TECHNICAL REPORT

on Stawell Gold Mines
Victoria, Australia

PURSUANT TO NATIONAL INSTRUMENT 43-101 OF THE
CANADIAN SECURITIES ADMINISTRATORS

28 March 2008

PREPARED FOR
NORTHGATE MINERALS CORPORATION

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3 Summary

This technical report (Technical Report on Stawell Gold Mine, Victoria, Australia) has been prepared to provide technical information underlying the 31 December 2007 estimates of Mineral Reserves and Mineral Resources for Stawell Gold Mines, Victoria, Australia. The beneficial owners of the project are Northgate Minerals Corporation who acquired the project in February 2008 as part of a takeover of ASX listed Perseverance Corporation.

Stawell Gold Mines is located in Western Victoria, Australia approximately 250km North West of Melbourne. The property is within the township of Stawell and consists of an active Mining Licence of 1003.58 ha, MIN5260 which for which title has recently been renewed and is current until 2020.

The Stawell operation has been in production since 1981 and is serviced by the rural township of Stawell with a population of approximately 6500. The operation consists of the main Magdala orebody which is accessed to a depth of approximately 1300m below surface by interconnecting 5.5m X 6.0m 1:7 - 1:9 declines. In addition to the Magdala Mine is a separate and currently inactive decline accessing the Wonga Mine and two inactive and partially backfilled open pit mines on the surface of the Magdala and Wonga deposits. The Stawell operation has a modern gold processing facility to processes all ore on site to produce dore bullion with appropriately permitted tailing facilities. There is sufficient power and water available to meet the requirements of the current life of mine plan.

The current mining tenement was granted in 1981 and the processing operation commenced in 1984. From 1984 to 31 December 2007 the project has produced some 1,794,786 ounces of gold. The initial discovery of the Stawell deposit was made in the 1850's and it has been estimated that total production from this period until 1926, when the mining activity ceased, was 2,700,000 ounces.

The Stawell Goldfield is located in the western Stawell Zone of the Lachlan Fold Belt. The Stawell Zone is a belt of predominantly deformed meta-sedimentary rocks representing the lower parts of the Cambro-Ordovician Lachlan Fold Belt.

There are three separate ore bodies defined at Stawell; the Magdala, Golden Gift and Wonga. All have differing characteristics but the same geological setting is apparent to all of them.

The stratigraphy at Stawell is divided into three principal units: Magdala Basalt; Albion Formation and Leviathan Formation. Intruded into this sequence are the Stawell Granite and a number of felsic and mafic intrusions. Adjacent to the Magdala Basalt is the Magdala Volcanogenics or Magdala Facies which is interpreted to be an alteration of a particular sequence of the Albion Formation. The Magdala Basalt, a large dome like feature is interpreted to form part of the Victorian Cambrian greenstone sequences, which are the oldest known rocks in the Palaeozoic Lachlan Orogen, and have inferred ages of 516-514 Ma.

There is significant structural evolution within the Stawell Goldfield with up to 7 separate deformation events recognised. Mineralisation is interpreted to be associated with the D_4 structural event which has subsequently been overprinted by brittle events which have displaced mineralisation along the South Fault. Magdala mineralisation is located above the South Fault and Golden Gift mineralisation is below the South Fault.

Mineralisation at Magdala is coincident with the largest and most significant mineralisation event in the western Lachlan Orogen at ca. 440 Ma contemporaneous in the Stawell and in the Bendigo-Ballarat zones with the mineralisation occurring during late D_4 at Stawell. The Wonga mineralisation occurred during late-stage magmatic events associated with the emplacement of the Stawell Granite ca 400 Ma. This mineralisation occurred in a series of brittle structures dependent on pre-existing weakness which are thought to be related to a fluid over-pressure event after the lockup of major regional structures.

The Magdala Facies are the primary and most important host rock for the sulphide replacement style of gold mineralisation at Stawell. Major mineralisation sulphides include arsenopyrite, pyrite and pyrrhotite, the latter two commonly occurring along cleavage planes and concentrated within shear zones.

Within the Magdala orebody the dominant ore types are Central Lode, Basalt Contact Lode and Stockwork Lodes. Central Lode, a quartz rich shear lode ranging from 0.5 to 10m in width dipping 55 - 65° to the west, was a significant production source in the Magdala Mine. Approximately 4km in strike with 1km of dip extent and continuously mineralised with economic shoots (> 4.0g/t) varying from 20 - 30m in strike up to 200 - 350m in strike. There is free gold in the quartz and also associated with pyrite, arsenopyrite and recrystallised pyrrhotite. The Basalt contact mineralisation typically 2 - 3 m wide is represented by arrays of quartz sulphide tension veins immediately adjacent to the volcanogenic Basalt contacts. Sulphides include pyrrhotite, arsenopyrite and pyrite and occur as alteration selvages on tension vein margins. Stockwork mineralisation consists of large quartz tension vein arrays with arsenopyrite and pyrrhotite dominant sulphide mineralisation. This mineralisation occurs in specific structural locations above Basalt Noses and shoots are limited in strike to 40 - 50m vertically 30 - 50m.

There is only one identifiable ore type in the Golden Gift termed the Golden Gift Stockworks which contains a spectrum of all Magdala styles. Typical widths range from 8 -12m up to 30m and the strike extents of shoots range between 150 and 400m. Quartz content is generally below 25%. Mineralisation includes abundant recrystallised pyrrhotite and coarse grained arsenopyrite, pyrite and visible gold with a quartz content generally less than 25%.

The Wonga deposit is hosted within the locally termed Wonga Schist (part of the Leviathan Formation) along two main fault systems. The two fault systems controlling the mineralisation are the Hangingwall structure which, strikes towards 350° and dips between 25° and 50° towards the east, and the Link structures which generally trend toward 240° and dip between 40° and 70° to the southeast. The mineralisation is represented by arsenopyrite disseminations to quartz veins within these structures. The main ore minerals present are anhedral fine grained pyrrhotite and arsenopyrite.

Mineral Resource estimation methodologies are consistent with industry practice with the majority of the current Mineral Resource estimated using 3D block modelling methodologies with gold grades estimated by ordinary kriging. Additionally components of the 31 December Mineral Resource estimate is based

upon manual 2D estimations. There is sufficient production history and reconciliation information to support these estimates.

The gold grade estimates are based on good quality assay datasets of diamond drillcore that has been spatially located, sampled and assayed using sound industry standard practices. Quality Control and Quality Assurance procedures are used to ensure the data returned from assay laboratories is of good quality and corrective actions have been implemented to maintain this standard. Data management systems are in place to ensure long term security of all geological information collected on site.

There is available and extensive coverage of diamond drilling reaching a drill spacing of 15m X 15m in areas that are subject to grade control drilling. Additionally face mapping information and "sludge sample" holes logged for geology are available to construct geological models for all Mineral Resource areas. The key control in on Mineral Resource estimation is accurate definition of the constraining geological models. Estimation of grade within the domains, whilst still very important, is of secondary importance to the first order geological domaining. The geological personnel have a sound understanding of the mineralized system and good practices in place to ensure quality models are produced.

These estimated Mineral Reserves and Mineral Resources at 31 December 2007 are based on accumulated diamond drilling and geological information collected over the life of the Stawell Project that was commenced in 1981 and are given in the following tables.

	Mineral Resources exclusive of Mineral Reserve						
	Indicated		Inferred		Total		
	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	In situ Au oz (000's)
Underground							
Magdala above 1250mRL	200	4.97	150	4.05	350	4.58	51
Golden Gif above 1360mRL	23	7.09	80	6.01	103	6.25	21
Wonga above 1000mRL			121	6.80	121	6.80	26
Sub-total U/G	223	5.19	351	5.44	574	5.35	99
Surface							
Magdala	2660	2.15			2660	2.15	184
Wonga	298	1.49	106	2.44	404	1.74	23
Sub-total Surface	2958	2.08	106	2.44	3064	2.10	207
TOTAL	3181	2.30	457	4.75	3638	2.61	306

Table 1 Stawell Gold Mines Mineral Resource summary table as at 31 December 2007

	Mineral Reserves						
	Proved		Probable		Total		
	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	In situ Au oz (000's)
Underground							
Magdala above 1250mRL	10	4.34	297	4.5	307	4.49	44
Golden Gif above 1360mRL	36	4.83	926	5.92	962	5.88	182
Wonga above 1000mRL							
Surface Stockpiles							
Sub-total U/G	46	4.72	1223	5.58	1269	5.54	226
Surface	0		0				
Davis Exte above 130mRL			325	2.12	325	2.12	22
LG Stockpiles			188	0.8	188	0.8	5
Sub-total Surface	0		513	1.64	513	1.64	27
TOTAL	46	4.72	1736	4.41	1782	4.42	253

Table 2 Stawell Gold Mines Mineral Reserve summary table as at 31 December 2007

Mineral Reserves				
Stawell	Proven	46,000	4.72	7,000
	Probable	1,736,000	4.41	246,000
	Total	1,782,000	4.42	253,000
Mineral Resources (exclusive of reserves)				
		Tonnes	Gold (g/t)	Gold (contained

				ounces)
Stawell	Measured	0	0.00	0
	Indicated	3,181,000	2.30	235,000
	Meas + Ind	3,181,000	2.30	235,000
	Inferred	457,000	4.75	70,000

Table 3 Condensed Mineral Resource and Mineral Reserves as at 31 December 2007

Notes:

1. *All Mineral Resources and Mineral Reserves have been estimated in accordance with the JORC Code and have been reconciled to CIM Standards as prescribed by National Instrument 43-101.*
2. *Mineral Resources for are exclusive of Mineral Reserves.*
3. *Mineral Reserves were estimated using the following economic parameters:*
    a. *Gold price of A$750/oz. Cut-off grade applied was variable for underground ore depending upon width, mining method and ground conditions. Dilution of 2-3m and mining recovery of 95-100% were applied to the underground reserves, dependent upon mining method.*
4. *Mineral Resources were estimated using the following parameters:*
    a. *Underground using Gold Price of A$750/oz.*
    b. *Magdala surface above 130mRL and above a nominal 0.8g/t cutoff*
    c. *Wonga surface within a A$850 optimised pit shell*
5. *Mineral Reserve estimates were prepared by:*
    a. *Glenn Miller, Mine Technical Superintendent, Northgate Minerals Corporation. Mr Miller is a member of the Australasian Institute of Mining and Metallurgy and has over 16 years of relevant engineering experience.*
    b. *Tamer Diner of Mining Solutions Consultancy Pty Ltd. Mr Diner is a member of the Australian Institute of Mining and Metallurgy and has 20 years of experience of relevant engineering experience.*
6. *Mineral Resource estimates were prepared by:*
    a. *Dean Fredericksen of Fredericksen Geological Solutions Pty Ltd. Mr Fredericksen is a member of the Australasian Institute of Mining and Metallurgy and has over 19 years of relevant geological experience.*
7. *Mineral Resources and Mineral Reserves are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summations may occur due to rounding.*

4 Introduction and Terms of Reference

This technical report has been prepared for Northgate Minerals Corporation the beneficial owner of Stawell Gold Mines. Northgate Minerals Corporation is listed on the Toronto Stock Exchange and acquired Stawell Gold Mines as a result of the takeover of Perseverance Corporation in February 2008.

This report is the first prepared for Stawell Gold Mines in accordance with the requirements of the National Instrument 43-101 standard and incorporates a detailed summary of the project, geological setting, and Mineral Resource and Mineral Reserve position as at the 30 December 2008.

The Mineral Resources and Mineral Reserve estimate for Stawell Gold Mines is a compilation of a number of individual estimates for various ore bodies or various geographically constrained areas. All of these estimates are contained within the Mining Lease MIN5260. Details of the locations and geographical constraints of the various ore body components as at December 2007 are given in Section 9.

The Stawell gold deposit was discovered in the mid 1850's during the Victorian gold rush which saw the discovery exploitation of the significant deposits at Bendigo and Ballarat. Mining activity eventually ceased in the 1920's and after a prolonged period of sporadic exploration mining operation recommenced in 1981. Mining operations and various levels of exploration and resource development activities have been continuous since 1981 and as such the project has significant past production and development history which is discussed in this report and also utilised during the compilation of the Mineral Resource and Mineral Reserve estimates.

This report has been prepared by a number of the site personnel with the assistance of Dean Fredericksen an independent geological consultant. The report utilises information available within company technical reports, published geological papers and internal Mineral Resource and Mineral Reserve documents completed by members of the SGM mine geological and mine engineering teams.

Dean Fredericksen of Fredericksen Geological Solutions Pty Ltd is a qualified person as defined by National Instrument 43-101 and accepts overall responsibility for the preparation of all sections of this report including the preparation of the Mineral Resources as reported in Section 19 other than the components of Section 19 which pertain to the 30 December 2007 estimate of Mineral Reserves and Section 25 which have been completed by Glenn Miller. Glenn Miller is a qualified person as defined by National Instrument 41-101 and is responsible for the preparation of and documentation of the estimation of Mineral Reserve and compilation life of Mine Plans that are outlined in Sections 19.

All information presented in this report was prepared in accordance with the requirements of National Instrument 43-101F1, Standards of Disclosure for Mineral Projects and is in the format prescribed by that instrument.

4.1Field Involvement of the Qualified Person

Dean Fredericksen has been involved in the Stawell Gold Mining operation since 1995. During the period 1995 - 2002 Dean Fredericksen was the Chief Geologist for Stawell Gold Mines and was responsible for all site based geological functions including near mine exploration, resource definition and mine geology functions and was the JORC competent person for reporting on Mineral Resources and Ore Reserves. From 2002 - 2004 Dean Fredericksen was the Group Mine Geologist for MPI Mines Ltd, owner of Stawell Gold Mines and maintained his involvement in the Stawell Project in an advisory and monitoring capacity. From June 2007 Dean Fredericksen has been consulting to Stawell Gold Mines and in this capacity has been providing mentoring to site geological personnel and audit and management advice to the operation. During this period regular monthly visits have been made to Stawell Gold Mines for either one or two weeks duration.

Glenn Miller is an employee of Stawell Gold Mines and has been in his current capacity as Mine Technical Superintendent since May 2006 and has been responsible for all mine planning functions including the estimation of Mineral Reserves since that time.

Tamer Dincer is Principal Consultant Mining Engineer from Mining Solutions Consultancy in Perth Western Australia. Tamer has completed the Davis Open Pit optimisation study and is responsible for Sections 19.8.3.2 and 24.2 of this report dealing with Davis open pit reserves.

4.2Definitions

Au	gold
AUD	Australian dollars
g/mt	grams per metric tonne
m	metre
km	kilometre
Ha	Hectare (10,000 sq m)
Ma	Million Years
mRL	Mine grid reduced level (metres)
ML	Mega litres
Northgate	Northgate Minerals Corporation
SGM	Stawell Gold Mines
Shotcrete	sprayed concrete slurry for support of underground mine openings
CRF	Cemented Rock Fill used as stope backfill
C7	Central Lode 7 ore block
U3	Upper South Fault ore block
GG1	Golden Gift 1 ore block
GG2	Golden Gift 2 ore block
GG3	Golden Gift 3 ore block
GG5	Golden Gift 5 ore block
GG5L	Golden Gift 5 Lower ore block
GG6	Golden Gift 6 ore block
GG South	Golden Gift South ore block

5 Reliance on Other Experts

The Authors of this report are Qualified Persons, and have relied on various datasets and reports that were provided by Stawell Gold Mines Ltd to support the compilation of the Mineral Resources and Mineral Reserves as documented in this report. It is the view of the authors that the data collection and data storage and data analysis methods utilised in estimating and compiling Mineral Resource and Mineral Reserve estimates at Stawell Gold Mines are of sufficient quality to ensure

the information is reliable and suitable for compilation of this Technical Report. The principal Author is not aware of any critical data that has been omitted so as to be detrimental to the objectives of this report. There was sufficient data provided to enable credible interpretations to be made in respect of the data. The principal author believes that no information that might influence the conclusion of the present report has been with-held from the study.

With respect to definition of tenement boundaries, legal status of the mining licence MIN5260, and statutory obligations the Principal author Dean Fredericksen has relied upon information supplied by Mr Jeff Dunwoodie (Environment and Community Coordinator, Stawell Gold Mines). The following documents have been reviewed during this process;

- Mining Licence 5260 -Variation dated 12 July 2000
- Stawell Gold Mines Environmental Management Plan, April 2007. Prepared by URS Australia Ltd
- Letter from Department of Minerals and Petroleum 22 May 2007 confirming acceptance of the current EMP and details of a site meeting on any outstanding issues with respect to the EMP.

In a review of this information with Mr Dunwoodie and subsequent discussions the principal Author has not been made aware of any material environmental or statutory issues that do or could impact on the ongoing operation of Stawell Gold mines and is satisfied that Stawell Gold Mines has in place a program of management to satisfy statutory requirements of the Mining Licence Min 5260.

Section 18 of this report summarises the results of past metallurgical performance for the operation and the results of ongoing metallurgical testing to determine future plant recoveries and the optimum treatment regime for the ore. This information has been prepared based on information supplied by Mr Les Dunn (Metallurgical Manager, Stawell Gold Mines). The Principal Author, Dean Fredericksen has verified sufficient information to be satisfied that there is a strong relationship between actual plant performance and predicted metallurgical performance from testwork to support the process utilised and the assumptions that have been utilised in preparation of this document.

6 Property Description and Location

6.1Property Location

Stawell Gold Mines is located in the State of Victoria, 250km north-west of Melbourne and 2km from the township of Stawell. Stawell is a rural township of approximately 6500 people and is within the Northern Grampians Shire.

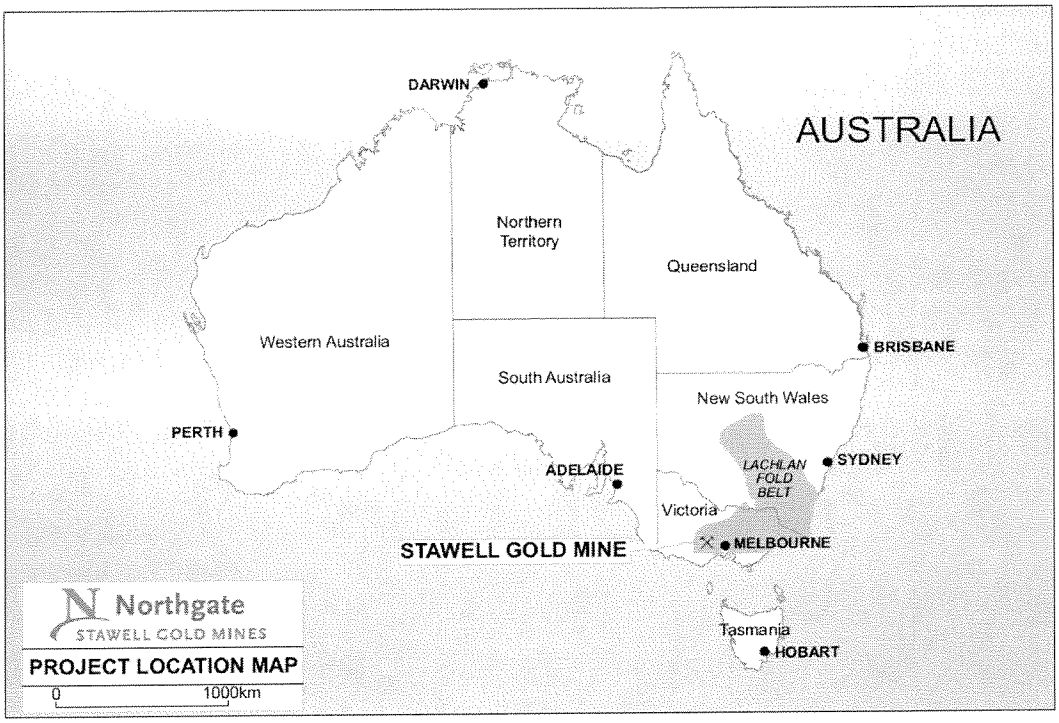


Figure 1 Map highlighting location of Stawell Gold Mine

6.2Property Description

SGM's principal approval is its Mining Lease (MIN5260) issued by the Victorian State Government under the *Mineral Resources (Sustainable Development) Act* 1990. This approval was first issued on the 31st May 1985 as ML1219 and has been amended on

at least six occasions since as a result of approved Work Plan variations. The current Mining Licence approval is active until 2020.

MIN5260 is located at centroid coordinates of 142.80° E and 37.06° S (GDA94)

The boundaries of the land covered by the mining licence are accurately surveyed and marked with posts and information plates in accordance with the Mineral Resources Development Regulations 2002.

Throughout its operational life, SGM has received at least sixteen local government, Planning Permits from the appropriate Council that relate directly or indirectly to environmental management.

MIN5260 (1003.58 ha) encompasses both the Magdala and Wonga mines and is located under and around the township of Stawell and covers 1000.58 Ha (Figure 2). All mineralised zones and mining related infrastructure is located within the boundaries of MIN5260 (Figure 5).

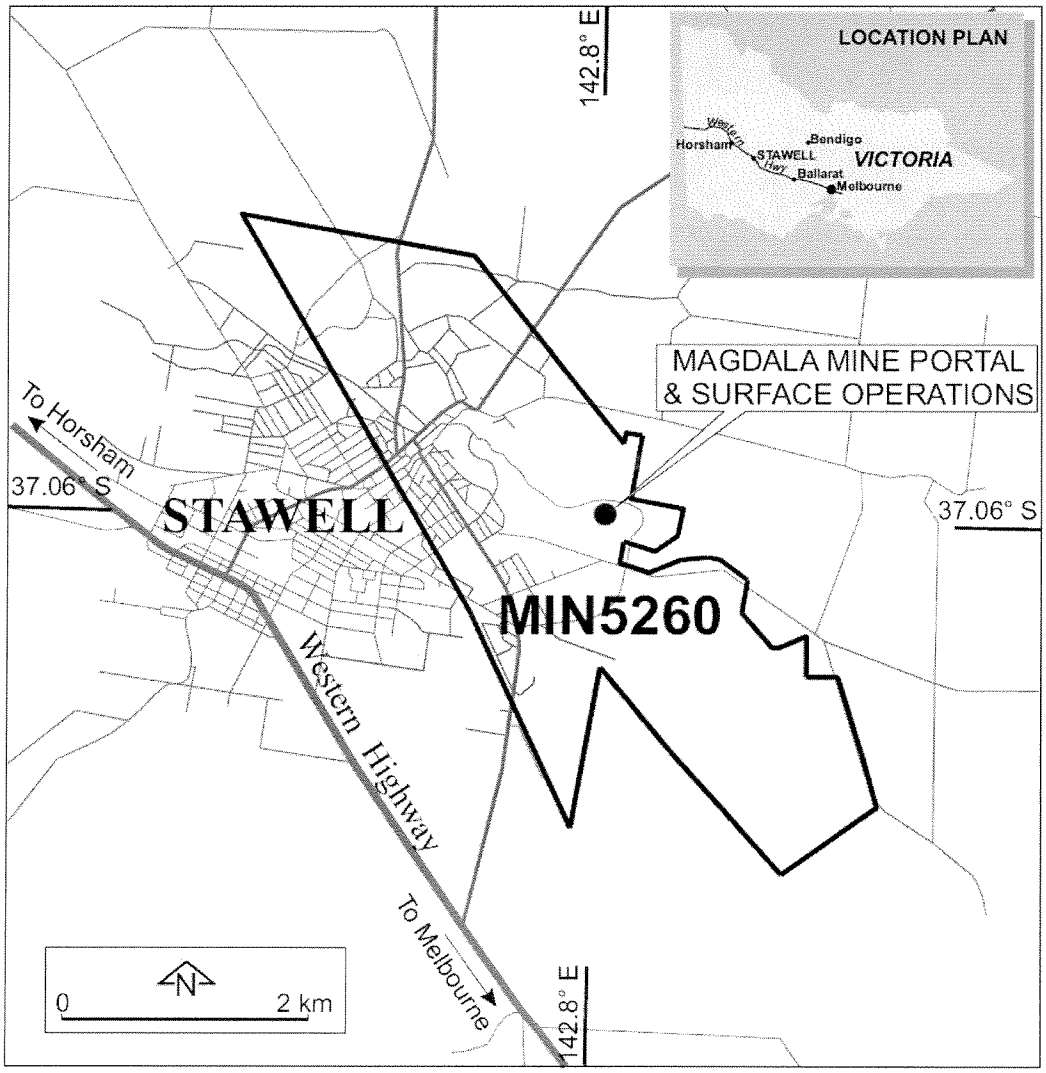


Figure 2 Location map of the MIN5260 lease. Grid is latitude and longitude as per GDA94

MIN5260 is comprised of private and crown land including designated crown land reserves (See Figure 3 for detail). Designated crown land reserves require particular consideration in relation to a number of documents including section 6 of the Mineral Resources Development Act 1990 and National Parks (Box-Ironbark and Other Parks) Act 2002.

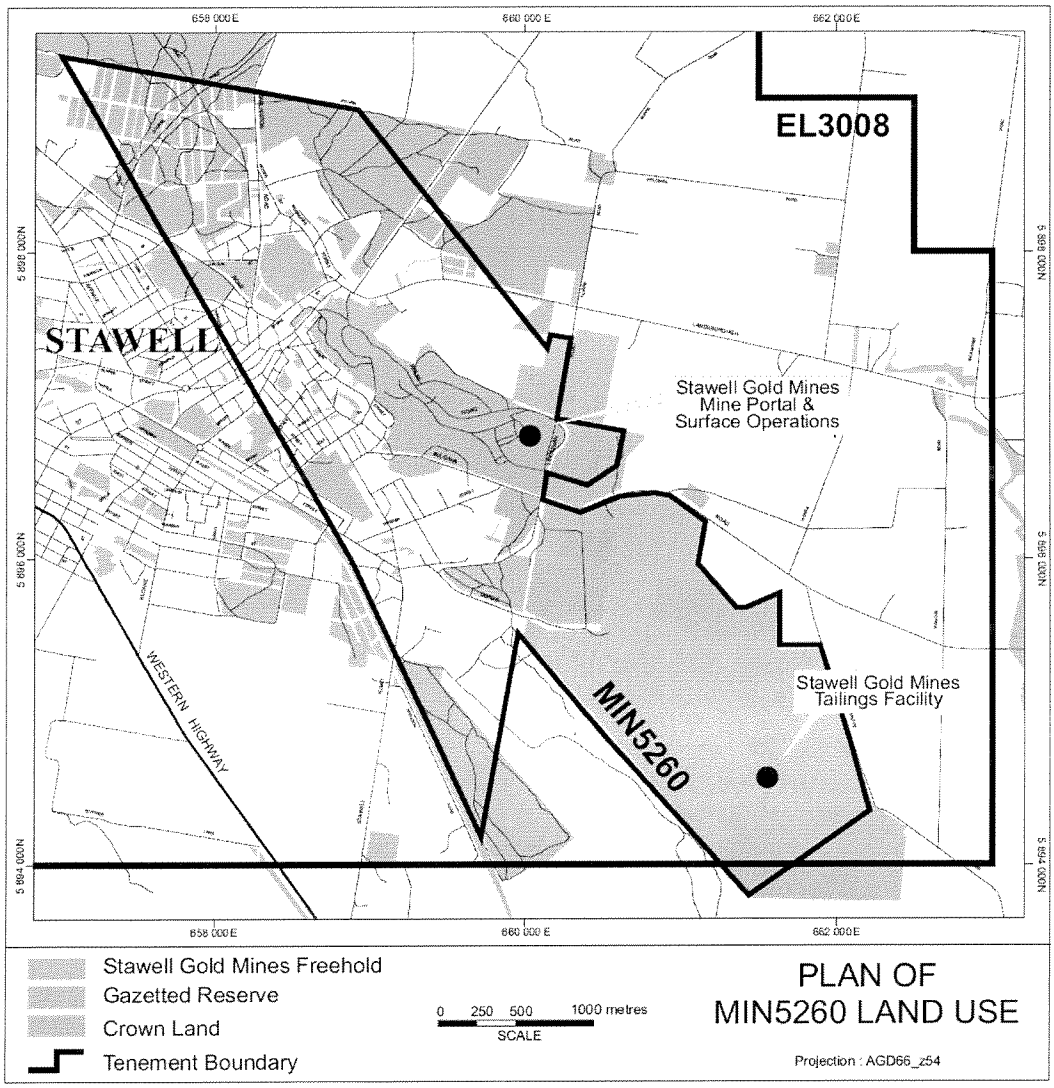


Figure 3 Map of Land use in relation to the MIN5260 lease.

6.3Legislation and Permits

SGM's principal approval, Mining license Min5260, is the applicable "right to mine" title over this land and is current to 2020. Attached to this title are a series of licence conditions that must be met and are the controlling conditions upon which an annual Work Plan and Work Plan variations are filed with the regulatory authority.

Apart from the primary mining legislation the Mineral Resources (Sustainable Development) Act 1990 (Vic) operations on MIN5260 are subject to the additional

following legislation and regulations for which all appropriate permits and approvals have been obtained:

Acts

- Extractive Industry Development Act 1995 (Vic)
- Environment Protection Act 1970
- Mines Act 1958
- Planning and Environment Act 1987
- Environmental Protection and Biodiversity Conservation Act 1999
- National Environment Protection Council (Victoria) Acts 1995
- Flora and Fauna Guarantee Act 1988
- Catchment and Land Protection Act 1994
- Archaeological and Aboriginal Relics Preservation Act 1972
- Heritage Act 1995
- Forest Act 1958
- Dangerous Goods Act 1985
- Mines Safety and Inspection Act 1994
- Drugs, Poisons and Controlled Substances Act 1981
- Health Act 1958
- Water Act 1989
- Crown Land (Reserves) Act 1978
- Radiation Act 2005
- Sustainability Victoria Act 2005
- Country Fire Authority Act 1958
- Conservation, Forests and Lands Act 1987
- Wildlife Act 1975

Regulations

- Dangerous good (Explosives) regulations 2000
- Dangerous good (Storage and Handling) regulations 2000
- Mines safety and Inspection regulations 1995
- Forest Fire regulations 1992
- Biodiversity Conservation Regulation 2000
- Drugs, Poisons and Controlled Substances 9Commonwealth Standard) Regulations 2001
- Mineral Resources (Infringements) Regulations 1991

- Environmental Protection (Vehicle Emissions) Regulations 2003

Stawell Gold Mines is operating under a Work Plan submitted as required under section 3 of the General Licence Conditions of Mining Licence (MIN5260). A key component of the Work Plan is an Environmental Management Plan the most recent of which was submitted to the Department of Primary Industries Victoria for approval in April 2007. A requirement of the General Licence Conditions of Mining Licence is to maintain the EMP.

Regular reviews of legislation and regulation requirements are completed and Stawell Gold Mines maintains all required statutory approvals to continue with mining operations.

An environmental bond for the project of AUD$4,547,738 is lodged with the Department of Primary Industries Victoria. Rehabilitation for the project is ongoing and SGM has entered into a cooperative research project with the University of Melbourne to conduct rehabilitation trials to prepare the rehabilitation program for eventual closure of the operations. Other than the rehabilitation bond the project is not subject to any other environmental liabilities.

6.4Royalties

Within M5260 there is an AUD$2.00 per Au ounce royalty payable to Mineral Ventures of Australia (MVA). This royalty agreement came into place in February of 2004 and is in place until the earlier of 15 year of production of 2.5 million ounces of gold. Furthermore this royalty agreement extends to Victorian tenements held by Leviathan Resources in February 2004 which included MIN5260.

6.5SGM Local Survey Grid Reference

All survey data on MIN5260 is collected and stored using modified AMG co-ordinates, based on Australian Map Grid AGD 66 (zone 54). The convention is to drop the first digit of the northing, so 5896000N becomes 896000N. Easting is unchanged.

The mine RL is calculated as AHD-300m (where AHD is Australian Height Datum) and displayed as a negative number below surface. The RL origin is at 303.60 AHD, measured at a trigonometric station located adjacent to the mine on Big Hill, Stawell.

The principal local grid in use within the Mine Lease is the Stawell Gold Mine Grid (also referred to as the "45 degree grid") as shown in Figure 4. This grid is orientated 45 degrees west of AMG north and has its origin at 5890137.479N and 659498.820E. It is convention to divide the northing by 20 and refer to this as the section northing line, i.e. northing 6200 becomes 310 section line. Additional local grids are used as required for presentation of geological information as required.

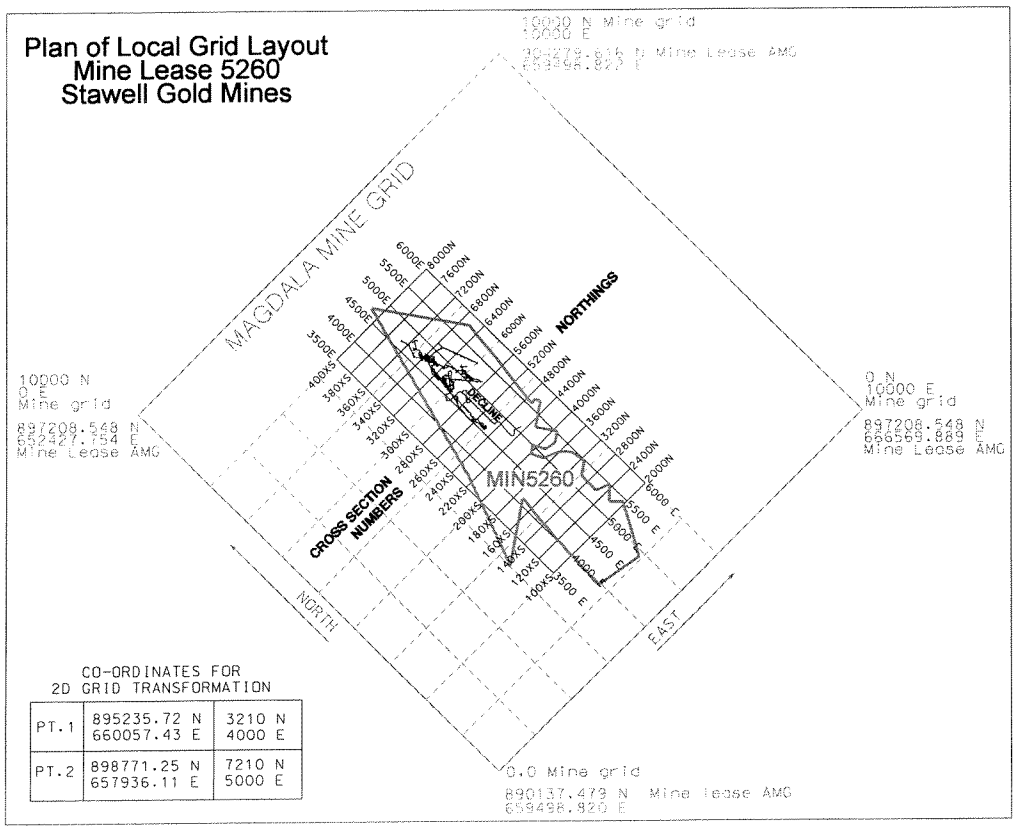


Figure 4 Local Mine Grid used at SGM

7 Accessibility, Climate, Local Resources, Infrastructure and Physiography

7.1Accessibility

Stawell Gold Mines is easily accessible from Melbourne via the Western Highway. Access closer to the mine site is provided through a network of sealed bitumen government roads. Roads within the mine site are unsealed and regularly maintained.

The main Melbourne to Adelaide rail line passes through Stawell and Stawell is also serviced by a local sealed airfield.

7.2Climate

Stawell is located within the southern part of the Wimmera where the climate is described as semi-arid, allowing for exploration and mining activities all year round. Since 1996, Stawell has recorded an annual daily average temperature of 20.5˚C. An extreme maximum of 43.6˚C was recorded on the 25th of January, 2003, and an extreme minimum of -3.7˚C on the 4th of August 1997. Mean annual rainfall is 480.2 millimetres with 70 days per year on average recorded as having rain.

7.3Local Resources

Stawell Gold Mines has been in operation for over 20 years, developing a highly experienced workforce. Many contractors, also having a long association with the mine, are available in the township of Stawell and surrounding regions.

Due to the mines close location to Stawell many facilities are available. Within the township is a police station, hospital, schools and shops. Mains Electricity and water is also accessible.

7.4Infrastructure

7.4.1 Surface Infrastructure

Stawell Gold Mines facilities are extensive and representative of a modern Gold Mining operation. The main site location comprises;

- Office and administration complex
- Store and storage facilities
- Heavy underground equipment workshop and Light vehicle workshop
- Surface Run of Mine stockpiles
- Gold Processing Plant and associated facilities
- On Site Assay and Metallurgical test work Laboratory
- Four freshwater storage dams to store rainfall run-off and mine dewatering which is used in the plant or around the mine site.
- Power for the plant is fed from a main transformer located adjacent to the administration complex.
- A batch plant for preparing shotcrete for underground support
- Core farm and core processing facility

Surface facilities include the gold processing plant, offices, core shed, laboratory and workshops. Larger infrastructure onsite includes tailings dams covering 96 ha and receiving all tailings from the processing plant. Three freshwater dams occur throughout the mine lease.

7.4.1.1 Tailings Storage Facilities

Since operations began in 1984, three tailings dams have been constructed and operated, two of which have since been decommissioned;

- Reserve Tailings Dam - decommissioned and rehabilitated to a Clay Target Shooting Complex.
- No 1 Tailings Dam - decommissioned and partially rehabilitated
- No 2 Tailings Dam - operational

All dams were constructed as earthen embankments with upstream sub-aerial deposition and are subject to annual integrity and operational review by an independent industry expert.

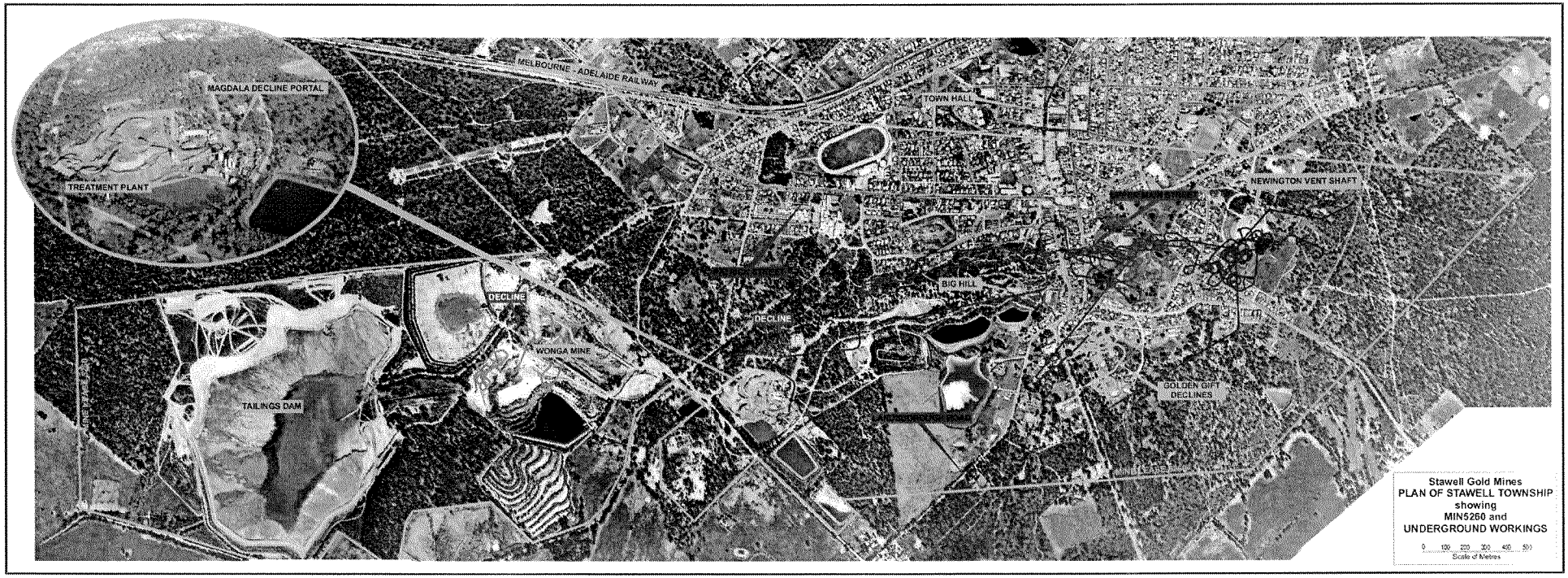


Figure 5 Plan showing the location of MIN 5260, Stawell Gold Mines operational infrastructure.

7.4.2 Underground Mining Infrastructure

Underground infrastructure within the Magdala mine is extensive and includes a 5.5m X 5.5m access decline extending from surface to around 1300mRL as at the end of December 2007. The gradient of the decline is 1:8 down to 468mRL then changes to 1:7 to 1300mRL. Two shafts, the Big Hill shaft (active intake) and Federal shaft (passive intake) supply fresh air to the mine. The exhaust system includes the Darlington shaft, which is the main exhaust. Albion shaft, Federal/Albion raisebore and Ulster shaft work as 3 surface connections in parallel making up the output of the south vent system.

Mine dewatering systems enable pumping to the surface although a significant portion is recycled and utilised within the mine.

An underground workshop and underground crib room is available at the 800m RL level within the mine.

In addition to the fixed plant Stawell Gold mines owns, operates and maintains all mobile mining equipment including Jumbo development drills, production drills, loaders, trucks, and ancillary equipment required to undertake mining operations.

7.4.3 Power

Stawell Gold Mines purchases power under contract from Origin Energy Australia. This 3 year contract expires in 2011.

Supply from the National Grid to SGM high voltage installations is in two locations, Moonlight Substation (10 Mega Watts feed) which supplies the Magdala Underground operation, and Reefs Road (7 Mega Watts feed) that supplies the Gold Processing Plant, administration, workshop facilities and parts of the upper levels of the Magdala Underground Mine.

Power to underground from the Moonlight substation is supplied through a 990m steel cased borehole and that from the Reefs Road substation via the Magdala decline.

Stawell Gold Mines management have provided information that the current power availability is sufficient to meet the needs of the current Life of Mine operating plan.

7.4.4 Water

Stawell gold mines water supply is from harvested rainfall runoff, dewatering, recycling of process water from the tailings facility and by way of a 60ML/month water right entitlement from Lake Bellfield located in the Grampians Mountains. The capacity of the site water storages is approximately 690 ML.

The Lake Bellfield water is potable and is preferentially used in the processing operations as it improves gold recovery.

7.5Physiography

The area surrounding Stawell is made up of flat to gently undulating farmland with the Grampians Mountain range and National Park 20km to the southwest. Close to the centre of Stawell is Big Hill, the town's highest point at a height of 303.6m above mean sea level. Stawell Gold mine is situated on the southern slope of Big Hill. Parts of the area adjacent to the mine are covered by Iron bark forest.

8 History

Stawell is a historic goldfield having produced 2.7Moz of gold between 1853 and 1926 from both alluvial and hard rock sources. There was little mining activity in the Stawell area from 1926 to 11th March 1976 when Western Mining Company (WMC) Resources Ltd was granted an exploration licence over the Stawell Goldfield. In 1981 Stawell Gold Mine was reopened by the WMC/Central Norseman Gold joint venture with commencement of the Magdala decline. By 1984 the operation had expanded with the construction of a processing facility and subsequent commencement of an open cut operation at the Wonga mine (2 km south of Magdala). A number of historical tailing dumps were retreated during this period. Towards the end of mining of the Wonga open cut (1987) the Davis open cut operation was commenced. The Davis open cut exploited the oxide material on up dip projection of the Magdala deposit.

The Wonga Open Cut operated from 1984 to 1987 and produced 778,847 tonnes recovering 69,159 Oz's of gold. The Davis Open Cut operated from 1987 to 1989 and produced 154,525 tonnes for 8,992 recovered ounces of gold.

Initial underground mining methods at Magdala were predominantly based on air-leg operations. There was very little diamond drilling and the basic mining method was to crosscut from the decline access through the ore body and strike drive until the ore ran out. At this point air-leg miners, or at a later stage production drills, would be brought in to work out the ore. In 1985 a decline commenced from the bottom of the Wonga open cut. Longhole Stoping methods were initially introduced in Wonga underground mine prior to a successful transition into the Magdala Mine. During this time not much capital was committed to the project but the geological understanding was generally good. A total of 336,000 ounces of gold were produce during WMC ownership.

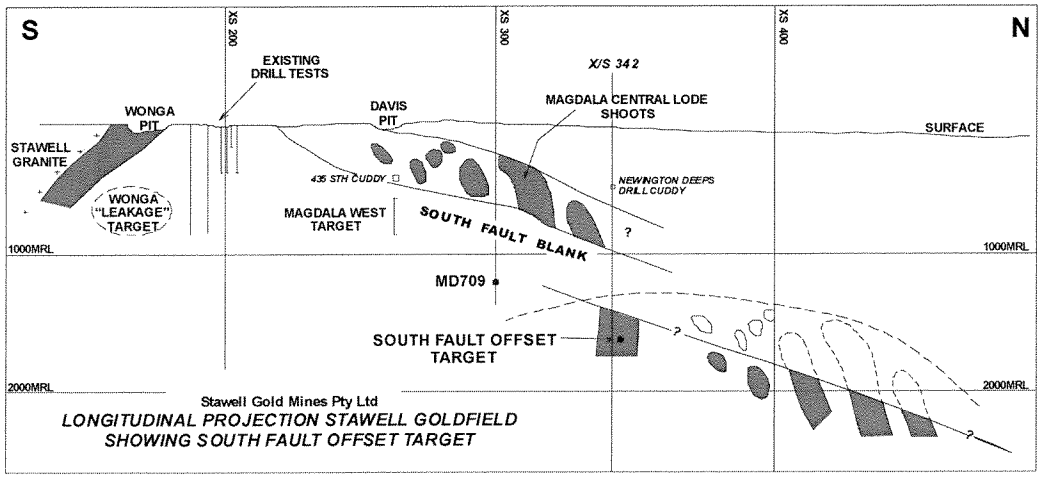


Figure 6 SGM Mine Long Projection in 1993

In December 1992 the operation was acquired in a 50/50 joint venture by Mining Project Investors (MPI) Pty Ltd and Pittston Mineral Ventures. At this stage the Magdala decline was approximately at 410mRL, while the Wonga decline was at 180-200mRL. With the acquisition there was a clear direction to increase expenditure on resource definition drilling and near mine exploration. This also included the development of the first hangingwall drill platform. This initiative

was a targeted program to increase the Mineral Resource and Probable Reserve position to enable more confident capital decisions about the operations future. During this time campaign treatment of Basalt Contact, Central Lode and Wonga ore types began as it was recognised that the different ore type had varying recovery rates for geological and metallurgical reasons.

From 1996 to 1998 there was improved ore body knowledge due to the additional diamond drilling. A third style of mineralisation was recognized in the Magdala ore body, Stockworks Style, which highlighted the importance of interpreting the geology and identifying basalt contacts and noses prior to mining. This initiated the development of integrated mining and mine geology functions to improve production outcomes. These initiatives included improved sampling methods, use of modernized laboratory and sample prep, and the initiation of grade control diamond drill programs to collect geological and geotechnical data with a reduced reliance on up-hole percussion sludge samples. The overriding objective of integrating mining and mine geology functions was to enable completion detailed mine design before initiating ore development. The other effect of the integration would be more accurate production forecasting.

During 1998 to 1999 many important initiatives where undertaken. An Australian Research Council (ARC) funded research project aiming to determine the timing and structural architecture of the faults at Stawell was initiated. At that stage the South Fault was the bottom of the known mineralisation and its movement direction and offset were not understood or constrained. Development on the South Fault provided the first real evidence of the movement vectors on the South Fault. Members of the ARC research project were able to model and indicate the transport vectors and projected offset across the structure.

Also during this time surface drilling into the North Magdala area with several 1200m directional drill holes commenced aiming to locate important future resource areas down dip of the existing mining area in the Magdala ore body. A second hangingwall drill platform was also developed at the 786mRL to enable continuation of the resource definition programs at depth. At this time a geostatistical study using Conditional Simulation methods was undertaken to

understand drill spacing and reliability. The outcomes from this study developed a Resource/Reserve classification criteria guide from SGM.

Drilling Category	Resource Classification	Resource Confidence	Reserves Confidence
	Measured Resource Level Development and / or Production Drilling	± 10%	Proven Reserve
Grade Control	**Indicated Resource** 15 x 15 m drill out	± 10%	Proven/Probable Reserve
Resource Definition	**Indicated Resource** 40 x 30 m drill out	± 20%	Probable Reserve
	Inferred Resource Geological continuity and 150 x 60 m drill out	± 35%	

Table 4 Resource/Reserve classification criteria guide developed for SGM

From 1999 to 2000 the first drill program, based on the ARC project findings, targeting mineralisation below the South Fault (Golden Gift) was successful. The third hole, MD2167 intersected three repeated zones of mineralised volcanogenics with visible gold on basalt contacts, **16.2m @ 5.15g/t Au, 19.0m @ 7.21g/t Au** and **2.85m @ 7.12g Au /t.** This intersection demonstrated that the mineralisation below the South Fault was very similar to the Magdala mineralisation above the South Fault, and fitted the conceptual model for a faulted repetition of the entire system.

Initial Golden Gift exploration involved "scoping" drilling on 320 metre centres from existing development underground. An exploration decline, East Decline, was commenced as a designated drill platform for resource definition drilling of the Golden Gift.

The North Magdala area was an important production source for continuity of ore feed to the processing facility. Stoping sequences were altered with the introduction of cemented rock fill (CRF) to progress stoping sequences to multi level primary and secondary retreat sequences.

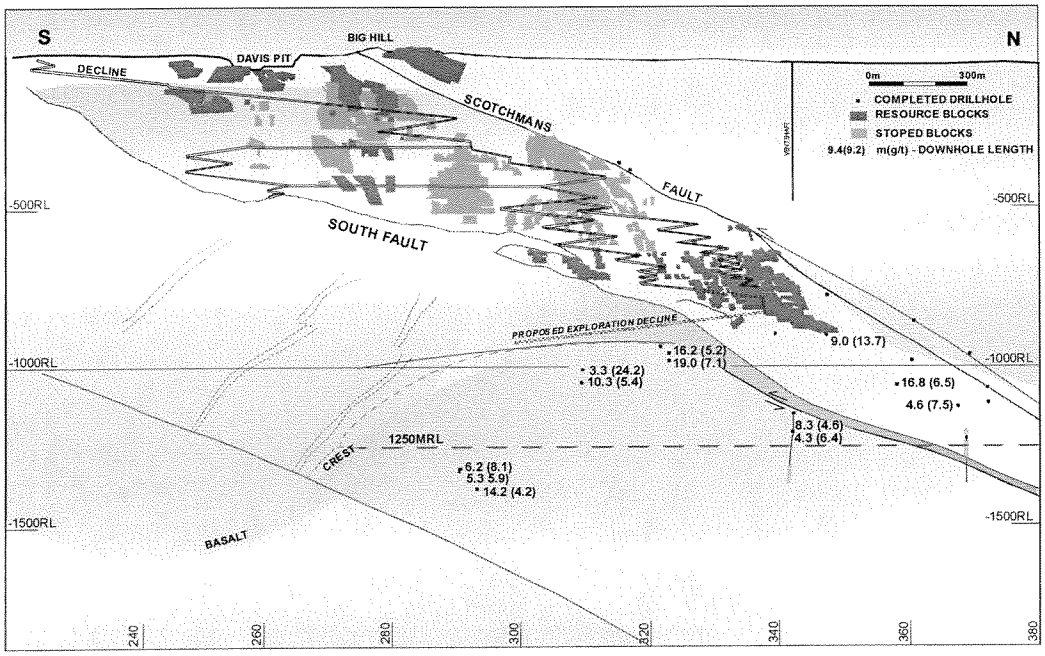


Figure 7 SGM Mine Long Projection June 2000

During 2001 and 2002 a geostatistical study of the coarse spaced (320m x 120m) drilling data obtained for the Golden Gift area were undertaken to confirm the potential size and more importantly the potential production grade. This study indicated that the grade of the Golden Gift would be approximately 1.0g/t Au higher than the Magdala areas. The development of the east decline drill platform finished and the detailed resource definition programs commenced into the Golden Gift. These programs highlighted that the geology was faulted to a significantly higher degree than originally anticipated creating isolated ore blocks.

In 2002 the decline into the Golden Gift 2 (GG2) area commenced with access established into the GG2 area and development of the first Golden Gift ore drives commenced in 2003.

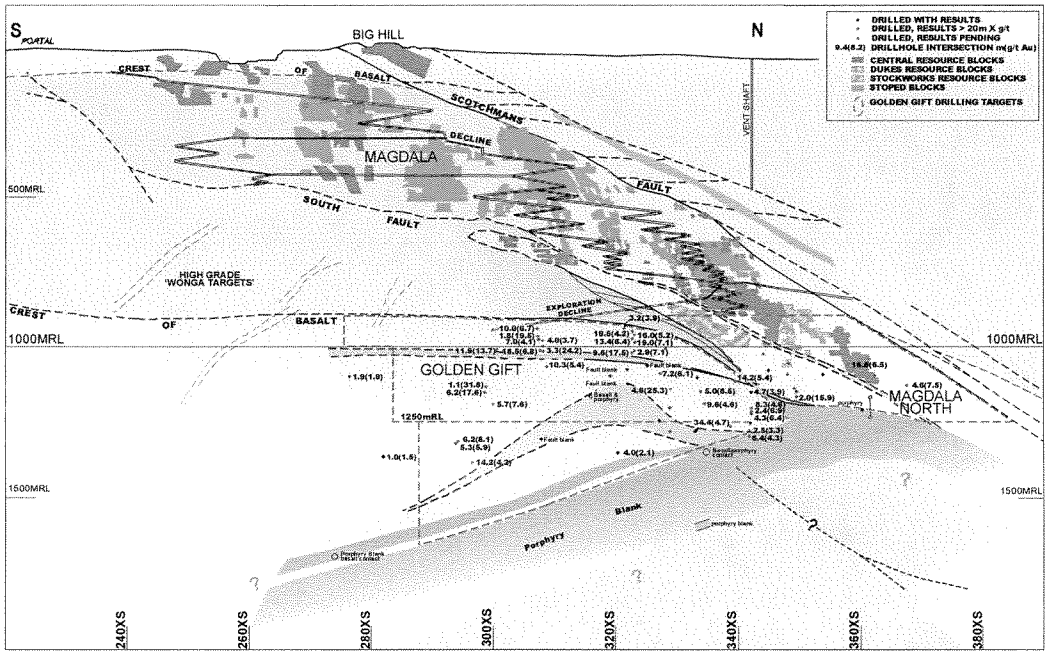


Figure 8 SGM Mine Long Projection December 2001

The resource definition and grade control drill program continued into different areas of Golden Gift confirming the original estimates on size and grade of the Golden Gift made using the coarse spaced "scoping" drilling

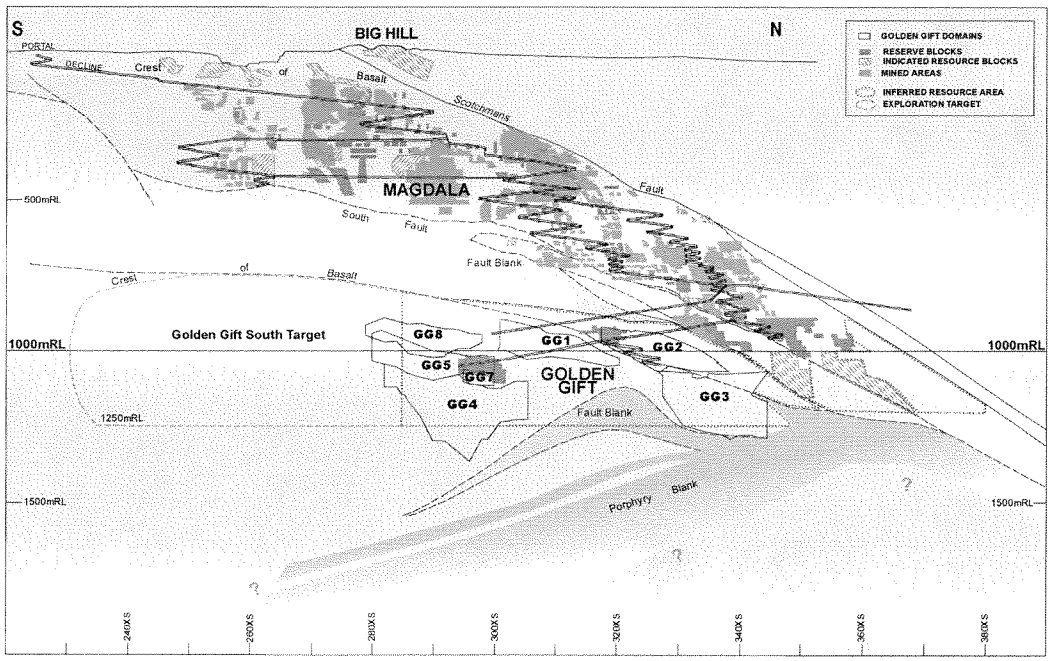


Figure 9 SGM Mine Long Projection December 2003

In February 2004 MPI acquired Pittston's 50% share of the project. Exploration continued in the Golden Gift during 2004 with the commencement of the Golden Gift South surface exploration program. This program drilled on three 400m spaced sections to a depth of 1200m but to limited success and in 2005 the program was abandoned. During this time a targeted exploration program was undertaken to target North Magdala on one section. This program had limited success with one intercept of **9.4m @ 8.3 g/t Au**.

In November 2004 a de-merger of the MPI gold business came into effect, and Leviathan Resources Ltd was floated in December 2004.

The resource drilling into the Golden Gift initially identified seven area of mineralisation offset from each other due to late faulting. Conversion of these areas of mineralisation into ore blocks wasn't universal but was successful in a majority of cases. The further drilling of the fault blocks also identified other mineralised surfaces previously unknown due to the faulted nature of the Golden Gift. From the increased geological understanding of the Golden Gift deposit, it was clear in the mine planning process that two declines were required, the GG5 and GG3 declines, to access the ore zones for continuity of supply.

In 2006 a decision to continue the GG5 decline down below 1200mRL to access the GG5 Lower ore block was made. By having two declines accessing several ore blocks in the Golden Gift the operation was able to maintain production from several fronts at any one stage.

In January 2007 Perseverance Corporation Ltd completed an off-market takeover of Leviathan Resources Ltd.

In 2007 exploration in the Golden Gift continued with the GG6 exploration program. This program was looking for the offset ore block under GG5 Lower. The understanding of the structural architecture built from previous drilling and mining in the Golden Gift environment identified the target area. Mineralisation was located with the second hole of the program intersecting ore grade mineralisation

on the basalt contact at approximately 1550mRL. Exploration and definition of the GG6 area is still in progress.

Perseverance Corporation Ltd was acquired by Northgate Minerals Corporation on the 18th February 2008.

8.1Historical and Modern Production

Stawell is a historic goldfield having produced 2.7Moz of gold between 1853 and 1926 from both alluvial and hard rock sources. Since the commencement of mining in the modern period, 1984, until December 2007 1,794,786 Ounces have been produce from the Stawell ore body. Summaries of annual gold production are shown in Figure 10. Ore treated and treated head grade are shown in Figure 11.

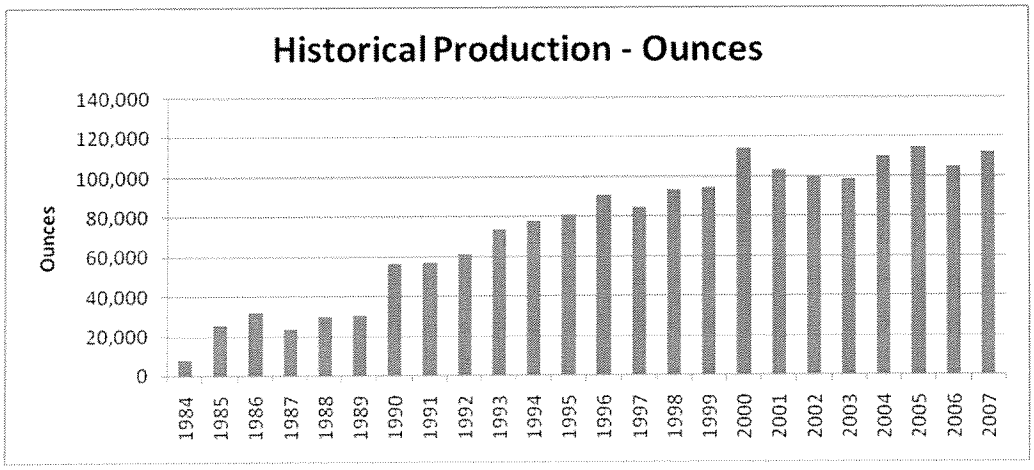


Figure 10 Annual Gold production in ounces since 1984

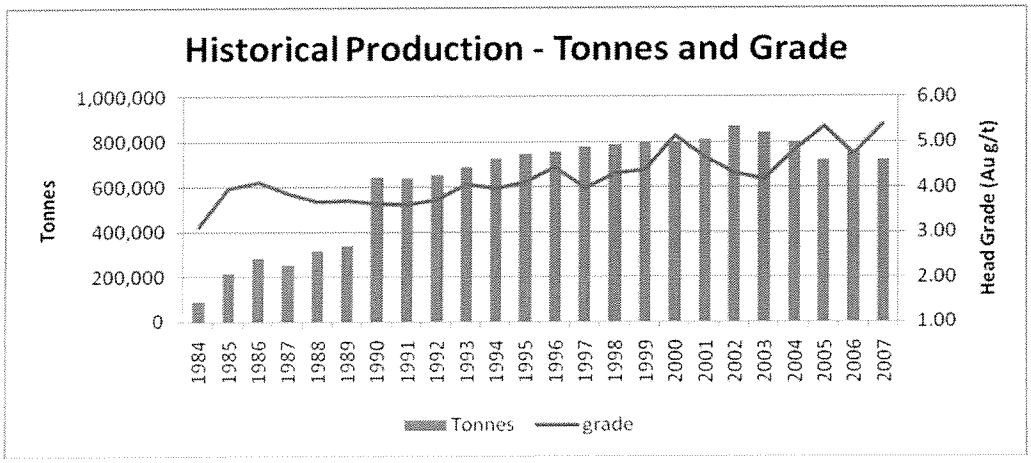


Figure 11 Annual Processing plant tonne throughput and head grade since 1984

9 Geological Setting

9.1Regional Geology

The Stawell Goldfield is located in the western Stawell Zone of the Lachlan Fold Belt. The Stawell Zone is a belt of predominantly deformed meta-sedimentary rocks representing the lower parts of the Cambro-Ordovician Lachlan Fold Belt stratigraphy bound to the west by the Moyston Fault and to the east by the Coongee Break (Vandenberg *et al.* 2000).

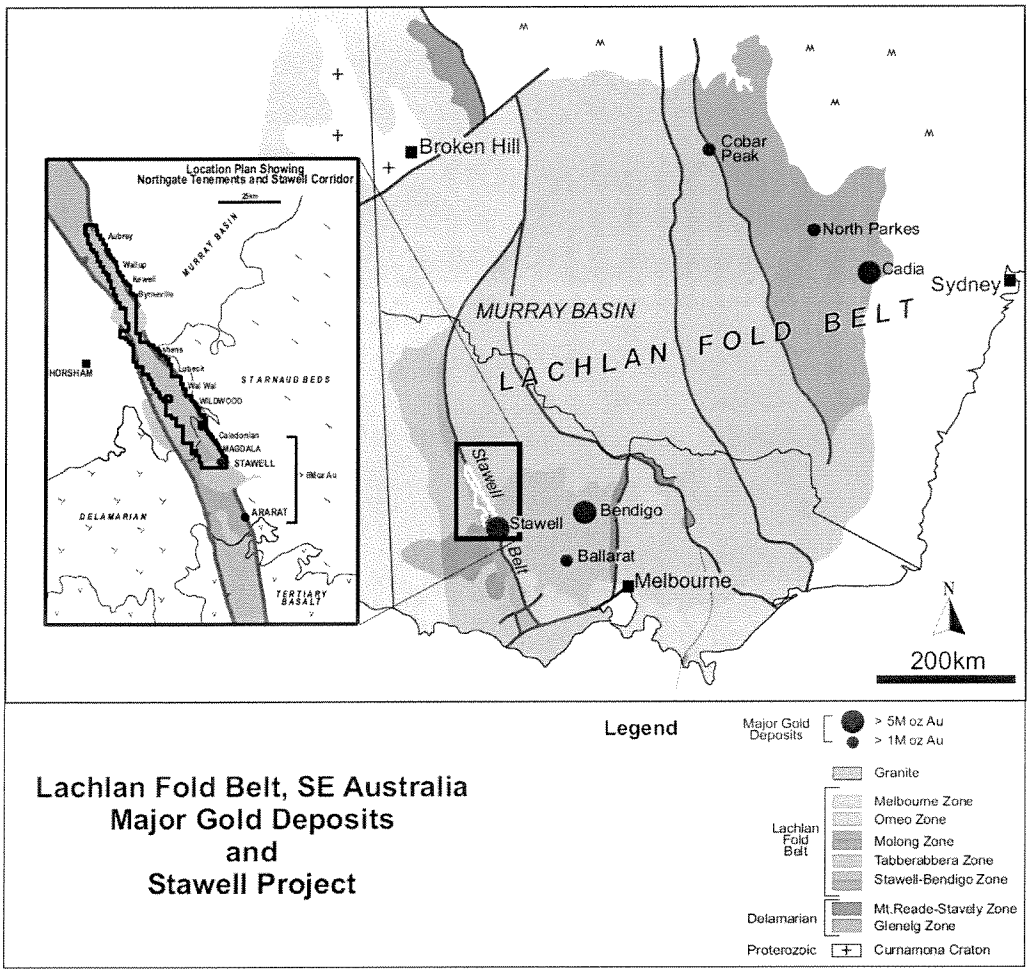


Figure 12 Image showing Lachlan Fold Belt, locating Stawell on the western boundary.

Interpretations from the Victorian Geological Survey present a thin skinned tectonics model where the Moyston Fault is an east dipping basal detachment which has juxtaposed higher metamorphic grade rocks of the Stawell Zone against lower grade Cambrian rocks of the Delamarian Glenelg Zone. The west dipping Stawell Fault, Coongee Break and other parallel west dipping faults represent back thrusts from the Moyston Fault. These back thrusts have progressively emplaced deeper stratigraphy against shallower stratigraphy with a generally west over east sense. An apparent anomaly in this sequence is the presence of deeper magnetic stratigraphy in the Stawell-Wildwood corridor. Vandenberg *et al.* 2002 interprets that the Pleasant Creek Fault, to the west of the Stawell Fault, actually dips east and has an east over west sense - similar to the Moyston Fault. The Stawell-Wildwood corridor therefore represents a significant structural high in an up-thrown block of deeper stratigraphy between the Coongee Break and Pleasant Creek Fault.

9.2Local Geology

There are three separate ore bodies defined at Stawell; the Magdala, Golden Gift and Wonga. All have differing characteristics but the same local geology is relevant to the genesis of them all.

9.2.1 Stratigraphy at Stawell

The stratigraphy at Stawell is divided into three principal units: Magdala Basalt; Albion Formation; Leviathan Formation, see Figure 15 (Squire and Wilson, 2005). Intruded into this sequence are the Stawell Granite and a number of felsic and mafic intrusions. Squire and Wilson (2005) interpret that the rock unit previously termed the Magdala Volcanogenics (Watchorn and Wilson, 1989) is an alteration facies that locally occurs adjacent to the basalt.

9.2.1.1 Magdala Basalt

The Cambrian Magdala Basalt is composed of subaqueous low-K tholeiitic lavas that exhibit an aphyric to sparsely plagioclase-phyric texture (Watchorn and Wilson, 1989; Squire and Wilson, in review). The Basalt body comprises flows ranging from 0.5 to 50 metres thick, pillows basalts with pillows ranging in size from 0.1 to 2

metres in size (Watchorn and Wilson, 1989) and monomictic basalt breccias varying proportions (Pritchard, 2001; Squire and Wilson, 2005).

The basalts are interpreted to form part of the Victorian Cambrian greenstone sequences, which are the oldest known rocks in the Palaeozoic Lachlan Orogen, and have inferred ages of 516-514 Ma (Squire and Wilson, 2005). The basalts near Stawell occur as dome-like units in the footwall and hangingwall of major faults (Miller and Wilson, 2002). The Magdala Basalt which closely resemble typical back-arc basin basalts (Kaufman, 2003; Crawford, 1988) has been interpreted to represent the medial to distal facies on the flank of a large basalt edifice upward of 500 m thick (Squire and Wilson, 2005) and has similar magmatic affinities to the known basalt bodies north of Stawell, Wildwood and Kewell Basalts (Kaufman, 2003; Jupp, 2003).

9.2.1.2 Leviathan and Albion Formations

Overlying the Magdala Basalt is a 200-300-metre-thick sequence of unfossiliferous turbidites (Squire and Wilson, 2005). The turbidite sequence has been subdivided into two different lithologies: the Albion Formation; and the Leviathan Formation. The differences between the two lithologies was first recognised but not explored by Gane (1998). He recognised the sediments on the western side of the Magdala Basalt graded from predominantly mud-rich to more sand-rich away from the basalt.

The Albion Formation is the lowest clastic sequence to the west of the Magdala Basalt. The unit varies in thickness with the top of the unit defined by a 20-100 metre sequence of black mudstone. Within the Albion Formation there are a number of facies which along with black mudstone include calcareous sandstone, siliceous siltstone and sulphidic black mudstone. Squire and Wilson (2005) suggested that the sediments were deposited predominantly due to suspension settling in a sediment-starved sedimentary basin. There were short-lived periods of oxygen-rich conditions shortly after volcanism recognised by the presence of siliceous siltstone but the dominance of black mudstone within the Albion formation indicates the basin of deposition was predominantly anoxic (Squire and Wilson, 2005). The provenance for the Albion Formation sediments has been

identified from detrital compositions to be a low-grade metamorphic terrane (Cas, 1983).

The Leviathan Formation overlies the Albion Formation and is dominated by fine-to medium-grained quartz-rich sandstones (Squire and Wilson, 2005). The contact between the two formations is gradational and conformable. Although the Leviathan Formation was deposited in a higher-energy environment than the underlying Albion Formation, the detrital compositions indicate little change in the provenance between the two formations (Squire and Wilson, 2005).

The Leviathan and Albion Formations are not segregated by the mine or exploration geologists at SGM and are referred to by the local name of 'mineschist'.

9.2.1.3 Magdala Facies

The Magdala Facies, termed Magdala Volcanogenics at SGM, distinguished by its dark green colour, is a result of intense chloritic alteration of mudstone and or shales located at the base of the Albion formation and immediately above the Magdala basalt. The Magdala Facies are the primary and most important host rock for the sulphide replacement style of gold mineralisation at Stawell. Major mineralisation sulphides include arsenopyrite, pyrite and pyrrhotite, the latter two commonly occurring along cleavage planes and concentrated within shear zones (Robinson, 2005).

9.2.1.4 Felsic Intrusions

Quartz±feldspar-phyric felsic intrusions cross-cut the turbidite sequence. The quartz±feldspar-phyric felsic intrusions vary in thickness from 50 cm to 12 m wide and showed chilled margins. They are predominantly composed of quartz and plagioclase with phenocrysts up to 3 mm in size. The feldspar phenocrysts have euhedral shapes and display multiple twinning while the quartz phenocrysts had a cloud-like appearance and were rimmed with fibrous quartz (Gedge, 1997). The groundmass is composed of ~80% quartz in anhedral grains and display undulose extinction (Gedge, 1997).

The felsic intrusions tend to follow northwest-trending shear zones (Wilson *et al.*, 1992) and the emplacement of the quartz±feldspar-phyric felsic intrusions post-dates the main Magdala mineralisation event. The intrusions have been dated at 413±3 Ma (Arne *et al.*, 1998).

9.2.1.5 Stawell Granite

The Stawell Granite was emplaced during the early Devonian, 401±4 Ma (Arne *et al.*, 1998), and is located about 2 km south of the Magdala deposit (Xu *et al.*, 1994) and adjacent to the Wonga deposit. The pluton is approximately 20 km wide and 13km long and intrudes the sandstone and shale units of the turbidite sequence. The pluton is an asymmetrically zoned, medium grained intrusion with contains diorites, granodiorites and magnetite-rich felsic granites (Wilson *et al.*, 1992). There is a 0.5 to 1 km contact aureole surrounding the Stawell Granite (Xu *et al.*, 1994).

9.2.1.6 Lamprophyric Intrusions

Lamprophyres intrude all the above lithologies and are hosted in D_4 shears (Gedge, 1997). The lamprophyre intrusions can range in thickness from 1 cm to 3 metres (Wilson *et al.*, 1992). The dykes vary in colour from dark grey to chocolate brown and display conchoidal fracture patterns (Gedge, 1997). The compositions of the intrusions vary from monchiquite (olivine bearing) to fourchite (augite and no olivine) (Wilson *et al.*, 1992). The lamprophyre dykes typical mineralogy is composed of Na-rich plagioclase (albite), clynopyroxenes, biotite, sulphides, ilmenite and Ti-rich magnetite (Gedge, 1997)

9.2.2 Structural history at Stawell

At least seven deformation events have been recognised at Stawell (Wilson *et al.*, 1992). These deformation events can be broadly split into two categories; early, ductile deformation (D_1 to D_4), and late brittle deformation (D_4 and later). A description of both categories of deformation and related structures are given below.

9.2.2.1 Early, ductile deformation

The ductile deformation events all occurred under a northeast-southwest shortening direction (see Figure 13) (Miller and Wilson, 2002). The earliest ductile

event recognised, D_1, is thought to be thrust-related with early shearing along detachment surfaces that produced a fabric, S_1, parallel to bedding (Wilson *et al.*, 1992). This event has been suggested to occur at about 510-504 Ma (Squire and Wilson, 2005). The second ductile deformation event, D_2, produced the most dominant ductile fabrics at Stawell and occurred at about 496-494 Ma (Squire, 2004). D_2 refolded F_1 closures and fabric into tight F_2 folds causing S_1 to appear predominantly parallel to S_2 (Miller and Wilson, 2002). The mesoscopic F_2 folds trend to the northwest and are generally asymmetric with hinges varying in size from centimetres to tens of metres (Wilson *et al.*, 1992). Peak metamorphism at Stawell is considered pre- to syn- D_2 and reached mid-greenschist grades (Miller and Wilson, 2002, Wilson *et al.*, 1992).

Overprinting both of the earlier fabrics at Stawell is an asymmetric differentiated crenulation cleavage, S_3. D_3 is a result of developing west-over east-shearing and folding (Watchorn and Wilson, 1989). This event is date at approximately 494-492 Ma (Squire and Wilson, 2005). The crenulation foliation is generally sub-horizontal and is associated with cleavage-parallel veins. There is evidence of a fourth ductile fabric at Stawell which is associated with D_4 and is interpreted as a ductile-brittle event (Miller and Wilson, 2002).

9.2.2.2 Late, brittle deformation

Superimposed on the ductile fabrics (D_1-D_4) are a number of brittle structures (see Figure 13 and Figure 14). The geometry and style of the brittle deformation is strongly dependent on the pre-existing geometry of the basalt (Miller and Wilson, 2002). The initiation of the brittle deformation occurred during late D_4 when the shortening direction changed from a northeast-southwest to an east-west orientation (Watchorn and Wilson, 1989; Miller and Wilson, 2002). The early D_4 shear zones have a northwest trend and dip to the southwest between 20° to 60° with a reverse sense of movement (Watchorn and Wilson, 1989).

There was a change in the regional stress field within the Lachlan Orogen during the Late Silurian which is expressed at Stawell as a change from east-west shortening to sinistral wrenching along pre-existing faults (Miller and Wilson, 2004a). Reactivation of the D_4 shears by sinistral wrenching is termed D_5 (Mapani

and Wilson, 1994). The sinistral wrenching was followed by another change in the regional stress field with the shortening direction changing to northwest-southeast. A set of major faults oblique to the earlier structural trends associated with this change in shortening direction are termed 'early South Fault' structures (Miller and Wilson, 2004a). The last major deformation event was associated with a final change to a northeast-southwest shortening. Faults associated with this event dip northwest and have a dip-slip sense of movement (Miller and Wilson, 2004a).

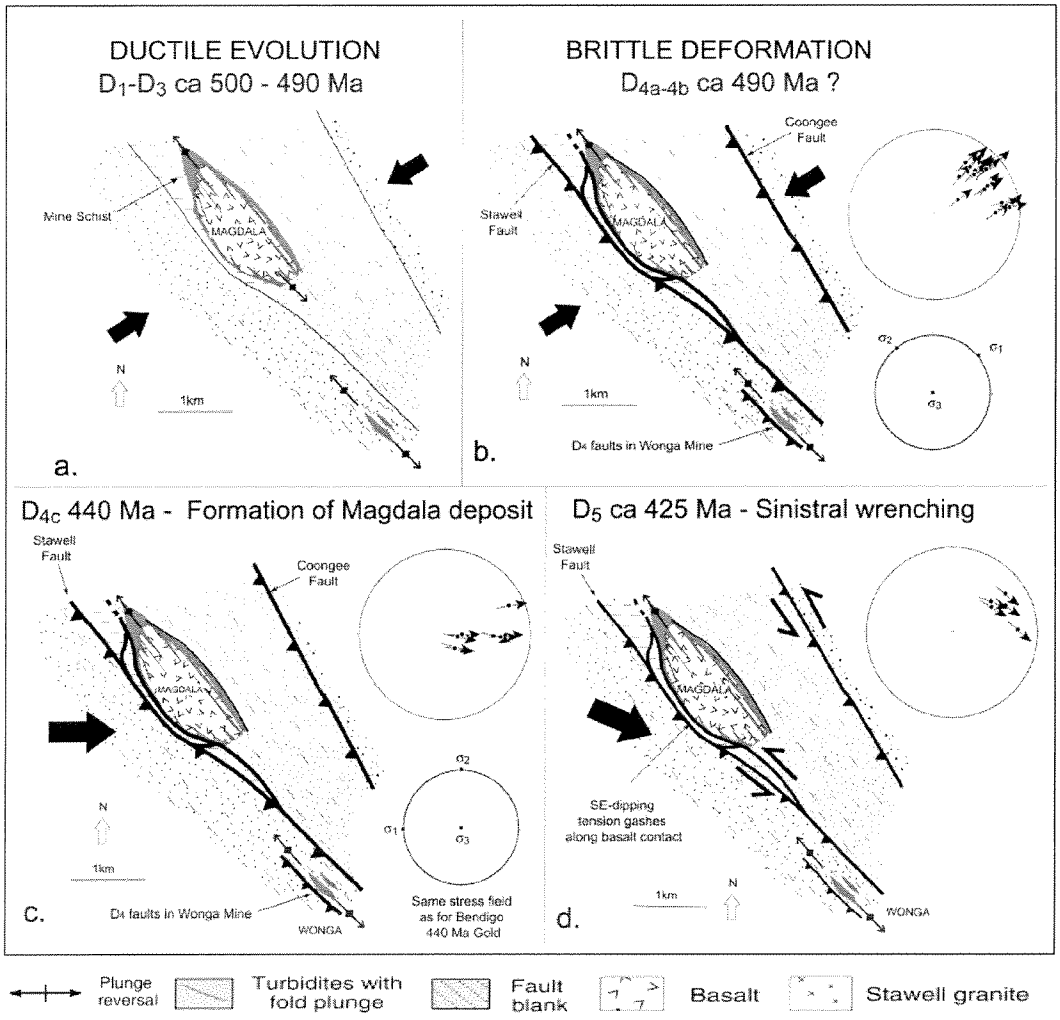


Figure 13 D_1 to D_5 ductile and brittle evolution of the Stawell System. Stereonets represent hangingwall transport direction calculated at pole to fault with the circle centre of each arrow representing a single fault pole (Miller & Wilson 2004a). These transport directions are the inferred maximum resolved shear stress along a fault for an applied stress tensor. A change in the hangingwall transport direction for similarly oriented faults represents a change in stress tensor. From Miller et al. 2006.

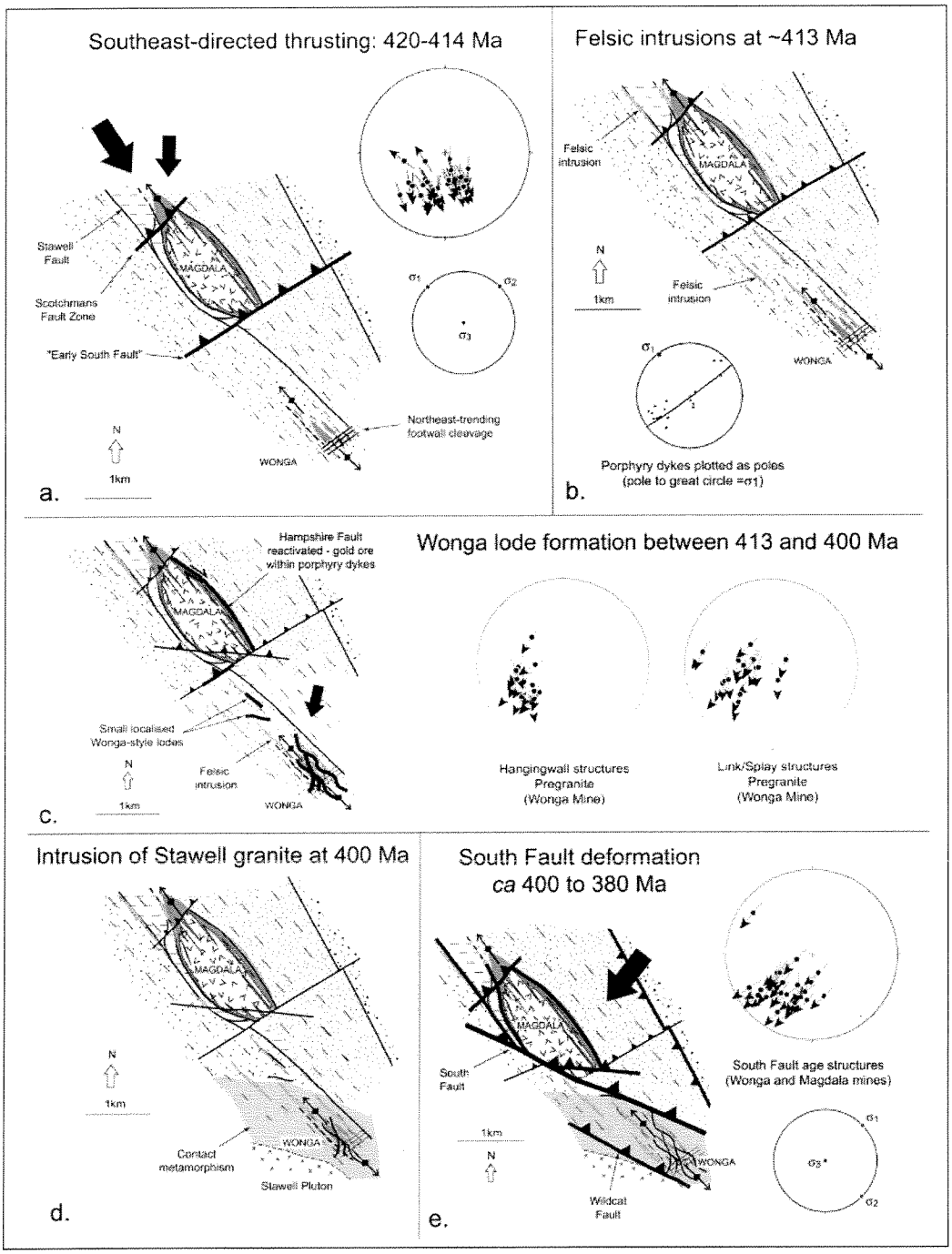


Figure 14 Evolution of the Stawell system from 420 to 380 Ma (modified from Miller & Wilson 2004a). Stereonets represent hangingwall transport direction calculated at pole to fault (Miller & Wilson 2004a). Map symbol are the same as those in Figure 13. From Miller et al. 2006.

9.2.3 Stawell Mine Geological Architecture

The dominant feature at Stawell is the 1.2Km wide doubly plunging northwest striking Magdala Basalt dome. The Magdala Basalt is made up of a series of basalt noses, interrupted to be flow sheets (Squire and Wilson, 2005), which dip to the southwest and plunge to the northwest. Areas of sedimentation are present between the basalt noses (interpreted flow sheets) and are locally termed 'waterloos'. The Magdala Basalt has been drilled and identified to a depth of 1.7Km and interpreted from existing drill information and from geophysical modelling to extend along strike at least 5Km.

Surrounding the basalt dome is the turbidite sequences of the Albion and Leviathan Formations (mineschist) which young to the west. The contact between the mineschist and Magdala Basalt on the western side is marked by the alteration package of the Magdala Volcanogenics. The Magdala Volcanogenics is weakly developed on the eastern surface of the Magdala Basalt.

This Magdala geology has been faulted and offset by later brittle deformation, the most notable of these offsets is the South Fault which has a northeast over southwest sense of transport (Figure 15 and Figure 16).

Above the South Fault is the Magdala ore body which contains limited offsets due to late faulting. The Basalt surface in the Magdala ore body dips to the west and strikes towards 340°. Beneath the South Fault is the Golden Gift ore body which is heavily offset by late faulting creating isolated ore blocks. Unlike the Magdala ore body the basalt in the Golden Gift dips to the east and strikes towards 315°. The late faulting as well as creating isolated ore blocks also complicates the ore geometry within the each block.

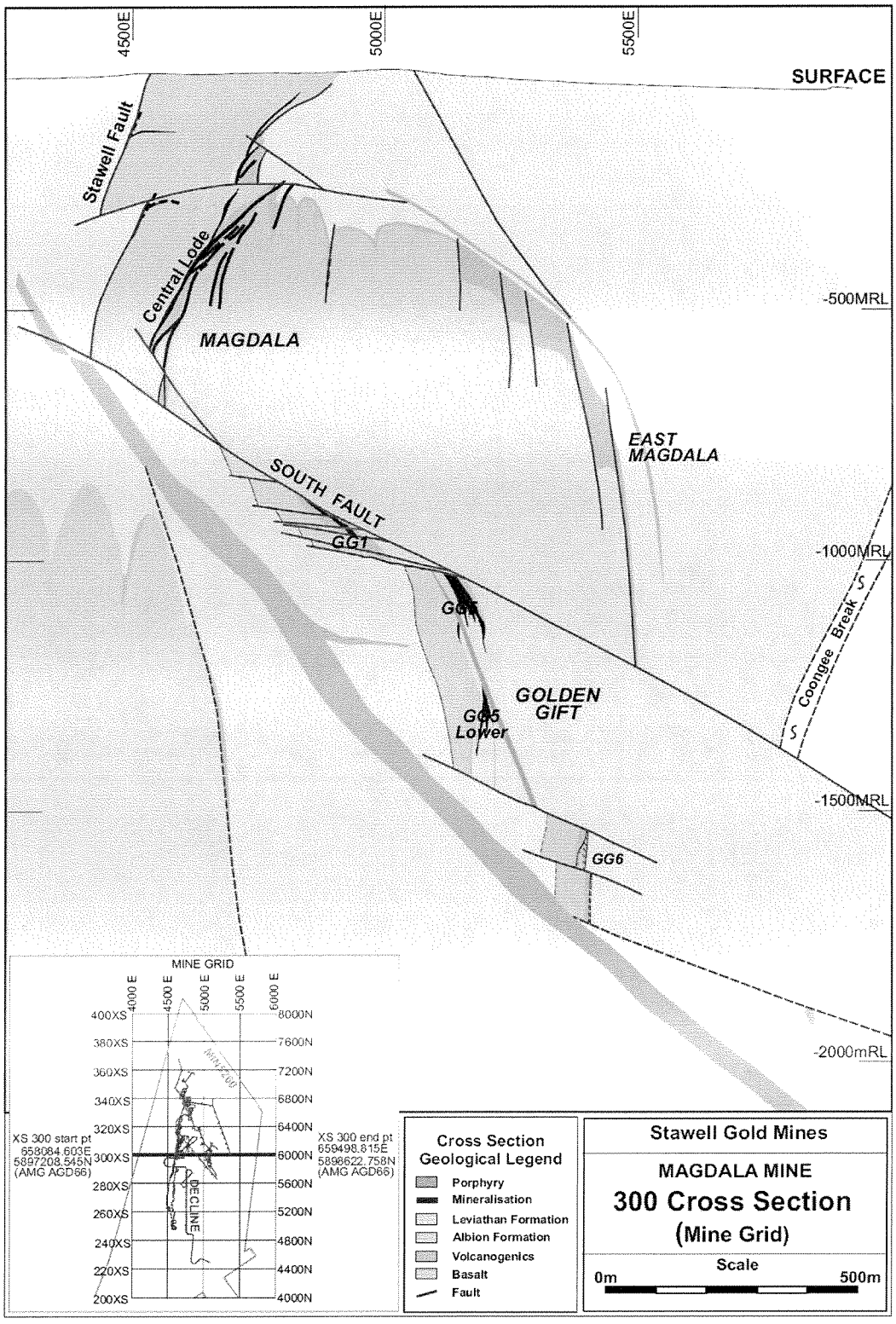


Figure 15 Mine Geology Cross-section highlighting architecture of the Magdala and Golden Gift ore bodies.

To the south of the Magdala Basalt is the Stawell Granite which structurally is situated below the South Fault (Figure 16).

Located close to an embayment in the Stawell Granite are a series of brittle structures. One of the structures termed the Hangingwall structure strikes towards 350° and dips between 25° and 50° towards the east, and the other structural set, termed Link structures, generally trend toward 240° and dip between 40° and 70° to the southeast (Xu *et al.*, 1994). Cross cutting these late brittle structures are a series of late felsic intrusive. This fault system hosts the Wonga ore body.

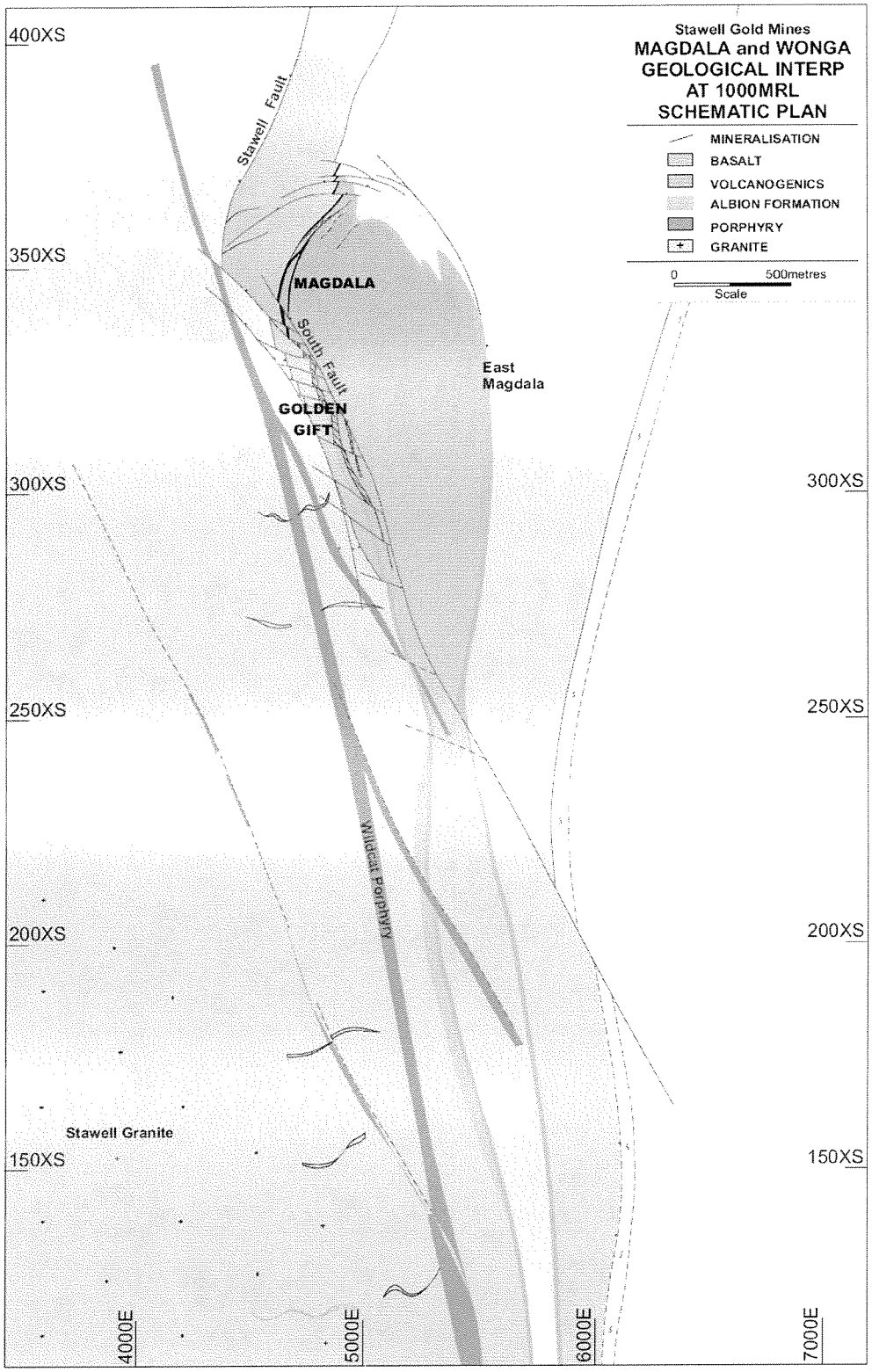


Figure 16 Plan view geological interpretation of the Stawell structural and stratigraphic architecture at 1000mRL.

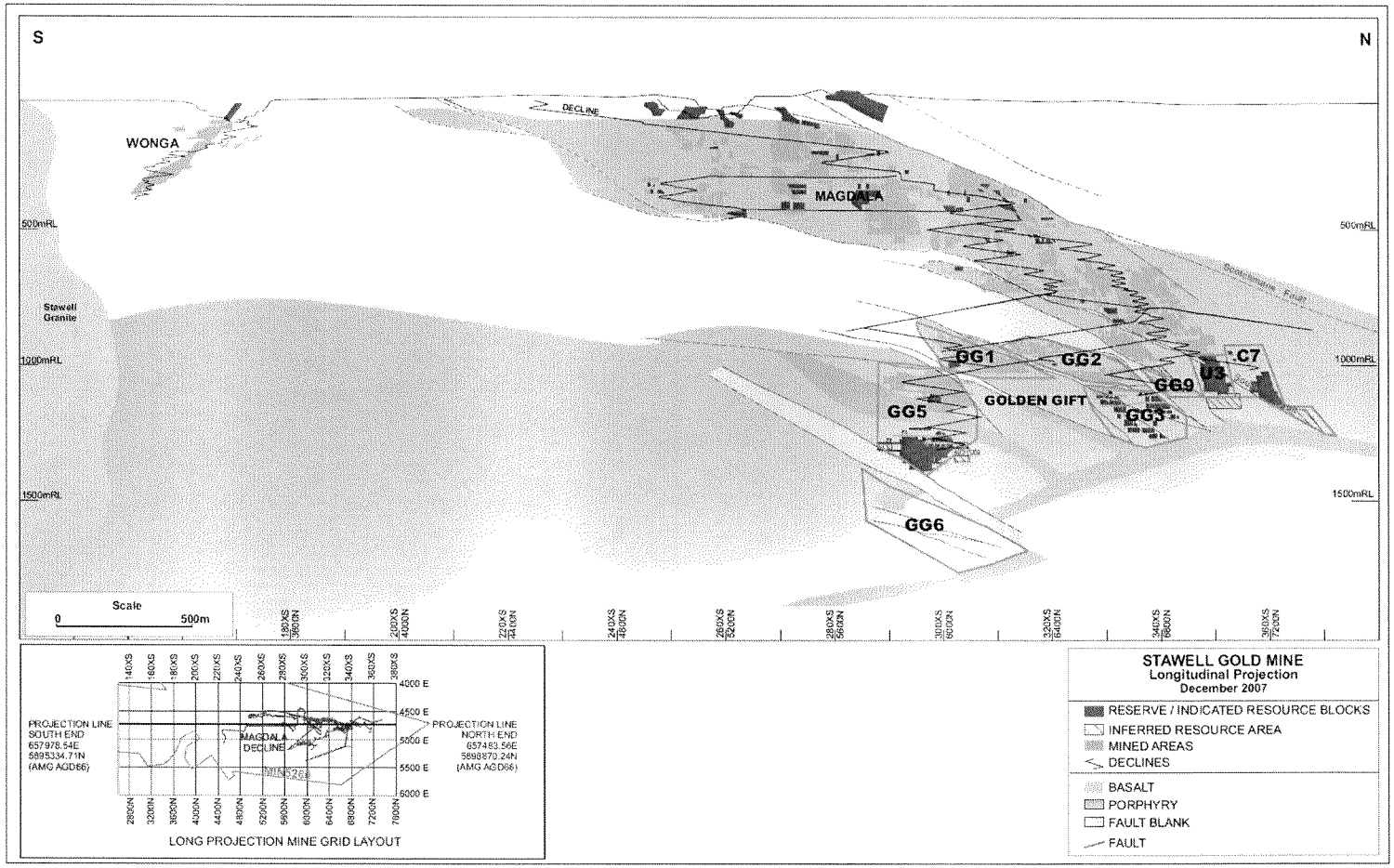


Figure 17 Stawell Mine Long Projection showing the location of the Mineralised ore block. The geological and spatial relationship between the Magdala, Golden Gift and Wonga deposits can seen clearly be seen

10 Deposit Types and Mineralization

Victorian mineralisation episodes have been dated to occur during the Devonian and Silurian with no gold mineralisation occurring prior to 440 Ma (Miller and Wilson, 2002). A description of the mineralisation episodes in Western Victoria is described below.

The largest and most significant mineralisation event in the western Lachlan Orogen occurred at ca. 440 Ma (Foster *et al.*, 1998). It occurred contemporaneous in the Stawell and in the Bendigo-Ballarat zones with the mineralisation occurring during late D_4 in the Stawell zone and occurring during late D_1 in the Bendigo-Ballarat zone. The mineralisation is hosted in D_4 brittle structures associated with east-over-west movement at Stawell while in the Bendigo-Ballarat zone the mineralisation occurs in saddle reefs in the hinges of D_1 folds and in reverse faults created via D_1 fold lock-up (Miller and Wilson, 2002; Schaubs and Wilson, 2002). This mineralisation event produced the largest endowments of gold within the western Lachlan Orogen (Miller and Wilson, 2002).

The next episode of gold mineralisation occurred at about 426-420 Ma (Foster *et al.*, 1998) and is associated with fault reactivation throughout western Victoria (Miller and Wilson, 2002). This episode of gold mineralisation produced significantly smaller endowments than the 440 Ma event (Miller and Wilson, 2002). The late Silurian mineralisation is associated with the D_5 sinistral wrenching at Stawell and has been recognised at the Percydale fields in the Stawell zone and at Tarnagulla in the Bendigo-Ballarat zone (Miller and Wilson, 2002).

The final episode of mineralisation recognised in western Victoria is the Wonga mineralisation at Stawell (Miller and Wilson, 2004a). The mineralisation at Wonga over prints the quartz- and felsic-rich intrusions and is overprinted by the Stawell Granite contact areole. Watchorn and Wilson (1989) suggested that this mineralisation is temporally and spatially related to the granites emplacement. Miller and Wilson (2004b) advocate the mineralisation event at Wonga formed at ca. 400 Ma (Foster *et al.*, 1998). The Wonga mineralisation occurred during a late-stage magmatic event within a long-lived orogenic system at shallow crustal levels

(Miller and Wilson, 2004b). This mineralisation occurred in a series of brittle structures dependent on pre-existing weakness which are related to a fluid over-pressure event after the lockup of major structures (Miller et al., 2004).

11 Ore Types

There are three different ore bodies at Stawell; the Magdala, Golden Gift and Wonga. Each of the differing ore and mineralisation types are summarized below. Both the Magdala and Golden Gift ore types are hosted within the Magdala Volcanogenics.

11.1 Magdala Deposit Ore Types

Within the Magdala deposit there are three main ore types; Central Lode, Basalt Contact Lodes, and Madgala Stockwork Lodes. These are summarized below.

11.1.1 Central Lode

Central lode mineralisation was a significant production source from Magdala. It is a quartz rich shear lode ranging from 0.5 to 10m in width and generally dips 55 - 65° to the west with a total strike length of 4km and a down dip extend of 1km. Whilst the overall structure is mineralised economic shoots are vary from 20 - 30m in strike up to 200 - 350m in strike. Free gold in the quartz is associated with pyrite, arsenopyrite and recrystallised pyrrhotite. Average mined grade for Central Lode is 4 - 7 g/t Au.

Figure 18 Example of central lode mineralisation.

11.1.2 Basalt Contact Lodes

Basalt contact lodes are located parallel to the Magdala Basalt and in 'waterloo' positions. They are typically 2m wide and are represented by arrays of quartz sulphide tension veins immediately adjacent to the volcanogenic Basalt contacts. Sulphides include pyrrhotite, arsenopyrite and pyrite and occur as alteration selvages on tension vein margins. The main alteration mineral is Stilpnomalane, resulting in its dark colour. The mineralisation is isolated to the Magdala Volcanogenic package with none present in the adjacent Magdala Basalt. Ore shoot lengths range between 50 and 450m. The average mined grade for Basalt Contact Lodes is 4 - 9 g/t Au.

Figure 19 Example of basalt contact mineralisation

11.1.3 Magdala Stockwork Lodes

The Magdala stockwork lodes are situated above major basalt noses and can be described as a hybrid between central and basalt contact lodes. They consist of large quartz tension vein arrays with arsenopyrite and pyrrhotite dominant sulphide mineralisation. The strike extent is limited to 40 - 50m and limited vertically 30 - 50m. Average mined grade for Magdala Stockwork Lodes is 4 - 7 g/t Au.

11.2 Golden Gift Deposit Ore Types

Unlike the Magdala deposit there is only one identifiable ore type in the Golden Gift and is termed the Golden Gift Stockworks. Though there is only one discernable ore type in the Golden Gift the Golden Gift Stockworks contain a spectrum of all Magdala styles. Typical widths range from 8 -12m up to 30m and the strike extents of shoots range between 150 and 400m. Areas of highest grades and largest widths are situated above major basalt noses which are present in most orebodies. Quartz content is generally below 25%. Mineralisation includes abundant

recystallised pyrrhotite and coarse grained arsenopyrite, pyrite and visible gold. Average mined grade is 4 - 10g/t Au.

11.3 Wonga Deposit Ore Types

The Wonga deposit is hosted within the locally termed Wonga Schist (part of the Leviathan Formation) along two main fault systems. The Wonga Schist has undergone contact metamorphism during the emplacement of the Stawell Granite (Xu *et al.*, 1994) and undergone three ductile deformation events similar to other areas of the Stawell region (Watchorn & Wilson, 1989). The two fault systems controlling the mineralisation are the Hangingwall structure which, strikes towards 350° and dips between 25° and 50° towards the east, and the Link structures which generally trend toward 240° and dip between 40° and 70° to the southeast. The mineralisation is represented by aresenopyrite disseminations to quartz veins within these structures. The main ore minerals present are anhedral fine grained pyrrhotite and arsenopyrite. The higher grade ore zones often show andalusite-sericite alteration with rutile and ilmenite associations (Xu *et al.*, 1994). Production grades from 4 - 6 g/t Au were common for Wonga ore.

12 Exploration

Section 8 details the significant exploration progress and successes that has been made over the life of the Stawell project including the discovery of the current Mineral Resources and Mineral Reserves. In addition to the programs that have lead to these discoveries in Magdala and Golden Gift a number of other near mine exploration programs have been undertaken over the past 20 years. A full summary of these activities are not relevant to and are beyond the scope of this report however they are documented in the annual mines Department reports filed for MIN5260 and the surrounding exploration tenement EL3008 that are available on site at Stawell Gold Mines.

The discussion in this section is limited to providing details on the current Exploration initiatives (2007 drilling) and the proposals that have been developed for further evaluation and testing during future exploration programs. Section 8

covers descriptions of the major discoveries of mineralisation that constitutes the existing Mineral Resource estimates reported in this document.

All Exploration activities conducted on MIN5260 are undertaken by Stawell Gold Mines employees utilising contract surface and underground diamond drill teams. The procedures adopted for exploration are as discussed in Sections 13 - 15 of this report.

12.1 Current Exploration

Apart from ongoing Resource Definition and Grade Control diamond drilling adjacent to the known mineralisation in Golden Gift and Magdala exploration drilling in 2007 has focussed on testing the extent of the Golden Gift mineralisation through two programs, Golden Gift South (GG South) and Golden Gift 6 (GG6). The location of the GG6 targets can be seen on Figure 17 with GG South target is essentially the projection of the Basalt surface south of GG5, GG5 Lower and GG6.

12.1.1 Golden Gift South

The target for Golden Gift South drill testing was Magdala Volcogenics located in the basalt flank positions to the south known mineralisation in the Golden Gift. A surface drill program had tested down to 1100 mRL with limited success in 2004/05. The aims of this new program were to test along the basalt flank below this RL. At the end of December 2007 three holes were complete (see Figure 20) Intersecting weak volcanogenics and returning the following downhole intercepts: MD5183: 1.6m @ 1.3 g Au/t; MD5136: 0.4m @ 1.0 g Au/t; MD5163A: NSR.

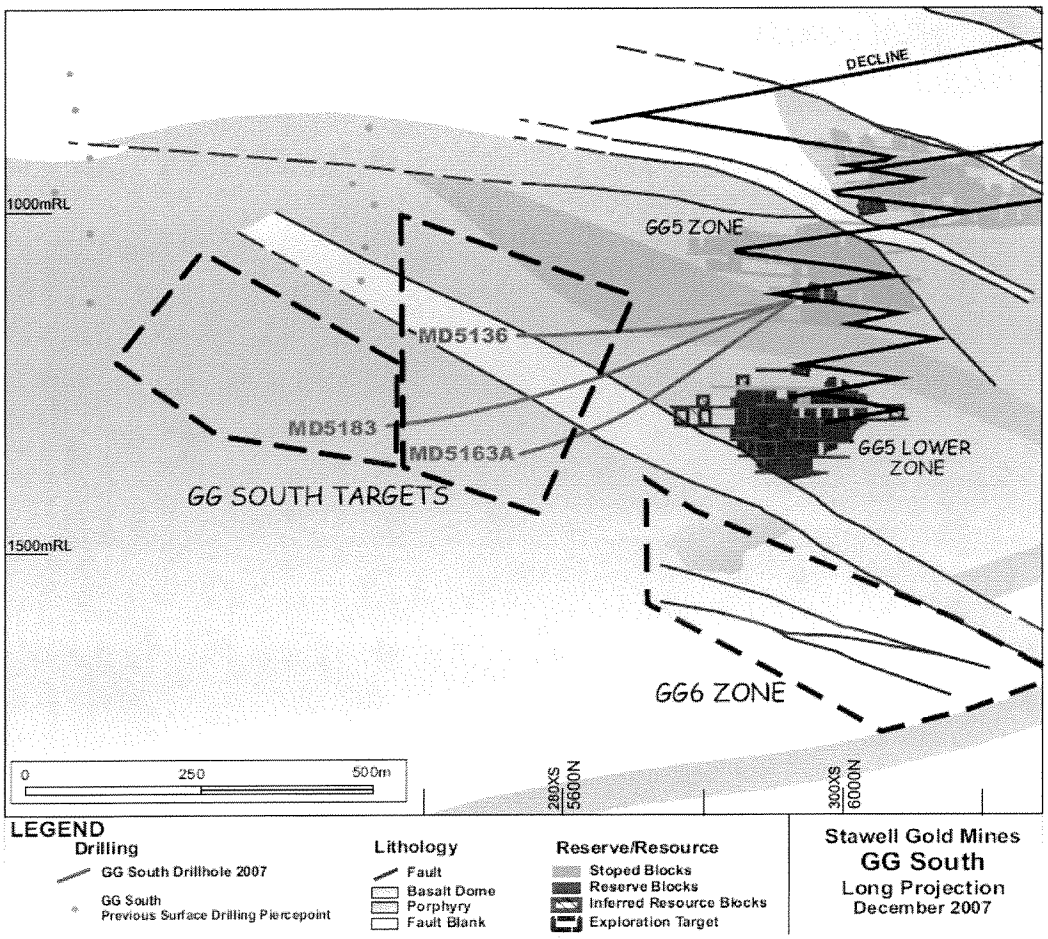


Figure 20 Longitudinal projection of the GG South target zone and completed drill holes.

12.1.2 Golden Gift 6

The Golden Gift 6 (GG6) exploration program began in 2007 targeting the area below the basal fault of the Golden Gift 5 Lower (GG5L) ore block to identify if an offset ore block existed. Thirteen diamond drillholes were drilled into the GG6 target zone during 2007 with the program successfully identifying mineralisation under the basal fault of GG5L on two surfaces. The results of the program are shown in Figure 21.

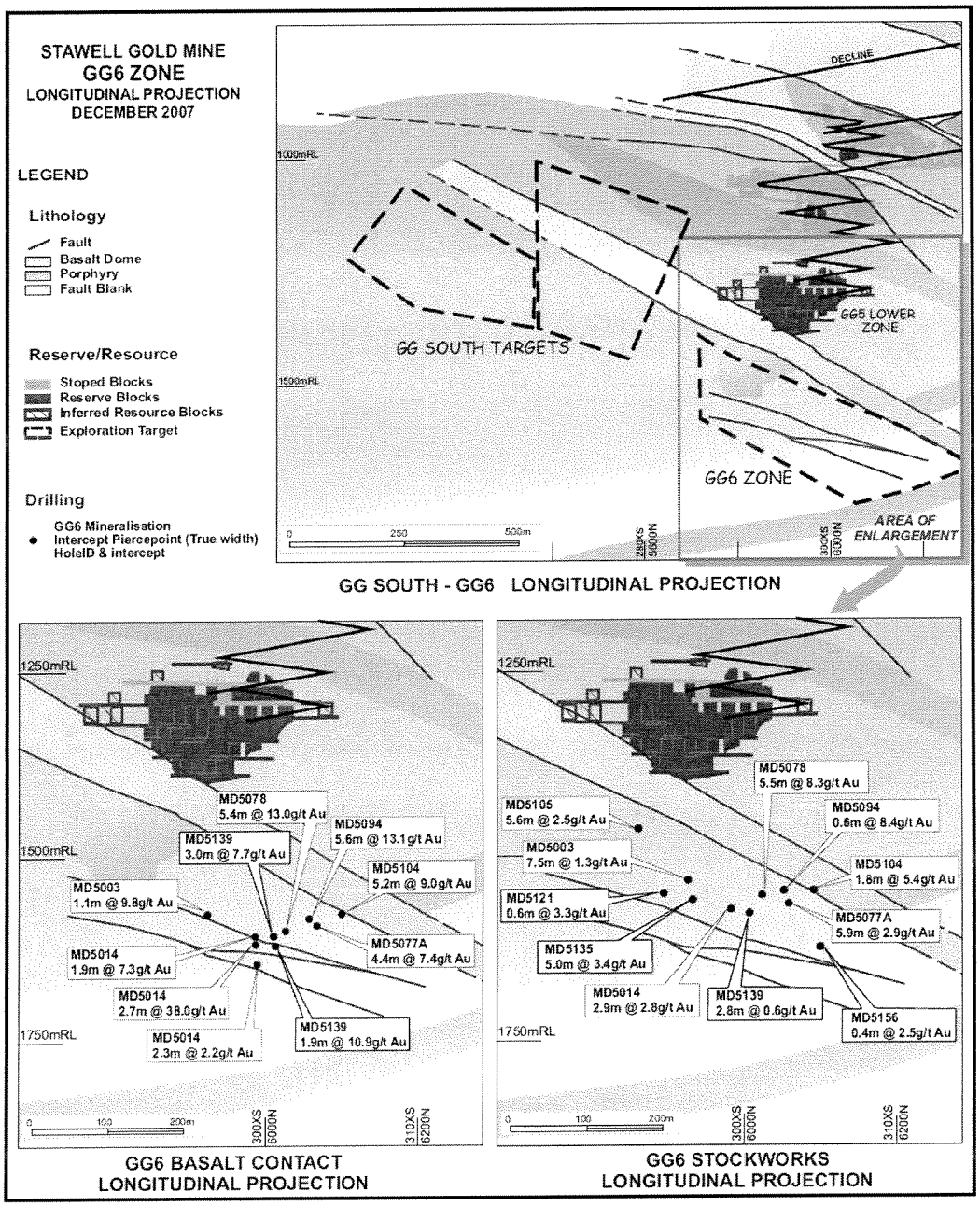


Figure 21 Longitudinal projections of the GG6 zone highlighting the results obtained on the basalt contact and stockworks surfaces. All intercepts are quoted in true width.

12.2 Proposed Exploration

A number of targets or project areas adjacent to the known deposits and contained within MIN5260 are proposed for ongoing exploration. These targets have all had various levels of exploration activity over recent years. Ongoing geological interpretation and data evaluation is in place to prioritise these targets and determine the appropriate programs for additional testing.

The targets listed in Table 5 are planned to be worked upon during 2008 by SGM personnel. Currently the ranking of the targets and scope of work is in discussion and with the recent corporate activities budgets are being agree upon and the budgeted allocations as shown in Table 5 are proposed and could vary pending final approval and receipt of exploration results. Locations of targets can be seen in Figure 22 and Figure 23.

Target	**Concept/scope of works**	**2008Budgeted Expenditure $M AUD**
Golden Gift 6	Follow up on 2007 drill results to develop a potential resource.	UG Drilling $0.66M
North Magdala	Following up on a surface drill hole intercept from 2005 in SD622 of 9.4m @ 8.35 g Au/t. This intercept is located down dip and along strike of current Mining areas on section 388mN. The follow up drilling wil be completed from underground and possibly the surface.	Surface Drilling $2.2M UG Drilling $1.2M
Wonga open pit and underground	To test potential of pit extensions to the North and South through a series of surface holes. Currently work is being undertaken to a 3D structural model of the Wonga deposit which will be used to identify deeper drill target surrounding the underground mine. These targets are also anticipated to be drilled from surface.	Surface Drilling $0.78M
East Magdala	Test the eastern flank of the Magdala Basalt Dome. Drill testing will occur from underground locations.	UG Drilling $0.65M
Other Targets	Testing of other targets including Golden Gift South	UG Drilling $0.57M
Total Program	**Exploration and Resource Definition**	**$7.06M**

Table 5 2008 MIN5260 Exploration targets

In addition to the Exploration program proposed for 2008 Stawell Gold Mines has budgeted $AUD 3.2M for Mine Geology activities including 26,500 metres of diamond drilling. These activities are required to complete final mining development stope design to support mining activities.

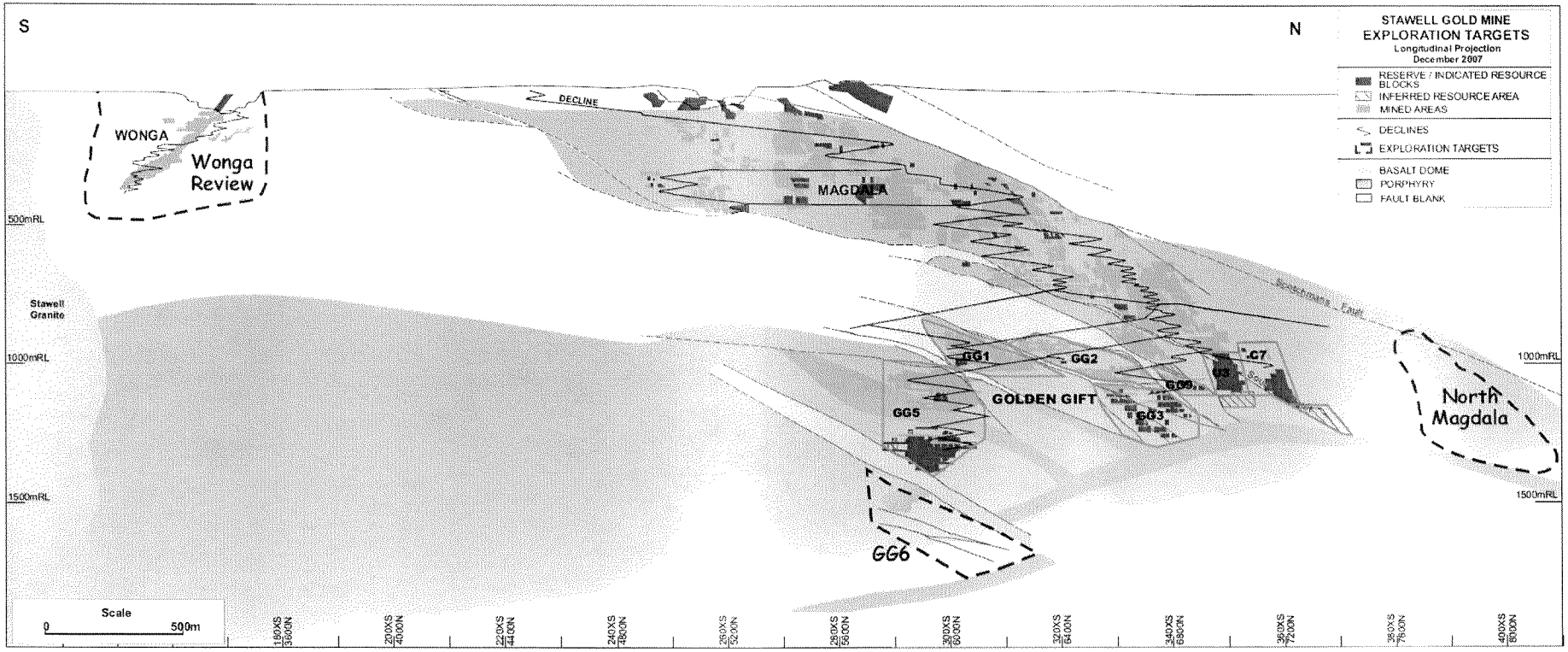


Figure 22 Longitudinal Projection highlighting the locations of 2008 exploration works

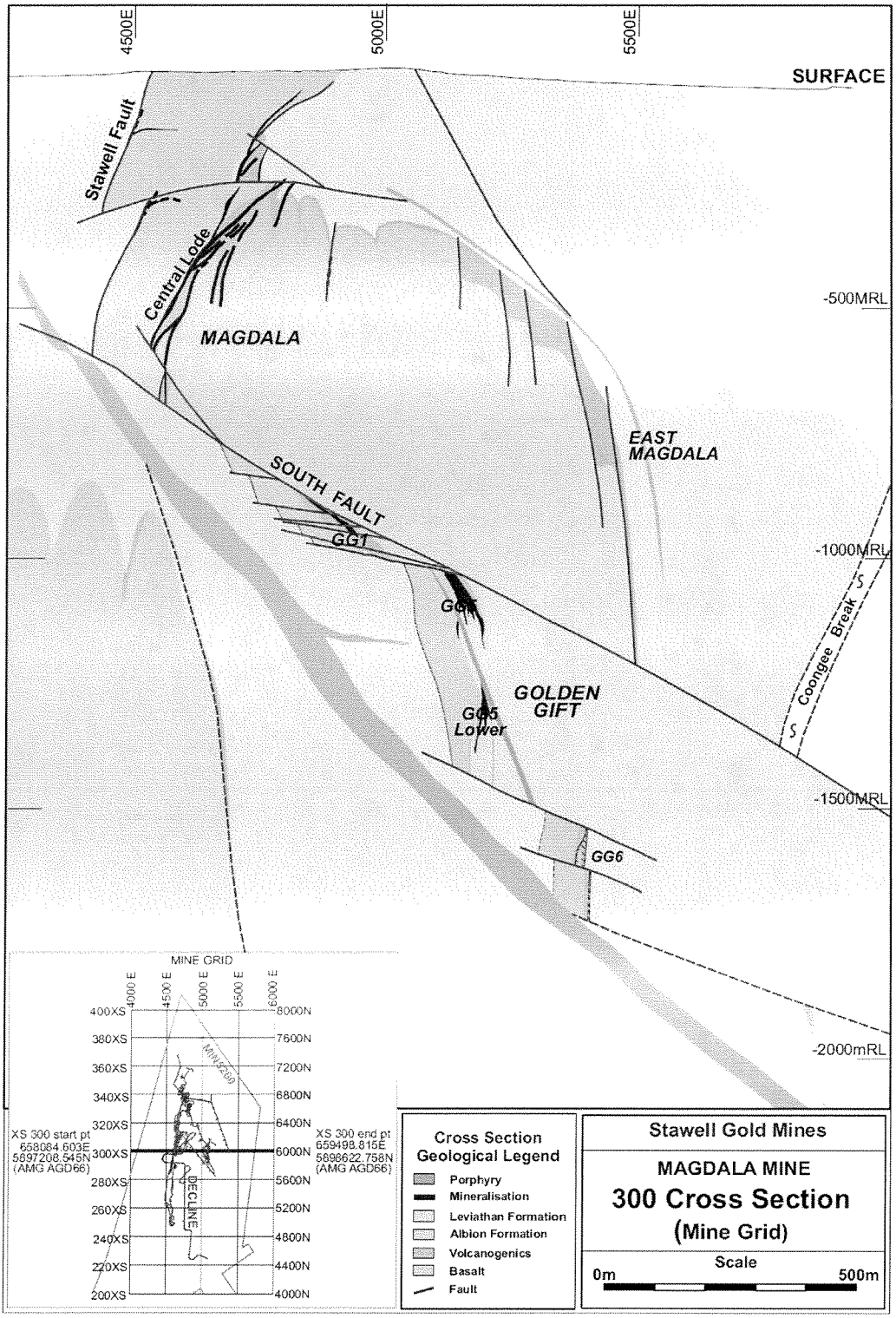


Figure 23 Cross Section highlighting the East Magdala exploration target for 2008

13 Drilling

13.1 Stawell Gold Mines Mineral Resource Definition Process

The Mineral Resource definition process at Stawell Gold Mines is a continuous process. Current Mineral Resources extend from surface to 1350mRL (effectively 1350 m below surface) and as such resource definition is an ongoing activity. Geological information is collected by a variety of methods with the objective of improving the confidence of the Mineral Resource estimates prior to and during the mining process including grade control drilling for development and stope definition. These geological processes and the data types gathered during each of these steps are summerised in Figure 24.

Activity Area	Target Type	Criteria	Resource Classification	Geological Data available
Exploration	Conceptual Targets	• Conceptual geological model • Geophysical anomaly • Geochemical anomaly		• Geophysics • Mapping • Wide spaced Exploration Drilling • Conceptual geological models
	Confirmed Targets	• Geological model confirmed by drilling • Mineralisation confirmed by drilling		• Wide Spaced Exploration drilling • Assay information • Drill logs
	Scoped targets	• Geological continuity established • Ore grade intersections established • Preliminary geological model established • Drill spacing 300 m X 120 m	Pre resource, Resource Target	• Broad spaced Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models
Resource Definition		• Geological continuity confirmed • Ore grade intersections continuous • Geological interpretation modelled • Geostatistical model established • Drill spacing 150 m X 60 m	**Inferred Resource**	• Regular Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis
		• Geological continuity confirmed • Ore grade intersections confirmed • Geological interpretation modelled • Geostatistical model • Economic analysis • Drill spacing 40 m X 30 m	**Indicated Resource**	• Regular Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis
Grade Control		• Confident Geological continuity • Ore grade intersections confirmed • Geological interpretation modelled • Geostatistical model • Economic analysis • Drill spacing 15 m X 15 m	**Indicated Resource**	• Close Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis
		• Confident Geological continuity • Ore grade intersections confirmed • Geological interpretation modelled • Geostatistical model • Economic analysis • Level Development above and below • Drill spacing 15 m X 15 m • Open-hole sludge drilling	**Measured Resource**	• Close Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis • Development face mapping sheets and ore runs • Open-hole sludge drill geological data

Figure 24 Stawell Gold Mines geological processes and approximate drill spacings

Given the continuous and ongoing nature of the Mineral Resource process the data utilised varies as the mining operations develop towards the resource area. A summary of the main drilling methodologies employees and data types utilised is as follows with specific details on the sampling and assaying methodologies given in Section 14 and sample intervals and the relationship to orebody geometries etc are discussed in components of Section 19.

- Surface RC Drilling
    - Utilised for definition of near surface resources where diamond drilling is not required for detailed structural definition
    - Utilised as a method for pre collaring deeper diamond drillholes
    - Drilling completed using 5 ¼ " Face Sampling Hammers
    - Samples collected from cyclone discharge
    - Hole Depths vary but are generally less than 200m
    - Drilling is conducted dry or with sufficient air to ensure collected samples are dry
    - In some project areas casing andvancing technologies have been utilised to ensure drilling through fill produces reliable samples
    - Undertaken by contract drilling personnel under the supervison of SGM geology
- Surface Diamond Drilling
    - Primarily used in initial Exploration programs, near surface Resource Definition and to provide structural and geological information in near surface RC drilling programs
    - Drilling by wireline methods.
    - Hole sizes PQ^3, HQ^3, NQ^3, HQ^2, NQ^2, BQ^2
    - Hole depths vary from <100m to >2000m
    - Directional drilling utilised for specific tasks Core Orientation devices often utilised to aid in structural interpretation
    - Undertaken by contract drilling personnel under the supervison of SGM geology
- Underground Diamond drilling
    - Utilised at all stages of the geological process, Exploration, Resource Definition and Grade Control

- Drillng by conventional and wireline methods
- Hole sizes HQ^3, HQ^2, NQ^2, BQ^2, LTK60, LTK48 (Not used post 1997)
- Hole depths vary from <50m to 1200m
- Directional drilling utilised for specific tasks
- Undertaken by contract drilling personnel under the supervison of SGM geology

- Open Hole percussion sampling "Sludge Sampling"
  - Utilised after development of Ore Drives for final stope definition
  - Hole sizes 89mm open hole
  - Hole depths vary from 5m - 25m
  - Samples of cutting of variable length are collected primariliy for geological logging of the chips to identify major faults and geological contacts
  - Undertaken by SGM production blasthole rigs

13.2 Extent of Drilling

The details of the drilling completed at Stawell Gold Mines for the life of the project are shown in Table 6. The extent of this drilling relative to the overal mineralised system at Magdala, Golden Gift and Wonga is shown and Figure 25. These holes have been progressively drillied and large portions of the material informed by this drilling has been mined to date.

Period	Surface Diamond	Magdala Underground Diamond	Wonga Underground Diamond	Surface RC	Total
	(m)	(m)	(m)	(m)	(m)
Pre 1997	107,809	151,533	33,508	14,497	**307,348**
1998	6,569	26,497	1,041	20,336	**54,444**
1999	11,973	37,104	1,210	1,147	**51,434**
2000	5,734	54,015			**59,750**
2001	9,192	53,603			**62,795**
2002		60,718			**60,718**
2003		25,070			**25,070**
2004	16,607	45,288		155	**62,049**
2005	25,039	56,080			**81,120**
2006	1,904	40,051		1,897	**43,852**
2007		39,268			**39,268**
Totals	**184,827**	**589,230**	**35,760**	**38,032**	**847,849**

Table 6 Drilling metres summary by drilling method for Stawell Gold Mines.This is the entire databse that informs Mineral resources reported for MIN5260.

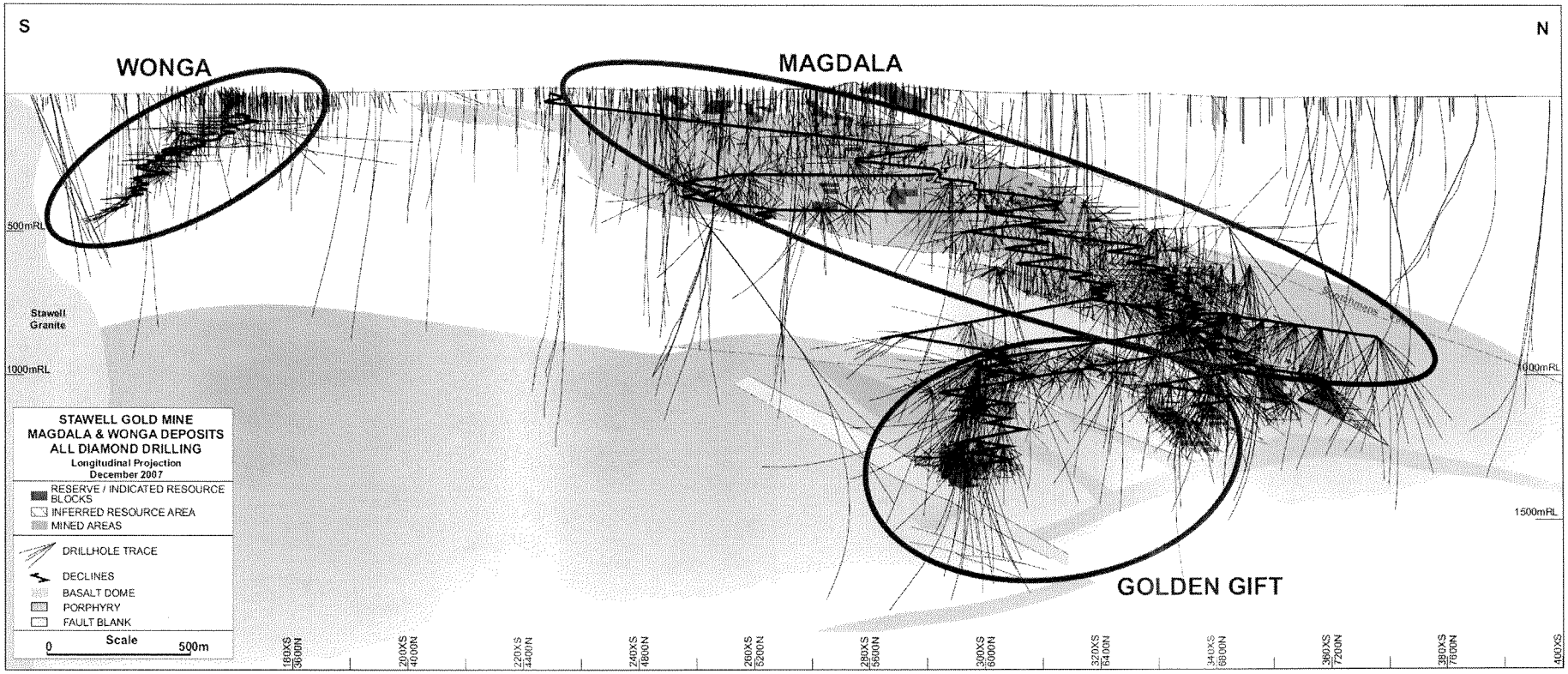


Figure 25 Stawell Longitudinal projection showing the extent of all drilling completed.

13.3 Drilling Process

A flowsheet of the diamond drill process from design to implementation is shown in Figure 26.

The diamond drill contract personnel provide a daily record of drilling activities for all drill rigs. A copy of a daily drill record sheet is shown in Figure 27. Data from the daily record sheet is entered daily to a site databse for tracking of drilling produciton and to enable tracking of drilling progress interogation at a latter date.

Geological personnel track the drillhole path and maintain in control of the daily activities of all drill rigs including which drillers were respnsible for various sections of the hole should there be issues with core presentation or downhole depths that require clarification. A regime of regular rig audits and inspections are also used to assist with maintaining the high level of core presentation and sample quality. These drill records are kept indefinetely enabling review of drill hole information many years after completion of drilling.

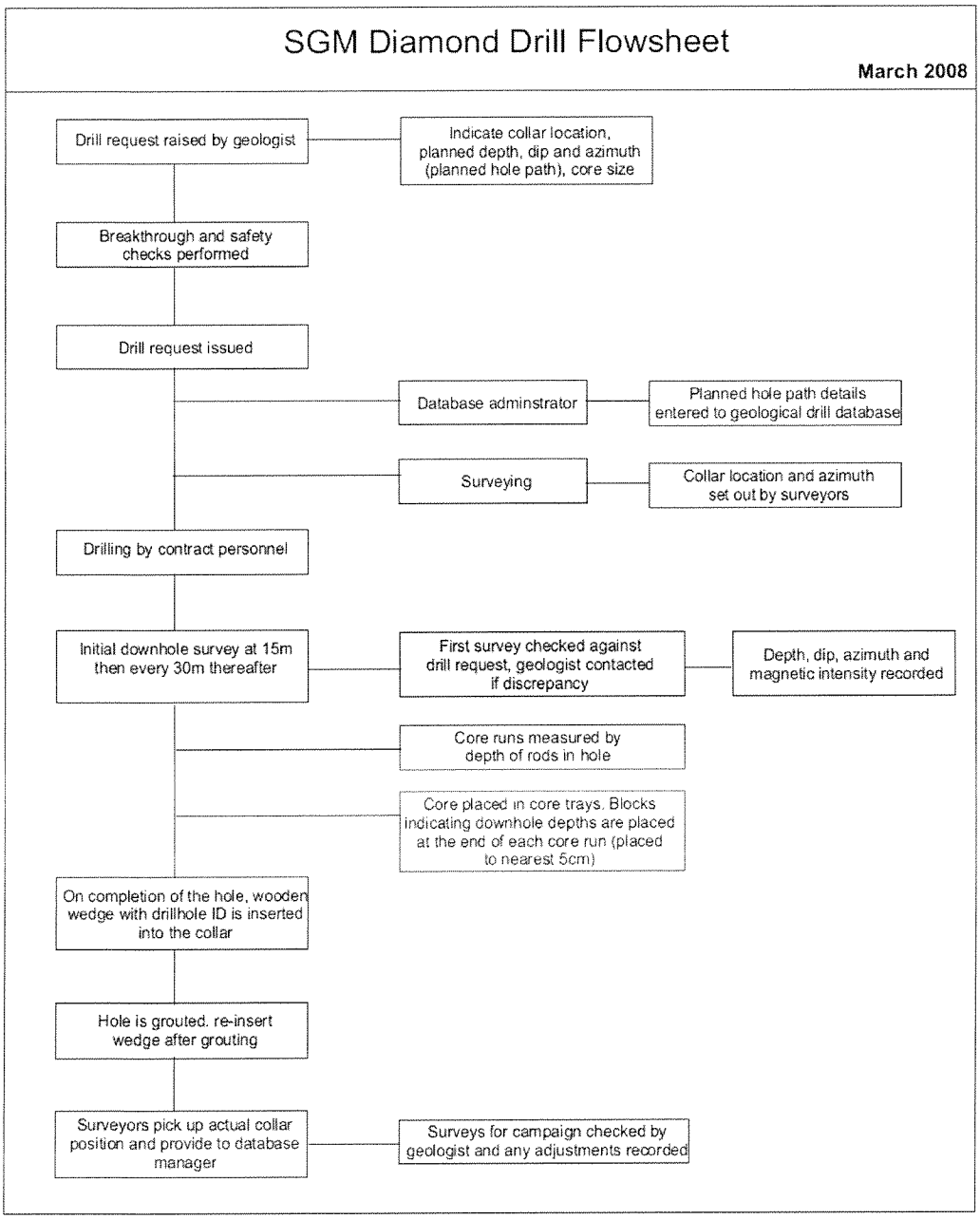
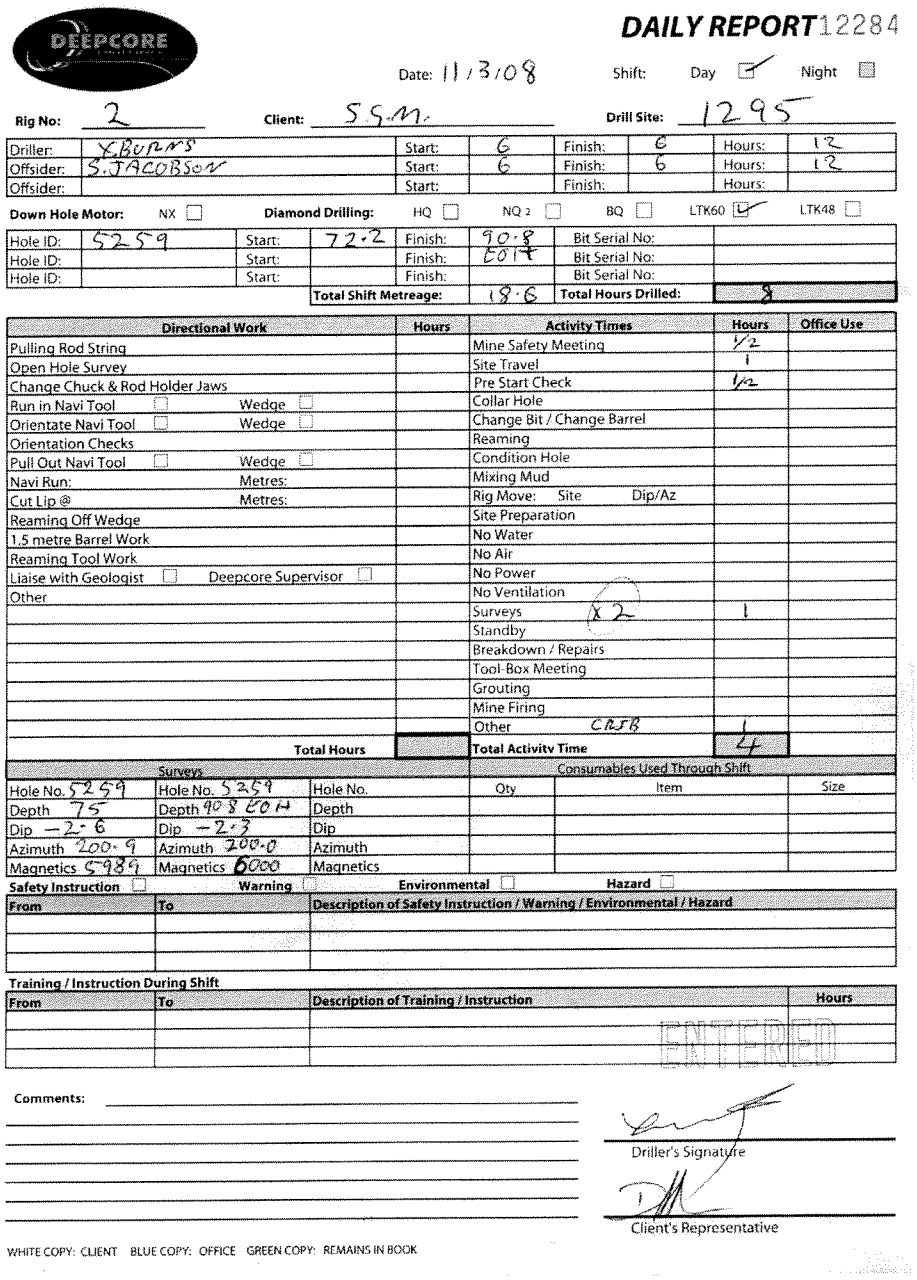


Figure 26 SGM Diamond drilling process flowsheet.



DEEPCORE

DAILY REPORT 12284

Date: 11/3/08 Shift: Day ☑ Night ☐

Rig No: 2 Client: SSM. Drill Site: 1295

Driller:	K.BURNS	Start:	6	Finish:	6	Hours:	12
Offsider:	S.JACOBSON	Start:	6	Finish:	6	Hours:	12
Offsider:		Start:		Finish:		Hours:	

Down Hole Motor: NX ☐ Diamond Drilling: HQ ☐ NQ 2 ☐ BQ ☐ LTK60 ☑ LTK48 ☐

Hole ID:	5259	Start:	72.2	Finish:	90.8 EOH	Bit Serial No:	
Hole ID:		Start:		Finish:		Bit Serial No:	
Hole ID:		Start:		Finish:		Bit Serial No:	
		Total Shift Metreage:		18.6	Total Hours Drilled:		8

Directional Work	Hours	Activity Times	Hours	Office Use
Pulling Rod String		Mine Safety Meeting	1/2	
Open Hole Survey		Site Travel	1	
Change Chuck & Rod Holder Jaws		Pre Start Check	1/2	
Run in Navi Tool ☐ Wedge ☐		Collar Hole		
Orientate Navi Tool ☐ Wedge ☐		Change Bit / Change Barrel		
Orientation Checks		Reaming		
Pull Out Navi Tool ☐ Wedge ☐		Condition Hole		
Navi Run: Metres:		Mixing Mud		
Cut Lip @ Metres:		Rig Move: Site Dip/Az		
Reaming Off Wedge		Site Preparation		
1.5 metre Barrel Work		No Water		
Reaming Tool Work		No Air		
Liaise with Geologist ☐ Deepcore Supervisor ☐		No Power		
Other		No Ventilation		
		Surveys X 2	1	
		Standby		
		Breakdown / Repairs		
		Tool-Box Meeting		
		Grouting		
		Mine Firing		
		Other CRSB	4	
Total Hours		Total Activity Time		

Surveys			Consumables Used Through Shift		
Hole No. 5259	Hole No. 5259	Hole No.	Qty	Item	Size
Depth 75	Depth 90.8 EOH	Depth			
Dip -2.6	Dip -2.3	Dip			
Azimuth 200.9	Azimuth 200.0	Azimuth			
Magnetics 5989	Magnetics 6000	Magnetics			

Safety Instruction ☐ Warning ☐ Environmental ☐ Hazard ☐

From	To	Description of Safety Instruction / Warning / Environmental / Hazard

Training / Instruction During Shift

From	To	Description of Training / Instruction	Hours

ENTERED

Comments:

Driller's Signature

Client's Representative

WHITE COPY: CLIENT BLUE COPY: OFFICE GREEN COPY: REMAINS IN BOOK

Figure 27 An example of a daily drill record from an underground rig. Note this record includes downhole survey information that matches the individual survey records

13.4 Drill Spacing

Drill spacing varies for exploration, resource definition and grade controls programs within the ranges as indicated in Figure 24. The appropriate Resource

classifications indicated in this table are a guide only and each Mineral Resource area is classified based on a number of criteria that will be discussed in Section 19 of the report. Each of the Mineral Resource areas discussed in Section 19 of this report will have drilling at various hole spacings depending upon the stage of resource development and mining activities.
Initially the exploration drilling is carried out on broad spaced targets, and if the continuity of the structure is apparent, as in the case of known Basalt bodies the targets will generally be tested on centres 300m along strike and 120m down dip (300m x 120m).

When exploration is succesful in locating appropriately mineralised environements this spacing is closed down to approximately 150m along strike x 60m up and down dip. If updated geological interpretations completed at this drill spacing are able to demonstrate geological continuity and define sufficient grade to complete and define a Mineral Resource it is possible to classify the defined Mineral Resources as Inferred.

Ongoing drilling will be completed once appropriate drill platforms can be established to enable the drill spacing to be reduced to 30m x 40m centres. At this spacing if the geological and grade continuity is well constrained a Mineral Resource could be classified as Indicated Resource. Generally at this stage final mine design and scheduling is possible and capital development infrastructure can be designed and commenced to access the area for mining including the design of appropriate platforms to complete ongoing Resource definition and Grade Control diamond drilling.

Grade Control diamond drilling targets a drill spacing of at least 15m along strike by 15m up and down dip on the mineralised structures. This drilling is a component of the mine production process and is required to identify any small scale changes in geometry which will affect mining shapes. Where the geometries are complicated by faulting or other geological features then the spacing can locally be closed to 10m x 10m. Following this work detailed stope and development desing is completed and ore development is designed and implemented under survey control. As a general rule only following development or sufficient close spaced diamond drilling will a Mineral Resource be classified as Measured.

As development is implemented every face or development round (3.3m Spacing) is visited by SGM geological personnel to map the location of the major contacted and structures exposed by the development. This informaiton is critical in ensuring development is in the correct location and also to provide the detailed geological informaiton required for final stope extraction and stope tonnes and greade determination. Sludge Sampling programs are completed only where there is a requirement to gain additonal geological information beyond that already available. Drilling is completed in fans of holes drilled up from the development locations (a typical fan is shown in Figure 28). These fans are only completed as required but may be as close as 8 - 10m along strike.

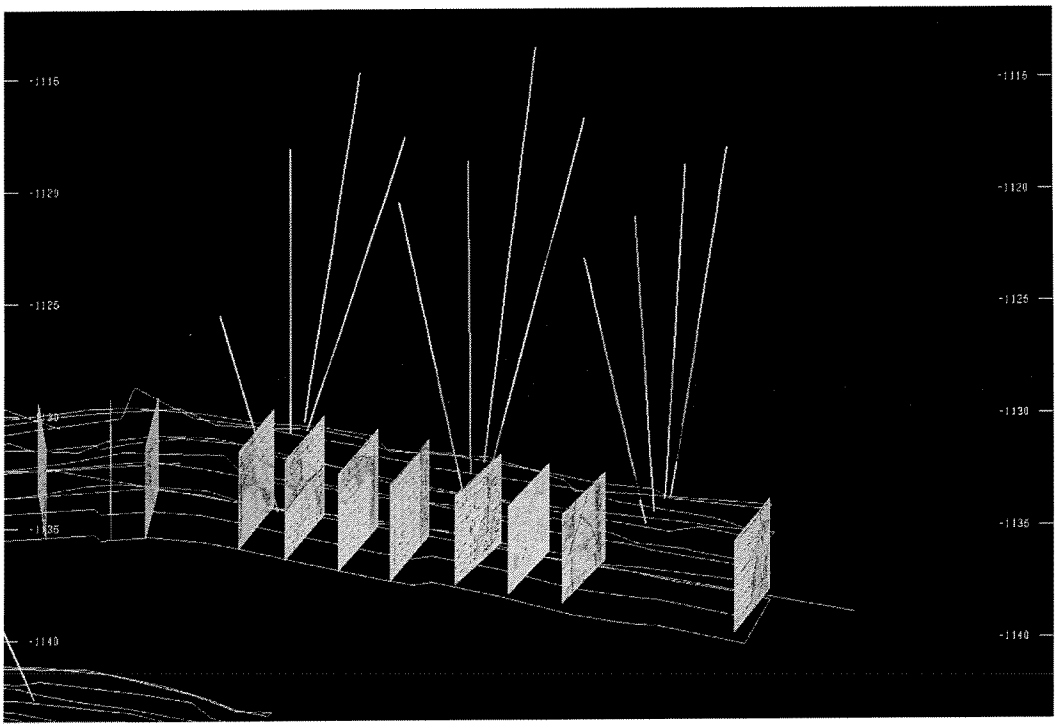


Figure 28. Typical sludge drilling fan completed from an ore development drive. The drive outline is shown as well as the geological mapping collected from each face location. The uphole drill fans are coloured by logged geology, green = potentially mineralised volcanogenics, yellow = Basalt.

13.5 Drillhole Orientation

Where possible drilling is oriented as perpendicular to the structures being tested as possible. The nature of the mineralisation at Stawell and the availability of

suitable drill platforms in the underground environment will always result in compromises in the ability to obtain near perpendicular tests of the mineralisation. An example of the orientations of the drillholes through the GG5 Lower Mineral Resource area is shown in Figure 29. Similar drill hole orientations relative to the strike and dip of the structure exist in many of the Mineral Resource areas.

An exception to this is the Big Hill Resource area where drilling has been conducted on a regular 20 metre along strike intervals with the drillholes oriented perpendicular to the strike and dip of the main mineralisation system with an up and down dip spacing of 25 m.

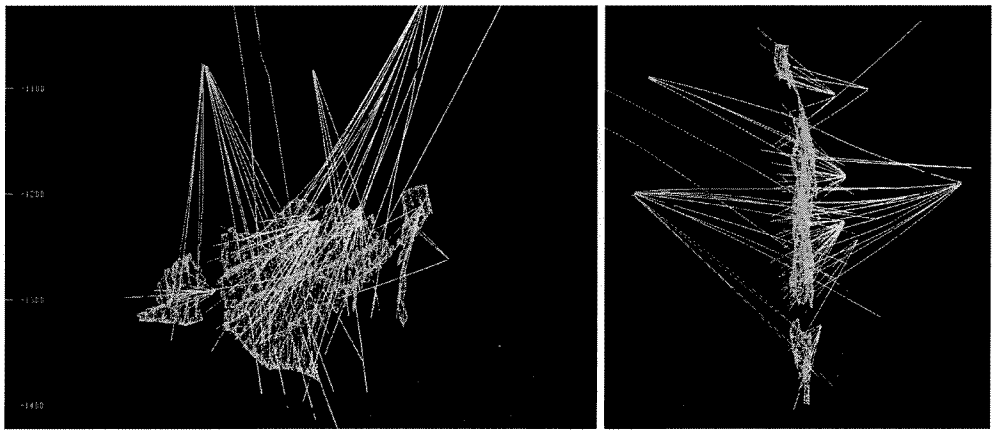


Figure 29 Perspective view (left) and plan view (right) of the GG5 Lower mineralised domain and all underground diamond drillholes used to constrain the most recent Mineral Resource Estimate as documented in Section 19.

13.6 Collar Survey Control

All survey control for the underground drill programs is established by SGM survey personnel. Survey control points are maintained in the underground decline by SGM Survey personnel and these locations provide the control for all mark out and pick-up surveying that is conducted in the underground environment. On conclusion of drilling and hole grouting diamond drilling personnel will insert wooden wedge labelled with the drill hole ID into the collar of the hole. This provides permanent identification of the drillhole collar to ensure matching of surveying information to the correct drill hole collars. The collar survey information is entered in the SGM database by data managers. An example of the information supplied by SGM

surveyors and the check list utilised to ensure appropriate information is collated is shown in Figure 30

The co ordinate system in use at Stawell is a modified version of AMG which is discussed in section 6.4. All survey data pertaining to the mining operation is stored in this co ordinate system.

p12752702 for loading.dat 12/03/2008

Drill Hole Summary

Azimuth(deg)	Dip(deg)	Northing	Easting	RL	Hole ID
		897956.419	659091.982	-1281.767	5244
		897956.441	659092.292	-1281.446	5245
		897956.281	659092.258	-1282.518	5246
		897955.491	659093.022	-1283.542	5247
		897955.396	659093.657	-1284.002	5248
		897955.731	659093.508	-1282.384	5249
		897955.433	659093.923	-1283.102	5250
		897955.561	659093.902	-1282.575	5251
		897955.494	659094.208	-1283.307	5252

Checklist

Designs Checked?

Checked if already picked up?

Collars Loaded - Date / /2007

DHS Loaded?

Drilling Data entry Excel sheet Updated?

Survey Comparision report done?

Recieved Survey Comparision report back from Geo?

DHS Log Sheets changed in folder?

Figure 30 Collar survey information and drillhole survey information checklist.

13.7 Downhole Survey Control

Downhole survey control is managed by utilising down hole cameras to survey the drillhole path. Electronic single shot instruments (REFLEX® and RANGER® tools) have progressively been used in preference to the Eastman® mechanical cameras since 2002 at Stawell as shown in Figure 31, and the vast majority of the downhole surveys of diamond drillholes that are utilised in the estimation of the Mineral Resource estimate detailed in this report have been made with Electronic single shot cameras. Some of the deeper surface diamond drillholes have been surveyed using a North Seeking Gyro instrument.

Downhole survey instruments routinely measure azimuth relative to magnetic north and declination (dip) relative to the horizontal. A correction is applied to convert

Magnetic North to Grid North. The details of how this correction is currently applied is shown in Figure 32.

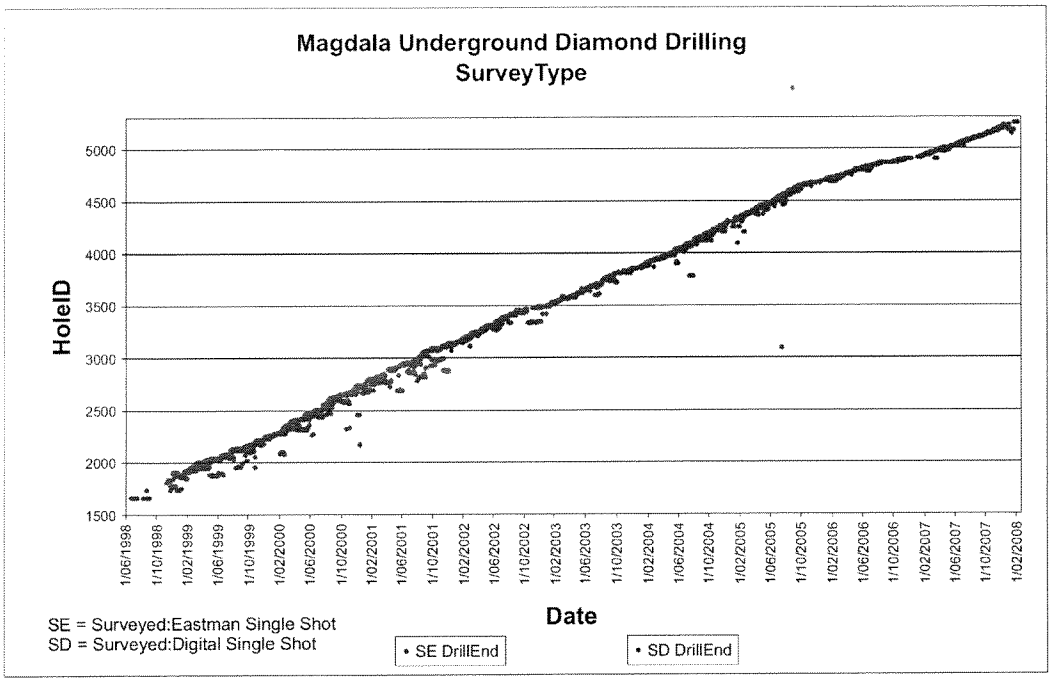


Figure 31 Plot of drillhole survey method (SE = Eastman Single Shot in Purple, SD = Electronic survey instrument in Blue) by time. Post the start of 2002 the standard survey instrument used has been an electronic single shot downhole survey tool.

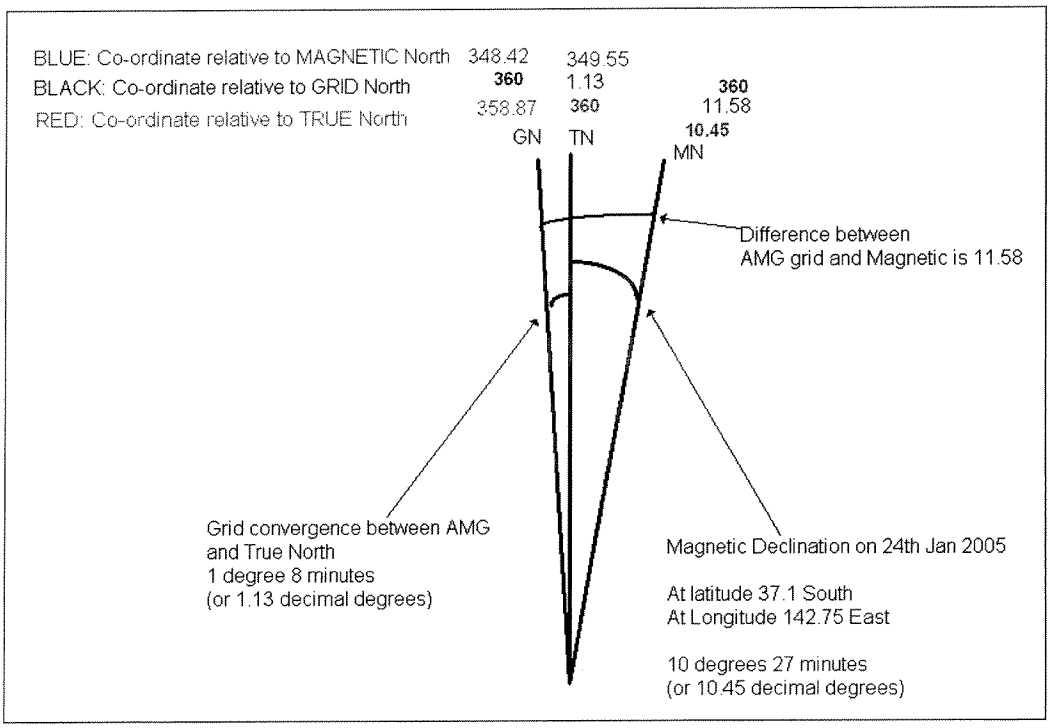
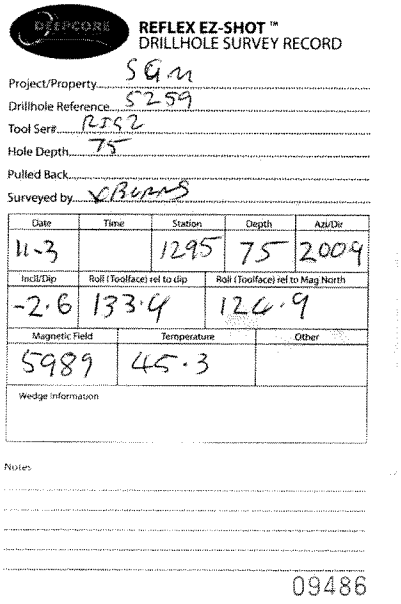


Figure 32 Magnetic declination correctiosn as currently applied to SGM drillhole data.

Contract drilling personnel are responsible for providing survey information at pre-determinated spacings down the hole. The first survey is taken at 15.0m downhole and is efectively used as the collar survey. This depth is used as it reduces the influence of magentics associated with the drill rig and associated equipment. Subsequent surveys are takern at 30m spacings or where deemed neccesary by the supervising geologist closer intervals. The contract personnel record survey details on the daily drilling record sheet and also on a separate survey record sheet (Figure 33) from which the information is entered to the acQuire database system. The electronic instruments provide a direct reading of the magnetic field intensity at the survey locations. This reading can be used to determine if survey readings have been influenced by magnetic material downhole.



REFLEX EZ-SHOT ™
DRILLHOLE SURVEY RECORD

Project/Property: SGM
Drillhole Reference: S259
Tool Ser#: RIS2
Hole Depth: 75
Pulled Back:
Surveyed by: K Burns

Date	Time	Station	Depth	Azi/Dir
11-3		1295	75	200.9
Incl/Dip	Roll (Toolface) rel to dip		Roll (Toolface) rel to Mag North	
-2.6	133.4		126.9	
Magnetic Field	Temperature		Other	
5989	45.3			

Wedge Information

Notes

09486

Figure 33 Current survey record sheets as supplied by the Underground diamond drill contract personnel

13.8 Down hole Survey Quality Control

Several quality control and quality assurance processes are in place to ensure that appropriate survey (downhole and collar) information is stored to the database. Apart from the Database Managers checklist as shown in Figure 30 a review sheet

for the downhole survey information is provided to the responsible geologists such that this information can be validated and where required adjustemnts made to the survey information. A copy of this sheet is shown in (Table 7).

For longer drillhole traces the survey information is plotted to provide a graphical review of the information method is utilised where adjustments to the survey information can be made using the overal trend of the drillhole trace, Figure 34.

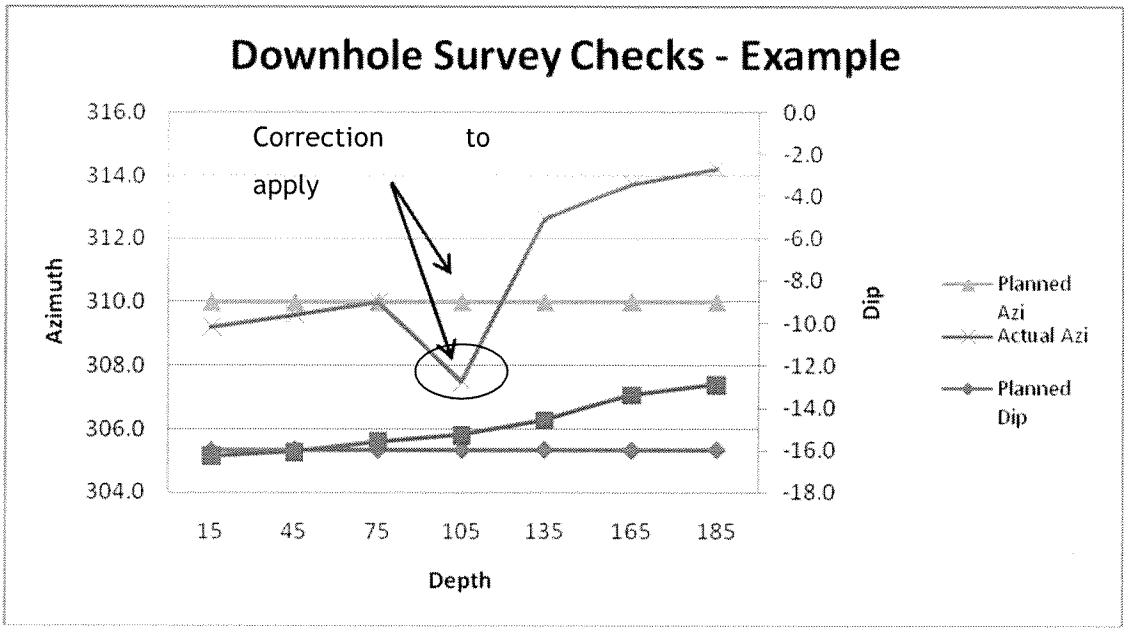


Figure 34 Example of the check plots used to correct downhole survey information.

Where clear issues have been identified with the validity of the survey information and adjusted surveys enetred the original surveys are given a lower priority in the database system. A record of survey methods and or adjustments are maintained in the main acQuire database as part of the audit trail.

Stawell Gold Mines personnel utilise a survey camera test bed with known azimuth and dip for for routine checking of downhole survey cameras. This test bed (Figure 35) is located on the surface well away from any potential magnetic sources and is utilised by contract drilling personnel to routinely check camera performance and determine if equipment requires servicing or re-calibration.

Report Date 24/07/2007

HoleID	Drill Platform	Surveyor collar_dip diff >1.5° highlighted	Surveyor collar_azimuth diff >3° highlighted	Depth	Camera Dip	Camera Azimut	Btot outside range 60.1 to 60.6 highlighted	Surv Type	Design Dip	Design Az
MD5045	1225 Egress			15.00	-60.80	192.20	60.34	SD	-61.4	192.7
				45.00	-60.00	192.60	60.04	SD		
				75.00	-59.40	193.70	60.05	SD		
				105.00	-58.60	195.40	59.38	SD		
				135.00	-58.10	196.10	59.78	SD		
MD5046	1225 Egress			15.00	-13.00	188.60	59.6	SD	-12	189.7
				45.00	-13.30	189.30	59.9	SD		
				75.00	-13.30	186.10	59.79	SD		
MD5047	1225 Egress			15.00	-65.40	185.40	60.02	SD	-64.6	186.2
				45.00	-65.00	185.60	59.79	SD		
				75.00	-64.50	186.10	60.01	SD		
				105.00	-64.10	186.40	59.68	SD		
				135.00	-61.70	189.60	59.78	SD		
				155.00	-61.50	189.40	59.89	SD		
MD5048	1225 Egress			15.00	-40.00	186.40	60.05	SD	-40.3	185.1
				45.00	-39.80	186.60	60.04	SD		
				75.00	-39.50	187.50	60.04	SD		
				95.00	-39.10	191.30	58.59	SD		
				111.00	-38.50	189.00	59.86	SD		

Table 7 An example of drillhole collar survey information provided by SGM survey personnel and the check list used the database manager to ensure appropriate information is loaded to the database. Note some corrections to erroneous data have been made as indicated by the penciled changes.

Figure 35 Photograph of the SGM survey camera test bench

14 Sampling Method and Approach

14.1 Diamond Drillcore Processing

All diamond drill core is delivered to the Stawell Gold mines core processing facility by the diamond drill contractor. Diamond drill core is washed to remove grease and individual core trays photographed in a light controlled installation (Figure 36) prior to laying out on benches ready for Logging by the site geologists. As part of the standard SGM geological procedures all core collected from diamond drill holes are photographed and a complete record of digital core photographs is available to assist in the geological interpretation process.

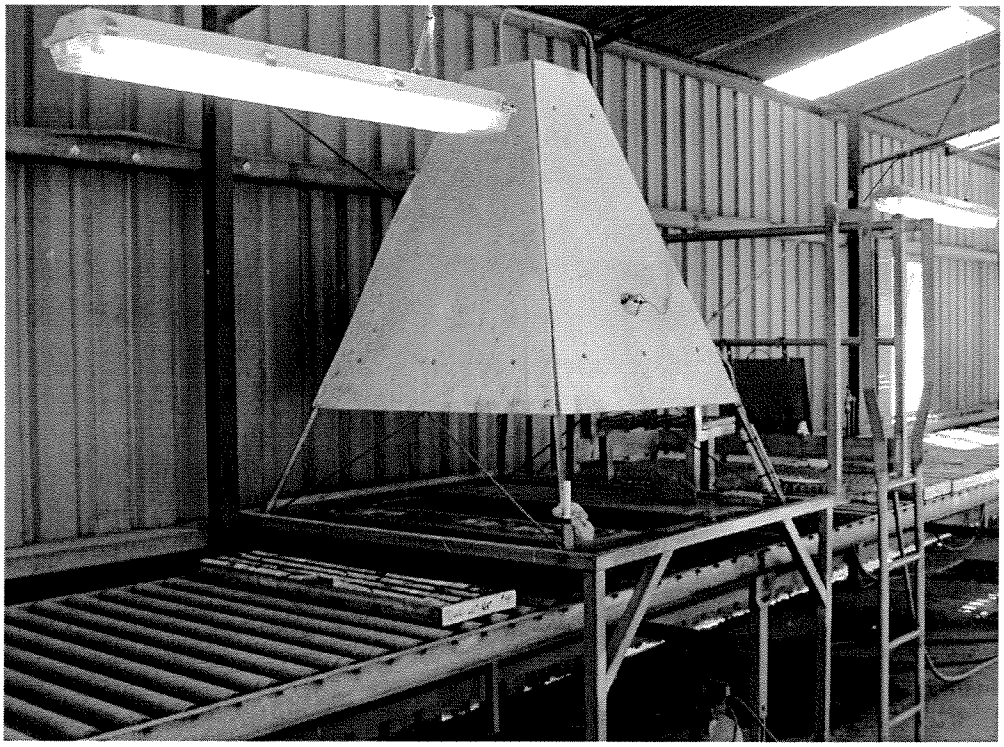


Figure 36 Stawell Gold Mines core photography installation.

A detailed flow sheet of core processing activities is shown in Figure 37. Prior to drillhole number MD2678 and SD607 the core photography was taken on film and stored as prints.

An example of the style of core photography that is available for all diamond drilling is shown in Figure 38.

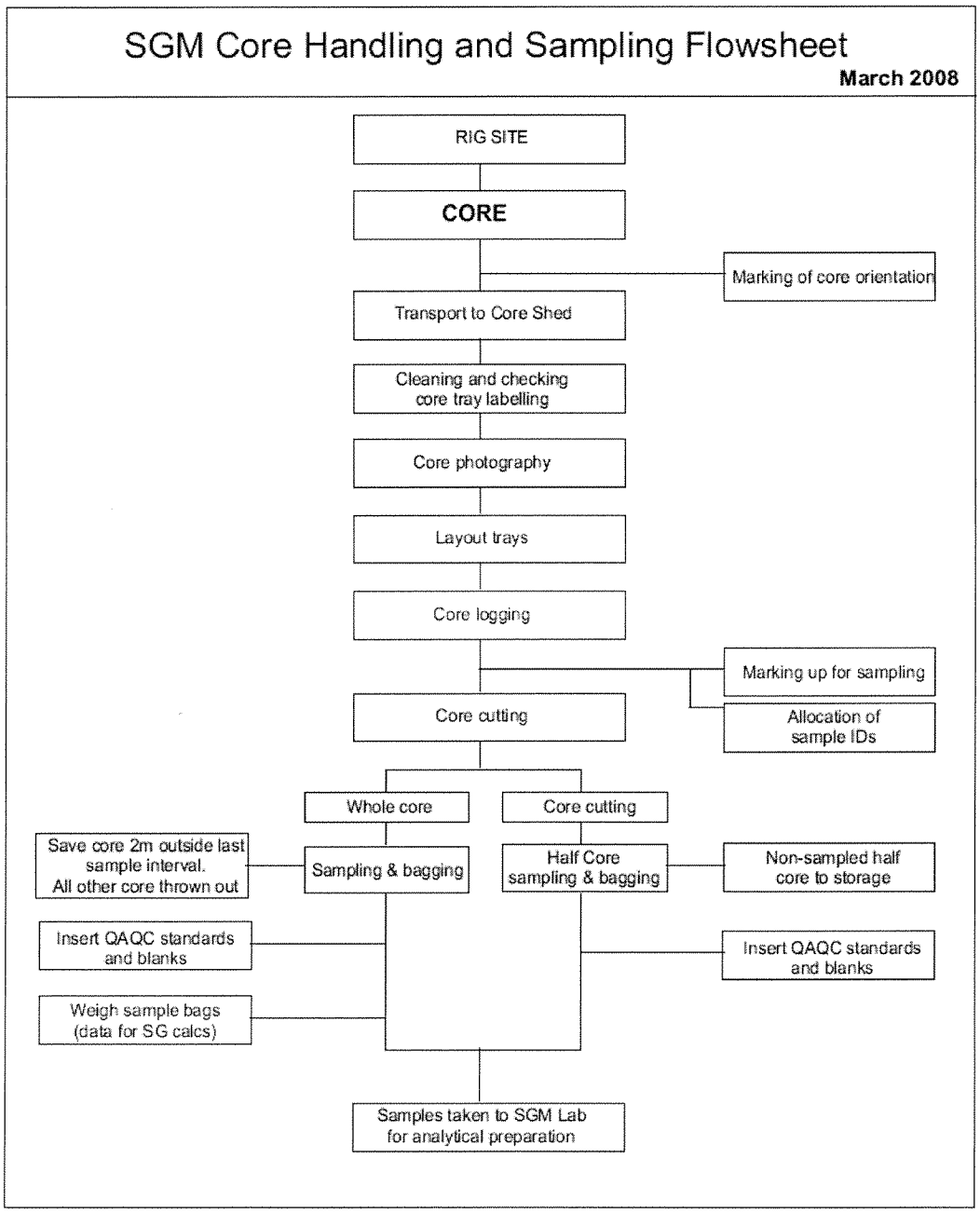


Figure 37 Diamond Drillcore processing operations

Figure 38 An example of the diamond drillcore photographs stored digitally for all diamond drillcore. This section of core belongs to Drillhole MD5063 which tests the GG5 Lower Mineral resource area.

14.2 Logging

All diamond drill core is logged by the site geological teams using a standardised logging legend. The data is captured electronically at the point of collection using a barcode logging "Datcol" software system. This system was developed on site in the mid 1990's and has remained the standard process since that time with the key tables for lithology, alteration, and structure and geotechnical information are populated during the logging process.

14.3 Core Recovery

During the logging process any lost core is estimated and logged as lost core with a specific start and end interval.

A review of database for recently drilled holes indicates excpetionally good core recovery throughout the deposit particulalry adjacent to the major mineralised zones. Where core is lost it is usually associated with significant faulting. Lost core is identified in the logging as "LOST" and as such there are very few if any assay

intervals utilised in the Mineral Resource estimate where core recovery is less than 100%.

14.4 Diamond Drillcore Sampling

During the logging process the geologist will mark up the intervals of core for sampling. Not all diamond core is sampled. Historical sampling has identified the key lithological and structural units that will host mineralisation and the selection of units for sampling follows the protocols shown below

- All Magdala Facies, also known as 'Magdala volcanogenics' are sampled for assay.
- A minimum of 2.0 m into the hangingwall and or footwall is sampled
- Fault zones and zones of sulphide are sampled at the geologists discretion
- Magdala Basalt and Albion formation rock units are sampled at the discretion of the logging geologist

Not all diamond core is cut in half prior to sampling. Sampling of diamond drillcore follows one of two methods as detailed below.

- Exploration and Resource Definition - HQ or NQ drill programs
    - Core is logged and geological derived intervals are marked up for sampling
    - Sample intervals are matched to geological boundaries (structural or lithological) and fall within the range of 0.10m - 2.0 m. The average sample interval is approximately 1.0m
    - For Resource Definition drilling programs 1 in 5 holes is cut with a diamond saw prior to sampling and one half of the core sent for assay. The remaining half core is retained as a record within the core library. All other holes are sampled as whole core which is sent for sample preparation and assay as per the flowsheet shown in Figure 40. All holes deemed to be Explorations are ½ core sampled and the entire remaining core retained in storage on site at SGM.
- Grade Control diamond drill programs - LTK60
    - Core is logged and geological derived intervals are marked up for sampling

- Sample intervals are matched to geological boundaries (structural or lithological) and fall within the range of 0.10m - 2.0 m. The average sample interval is approximately 1.0m
- For grade control drilling programs drill holes are sampled as whole core with the entire sample sent for sample preparation and assay as per the flowsheet shown in Figure 40.

Detailed operating procedures for sampling of diamond drillcore are used at Stawell Gold Mines to ensure uniformity of process and prevent errors.

14.5 Diamond drillcore Sample Intervals

During the 2007 Calender year a total of 39,361.8 metres of diamond core was poduced of which 8918.6 metres was sampled for assay.

The sample interval statistics for the 2007 calendar year are shown in Table 8 and a histogram of the sample intervals for 2007 shown in Figure 29. The average sample intervals demonstrated by the 2007 data is indicative of complete data set utilised to estimate Mineral Resource at Stawell.

Average Diamond Core Sample Length (m)	0.93
Minimum Sample Length (m)	0.1
Maximum Sample Length (m)	2.1
Total Sampled metres	8989.6

Table 8 Diamond drill core sample interval statistics for samples taken during the calendar year 2007.

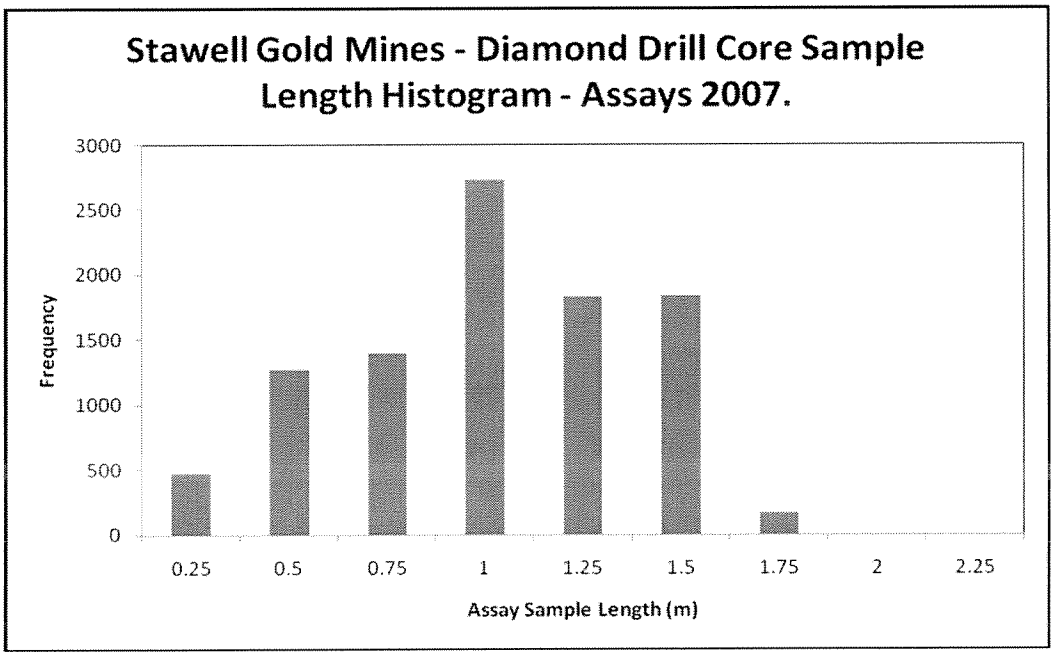


Figure 39. Histogram of sample intervals for diamond drillcore taken during the calendar year 2007.

14.6 RC Sampling

Specific details of the RC sampling methodologies for the Magdala Surface Mineral Resource estimate are outline in the documentation in Appendix B. A summary of the RC sampling protocols that were utilised for this program and for other RC drilling that has been conducted on site are given below;

- All SGM RC sampling was carried out using the following protocol
    - Generally the entire hole was sampled from the collar unless it was recognized as recent fill or material associated with the construction of the drill pads
    - Samples were collected at 1.0 m sampling interval, bit pulled back and flushed between intervals
    - Samples discharged into tightly fitting plastic sample bag from cyclone
    - Sample transferred to a rectangular plastic tub the same size as the splitter
    - Sample tipped into three tier splitter from plastic tub to ensure equal quantities available to all vanes

- 1/12 subsplit (nominally 3kg) collected in calico sample bag, tied and placed in lots of five into plastic bags
- Residue sample collected in original sample bag and transported to bag farm for storage
- Splitter cleaned by shaking/banging/brush/air compressor as necessary between samples
- Cyclone cleaned at regular intervals (completion of each hole minimum) by banging/checked by hand/arm
- Every 20^{th} sample resplit to give a 2^{nd} 3kg sample, field splits dispatched along with first splits
- Samples dispatched to Laboratory for analysis by Fire Assay. In the case of the Big Hill Mineral resource estimate all assaying was conducted at Aminya Laboratories Ballarat.

14.7 Reliability of Samples

It is the opinion of the Qualified Person that the drilling and sampling methodologies employed by Stawell Gold mines are of a high standard and provided representative tests of the ore body for the estimation of Mineral Resources. Standard drill spacing's adopted by Stawell Gold Mines are appropriate for the stages of Mineral Resource development and whilst other factors contribute to decisions regarding classification of the Mineral Resources the drill spacing's discussed in this section enable appropriate geological interpretation and Mineral Resource classification decisions to be made.

15 Sample Preparation, Analysis and Security

The Qualified Person has reviewed the sample preparation, assay and sample security processes utilised at Stawell Gold Mines a summary of which is included in this section. All sampling and sample preparation is currently completed by employees of Stawell Gold Mines, the issuer of this report. The employees follow appropriate written procedures as documented in Sections 14 and 15 of this report.

15.1 Assay Laboratories

During the life of the Stawell Gold Mines a number of laboratories have been utilised for routine assaying of diamond drillcore and RC samples. The details of the laboratories and the periods for which assaying has been conducted are as follows;

- **SGM site laboratory**. Utilised intermittently prior to 1995 for assaying of diamond drill core and RC samples
    - Non accredited company assay laboratory
    - Assay method was 10g Aqua Regia with pre digest roasting and AA finish.
    - Assaying of diamond drillcore and RC samples was discontinued in 1995 and the laboratory sample preparation and assay methods updated to industry standard practice.
    - Now utilise a 25 g Aqua Regia method with pre digest roasting and AA finish
    - Utilised for Metallurgical assaying, UG Face sample and other geological grade control sampling
    - Undertake routine sample preparation of diamond drillcore samples - post 2004.
- **WMC Ballarat Assay** laboratory utilised prior to 1995 for assaying of diamond drillcore and RC samples
    - Non accredited company assay laboratory
    - Assay method was 10g Aqua Regia with pre digest roasting and AA finish.
    - Assaying of diamond drillcore and RC samples was discontinued in 1995 and the laboratory sample preparation and assay methods updated to industry standard practice.
- **AMDEL Laboratories - Adelaide** SA
    - ISO 9001 accredited
    - utilised intermittently from 1995 through to present day
    - Primary supplier of assay services during from 2004 - mid 2007
    - Ongoing utilization for check assays
    - SGM reduced reliance on AMDEL mid 2007 as a result of very slow turnaround of assays results
- **AMINYA Laboratories - Ballarat VIC**
    - Not accredited

    - Primary supplier of Assay services for Diamond drillcore and RC samples period 1995 - 2004
    - Discontinued in 2004
- **Genalysis Laboratory Services - Perth WA**
    - Genalysis is a NATA accredited laboratory to ISO 17025.
    - Provider of assay services for the period XXXX - XXXX
- **ALS Laboratory Group - Orange NSW**
    - ALS is accredited to ISO 9001 and ISO 17025
    - Primary provider of assay services August 2007 to present day

15.2 Sample Preparation

The sample preparation protocol for diamond drillcore is shown in the flowsheet given in Figure 40. This sample preparation flowsheet was developed in 1995 and has been in operation for all Stawell Gold Mines diamond core and RC samples since that time.

During the period 1995 to 2004 all sample preparation was conducted by the assay laboratory facilities as detailed in section 15.1. In 2004 it was decided by site personnel to complete this task on site at the Stawell Gold Mines laboratory facility. The sample preparation follows the same process utilising modern sample preparation equipment.

- Primary Crushing to 75% passing 2.0 mm using a Boyd Crusher
- Splitting using a vibrating feed cone splitter
- Pulverising to 95% passing 75um using Labtechnics LM5 pulversing mills

By retaining responsibility for this work through the existing site based facility SGM has flexibility in sending the pulps only to a variety of assaying laboratories and also retain the coarse rejects on site for ongoing metallurgical testwork programs.

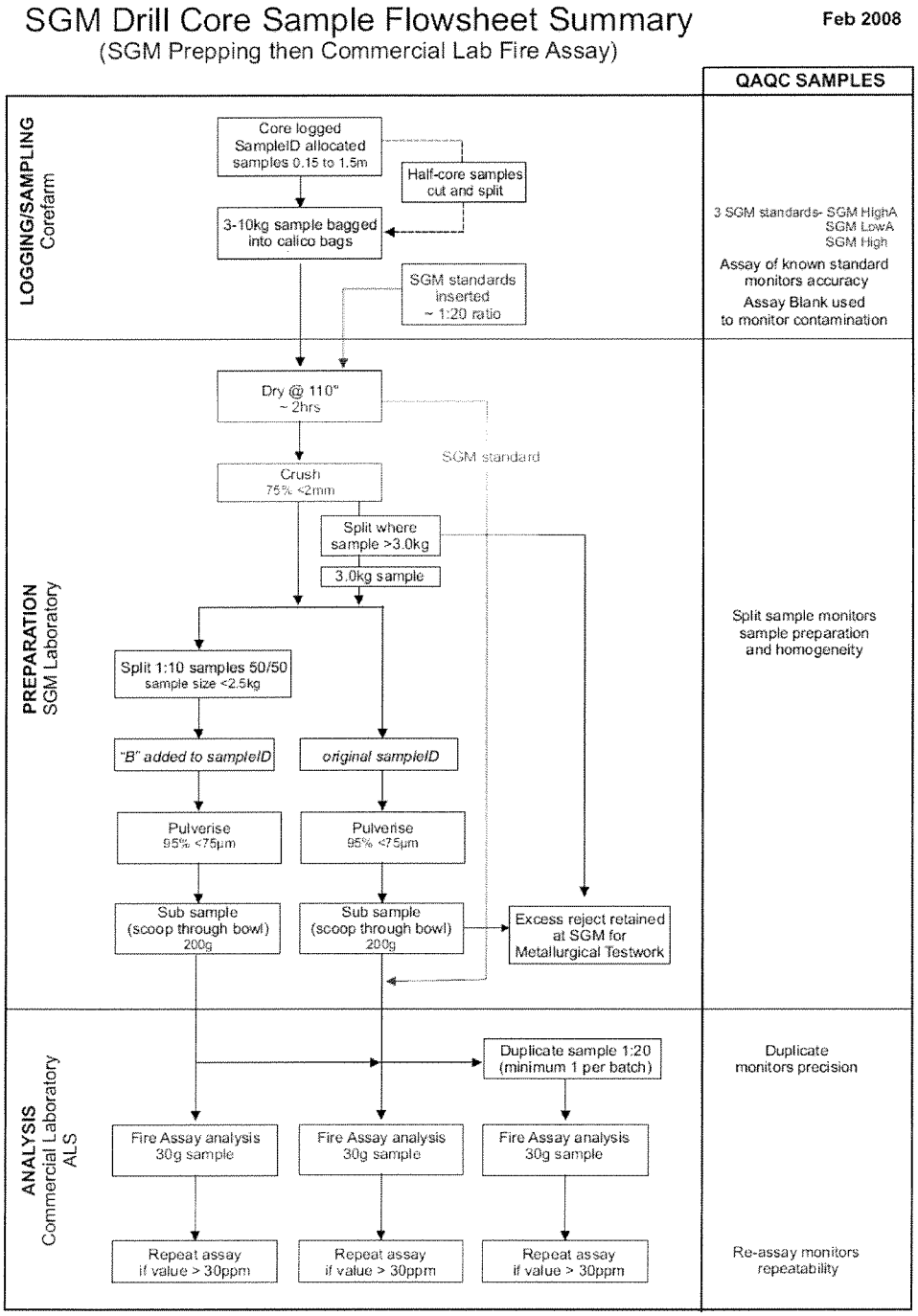


Figure 40 Stawell Gold Mines drill core sample preparation, assay and QA/QC flowsheet.

15.3 Sample transport and Security

Security of drillcore and samples is managed by maintaining records throughout the complete process from drilling, core processing, logging, sampling, sample preparation and assaying through to return of results.

Key record keeping utilised in managing sample and data security are;

- Daily drilling records are entered to the database which provide records of drillcore produced.
- Core is photographed within 24 hours of being delivered to the core processing facility.
- The Stawell Gold Mines sample processing facility is located on the Mine Lease within a security fenced area. All core stored here is only able to be accessed by SGM personnel.
- At the conclusion of logging a sample requisition sheet is generated listing sample numbers, assay standard insertion and assay requirements. This is loaded directly to the acQuire database enabling tracking of samples after this process.
- SGM personnel are trained in appropriate procedures for logging and sampling of the diamond drillcore and generate an Analytical Request sheet sheet outlining sample ID and assay requirements.
- The production of carefully labelled sample pulps for dispatch by registered posts.

The pulps are sent from the SGM prep laboratory to the assay laboratories using registered post. Consignments travelling by registered post are required to be signed off by each leg of the postage route on arrival and can be tracked online. The assay laboratories are also required to send a statement informing SGM that the pulps have arrived and that the samples as detailed on the analytical request sheet can be accounted for.

15.4 Assay Methods

A summary of the laboratory methods utilised by the various laboratories is given in following Table 9. All assaying for gold that are utilised in the Mineral Resource

estimates have been completed by Fire Assay method (30 - 50g charge weights) with AAS finish.

Laboratory	Lab Method Code	Description	Limit of detection (ppm)
Aminya	PE01S	50g Fire Assay with AAS finish	0.01
AMDEL	FA1	40g Fire Assay with AAS finish	0.01
Genalysis	FA25 AAS	25g Fire Assay with AAS finish **(used for repeats only)**	0.01
Genalysis	FA50 AAS	50g Fire Assay with AAS finish **(Standard method used)**	0.01
ALS	Au-AA26	50g Fire Assay with AAS finish	0.01
ALS	Au-AA25	30g Fire Assay with AAS finish - **Used post January 2008**	0.01

Table 9 Laboratory assay method codes, descriptions and limits of detection.

For samples reporting below LLD, a value of 0.5xLLD is utilised as standard in resource estimation.

15.5 Database Storage and Integrity

All SGM drilling data is stored within the "acQuire" Database Management System. The database operates in an SQL Server framework and data security is established by having various levels of user access rights. SGM maintain a security access system where loading and manipulation of data is only conducted by one of two data managers. All geological personnel have access to the database for read only purposes.

Data validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth

- Checks are performed for overlapping samples
- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sample ID created during logging procedure.

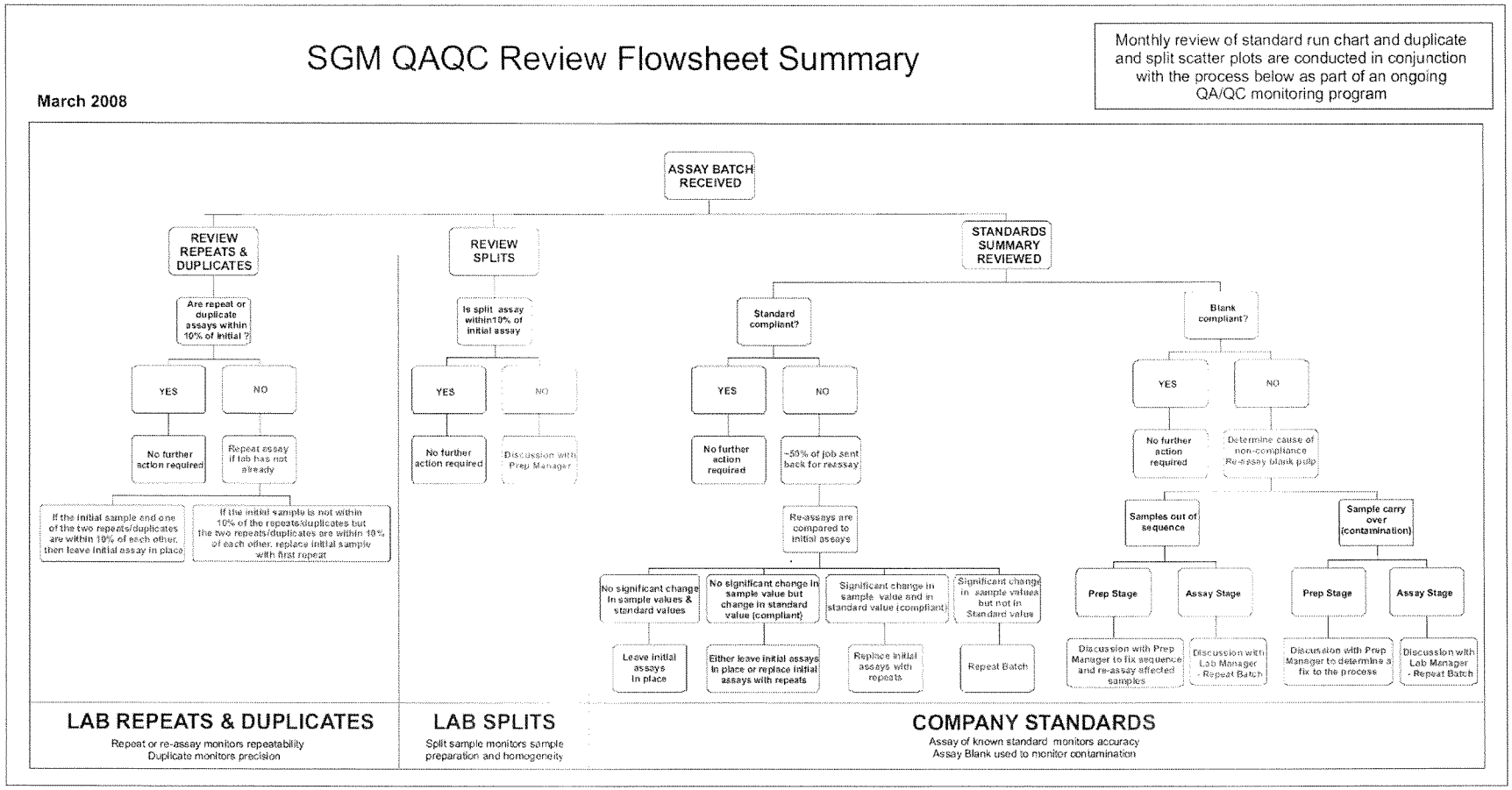


Figure 41 Stawell Gold Mines QAQC review and actions flow sheet

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (e.g. MineSight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted.

After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

15.6 Stawell Gold Mines Assay QA/QC Process

The general flowsheet for the sample preparation and assaying including the Quality Assurance and Quality Control samples submitted to ensure this compliance is shown in Figure 40.
A range of checks and resulting actions are in place to monitor the QA/QC of the Stawell data set as set out in the QA/QC flowsheet shown in Figure *41*.

When monitoring these checks the following guidelines are followed:

- The first checks are regular submission of standards and blanks at the sampling stage
    - If the standard assay is outside ±4 standard deviations from the known value then ~50% of that batch is automatically repeated. Entry is made into the SGM QA/QC diary to indicate which sample IDs have been repeated and the results of the re-assay
    - If the blanks are not compliant checks are initiated to determine the cause of the non compliance and what remedial action is required
- This is followed by the use of splits of the crushed stage
    - If the splits are outside of 10% of the initial assay investigations into the quality of the sample preparation for the batch is conducted
- The final check is the quality of the repeats and duplicates
    - Where repeats or duplicates are outside of 10% of the original sample additional repeats will be requested to determine if it is a laboratory issues or associated with coarse gold within the sample

- Repeated assays are compared to initial assays with any action to be taken entered into the SGM QA/QC diary to indicate what actions respond to Batch and sample IDs.

- Monitoring of these checks is done within two days of the sample batch return and actioned generally no later than seven days after the return date.
- Any actions taken during the monitoring process are recorded in the SGM QA/QC diary which was set up in September 2007 (Table 11).

15.6.1 SGM Standard Reference Material

SGM Standard reference material has been prepared and certified by Ore Research of Melbourne from samples of ore from the Magdala Mine. The details of the standards and when they were introduced to the system are shown in Table 10. Certification certificates are available in the SGM records.

Standard ID	Start/End Date Used	Gating Values			Comment
		Lower Limit 1σ	Recommended Value	Upper Limit σ1	
SGM Low A	June Ongoing	3.18	3.34	3.50	
SGM Low B	June 2001 2001/Sept2004		3.54		Discontinued - No sample left
SGM High A	June 2001 Ongoing	4.04	4.20	4.36	
SGM High B	June 2001/Sept 2004		4.54		Discontinued - No sample left
SGM High	New Standard available	9.30	9.36	9.42	

Table 10. SGM assay standards and certified values reported by ORE

Stawell Gold Mines

QAQC Request Record

Mine lease U/G drilling

Date Re-assay request	Requested by	HoleID	Orebody	Non-compliant sampleID	Lab Name	Lab Job No.	Repeated samples	Date Re-assays reported	Repeats Lab Job No.	Date loaded to DB	Reviewed by	comments	Action requested	Actioned by
?	DH -> JC	MD5062		MD5062-022/022B	ALS	OR07093156	MD5062-020 to -023	06-Sep-07	OR07098050	06-Sep-07		diff between sample and split 2.03/26.3		
18/09/2007	DH -> JC	MD5090		MD5090-042	ALS	OR07099872	MD5090-034 to -047	26-Sep-07	OR07107140	05-Oct-07	DH	non-compliant standard	Change database values to re-assay	CD
?	DH -> JC	MD5089		MD5089-021	ALS	OR07102944	MD5089-016 to -028	27-Sep-07	OR07107141	05-Oct-07	DH	non-compliant standard	Change database values to re-assay	CD
12/10/2007	DH -> JC	MD5116		MD5116-008	ALS	OR07113517	MD5116-004 to -011	21-Nov-07	OR07109857	26-Nov-07		non-compliant standard		
12/10/2007	DH -> JC	MD5116		MD5116-053	ALS	OR07113517	MD5116-050 to -056	21-Nov-07	OR07109857	26-Nov-07	DH	non-compliant standard	Change database values to re-assay	CD
12/10/2007	DH -> JC	MD5111		MD5111-012	ALS	OR07112313	MD5111-005 to -015	21-Nov-07	OR07109858	26-Nov-07	DH	non-compliant standard	Change database values to re-assay	CD
22/10/2007	DH -> JC	MD5100		MD5100-021	ALS	OR07106068	MD5100-015 to -030	21-Nov-07	OR07116775	26-Nov-07	DH	non-compliant standard. Assays repeated MD5096-014 to -028	Change database values to re-assay	CD
23/10/2007	DH -> JC	MD5122		MD5122-020	ALS	OR07117694	MD5122-015 to -024	26-Nov-07	OR07116776	26-Nov-07	DH	Assays repeated MD5110-014 to -022	Leave first assay batch as priority 1	
24/10/2007	DH -> JC	MD5124		MD5124-018	ALS	OR07116740	MD5124-008 to -019	21-Nov-07	OR07116777	26-Nov-07	DH	Assays repeated MD5114-008 to -019	Leave first assay batch as priority 1	
05/11/2007	DH -> JC	MD5132		MD5132-021	ALS	OR07123244	MD5132-001 to -068	21-Nov-07	OR07129002	26-Nov-07	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5142		MD5142-028	ALS	OR07139007	MD5142-017 to -030	17-Dec-07	OR07143758	09-Jan-08	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5152		MD5152-015	ALS	OR07139007	MD5152-003 to -017	17-Dec-07	OR07143758	09-Jan-08	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5158		MD5158-019	ALS	OR07139007	MD5158-015 to -024	17-Dec-07	OR07143758	09-Jan-08	DH		Leave first assay batch as priority 1	--
03/12/2007	DH -> JC	MD5145		MD5145-022	ALS	OR07139008	MD5145-017 to -023	17-Dec-07	OR07143757	09-Jan-08	DH	holeID is MD5145 but samples for repeat are MD5143-017 to -023	Leave first assay batch as priority 1	
03/12/2007	DH -> JC	MD5153		MD5153-021	ALS	OR07139008	MD5153-016 to- 028	17-Dec-07	OR07143757	09-Jan-08	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5135		L039800	ALS	OR07135650	L039795 to L039809	17-Dec-07	OR07143759	09-Jan-08	DH		Change database values to re-assay	
05/12/2007	DH -> JC	WD164		L039362	ALS	OR07127816	L039358 to L039370							
05/12/2007	DH -> JC	WD119		L039423	ALS	OR07127816	L039418 to L039430							
05/12/2007	DH -> JC	MD5131		MD5131-012	ALS	OR07127621	MD5131-10 to -028							
13/12/2007	DH -> JC	MD5155		MD5155-042	ALS	OR07142500	MD5155-033 to -045	18-Dec-07	OR07147701	09-Jan-08	DH		Change database values to re-assay	CD

Table 11 QA/QC diary entries for 2007

15.7 Assay QA/QC Calendar year 2007

For this report an analysis of the QA/QC data returned for the period 1st January 2007 to 31st December 2007 has been completed. It is possible to complete this analysis for all data back to 2001 however the bulk of the mineral Resource estimate is supported by the most recent assaying.

For details of QA/QC results for the Big Hill surface Mineral Resource refer to documentation for this model which is given in Appendix B of the main report.

This analysis encompasses all QA/QC data returned to SGM during that period and serves to demonstrate that a responsible and ongoing approach to managing assay data quality is maintained at SGM and that assaying information is of a good quality for Mineral Resource estimation.

As a result of QA/QC processes carried out during 2007, 197 initial assays from a total of 9697 assay returned where replaced by repeat assays as a result of the QA/QC monitoring process.

Figure 42Figure 47 show the results of QA/QC data for calendar year 2007 data

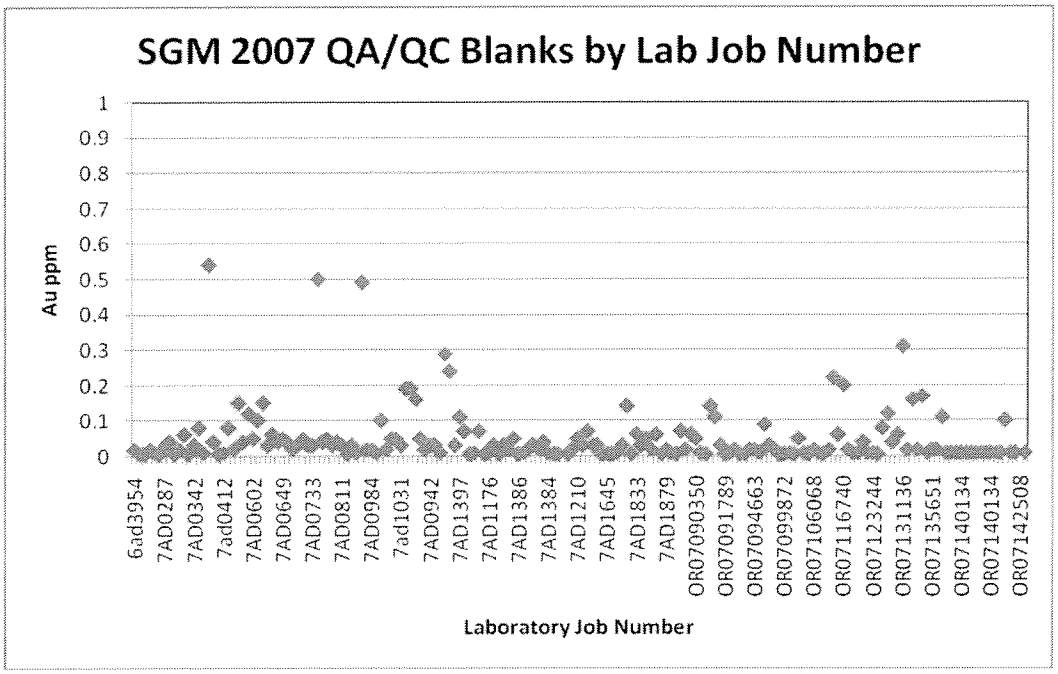


Figure 42 QA/QC plot of Blanks assayed during 2007

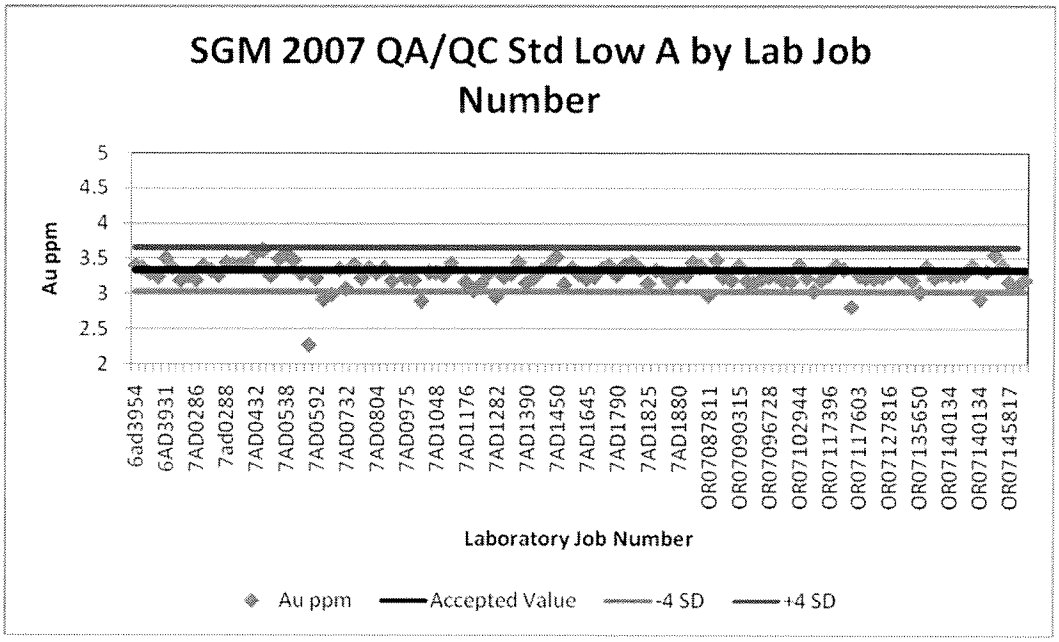


Figure 43 SGM Standard Low A by job number for the calendar year 2007

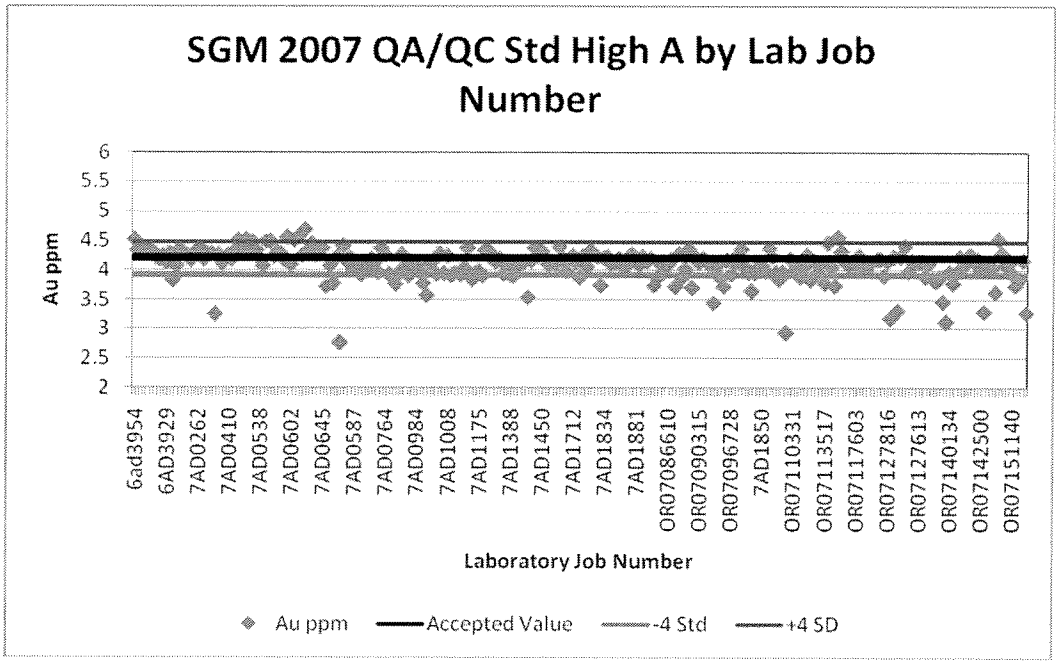


Figure 44 SGM Standard High A by Job number for the calendar year 2007.

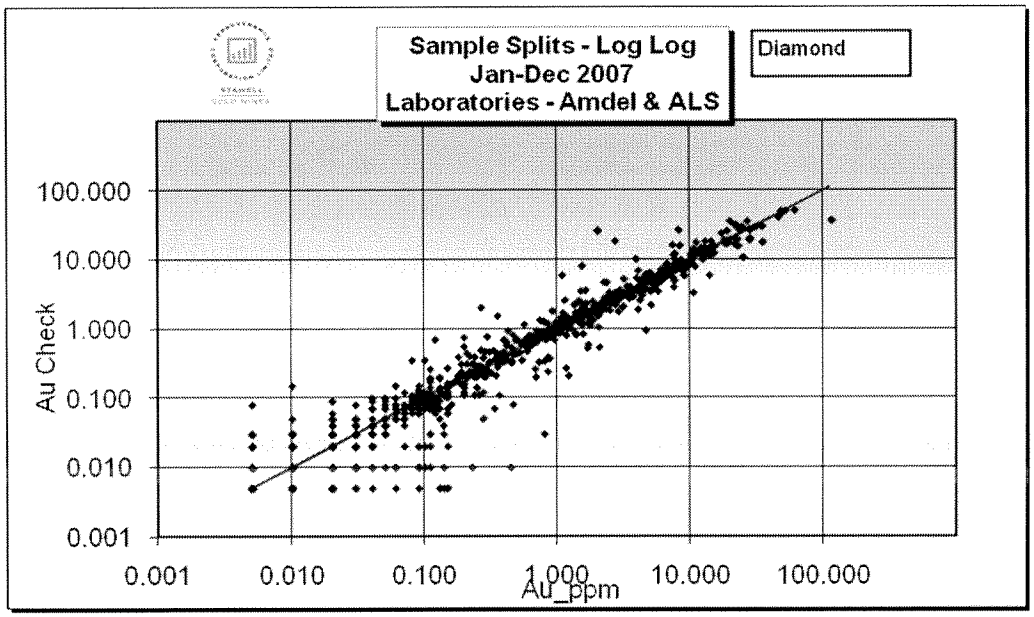


Figure 45 Sample Splits comparison for 2007

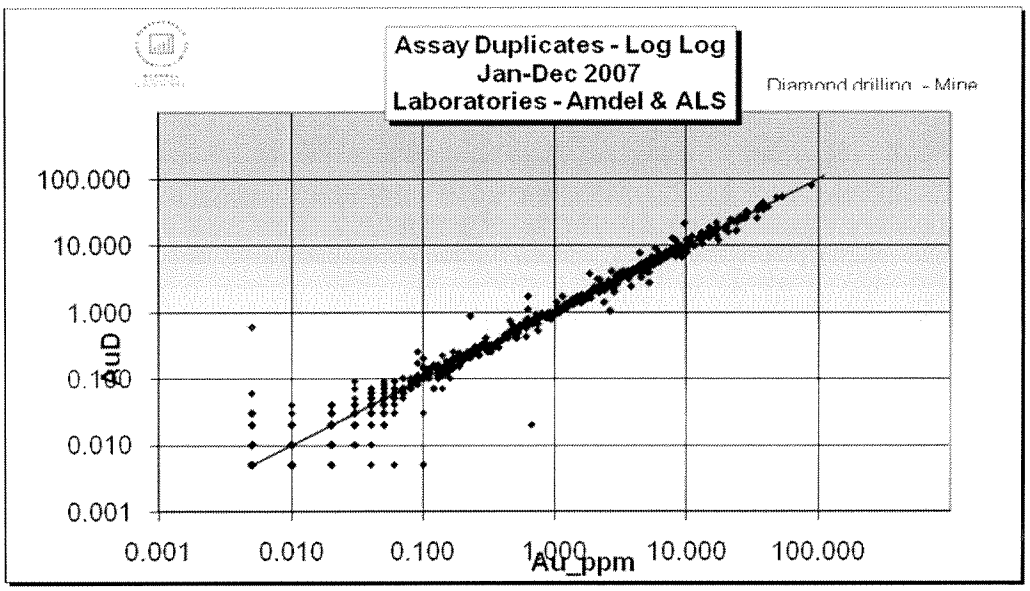


Figure 46 Assay duplicate comparison for 2007

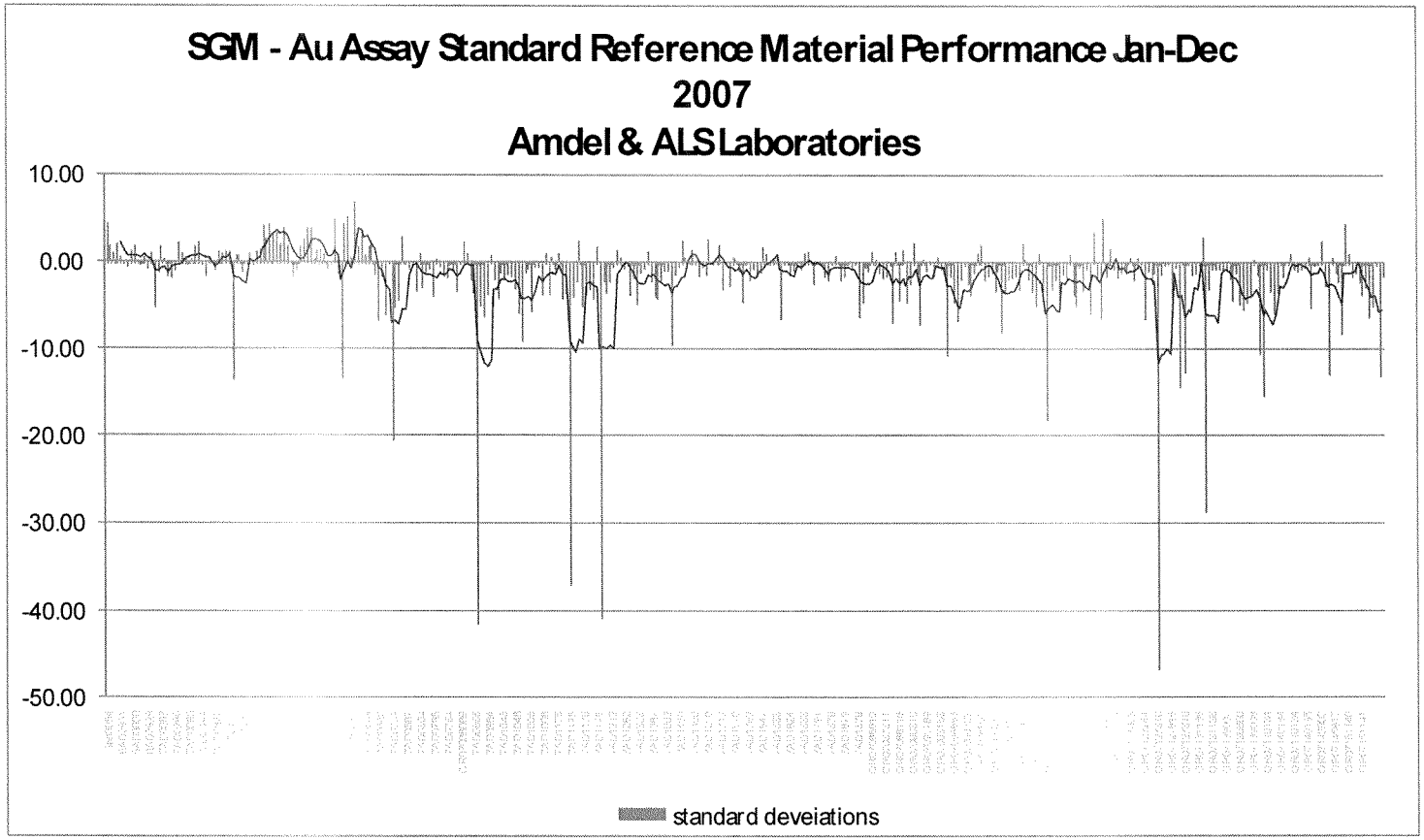


Figure 47 showing 2007 Original (first) Assay Standard performance plotted as standard ±deviations relative to the recommended value

15.7.1 QA/QC Discussion

Two external laboratories were used during 2007 for fire assay of all SGM drill core. These were Amdel in Adelaide SA and ALS laboratories in Orange NSW. During the first half of the year Amdel was used for Stawell fire assay work with 40g charge weights. As demonstrated in Figure 47 cycling can be seen within the results for the combined assay results indicating poor performance with the laboratories.

Limited work was conducted between SGM and Amdel to improve the assaying process. Towards the end of SGM and Amdel's relationship the current SGM QA/QC system was being developed but the turnaround times for samples sent to Amdel in some cases were out to 60 days. This prolonged turnaround time impeded QA/QC interaction but also severely impacted on production requirements. Due to this in August 2007 SGM changed to ALS laboratories in Orange NSW for all fire assay work.

The same cyclic nature to the standard assaying was seen by ALS at first. ALS sample turnaround averaged two weeks which made QA/QC dialog a smoother process. To react to the cyclic nature of the Standard assaying ALS in November 2007 added extra litharge to the flux. The extra litharge was implemented due to concerns that the high sulphide component to the Stawell ore could be interfering with the flux during the fusion process. This proved to have limited success so towards the end of December 2007 ALS changed the charge size from 50g down to 30g as another attempt to combat the high sulphide nature of the Stawell ore. The success of this reduced charge size is currently under review. ALS also highlighted that the Standards were much finer than the sample product and could pose issues in mixing with the flux.

Overall for 2007 based of the original (first) assay received the laboratories have been running at a negative 4% bias. However the QA/QC process identifies batches that require check assays. During 2007 a number of check assays on batches were completed with 10 Standards previously out of specification being replaced with re assays. Following replacement it is estimated that the laboratory bias for 2007 is negative 3% for gold.

In the opinion of the Qualified Person the quality control procedures in place at Stawell Gold Mines including check assays, insertion of Standard reference materials and the results of corrective actions for the data sets utilised to estimate

Mineral Resources are to a standard that provides quality assays for the estimation of Mineral Resources.

Data management procedures and personnel and subsequent data security ensure sufficient rigour is applied to the validation of the data prior to storing in the database and as such downstream estimation of Mineral Resources is utilising data of a high quality.

16 Data Verification

Dean Fredericksen the Qualified Person has personally been involved in the collection and processing of Geological data and information during the course of several engagements previously as a direct employee of the then owners of the operation and recently in the capacity of a consultant to Stawell Gold Mines. As such completed sufficient verification of the sampling, sample preparation and analytical procedures and resulting assay and geological data on which the Mineral Resource estimates are based and is comfortable with the quality of the information available.

To further test the integrity of the information kept within the SGM database, four drill holes were randomly selected from each of two mining areas: Golden Gift 5 lower (MD5063, MD4383, MD4215W1 and MD4693) and Golden Gift 3 (MD4782, MD3095, MD5027 and MD4372) Table 12.

Checks were made to locate the original data for collar locations, down hole surveys, core photographs, and end of hole and assay records each of the drill hole includes.

16.1 Collar Locations

The collar locations in the database were compared to co-ordinates on the original drill request form. The surveyed collar of the holes is very close to the original drill request coordinates

16.2 Downhole Survey

Downhole survey information in the database was compared to the original drill survey forms. An error occurred in MD4372 with one azimuth record shown to be out by one degree. This has now been rectified.

16.3 Geology

Geology information in the database was compared to the observed geology in the core photographs. An error was found in MD4215W1 with the geology log starting at 70 metres compared with the core photos and plods which start at 77 metres. This error is likely to have occurred due to this being a wedge hole that was stopped and re-started at a later date. This has been discussed with SGM personnel to highlight that in the rare case where wedge holes are completed that due care must be taken in verifying the start depths of the drill core.

16.4 EOH

End of hole within the database was compared using collar, geology and assay files and then compared outside the database against the core photos. All end of holes were confirmed

16.5 Assay records

The assay records in the database were compared with the lab reports in electronic format for all holes with verified hardcopies also available for MD5063, MD4963 and MD4782. Several apparent irregularities were highlighted in the exercise that upon further investigation were explainable and no errors were located with the assay information reviewed.

Hole	Collar locations	Downhole Survey	Core Photos	EOH	Assay Records
GG5L					
MD5063	verified	verified	verified	verified	Verification checks are OK
MD4383	verified	verified	verified	verified	No Hardcopy digital only - Verification checks are OK
MD4215W1	verified	verified	core photos and drill plods start at 77, whereas geology log starts at 70m. All major contacts match	verified	No Hardcopy digital only - Verification checks are OK
MD4963	verified	verified	verified	verified	Verification checks are OK
GG3					
MD4782	verified	verified	verified	verified	Additional samples were on a different Job #
MD3095	verified	verified	verified	verified	Verification checks are OK
MD5027	verified	verified	verified	verified	No Hardcopy digital only - Verification checks are OK
MD4372	verified	one azimuth is out by 1 degree - Typing error	verified	verified	Verification checks are OK

Table 12 Table showing data verification work completed for the 2007 Mineral Resource update.

17 Adjacent Properties

There are no adjacent properties other than those controlled by Northgate.

18 Mineral Processing and Metallurgical Testing

The gold processing facilities utilised at Stawell comprise a standard Carbon-In-Leach (CIL) gold recovery circuit following crushing and grinding and sulphide flotation . The treatment plant consists of five unit processes. These are:

- size reduction (crushing and milling),
- gravity gold recovery,
- flotation/ultra fine grinding,
- leach-adsorption, and

- gold recovery.

Geographically the plant can be split up into five main areas. These are:

- the primary crushing circuit,
- the milling circuit,
- the flotation/ultra fine grinding circuit,
- the leach-adsorption circuit, and
- the elution/electrowinning circuit.

A current processing flowsheet is shown in Figure 48. A history of tonnage and grade throughput for the processing facilities is shown in section 8 of this report.

Coarse gold (up to 30% of the gold in mill feed) is recovered from the milling circuit in self-cleaning centrifugal gravity concentrators.

Approximately 75% of the ore requires further liberation of the gold from sulphides and this is achieved in a two stage flotation circuit where gold bearing sulphides (pyrite, arsenopyrite and some pyrrhotite) are concentrated. The sulphide is ground to approximately 0.01mm in an ultra-fine grinding mill to liberate enclosed gold (up to 20% of the gold in mill feed). The ground sulphides and flotation tail are recombined and sent to the carbon-in-leach (C.I.L.) circuit.

Stawell ore exhibits various degrees of Preg-robbing of gold. Preg-robbing occurs when naturally occurring carbon species (graphite) in the ore rob gold from the pregnant liquor in the leach circuit, thus reducing the gold recovery. To combat this, Kerosene is added to foul the naturally occurring carbon before it enters the leach circuit and a simple preg-rob index developed at Stawell indicates the rate of addition needed for the kerosene to be most effective.

Total gold recovery for 2006/2007 financial year averaged 90.52%.

SGM Treatment Plant Flowsheet

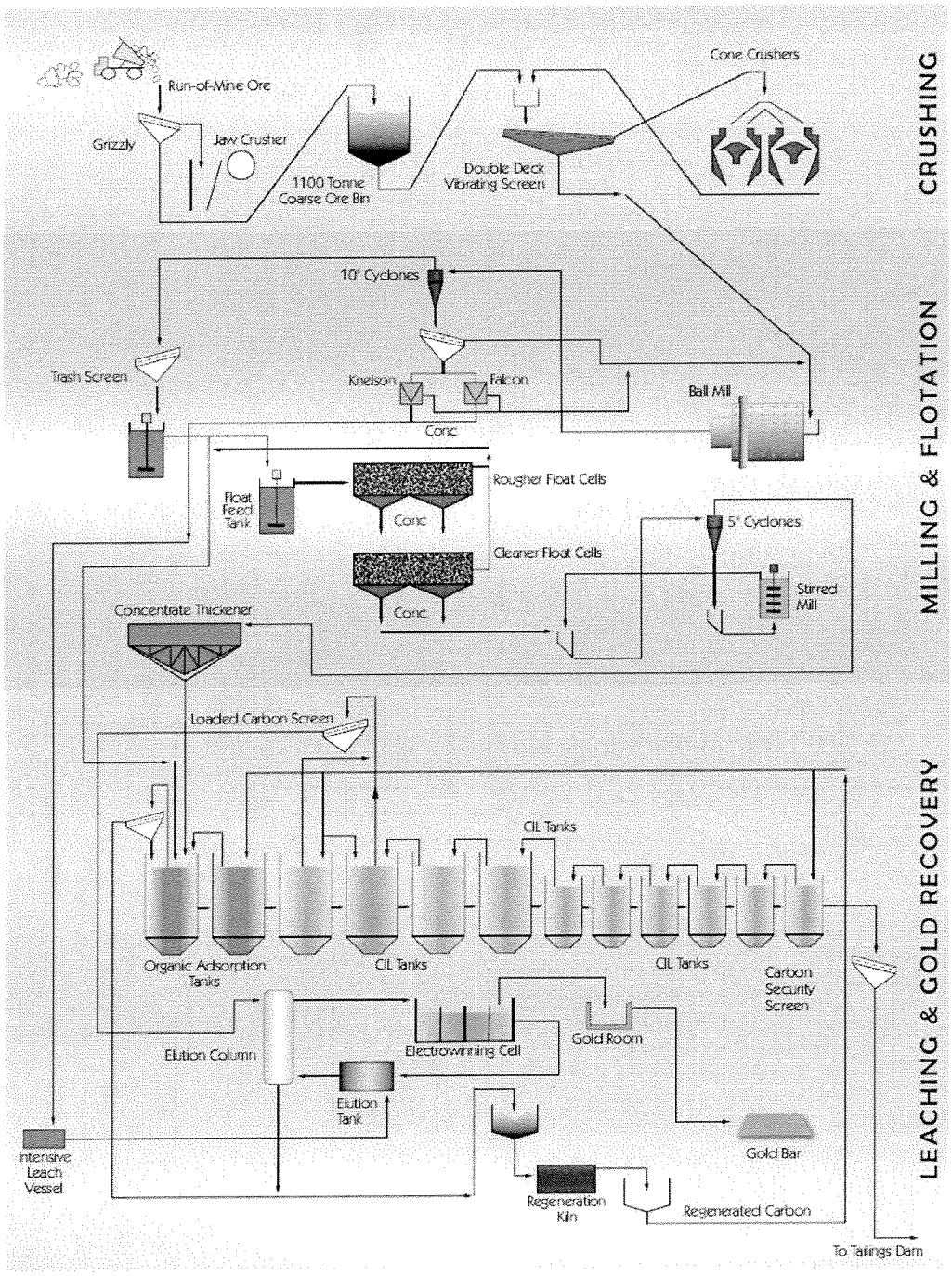


Figure 48 SGM Treatment Plant Flowsheet

18.1 Metallurgical Testwork

An ongoing program of metallurgical testwork is conducted at Stawell Gold Mines. The program utilises diamond drill core to determine the expected plant recovery for all ore blocks at a stope scale within the immediate and long term mine plan.

Samples of the ore and estimated dilution are tested to determine the expected preg-rob index and expected gold recovery through the Stawell Gold processing circuit. Samples for metallurgical testwork are selected from each ore lode; desirably samples are tested for each stope block. In this way expected rates of recovery can be determined for individual stope blocks, levels and ore lodes. As the metallurgical testwork program is an ongoing process the samples being tested to determine recovery rates can be said to be representative of the current and future production areas.

The results of the testwork program provide an expected plant recovery on a campaign basis. SGM metallurgists are able to plot the actual versus predicted plant recoveries using the test work results. Figure 49 shows the relationship between actual plant recovery and expected plant recoveries for all float ore treated project to date. This validates the robustness of the metallurgical testwork programs utilised by SGM and as such the robustness of the forecast metallurgical assumptions used in developing project schedules and financial forecasts.

Figure 49 also shows the recovery difference that can be obtained by using fresh water as opposed to using process water returned from the Tailings Storage Facility.

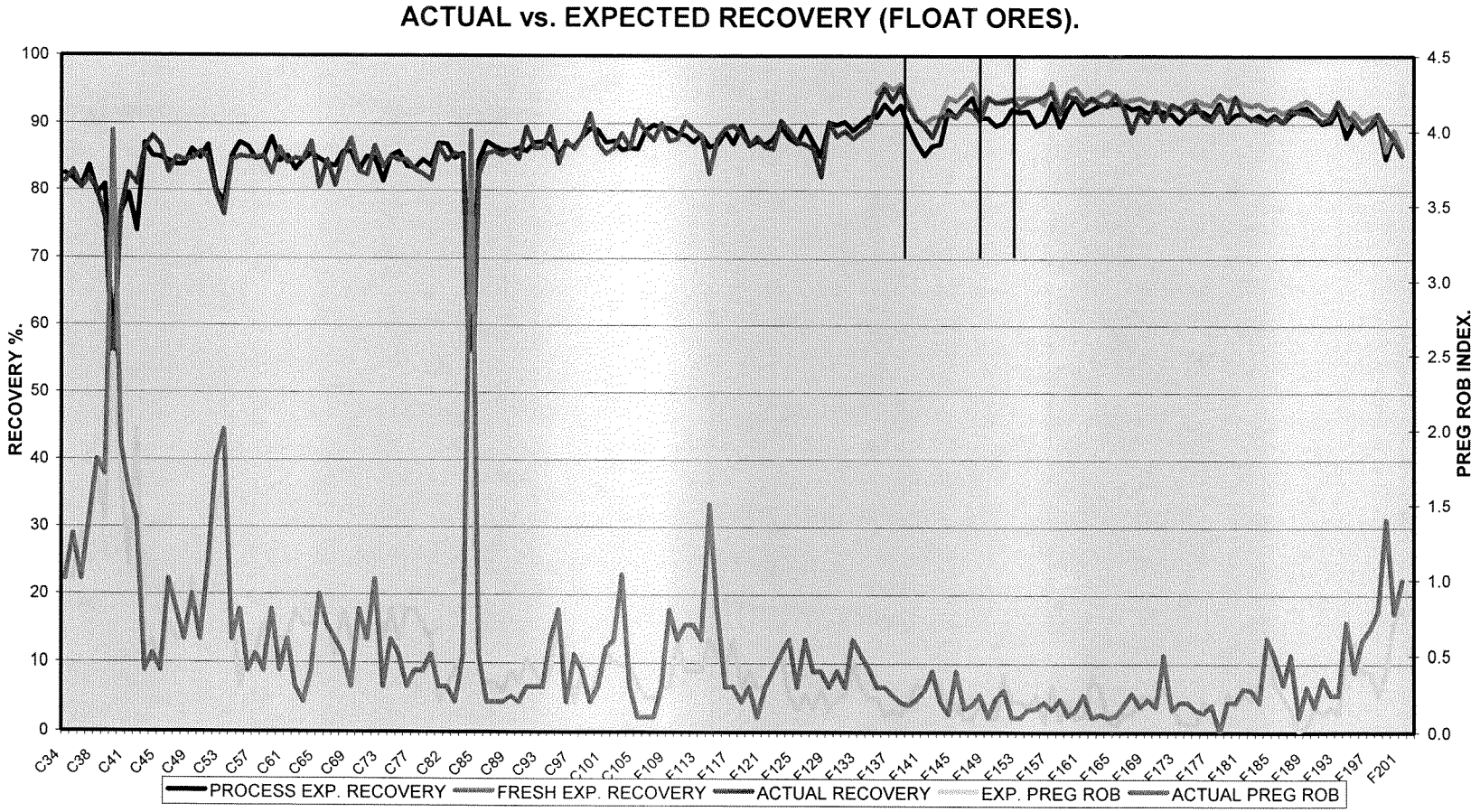


Figure 49 ***Actual versus Expected Recovery (Float Ore).***

19 Mineral Resource and Mineral Reserve Estimates

19.1 Introduction and Scope of Mineral Resource and Mineral Reserve Estimates

The geographical locations of the various Mineral Resource and Mineral Reserve areas for Stawell Gold Mines are shown in Figure 50.

This estimate is a compilation of a number of separate models that have been completed and updated over the past 15 years at Stawell Gold Mines.

Prior to 1998 the majority of the Stawell Gold Mines Mineral Resource and Mineral Reserve estimates were completed using manual 2 dimensional estimation techniques. A portion of the existing estimate consists of separate Mineral Resource areas that have been estimated by this methodology. These individual areas have been reviewed as part of the 30 December 2008 update and comply with the relevant Mineral Resource classifications as defined by NI43 101.

Post 1998 the majority of the Mineral Resources and Mineral Reserves have been estimated using 3 dimensional geological block models. A number of block models have been created for separate geographical areas. The Key Mineral resource and Mineral Reserve areas are as follows;

- Wonga Underground - Manual 2D methods - Historical Estimate based on information from 1998.
- Magdala Upper levels - Manual 2D methods - reviewed 30 December 2007
- Wonga Open pit - 3D Block model, separate documentation available (Appendix A)
- Magdala Surface - 3D Block model, separate documentation available Magdala (Appendix B)
- Magdala Lower Levels
    - C7 (Central Lode) - 3D Block model, separate documentation available (Appendix C)
    - U3 - 3D Block model, separate documentation available (Appendix D)
- GG1 - 3D Block model, separate documentation available (Appendix E)
- GG2 - 3D Block model, separate documentation available (Appendix F)
- GG3 - 3D Block model, separate documentation available (Appendix G)
- GG5 - 3D Block model, separate documentation available (Appendix H)
- GG5 Lower - 3D Block model, separate documentation available (Appendix I)
- GG9 - 3D Block model, separate documentation available (Appendix J)

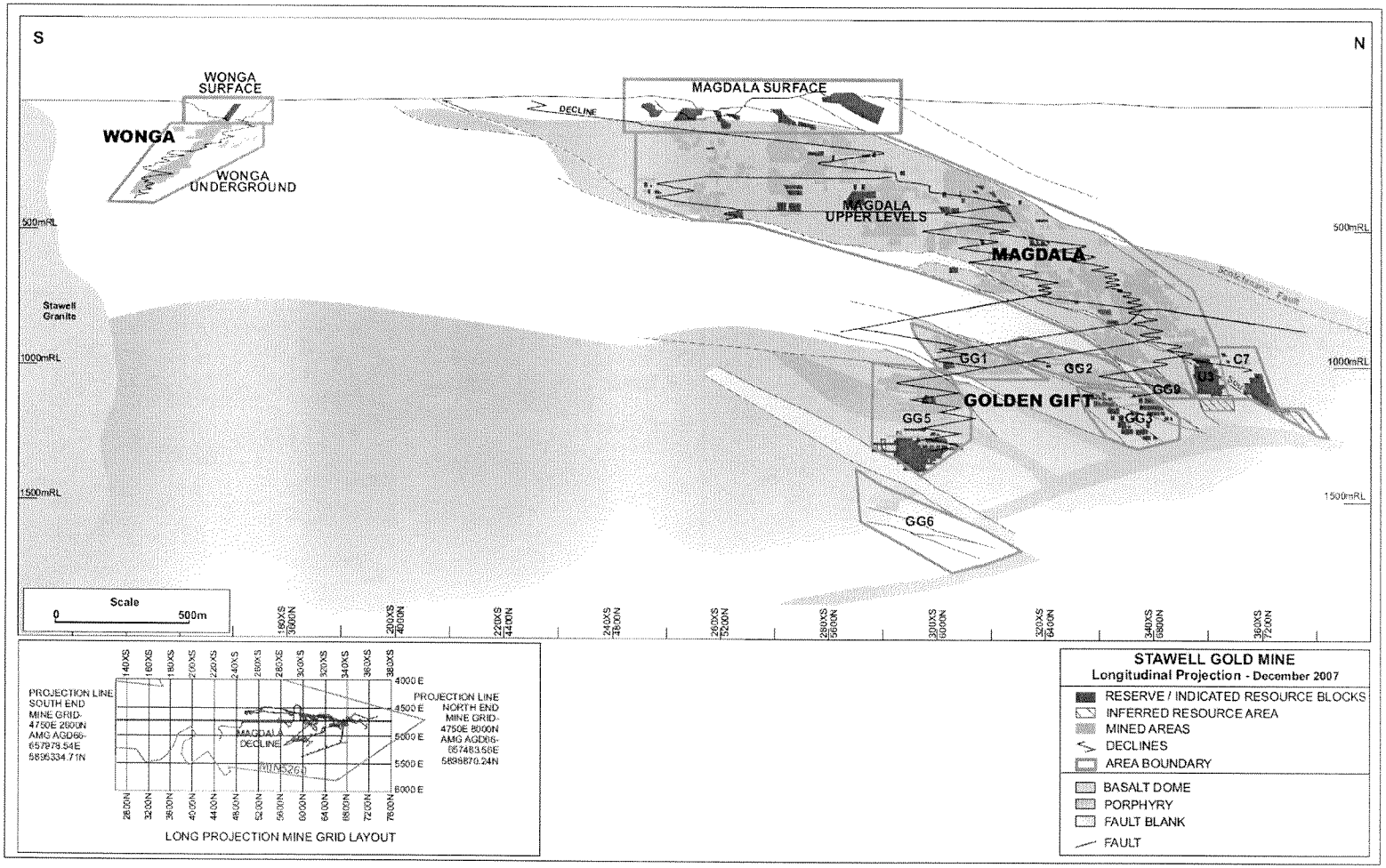


Figure 50 Longitudinal Projection showing the location of Mineral Resource and mineral Reserve areas as the 30 December 200. Note GG6 is not reported as a component of the December 2007 Mineral Resource estimate.

This section of the document summarises the methodologies used to estimate the Mineral Resources and Mineral Reserves and provide a summary of the key assumptions and results for each Mineral Resource/Mineral Reserve Area.
The methodologies used at Stawell are supported by a significant period of mining and reconciliation information. A summary of the results of stope reconciliations since 2003 are included in section 20 of this report.

19.2 Manual 2 Dimensional Mineral Resource and Mineral Reserve Estimation Methodologies

Mineral Resources and Mineral Reserves calculated using manual 2D methodologies are confined to sections of the upper levels at the Magdala deposit and to remaining Mineral Resources for the underground portion of the Wonga deposit.

A detailed breakdown of the quantity of the December 2007 Mineral Resource and Mineral Reserve estimates completed using this methodology is given below.

Prior to 1996 this was the primary estimation methodology for all Mineral Resources and Mineral Reserves estimated for the Stawell deposits. Progressively 3D block model methodologies were introduced and the remaining areas estimate using 2D methods are remnant parcels for which detailed 3 dimensional models have not yet been created. There is an expectation that prior to mining of these areas additional geological information will be collected and processed to produce 3D block models upon which final economic evaluation and mine design will be completed.
The validity of estimation methodology is supported by many years of mining experience at Stawell where with the addition of appropriate dilution and grade cutting have produced reliable estimates of tonnes grade and contained metal.
The Mineral Resource Estimation methodology is based on compiled 1:500 scale longitudinal projections for individual mineralised structures showing the locations of all diamond drill or sample intersections.

- Each mineralised intercept is firstly interpreted on cross section and or level plan or in 3D to determine the true width of the mineralised structure and to ensure the intercept is plotted on the longitudinal projection for the appropriate structure.
- Individual assays within the mineralised structure are top cut to 17.0 g/t Au for Magdala and 15.0 g/t Au for Wonga mineralisation respectively. The assays are accumulated by length weighting of the individual assay grades.

- Wonga mineralisation is diluted to a minimum width of 2.0m by adding dilution at 0.5g/t Au
- Magdala mineralisation is diluted to a minimum width of 3.0m by adding dilution at 1.0g/t Au
- Additional interpretation is completed to confirm the strike and dip of the mineralised structure being modelled to calculate a Strike/Dip correction factor to be applied to the individual Polygon being estimated.
- Polygon areas are measured for each block to be estimated
- The tonnage is calculated by multiplying AREA X Mean Width X SG X Strike/Dip Factor
- The grade assigned to the block is estimated based on the diluted length weighted intersections deemed to be informing each polygon.

The conversion of the Mineral Resource estimate to Mining Reserve takes into account historical dilution factors and expected mining recovery based upon known pillar spacing and mining requirements. The following are the historical mining factors applied

- Wonga lodes are diluted by variable amounts to account for the expected mining method.
- Magdala ore structures are diluted by 50% at 1.0g/t Au to account for planned mining dilution. This is effectively to account for irregular geometry.
- An additional 25% dilution at 1.0 g/t Au is added to account for unplanned mining dilution.
- Mining recovery factors to account for pillars are applied.

19.3 Computer 3 Dimensional Mineral Resource and Mineral Reserve Estimation Methodologies

Post 1997 3D block modeling methodologies have progressively become the standard method for estimating Mineral Resources and have provided the basis for detailed mine design and the estimation of Mineral Reserves.

This work is carried out in the MineSight software suite which is an industry standard geology and mine planning software package. Detailed Mineral Resource documents for each individual Mineral Resource area are included in Appendices A - J. The description below describes the general process that is common to each model.

19.3.1 Geological Modelling

Geological modeling is carried out on individual resource areas by the geological team at Stawell. The data available for this varies depending upon the stage of the development and understanding of the deposit area. The general process is as follows

- Wireframe models of major geological units are interpreted and created using MineSight Software.
- Less advanced deposit areas are first interpreted on paper while more advanced deposits are often interpreted directly within the software.
- All available geological information is utilised in the interpretation process.
    - Diamond Drill core logs, Core photographs, Face mapping and photographs and sludge drilling geology logging.
    - The drillhole logging information available to the mine geologists includes
        - Lithology
        - Alteration
        - Quartz veining percentage and veining style
        - Sulphide percentages, type and style
        - Location and orientation of lithological contacts, shears and fault structures
        - Core texture - indicating faulting, shearing etc
        - Core photographs
    - Underground development face mapping and sludge sampling is incorporated into the models where available. An example of this is shown in Figure 28 in section 13.
    - Development Face sketches and or Face photographs are scanned and registered in 3d to enable the detail to be incorporated into the interpreted 3D models, also shown in Figure 28
    - Face sketches and underground mapping observation provide details on significant offsetting faults and structures
    - Sludge hole drilling produces rock chip cuttings that are logged by the mine geology team. This information is stored in a separate drilling data set and displayed in 3D to add additional information on the location of major contacts and fault structures
- Geological modelling is an ongoing process and models are progressively updated to reflect the most up to date information. Model updates are generally made upon completion of infill drill programs and completion of development levels, where this information results in a material change in the amount and quality of data involved and or material changes in geological understanding.

The gross geological architecture of the mineralisation systems is well understood and described in detail in Section 9 - 11 of this report. Mineralisation is hosted by relatively distinctive and predictable geological units that are modelled by the area geologists. The key units that are modelled in each area are;

- Magdala Basalt - unmineralised. Geometry of the basalt is required to estimate the quantity of dilution that will be incorporated into the mine designs
- Mine Schist - unmineralised. Also important for estimating the quantity of dilution that will be incorporated into the mine designs

- Weakly mineralised volcanogenic - important to estimate dilution in the mine designs
- Mineralised domains. For individual model areas these may be either in the Central Lode position, Basalt Contact positions and contain zones of Stockwork mineralisation.
- Key fault structures are modelled as they can have a significant impact on the shape of the mineralised domains (particularly in Golden Gift)
    - Structures (particularly in Golden Gift) have a significant impact on the outcomes of the geological modelling and improving the understanding of the location of these is a key component to producing reliable estimations of the insitu Mineral Resources
    - Structures (particularly in Golden Gift) have a significant impact on the insitu Mineral Resources
- Wireframes are where possible snapped to drillhole intervals.

The key control in on Mineral Resource estimation is accurate definition of the constraining geological models. Estimation of grade within the domains, whilst still very important, is of secondary importance to the first order geological domaining.

19.3.2 Block Modelling

The process of Block modeling at Stawell gold Mines is relatively standard across all model areas. This description is drawn from the details for the individual model areas as documented in appendices A - J.

Stawell Gold mines has retained the services of several key consulting groups over time to ensure Mineral Resource estimation processes have been maintained to a high standard. Quantitative Geosciences (QG) have provided ongoing coaching, training, mentoring and Mineral Resource estimation services on an as required basis since the late 1990's. This has ensured consistency of process over this time as key site personnel have changed and supported by the Qualified Person.

19.3.2.1 Block Model Dimensions

Block model dimensions vary on an area by area basis as indicted in Table 13

Model Area	Block Dimensions		
	Along Strike	Across strike	Dip Extent
Magdala Surface - Big Hill	10	5	5
Wonga Surface	10	5	10
U3	20	2	20
C7	15	2	15
G1	10	2	15
G2	7.5	2.5	7.5
G3	10	5	10
G5 and G5 Lower	10	5	10

Table 13 Block sizes utilised in Stawell Gold Mines local area area block models

All block models other than the Magdala Surface and Wonga surface models are rotated to align with the strike and dip of the area being modelled. The details of the rotations applied are shown in the individual model documentation as given in appendices A - J. Alignment of the surface models into the plane of the structure is not completed as current mine planning software is not readily able to use the models and subsequent mine planning is impeded.

Block model dimensions vary based upon the density of the available drilling data and also on the overall geometry of the mineralised structures.

19.3.2.2 Block Model Coding

Block models are coded with the key structural/mineralisation domains. This process is completed in MineSight software. The details of the coding also vary by Mineral Resource area and are determined based on the number and geometry of the mineralisation domains. Given the block sizes utilised by SGM, multiple domain codes and percentages are stored in each block.

19.3.2.3 Drillhole Coding

A diamond drillhole set is coded with the key structural/mineralisation domains. Importantly this coding is checked manually by the area geologists to ensure that they match the wireframes created. Where required some manual adjustment is made and if suspect drillhole locations are noted these drillholes may be excluded from the estimate. These details are included in the documentation for each mineral Resource area

19.3.2.4 Compositing

The drillhole files are composited downhole to fixed length composite intervals. The composites are matched to the geological domains as coded in the drillhole files. Composite intervals used are either 1.0m as in the case of Magdala Surface, C7, U3 models or 2.0m for all other areas. The choice of composite intervals is made on a model by model basis that will best reflect the block size, data available and geometry of domains.

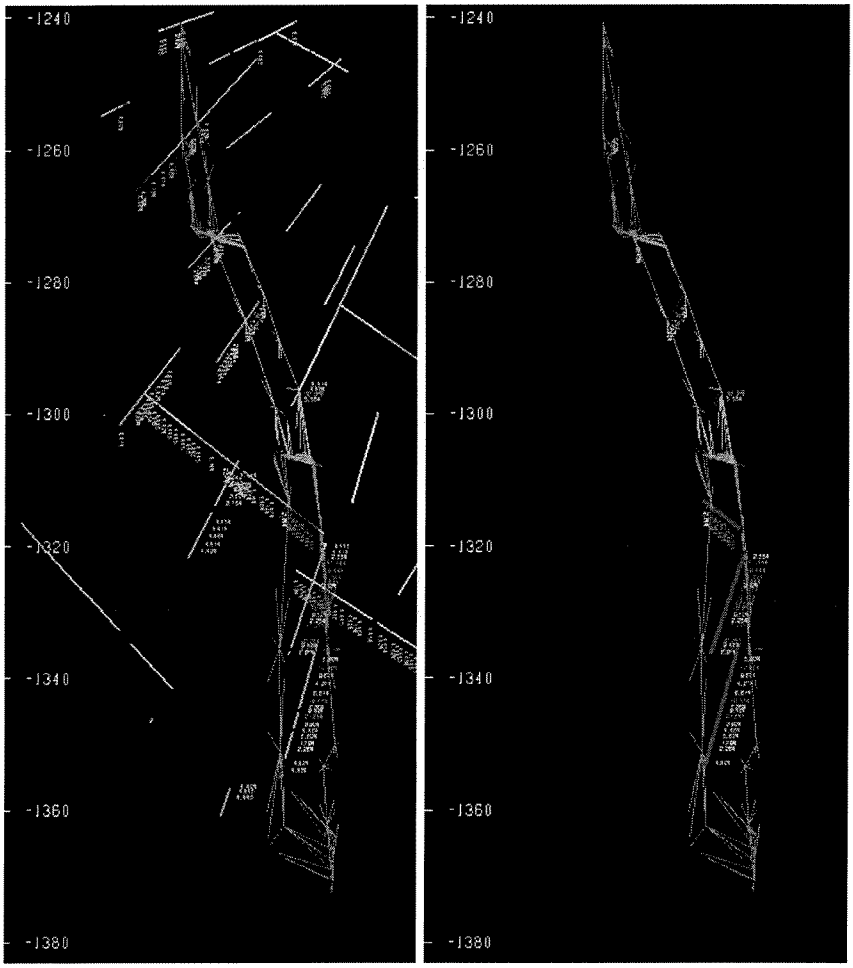


Figure 51 Cross Section showing an example of the compositing process used at Stawell Gold mines. This example is from the GG5 Lower resource area, Appendix I.

19.3.2.5 Geostatistical Parameters

For all of the Mineral Resource areas that have been modelled in 3D Gold Grades (Au ppm) have been estimated by ordinary kriging. Over time this methodology when coupled with detailed and robust geological models provided reliable estimate of insitu gold grade. A separate section and analysis of reconciliation information for the mine is presented in Section 20 of this report.

The key geostatistical parameters are modelled for each project separately. Variography studies are completed using either ISATIS software or MineSight software. Individual variogram studies are conducted for each domain and modelled separately.
Key variogram parameters, nugget and sill and variogram ranges are modelled on either normal variograms or Gaussian transformed data. Whilst the estimated values for the nugget and ranges vary for each modelled area they are generally relatively consistent.
In addition to modeling variograms studies have been performed to optimise the kriging neighborhood as per the methodologies presented in Vann et al (2003). This enables quantitative evaluation of the results of the kriging to be performed and as well as enabling the search neighborhoods to be optimised provides numerical outputs from the kriging runs (Slope of Regression of the Estimate). The Slope of Regression is used in part to aid in classification of the Mineral Resource estimates as per the definitions contained within NI43-101.
All geostatistical parameters used for each model area are documented within the individual Mineral Resource estimate documents as appended

19.4 Density

As per the Stawell Gold Mines core processing flowchart (section 13 Figure 37) all whole core sample intervals that are sent for assay are bagged and weighed prior to leaving site. The weights of the core are stored in the drillhole database and an apparent dry bulk density for each interval is calculated based upon a theoretical volume for the specific core diameter for each sample interval.

Whilst there are potential limitations in this process given that the volume for each core is based upon the theoretical drillcore diameter and the measured length of each interval it can be demonstrated by reconciliation of stope volumes as surveyed by cavity monitoring systems and tonnage measured by trucked ore that the estimated Density applied to specific areas is a reliable estimator of insitu density. Historical reconciliation supports the density values that are currently being used in the estimates at Stawell Gold Mines.

This apparent density measurement includes some residual moisture which by measurement has been estimated at approximately 2% or less which is generally not accounted for in the estimates.

Table 14 shows the density values that have been applied by Mineral Resource area. Specific details of the data and analysis and assumptions used to derive these values are given in each of the area specific reports.

Table 14 Compilation of Density applied by Resource Model area

Model Area	Ore SG Applied (As Estimated)	Basalt SG	Mine Sc
Magdala Surface - Big Hill	See Table 15		
Wonga Surface	2.45 - 2.85		
Wonga Underground - Manual Estimates	2.65	NA	NA
Magdala underground Manual Estimates	2.85	NA	NA
U3	2.7 - 2.94	2.8	2.7
C7	2.85	2.8	2.7
G1	2.85	2.8	2.7
G2	2.66 - 2.74	2.75	2.55
G3	2.85 - 2.97	2.8	2.7
G5 and G5 Lower	2.9	2.8	2.7
G5 Lower	2.95	2.8	2.7

Table 15 Magdala surface Mineral Resource estimate density values applied

Domain		Oxide	Transition	Sulphide
Mariner's		2.3	2.5	2.85
Allen's	Constrained	2.15	2.3	2.85
	Volcangenics	2.1	2.3	2.85
Iron Duke	Constrained	2.15	2.3	2.85
	Volcangenics	2.1	2.3	2.85
Magdala	Constrained	2.1	2.3	2.7
	Volcangenics	2.0	2.3	2.85

19.5 Resource Classification

Mineral Resources and Mineral Reserves are classified in accordance with NI43-101 disclosure rules.

The practice adopted at Stawell uses general guidelines for classification that utilise the following information as outlined in Figure 52.

- Drilling Density
- Stage of development - Ore Development and final data gathering in place
- Demonstrated geological continuity of structure and mineralised domains
- Slope of regression of the estimate (Calculated value during the Kriging Process

The classification as applied to the Mineral Resources disclosed in this document have been reviewed in detail on an area by area basis and are considered to be appropriate and within the guidelines as per the CIM Definition Standards on Mineral Resources and Mineral Reserves.

Activity Area	Target Type	Criteria	Resource Classification	Geological Data available
Exploration	Conceptual Targets	• Conceptual geological model • Geophysical anomaly • Geochemical anomaly		• Geophysics • Mapping • Wide spaced Exploration Drilling • Conceptual geological models
	Confirmed Targets	• Geological model confirmed by drilling • Mineralisation confirmed by drilling		• Wide Spaced Exploration drilling • Assay information • Drill logs
	Scoped targets	• Geological continuity established • Ore grade intersections established • Preliminary geological model established • Drill spacing 300 m X 120 m	Pre resource, Resource Target	• Broad spaced Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models
Resource Definition		• Geological continuity confirmed • Ore grade intersections continuous • Geological interpretation modelled • Geostatistical model established • Drill spacing 150 m X 60 m	**Inferred Resource**	• Regular Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis
		• Geological continuity confirmed • Ore grade intersections confirmed • Geological interpretation modelled • Geostatistical model • Economic analysis • Drill spacing 40 m X 30 m	**Indicated Resource**	• Regular Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis
Grade Control		• Confident Geological continuity • Ore grade intersections confirmed • Geological interpretation modelled • Geostatistical model • Economic analysis • Drill spacing 15 m X 15 m	**Indicated Resource**	• Close Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis
		• Confident Geological continuity • Ore grade intersections confirmed • Geological interpretation modelled • Geostatistical model • Economic analysis • Level Development above and below • Drill spacing 15 m X 15 m • Open-hole sludge drilling	**Measured Resource**	• Close Grid Drilling • Detailed cross sectional interpretations • Assay information • Drill logs • Geological models • Geostatistical model • QA/QC analysis • Development face mapping sheets and ore runs • Open-hole sludge drill geological data

Figure 52 Stawell Gold Mines geological processes and general guidelines for classification

19.6 Underground Design methodology

19.6.1 General Design Parameters

- Level spacing up to 25 metre maximum for the orebodies.
- The stoping method for all areas is longhole stoping with backfill. The backfill may be cemented rockfill (CRF) or unconsolidated fill, depending on the grade of the associated rib pillar necessitated by unconsolidated fill.
- Rib pillar aspect ratio is not designed to be more slender than 1:1.
- Minimum stoping width is deemed to be the width of the mined development.
- Fill triangle pillars were assumed to allow for backfilling rill angle placement of 45 degrees.
- HW dilution parameters are determined by the geotechnical department from hangingwall characterisation based on geological and geotechnical logging of diamond drill
- Strike dimension of individual stopes is nominally 18 metres for single-lift stopes and 15 metres for double-lift stopes. Initial dimensions may be altered in consultation with geotechnical personnel to account for structure, sequence or abnormal hangingwall conditions.
- Sequences are designed to minimise the creation of closure pillars. These sequences are predominantly bottom up, with footwall lodes preceding the hangingwall lodes.
- Where stopes are to be mined under CRF, a stope height of 12m has been used to try and limit stope and pillar size and increase potential recovery.

19.6.2 Financial Considerations

- The economic viability of the stoping regions was determined with a spreadsheet distilled from the approved 2006 Budget. Marginal stopes were not used in determining the viability of the regions under consideration, but were included in the reserve if they paid the operating costs associated with their extraction, and were developed in the process of accessing fully-funded stopes. Each region was assessed with full financial consideration and as such no particular cut-off grades have been assigned to specific areas.

- Capital costs in the 2006 Budget draw capital depreciation and amortisation figures for vertical and horizontal development from the approved Life-of-Mine Plan.
- Only designs created on an Indicated or Measured Resource were considered for reserve classification. No Probable reserves have been derived from Measured Mineral Resources.
- All Ore Reserves are estimated by querying 3-dimensional; block models with 3-dimensional stoping design voids. The stope designs include internal and external dilution. The resulting in-situ reserve then has modifying factors for grade and tonnage applied to estimate the Ore Reserve.
-

Design criteria; include dilution factors for Magdala and GG styles

	Hangingwall Dilution	Foot wall Dilution	Hydraulic Radius	Strike Length of Stopes
Magdala	2-3m	1m	6	12-15m
GG 1	2m	1m	6	15m
GG 3	1-2m	0m	6	15-18m
GG 5	1.5m	1m	6	15-18m
GG 5 Lower	1-2m	1m	6	15-18m
GG 9	2m	1m	6	12-15m
U3	2-3m	1m	6	12m
C7	2-3m	1m	6	12m

Table 16 Summary of design factors for Magdala and GG styles

19.7 Mineral Resource and Mineral Reserve Summary

Mineral Resources and Mineral Reserves for the Stawell ore bodies are combined together into a single detailed database containing all relevant information for individual Mineral Resource and Mineral Reserve areas. Whilst this detail is available the information presented in this report summarise the Mineral Resources by area as shown in Figure 53 which reports contain ounces only.

The Mineral Resources exclusive of Mineral Reserves is shown in Table 17. The Mineral Reserve summary is given in Table 18. A condensed summary table of Mineral Resource and Mineral Reserves for Stawell Gold Mines is shown in Table 19.

Stawell Gold Mines has compiled the estimate such the Mineral Resources are reported exclusive of Mineral Reserves.

	Mineral Resources exclusive of Mineral Reserve						
	Indicated		Inferred		Total		
	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	In situ Au oz (000's)
Underground							
Magdala above 1250mRL	200	4.97	150	4.05	350	4.58	51
Golden Gif above 1360mRL	23	7.09	80	6.01	103	6.25	21
Wonga above 1000mRL			121	6.80	121	6.80	26
Sub-total U/G	223	5.19	351	5.44	574	5.35	99
Surface							
Magdala	2660	2.15			2660	2.15	184
Wonga	298	1.49	106	2.44	404	1.74	23
Sub-total Surface	2958	2.08	106	2.44	3064	2.10	207
TOTAL	3181	2.30	457	4.75	3638	2.61	306

Table 17 Stawell Gold Mines Mineral Resource summary table as at 31 December 2007

	Mineral Reserves						
	Proved		Probable		Total		
	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	tonnes (000's)	grade g Au/t	In situ Au oz (000's)
Underground							
Magdala above 1250mRL	10	4.34	297	4.5	307	4.49	44
Golden Gif above 1360mRL	36	4.83	926	5.92	962	5.88	182
Wonga above 1000mRL							
Surface Stockpiles							
Sub-total U/G	46	4.72	1223	5.58	1269	5.54	226
Surface	0		0				
Davis Exte above 130mRL			325	2.12	325	2.12	22
LG Stockpiles			188	0.8	188	0.8	5
Sub-total Surface	0		513	1.64	513	1.64	27
TOTAL	46	4.72	1736	4.41	1782	4.42	253

Table 18 Stawell Gold Mines Mineral Reserve summary table as at 31 December 2007

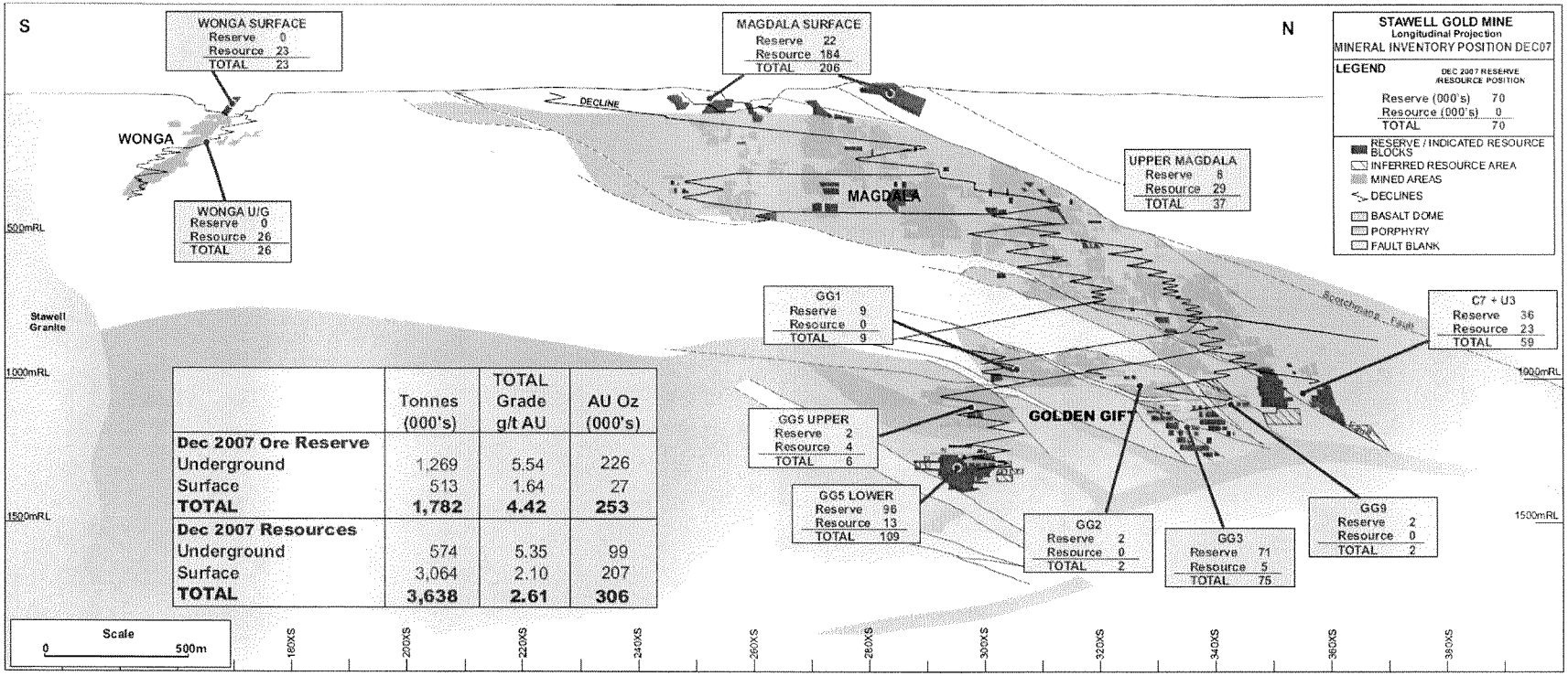
Mineral Reserves				
Stawell	Proven	46,000	4.72	7,000
	Probable	1,736,000	4.41	246,000
	Total	1,782,000	4.42	253,000

Mineral Resources (exclusive of reserves)				
		Tonnes	Gold (g/t)	Gold (contained ounces)
Stawell	Measured	0	0.00	0
	Indicated	3,181,000	2.30	235,000
	Meas + Ind	3,181,000	2.30	235,000
	Inferred	457,000	4.75	70,000

Table 19 Condensed Mineral Resource and Mineral Reserves as at 31 December 2007

Notes:

1. *All Mineral Resources and Mineral Reserves have been estimated in accordance with the JORC Code and have been reconciled to CIM Standards as prescribed by National Instrument 43-101.*
2. *Mineral Resources for are exclusive of Mineral Reserves.*
3. *Mineral Reserves were estimated using the following economic parameters:*
    a. *Gold price of AUD$750/oz. Cut-off grade applied was variable for underground ore depending upon width, mining method and ground conditions. Dilution of 2-3m and mining recovery of 95-100% were applied to the underground reserves, dependent upon mining method.*
4. *Mineral Resources were estimated using the following parameters:*
    a. *Underground using Gold Price of AUD$750/oz.*
    b. *Magdala surface above 130mRL and above a nominal 0.8g/t Au cutoff*
    c. *Wonga surface within a AUD$850 optimised pit shell*
5. *Mineral Reserve estimates were prepared by:*
    a. *Glenn Miller, Mine Technical Superintendent, Northgate Minerals Corporation. Mr Miller is a member of the Australasian Institute of Mining and Metallurgy and has over 16 years of relevant engineering experience.*
    b. *Tamer Diner of Mining Solutions Consultancy Pty Ltd. Mr Diner is a member of the Australian Institute of Mining and Metallurgy and has 20 years of experience of relevant engineering experience.*
6. *Mineral Resource estimates were prepared by:*
    a. *Dean Fredericksen of Fredericksen Geological Solutions Pty Ltd. Mr Fredericksen is a member of the Australasian Institute of Mining and Metallurgy and has over 19 years of relevant geological experience.*
7. *Mineral Resources and Mineral Reserves are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summations may occur due to rounding.*



	Tonnes (000's)	TOTAL Grade g/t AU	AU Oz (000's)
Dec 2007 Ore Reserve			
Underground	1,269	5.54	226
Surface	513	1.64	27
TOTAL	**1,782**	**4.42**	**253**
Dec 2007 Resources			
Underground	574	5.35	99
Surface	3,064	2.10	207
TOTAL	**3,638**	**2.61**	**306**

Figure 53 Long Projection showing a summary of the Mineral Resource and Mineral Reserves on an area by area basis

19.7.1 Magdala Underground

Magdala Mineral Resources and Mineral Reserves are contained around previously mined areas in the upper levels. The majority of the Mineral Resources in this area have been estimated using conventional 2D estimation methodologies. There are ongoing data gathering and evaluation programs to convert the current Mineral Resources to Mineral Reserves and appropriate resource are allocated to these programs.

Reserves are contained on Central, and Basalt Contact structures. Rehabilitation and access considerations have been taken into account when assessing the viability of each stoping block. Stopes generally have a weak hangingwall of Mine Schist or weakly mineralised volcanogenics.

19.7.2 C7

Mineral Resources have been estimated by 3D estimation methodologies. The details of which are outlined in appendix C.

Mineralisation of the C7 orebody is characterised by two main lodes, the Duke's lode and Central lode. Duke's lode is volcanogenic mineralisation against a basalt contact with Central lode a quartz vein in the hangingwall of Duke's lode. The separation between the two lodes varies between 1m and 10m with weakly mineralised volcanogenics separating the two lodes. The hangingwall of the Central lode is mineschist and is of varying rock strength.

19.7.3 U3

Mineral Resources have been estimated by 3D estimation methodologies. The details of which are outlined in appendix D.

The U3 orebody is characterised by a single lode of mineralisation along the contact of the Oriental basalt. The mineralisation style includes both volcanogenics and quartz veining. The South fault is the upper bounding fault of the orebody and the Merlot fault is the lower bounding fault. The Oriental basalt forms the footwall with mineschist as the hangingwall. Ground condition encountered in the ore and hangingwall are considered extremely poor.

19.7.4 GG1

Mineral Resources have been estimated by 3D estimation methodologies. The details of which are outlined in appendix E.

The GG1 orebody contains two main lodes separated as East Patron and West Patron Lodes. The lodes are separated by internal basalt and are bounded by a basalt hangingwall and mineschist footwall. The orebody has a typical dip of 50° to 55°, and in some cases reduces to 35°, this flat nature can lead to increased waste dilution. The top of the orebody is constrained by the South Fault with several flat faults offsetting the orebody horizontally.

19.7.5 GG3

Mineral Resources have been estimated by 3D estimation methodologies. The details of which are outlined in appendix G.

The main mineral bearing lode in GG3 is the Dalray Lode which has both a hangingwall and foot wall component. The GG3 Geology is characterised by a series of flat planar faults that have resulted in significant structural complexity. Ongoing Grade Control drill programs are targeted to unravel this structural complexity and to ensure that Ore Reserve extraction is optimised. Offsetting of ore structures has required additional mullock to be included in Stope Reserves to access available ore. Hangingwall conditions are considered good with basalt, though faulting may result in some dilution.

19.7.6 GG5

Mineral Resources have been estimated by 3D estimation methodologies. The details of which are outlined in appendix H.

The GG5 orebody is characterised by two main steeply dipping lodes, the Empire (HW) and Statesman (FW) Lodes alongside each other, bounded by a basalt hangingwall and porphyry footwall. An internal basalt zone pinches in between the lodes, creating a high grade waterloo zone and an offset zone west of the porphyry. The Tarrango fault constraints the orerbody at the northern end.

Hangingwall conditions are considered good with basalt hangingwalls. Mineralisation exists from the 1045RL to the 1165RL with a central retreat fashion to avoid the need for closure stopes and allows for a dual production front.

19.7.7 GG5 Lower

Mineral Resources have been estimated by 3D estimation methodologies. The details of which are outlined in appendix I.

Mineralisation in the GG5 Lower is characterised by a single steeply dipping lode that has the lower fault as a boundary. Two basalt noses cause areas of increased ore width within the structure. Hangingwall conditions are considered excellent with basalt hangingwalls and volcanogenic footwall. Mining of GG5 Lower has required a temporary crown pillar to be established to allow for continuity of ore supply while vertical depth is achieved. Mining is generally in a central retreat fashion to avoid the need for closure stopes. Mineralisation extends from the 1260RL to 1375RL with temporary crown established at the 1310 RL.

19.7.8 GG9

Mineral Resources have been estimated by 3D estimation methodologies. The details of which are outlined in appendix J.

The GG9 resource is characterised predominantly by a single steeply-dipping Kingston lode which reduces in width with distance from the Merlot Fault. The Merlot fault forms the upper boundary of mineralisation and the Durif Fault the lower. The expected poor conditions of the weakly mineralised volcanogenic footwall have required that 1.5m of dilution be attributed to this, though the basalt hangingwall is expected to have good ground conditions. The GG9 contains a number of flat structures that have increased geological complexity.

19.8 Open Pit Design methodology

19.8.1 Design criteria

	Overall slope angle	Bench Height	Depth
Wonga	37	10m	120
Davis	43	10m	70
Wildwood	45	10m	70m

Table 20 Summary of design factors for open pits

19.8.2 Wonga Surface

The Wonga Surface mineral Resources have been estimated using 3D block modelling methodologies as described in Appendix A. The Mineral Resources are reported within an optimised pit shell at a gold price of AUD$850 (see Appendix K).

The Wonga Open pit is located 1.5km from the Stawell Mill and consists of a mined open pit which has then progressed to an underground mine reaching the 400RL. The Wonga ore zone consists of 2 structures, the flat Hanging-wall lode and the steep link Structure. Mining has historically mined both structures, with the majority of work focussed on the Link Structure. Open pit Mining will be centred on the Hangingwall lode through a cutback of the Eastern Pit Wall. A high risk of ore loss is associated with mining due to open voids beneath the pit. Access to the bottom of the pit is through an established ramp, and mining will be using conventional drill and blast techniques.

19.8.3 Magdala Surface

19.8.3.1 Resource

The Magdala Surface Mineral Resource (referred to also as Big Hill Mineral Resource) was estimated in 1998 using 3D estimation methodologies as documented in Appendix B.

The Mineral Resources are reported above 130m RL above a nominal cutoff of 0.8 g/t Au.

The Magdala Surface Mineral Resources have been subject of detailed reviews by Mike Stewart from Quantitative Geosciences including a recommendation that alternative estimations methodologies be adopted for this deposit. These recommendations have been reviewed by a Qualified Person and whilst are sound advice it is considered the outcome in respect of the estimated Mineral Resources is not material to the information disclosed in this document.

19.8.3.2 Davis Open Reserve

The Davis Open Pit is located 1km North West of the Stawell Mill. Work will be to extend the pit to depth. Historical underground workings, as well as proximity to local dwellings and water infrastructure limit the extent to which the pit can be extended. Mining will be through conventional drill and blast mining, with consideration to houses located close to the edge of the mining pit.

19.9 External Factors Effecting Extraction of Mineral Resources and Reserves

19.9.1 Magdala Surface Mineral Resources

The Magdala surface Mineral Resources are located within a segment of Crown Land called the Big Hill ridge which defines the up dip portion of the Magdala mineralised system. This area is bordered by residential areas on two sides. An extensive RC and Diamond drilling program was completed during 1997 and 1998 to define the extent of the mineralised system and define the mineral Resource.

A feasibility study to establish and open pit mining operation on this section of the deposit was completed and a detailed Environmental Effects Statement (EES) prepared. The EES was submitted to the Victorian State Government in 1999 seeking approval to commence a mining operation within the Big Hill environment. In November 2000 the proposal was rejected by the Minister for Planning and the project was not able to proceed.

Stawell Gold Mines believes there is potential to re-examine the project and the methodologies that were proposed at the time and thus exploit portions of the Mineral Resource as defined and reported in this document.

Further open pit optimisation studies have been completed and a smaller open pit designed as an extension of the existing Davis Open Pit. Stawell Gold Mines management has received advice that this proposal could be completed by applying to the Victorian State Government for a variation to the existing "Work Plan" and a planning permit from the local Grampians Shire Council. As such a small open pit Mineral Reserve has been defined and reported as part of the Mineral Reserve estimate.

During 2007 the waste dump created from the initial mining of the Davis open pit has been progressively reclaimed and treated through the Stawell Gold Mines processing facility. As such there is precedent for mining operations in this area.

Additionally some studies have been completed looking at the potential to extract portions of the Mineral Resource by underground methods. At the time these studies were completed (2001 - 2002) the economic scenarios were not favorable. A significant change in the gold price during 2007 has not been factored into this analysis and further reviews are planned.

Whilst there are currently permitting, environmental and political constraints preventing mining this portion of the Mineral Resource estimate there are potentially significant economic benefit to the Stawell operation if this Mineral Resource could be exploited by some form of mining activity. Stawell Gold Mines will continue evaluation of potential extraction methodologies.

19.9.2 Wonga Surface Mineral Resource

The Wonga surface Mineral Reserves are based upon an open pit optimisation study completed in mid 2007. The resulting optimisation shells at a gold price of AUD$ 850 create an extension to the existing partially filled Wonga Open Pit void.

There are existing Mining Licence approvals and conditions that permit open pit mining in this area. Stawell Gold Mines management has advice that a Work Plan variation would need to be obtained prior to the commencement of any mining activity in the Wonga area. The exploration program proposed for 2008 includes an allocation to complete additional exploration on near surface and underground targets within the Wonga deposit area and as detailed plans for mining operation have yet been completed.

Stawell Gold mines have received a Work Plan variation enabling mining and trucking of portions of the Wonga waste dump to the processing plant for milling. These trials will be carried out during 2008 to determine the potential processing grades and viability of treating this material through the Stawell processing facility.

19.9.3 Wonga Underground Mineral Resources

The Wonga Underground Mine was placed on care and maintenance in 1999. After extraction of some of the remnant Mineral Reserves and resources in subsequent years the mine pumping was stopped and the mine allowed to flood. The water level is currently 150mRL and the majority of the existing working underwater.

Should economic conditions change such the remaining Mineral Resource was economic, current licence conditions permit Stawell Gold Mines to recommence operations and extraction of any remaining or newly defined Mineral Resources. During 2008 it is proposed to re-evaluate the exploration potential within and adjacent to the existing Wonga underground mine.

19.9.4 Magdala Underground mineral Resources

Mining and extraction of the Magdala Mineral Resources and Reserves has been continuous since 1981. The authors are not aware of any known environmental, permitting, legal, title, socio-economic, political or other issues that will prevent extraction of the remaining existing Mineral Reserves and Mineral Resources.

Stawell Gold Mines has been able to carry out its mining activities as defined by current Work Plans and approvals without intervention since 1981.

20 Other Data and Information

20.1 Stoping Reconciliation

Stawell Gold Mines personnel maintain a detailed reconciliation register of stoping performance. This reconciles Ore reserve versus Mine recorded versus Mill production. This process has been reviewed by the competent person and is a robust process for determining the applicability of the Mineral Resource and Mineral Reserve estimation processes.

A summary of the reconciliation or Stoping Mineral Reserves to Mill production is shown in Figure 54

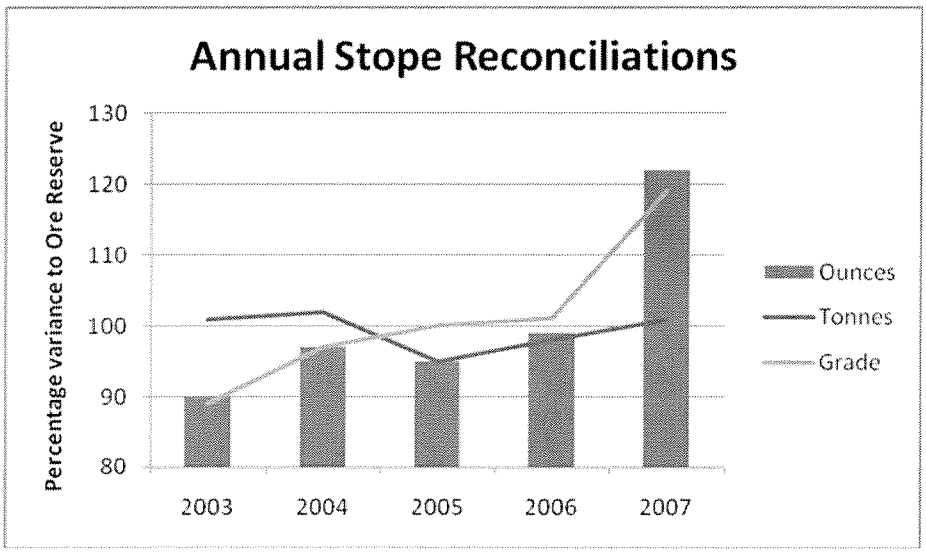


Figure 54 Annual Stoping Mineral Reserve reconciliation for the period 2003 - 2007

2007 reconciliation information has shown significant additional gold has been produced from the Stope portion of the Mineral Reserve. This can be explained by local geologically complex and abnormally high grade mining areas in GG2 and GG3 and is not interpreted to be a systematic Mineral Resource and Mineral Reserve estimation issue. Based on the data currently available it is not expected that this will continue other than local fluctuations in the reconciliation performance.

21 Interpretation and Conclusions

Stawell Gold mines has in place rigorous processes for the estimation of Mineral Resources and Mineral Reserves. These processes are managed on site and appropriate resources (personnel and drilling) are allocated to this function.

The conclusion of the Qualified Persons as defined in this report are that the Mineral Resources and Mineral Reserves as stated in this document are valid and supported by appropriate data collection, sampling, processing, interpretation and estimation methodologies and conform to NI43-101 guidelines.

The Stawell project has significant production history and the site personnel significant knowledge of the orebody and the mining process which adds additional support to the methodologies used to estimate the Mineral Resources and Mineral Reserves.

2007 reconciliation information has shown significant additional gold has been produced from the Stope portion of the Mineral Reserve. This can be explained by local geologically complex and abnormally high grade mining areas in GG2 and GG3 and is not interpreted to be a systematic Mineral Resource and Mineral Reserve estimation issue. Based on the data currently available it is not expected that this will continue other than local fluctuations in the reconciliation performance.

Exploration activities are incorporated into ongoing Life of Mine plans and whilst the details of the proposed plan for 2008 and beyond are still being developed it is the opinion of the Qualified Person that additional exploration is warranted as per the descriptions included in Section 12 of this report.

22 Recommendations

The key recommendations of the Principal author are in respect of ongoing geological testing and evaluation of the Mineral Resources and Exploration areas at Stawell Gold Mines. A detailed exploration program is presented in section 8 of this report including a proposed Exploration program. It is the opinion of the author that there remains sufficient prospectively within the Stawell Gold Mines tenement to support the proposed program and budgeted expenditure and it is recommended

that ongoing evaluation of this potential be continued and re-evaluated as exploration results are received.

It is recommended that Stawell Gold Mines maintain the Mine Geological program as currently implemented to supplement currently available geological information in the preparation of detailed mine design.

A copy of the 2008 MIN5260 Exploration targets and Budget allocations is given in Table 21 2008 MIN5260 Exploration targets and Budget allocations

Target	**Concept/scope of works**	**2008Budgeted Expenditure $M AUD**
Golden Gift 6	Follow up on 2007 drill results to develop a potential resource. Resource Definition	UG Drilling $0.66M
North Magdala	Following up on a surface drill hole intercept from 2005 in SD622 of 9.4m @ 8.35 g Au/t. This intercept is located down dip and along strike of current Mining areas on section 388mN. The follow up drilling wil be completed from underground and possibly the surface.	Surface Drilling $2.2M UG Drilling $1.2M
Wonga open pit and underground	To test potential of pit extensions to the North and South through a series of surface holes. Currently work is being undertaken to a 3D structural model of the Wonga deposit which will be used to identify deeper drill target surrounding the underground mine. These targets are also anticipated to be drilled from surface.	Surface Drilling $0.78M
East Magdala	Test the eastern flank of the Magdala Basalt Dome. Drill testing will occur from underground locations.	UG Drilling $0.65M
Other Targets	Testing of other targets including Golden Gift South	UG Drilling $0.57M
Total Program	**Exploration and Resource Definition**	**$7.06M**

Table 21 2008 MIN5260 Exploration targets and Budget allocations

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24 Additional Requirements for Technical Reports on Development Properties

24.1 Underground Mining method

The mine is accessed by a decline from a portal located adjacent to the mill. The mine access development and services are located mainly within basalt. Ground conditions are good and there is no history of major seismic activity. Development follows the Magdala lode system down plunge, and between 470 RL and 786 RL the decline splits into a north and a south decline to access the Golden Gift ore bodies To facilitate ore access, extraction levels are developed at approximately 20m to 25m vertical intervals. The mining areas currently extend over approximately 3km of strike to more than 1,300m below surface, measured from the top of Big Hill.

The mining method used in the Magdala was bench stoping with cemented rock fill pillars in primary stopes, and rock filled secondary stopes

The mining method used in the Golden gift and narrow Magdala ore zones is retreat open stoping with either cemented rock fill (CRF) if full extraction, or ½ CRF/½ rockfill or all rockfill stope with pillars.

In the Magdala orebody, stope sizes typically range from 2,000 to 10,000 tonnes. In Golden Gift where the width and tenor of the reserves have so far been of higher quality, generally larger stopes up to 15,000 tonnes are expected.

Stope ore is recovered using loaders under direct or remote control of an operator, with haulage by 50 tonne trucks.

The access decline is used as an intake airway, and a chilled water plant delivers conditioned air via an intake shaft. Local spot refrigeration plants are also used for decline advance. Exhaust air is drawn through the workings by a series of ventilation rises and drives by two primary ventilation fan installations located at the northern and southern ends of the mine. The mine is relatively dry. Water pumped from the workings is recycled for use in the mine or the treatment process.

24.2 Open Pit Mining method

Open Pit Mining will utilize conventional benching techniques of drill and blast. Loose overburden will be removed with either a combination of Ripping and Scrapers or conventional truck and shovel / front end loader. The discrete nature of Open pits will necessitate the use of small scale equipment. Ramp widths are kept to a minimum of one lane at 12m as the mining fleet will consist of few trucks. Ore delivery from satellite pits will be through the use on road registered truck and trailer combinations. These will transport ore to the mill during ordinary work hours.

Ramp gradient of 1 in 9 (with a final ramp section at 1:8) will be used and a berm interval of 20m vertical. Batter slope angles are typically 50 deg in weathered material and reach 60deg in fresh rock. Batter slope angles are determined on a site by site basis using geotechnical advice from diamond drill and rock sampling data. Whittle modelling of the resource is used to determine the economic pit shape, with recovery factors of 95% and dilution factor of 10% typically used.

Work-shifts will be typically Monday to Friday during day light hours to minimize environmental impacts to the surrounding community.

Open Pit operations are designed to be backfilled at the end of Mine life to further minimize community impact.

24.3 Production Forecast

Refer to Table 23.

24.4 Recoverability

Refer to Section 18.

24.5 Markets

SGM produces gold doré bars at the Stawell Gold Mine, which are transported to AGR Matthey in Western Australia and refined to produce gold bullion. The Corporation sells the gold bullion over the counter according to its treasury policy through either AGR Matthey or an Australian based bank.

24.6 Contracts

The Corporation sells the gold bullion over the counter according to its treasury policy through either AGR Matthey or an Australian based bank.

The author is not aware of any agreements that are not within market parameters.

24.7 Environmental Considerations

The Corporation is committed to maintaining effective management systems with respect to environmental matters at the Stawell Gold Mine. At SGM, site management monitors and regularly reviews the environmental and social impacts of the operations, such as water quality, air quality, blast vibration and noise. Monitoring and site environmental performance results are shared with regulatory authorities and local communities and are reported quarterly to the board.

SGM promotes responsible environmental behaviour among all employees and contractors. Operations are managed in accordance with SGM's Environmental Management Plan which provides the standards, procedures and guidelines required to achieve that aim.

SGM regularly updates its estimate of future expenditures. The provision at December 31, 2007 for the site closure and reclamation was AUD$A4.547 Million. This estimate is based on available information including preliminary closure plans, alternatives and applicable regulations.

24.8 Taxes

SGM is currently subject to the following taxes and duties:

- Victorian Payroll tax of payable at the rate of 5.05% calculated on wages paid by SGM to its employees. SGM is liable for Victorian Payroll Tax when its total Australian wages exceeds the Victorian general deduction threshold of AUD$45,833 a month or AUD$550,000 over a full financial year; and / or when grouped with other businesses of the Corporation, the combined Australian wages of the group exceed the general deduction threshold level.

- Minor taxes and duties including land tax, insurance duty, mortgages duty, motor vehicle duty debits tax and local council rates.

The Corporation is currently subject to the following taxes:

- Federal Income tax is levied on the taxable income of the Corporation at a rate of 30%. In general terms, taxable income is calculated on assessable income less any allowable deductions.
- Capital Gains Tax (CGT) is paid on any capital gain that the Corporation includes in its annual income tax return. CGT is not a separate tax; it's a component of income tax.
- Fringe benefits tax (FBT) is payable for benefits paid to an employee or the employee's associate by the Corporation. FBT is separate from income tax and is based on the taxable value of the various benefits provided.
- Goods and Services Tax (GST) is a broad-based tax of 10 per cent on the sale of most goods and services and other things in Australia. Being registered for GST enables the Corporation to claim input tax credits for the GST included in the purchase price of goods and services used in the business.
- Victorian Payroll tax of payable at the rate of 5.05% calculated on wages paid it and its subsidiaries to its employees. Corporation is liable for Victorian Payroll Tax when its total Australian wages exceeds the Victorian general deduction threshold of AUD$45,833 a month or AUD$550,000 over a full financial year; and / or when grouped with other businesses, the combined Australian wages of the group exceed the general deduction threshold level.
- Minor taxes and duties including land tax, insurance duty, mortgages duty, motor vehicle duty debits tax and local council rates.

24.8.1 Royalties

Within M5260 there is an AUD$2.00 per Au ounce royalty payable to Mineral Ventures of Australia (MVA). This royalty agreement came into place in February of 2004 and is in place until the earlier of 15 year of production of 2.5 million ounces of gold. Furthermore this royalty agreement extends to Victorian tenements held by Leviathan Resources in February 2004 which included MIN5260.

24.9 Capital and Operating Expenses

Operating Costs (Cash Costs)				
Mining - Underground	30,576	30,042	14,566	**75,184**
Mining - Surface	-	-	7,194	**7,194**
Mining - Mine Geology	3,794	1,229	250	**5,272**
Processing	15,981	14,467	11,116	**41,564**
Administration	4,883	5,693	3,437	**14,013**
Total	55,233	51,431	36,562	**143,226**
Capital Costs				
Mining	19,551	6,575	-	**26,126**
Plant & Equipment	6,515	2,842	1,190	**10,548**
Total	26,066	9,418	1,190	**36,674**
Total Operating and Capital Costs	81,299	60,849	37,752	**179,900**

Table 22 Annual Operating and Capital expenses based on LOM

24.10 Economic Analysis

Table 23 sets forth an economic analysis for the SGM reflecting a cash flow forecast on an annual basis from 1 July 2007. In addition, certain sensitivity analyses are described in Table 24 below. The cash flow forecast in Table 23 has been based upon only the mineral reserve estimate for the SGM as reflected in this Report, representing approximately a three year life.

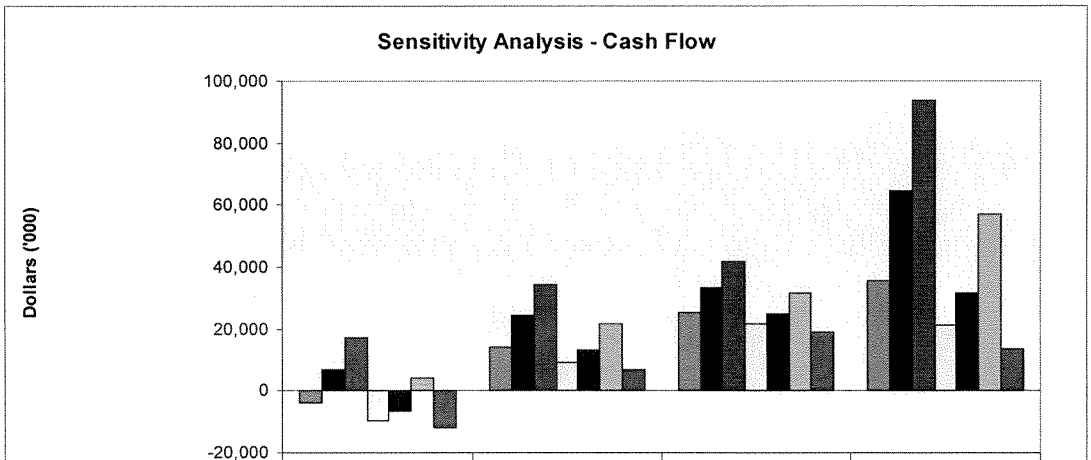
Production				
Underground Production				
Tonnes mined	686,828	666,836	306,974	**1,660,638**
Head grade	5.03	5.20	5.93	**5.26**
Production (oz)	110,978	111,382	58,540	**280,900**
Surface Production				
Tonnes mined	-	-	325,383	**325,383**
Head grade	-	-	2.12	**2.12**
Production (oz)	-	-	22,178	**22,178**
Total Production				
Tonnes mined	686,828	666,836	632,357	**1,986,021**
Head grade	5.03	5.20	3.97	**4.75**
Production (oz)	110,978	111,382	80,718	**303,078**
Surface ore ratio (%)	0%	0%	51%	**16%**
Total Processed				
Tonnes processed	731,199	676,818	955,880	**2,363,897**
Head grade	5.10	5.23	3.18	**4.36**
Production (oz)	120,001	113,759	97,707	**331,467**
Recovery	89%	89%	86%	**88%**
Recovered gold (oz)	106,527	100,681	83,971	**291,179**
Revenue	**$000**	**$000**	**$000**	**$000**
Gold Sales	79,895	75,510	62,979	**218,385**
Interest Received and other revenue	-	-	-	**-**
Total	79,895	75,510	62,979	**218,385**
Spot Price (A$)	$750	$750	$750	
Operating Costs (Cash Costs)				
Mining - Underground	30,576	30,042	14,566	**75,184**
Mining - Surface	-	-	7,194	**7,194**
Mining - Mine Geology	3,794	1,229	250	**5,272**
Processing	15,981	14,467	11,116	**41,564**
Administration	4,883	5,693	3,437	**14,013**
Total	55,233	51,431	36,562	**143,226**
Operating Cash Flow	**24,663**	**24,079**	**26,417**	**75,158**
Capital Costs				
Mining	19,551	6,575	-	**26,126**
Plant & Equipment	6,515	2,842	1,190	**10,548**
Total	26,066	9,418	1,190	**36,674**
Total Operating and Capital Costs	81,299	60,849	37,752	**179,900**
Operating and Capital Cash Margin	**(1,404)**	**14,661**	**25,227**	**38,484**
Corporate Costs	213	183	120	**516**
Resource Definition Costs	2,122	255	-	**2,377**
Borrowing Costs	234	17	(0)	**252**
Cash Margin Before Tax	**(3,972)**	**14,206**	**25,107**	**35,340**

Table 23 Cash Flow Forecast

Notes:

1. *Cash flow excludes rehabilitation, redundancies and leave payout, residual value for sale of plant and equipment, balance sheet wind down, lease payments and corporate costs; and*

2. *Gold price assumptions used in the calculation of cash flow are consistent with SGM's 2007 Mineral Reserve assumptions.*



	Jun-08	Jun-09	Jun-10	Total
Base Case	-3,972	14,206	25,107	35,340
Gold Price A$850	6,680	24,274	33,504	64,458
Gold Price A$950	17,333	34,342	41,901	93,576
Operating cost increase 10%	-9,496	9,063	21,451	21,017
Capital cost increase 10%	-6,579	13,264	24,988	31,673
Grade increase 10%	4,017	21,757	31,405	57,179
Grade decrease 10%	-11,962	6,655	18,809	13,502

Table 24 Cash Flow Sensitivity Analysis. Figures stated in AUD$'000.

24.11 Payback

There is no outstanding debt on the property. Stawell does however have Life of Mine model forecast lease principal payments of $6.15 million. A net present value (NPV) sensitivity analysis over varying discount rates (Table 25) resulted in a LOM NPV range from $29.1 million (base case assuming 0% discount rate) to $19.1 million (14% discount rate). The NPV amount of $29.1 million assuming a 0% discount rate varies to the $35.3 million by the forecast lease principal payments of $6.15 million. Because of the limited mine life at Stawell, variations to the discount rate has a relatively minor impact on the total NPV.

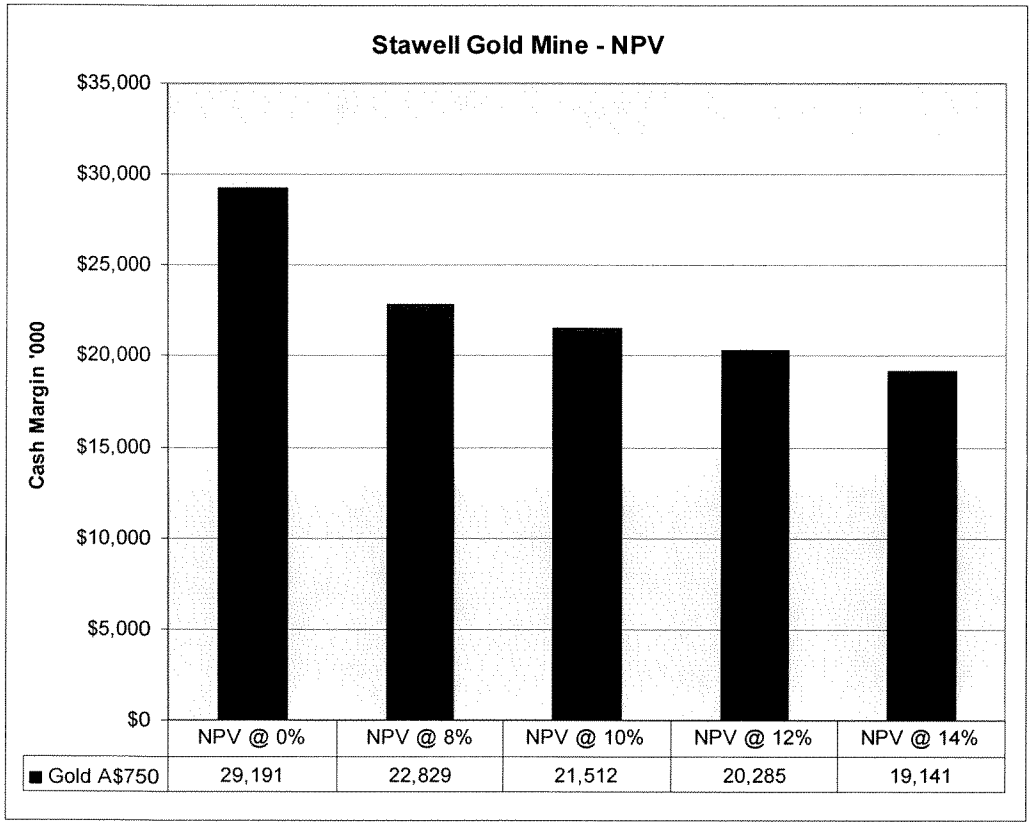


Table 25 SGM NPV sensitivity analysis over varying discount rates.

24.12 Mine Life

Proven and probable mineral reserves are projected to sustain the mining operation until the end of the second quarter of 2010.

The most immediate potential to extend the mine life of the property is the Golden Gift 6 area. This area currently is not in resource but work is continuing during 2008 to examine its potential.

25 Date and Signatures

25.1 Certificate of Qualified Person

Name: Dean Fredericksen
Address: 629 Huntly Road, Orange NSW Australia 2800

I, Dean Fredericksen, MAusIMM do hereby certify that:

1. I am employed as the Principal Consultant, Fredericksen Geological Solutions Pty Ltd.

2. I am the Principal author responsible for the preparation of the report titled "Technical Report on Stawell Gold Mine, Victoria Australia", dated March 28, 2008, and am wholly responsible as the "qualified person" for the Mineral Resources Estimates as at 31 December 2007. I accept responsibility for the preparation of Sections 4 - 18 and 20 and joint responsibility for the preparation of sections 19, 21, 22, 23 and 25.

3. I graduated from the University of Waikato with a Bachelor of Science degree in 1988 and a Master of Science Honours degree in 1989.

4. I am a member in good standing of the Australasian Institute of Mining and Metallurgy and have over 19 years of professional experience.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I have been involved in the Stawell Gold Mining operation since 1995. During the period 1995 - 2002 I was the Chief Geologist for Stawell Gold Mines and was responsible for all site based geological functions including near mine exploration, resource definition and mine geology functions and was the JORC competent person for reporting on Mineral Resources and Ore Reserves. From 2002 - 2004 I was the Group Mine Geologist for MPI Mines Ltd, owner of Stawell Gold Mines and maintained an involvement in the Stawell Project in an advisory and monitoring capacity. From June 2007 I have been consulting to Stawell Gold Mines, providing mentoring to site geological personnel and audit and management advice to the operation. During this period regular monthly visits have been made to Stawell Gold Mines for either one or two week's duration.

7. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I consent to the filing of the Technical Report with any stock exchange and any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated 5th June, 2008

Dean Fredericksen

Signature of Qualified Person

Name of Qualified Person

25.2 Certificate of Qualified Person

Name: Glenn Miller
Address: c/o Stawell Gold Mines, Leviathan Road, Stawell, Victoria, Australia, 3380

I, Glenn Miller, MAusIMM do hereby certify that:

1. I am employed as the Mine Technical Superintendent at Stawell Gold Mines.
2. I have been jointly responsible for the preparation of the report titled "Technical Report on Stawell Gold Mine", dated March 28, 2008. I accept joint responsibility for the preparation of sections 19, 24, and 25.
3. I graduated from Ballarat University College with a Bachelor of Engineering (Mining) degree in 1991.
4. I am a member in good standing of the Australasian Institute of Mining and Metallurgy and have over 16 years of professional experience.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have worked continuously at the Stawell Gold Mines since May 2006.
7. I am not independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I consent to the filing of the Technical Report with any stock exchange and any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated 5th June, 2008

Signature of Qualified Person

Glenn Miller

Name of Qualified Person

25.3 Certificate of Qualified Person

Name: Tamer Dincer
Address: 5 Salter Point Parade, Salter Point 6152

I, Tamer Dincer, BSc & MSc in Mining Engineering, MAusIMM do hereby certify that:

1. I am employed as the Principal Consultant Mining Engineer, Mining Solutions Consultancy Pty Ltd.
2. I have been jointly responsible for the preparation of the Section 19.8.3.2 and Section 24.2 of the report titled "Technical Report on Stawell Gold Mine, Victoria Australia", dated March 28, 2008. I am wholly responsible as the "qualified person" for the Davis Open Pit Extension Mineral Reserve Estimate included in the technical report.
3. I graduated from the Mining Engineering Department of the University of METU in Ankara, Turkey with a Bachelor of Science (Honours) degree in 1986. I have also received the Master of Science degree form the same university in 1988.
4. I am a member in good standing of the Australasian Institute of Mining and Metallurgy and have over 20 years of professional experience.
5. I have read the definition of "qualified person " set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
7. I have read National Instrument 43-101 and Form 43-101Fl, and the Technical Report has been prepared in compliance with that instrument and form.
8. As of the date of this certificate, limited to the sections listed in item 2 above, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I consent to the filing of the Technical Report with any stock exchange and any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated 5th June, 2008

Signature of Qualified Person

Tamer Dincer
Name of Qualified Person

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX A

Wonga Resource Model

July 2007

Ravensgate
Minerals Industry Consultants

WONGA GEOLOGICAL MODEL

for

STAWELL GOLD MINES PTY LTD

RAVENSGATE

01 July 2007

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Ravensgate
Minerals Industry Consultants

WONGA GEOLOGICAL MODEL

Prepared by RAVENSGATE on behalf of:

Stawell Gold Mines Pty Ltd

Author:	Dean Fredericksen	Principal Consultant	MSc Hons, MSAIMM
Reviewer:	Stephen Hyland	Principal Consultant - Director	BSc Geology, MAusIMM, CIMM, GAA
Date:	01 July 2007		
Copies:	Stawell Gold Mines	(2)	
	Ravensgate	(1)	

Dean Fredericksen
For and on behalf of:
RAVENSGATE

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1. EXECUTIVE SUMMARY AND RECOMMENDATIONS

The Wonga mineralisation has been separated into four major structural domains.

The porphyry intrusive have been modelled but not estimated as separate domains.

It is recommended that additional geological interpretation of the porphyry structures is completed with respect to discovering additional mineralisation at depth. The additional interpretation review may reveal additional details associated with structural geometry and mineralization evolution of the Wonga gold deposit system including any other prospective corridors for additional mineralisation.

There is a zone of mineralisation identified from -120mRL to -170mRL that has not been incorporated into this model. If preliminary open pit mining study indicates a pit floor closure at -120m RL then the additional mineralisation may contribute to making economic pit shells larger.

It is recommended that additional large diameter diamond drilling be carried out through this area with the aim of identifying the dominant orientations of the mineralised structures present. In addition all un-assayed drill core for these identified areas should be re logged and assayed where possible.

There remains a significant number intervals of un-assayed core some of which is inside the existing domain envelopes.

It is recommended when the preliminary mining studies have been completed that any un-assayed core in the vicinity of potential mining areas be assayed (if possible) and then updated the modelling process to reflect the additional information.

Variogram studies were completed on 2.0m composites. The 2.0m interval was optimum based on localized numerical studies.

Downhole variogram models clearly defined the short range structures.

Directional Variography on domain 101 identified clear anisotropic structures generally. These were not so apparent for the domain 104.

The density model data was supplied by SGM personnel and include a number of measurements on half diamond drill core and rock chip samples collected within the open pit area. The measurement process used sound techniques and the results are considered appropriate.

The quality of the estimate as measured by the slope of regression of the estimate is poorer when the maximum distance to the nearest composite increases. This limit was set at 30m for the Main Link structures and increased to 50m for the Hanging wall structure.

Based on a number of criteria Mineral Resource classification categories were applied. The Main Link structure within the modelled area can be considered to be Indicated Resource and the remaining structures Inferred Resource.

No "formal" reporting has been completed for this resource review as additional work is required to define the un-mined material that still has "reasonable prospects" of being economic and being mined. An open pit optimisation study commissioned by SGM will assist in this regard.

It is recommended that a pit shell at an inflated gold price be used as a primary reporting tool for the "in-situ" resources.

2. INTRODUCTION

2.1 Qualifications, Experience and Independence

Ravensgate was incorporated in 1997 and specialises in resource modelling and resource estimation services. The company's major global clients include Freeport at Grasberg Mine, Ok Tedi Gold Mine in Papua New Guinea, Goldfields and Newmont in Ghana and many junior resource companies which are ASX (Australian Stock Exchange), TSX (Toronto Stock Exchange) or AIM (London Stock Exchange) listed. Ravensgate has focused upon providing resource estimations, valuations, independent technical documentation, and has been involved in the preparation of Independent Reports for Canadian, Australian and the United Kingdom listed companies.

Dean Fredericksen, MSc Hons, MSAIMM.

Principal Consultant, Ravensgate - Principal Author

Dean Fredericksen is a geologist with over 18 years experience in mining and exploration geology and resource evaluation. Dean has worked on gold and base metal projects in New Zealand and Australia and has spent the last 12 years in senior positions with Stawell Gold Mines, MPI Mines, Newcrest Cadia Valley Operations and Tritton Resources. During this time Dean has been responsible for Mine Geology functions including compilation of Mineral Resource and Ore Reserve estimates, production geology, near mine exploration and has participated in a number of pre-feasibility and feasibility studies for open pit and underground mining projects. Dean Fredericksen holds the relevant qualification and professional associations required by the ASX, JORC and Valmin Codes in Australia.

Stephen Hyland, BSc Geology, MAusIMM - CIMM - GAA.

Principal Consultant, Ravensgate - Reviewer

Stephen Hyland - Principle Consultant - Ravensgate Resource Consultants
BSc Geology, Member of Australasian Institute of Mining and Metallurgy - CIMM - GAA

Stephen Hyland has extensive experience of over 20 years in exploration geology and resource modelling and has worked extensively within Australia as well as offshore in Africa, Eastern and Western Europe, Central and South East Asia, modelling base metals, gold, precious metals and industrial minerals. Stephen's extensive resource modelling experience commenced whilst working with Eagle Mining Corporation NL in the extensive Yandal Gold Province where for three and half years he was their Principal Resource Geologist. Whilst the majority of his time there had been developing the now successful Nimary Mine, he also assisted the regional exploration group with preliminary resource assessment of Eagle's numerous exploration and mining leases. Since 1997 Stephen has been a full time Consultant with the consulting firm Ravensgate where he is responsible for all geological modelling and reviews, mineral deposit evaluation, computational modelling, resource estimation, resource reporting for ASX / JORC and other regulatory compliance areas. Primarily Stephen specialises in Geological and Resource Block Modelling generally with the widely used Minesight 3D mine-evaluation and design software. Stephen Hyland holds the relevant qualifications and professional associations required by the ASX, JORC and ValMin Codes in Australia. He is a Qualified Person under the rules of the CIMM and NI43-101.

2.2 Disclosure

Perseverance Corporation Limited is the beneficial owner of the Wonga Gold Project. This resource block model report for describes the resource definition and project review assumptions and is based on information provided by the title holders and project owner, along with technical reports by consultants, associated contractors, previous tenement holders, and other relevant published and unpublished data for the area.

In accordance with the JORC Code (2004) guidelines, this report compiled by Ravensgate discusses technical aspects of the project and provides information on data interpretation and usage. The Ravensgate Resource Modelling team has acted as the primary auditor for aspects relating to data quality for the Wonga Gold Project and ongoing Development Program and as such, attest to its integrity.

The Ravensgate Resource Modelling team has endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the technical data upon which this report is based. It should be noted, however, that some aspects related to data quality require attention prior to any further work being carried out.

Ravensgate's Resource Modelling personnel, associated site personnel and contractors involved in the preparation of this Independent Resource Report are all considered Qualified Persons according to Appendix 5A of the ASX listing rules and the JORC Code (December 2004).

Neither Ravensgate nor any of its employees or associates is an insider, associate or affiliate of Stawell Gold Mines ('SGM') or any associated company. Neither Ravensgate nor any of its affiliates have acted previously in any capacity for Ironbark or any of its associates or affiliates.

Ravensgate's professional fees are based on time charges for work actually carried out, and are not contingent on any prior understanding concerning the conclusions to be reached.

2.3 Scope of Works

Michelle Voss Principal Resource Geologist, Perseverance Corporation requested that Ravensgate complete a Geological Model and a Mineral Resource estimate model based upon a recent recompilation of the geological data for the Wonga Deposit, Stawell Gold Mines.

The scope of works for the estimate was detailed as follows:

Complete a geological model for the Wonga Project and deliver such model to the Perseverance Corporation representative in a format suitable for open pit mine planning prior to 15 June 2007. The modelling work will include:

- Review of status of interpretations
- Wire-framing as required based on interpretations supplied
- Drill-hole coding and compositing
- Domain statistics
- Variography and neighbourhood Analysis
- Block model estimation
- Validation
- Classification
- Documentation
- Project debrief and Management review

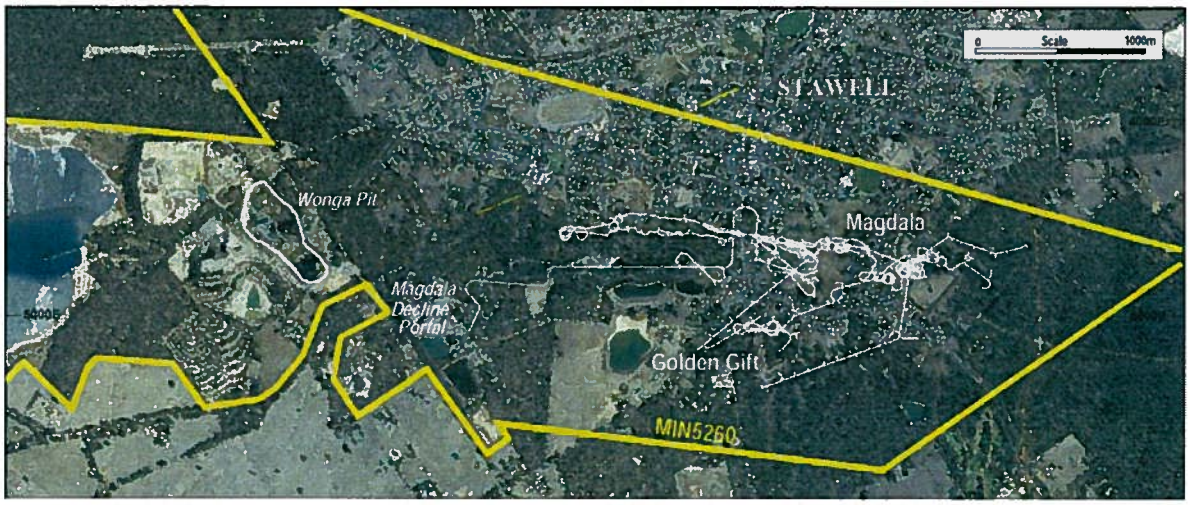
2.4 Project Location

The Wonga deposit is located on the southern margin of the township of Stawell in Victoria within mining lease MIN5260 (Figure 1) * (~Lat -37.15 S, Long 142.85 E). The deposit is located some 1km south from the processing and mine facilities and is readily accessible from public roads adjacent to the mine site and facilities.

The Wonga deposit has been previously mined by both open pit (1982 - 1985) and by underground methods (1985 - 1999). Underground mining was completed in 1999 after which the mine was placed on a care and maintenance program. Eventually the decision was made to stop mining on the deposit and the mine pit has been allowed to flood with water.

A waste refuse station and rubbish tip has been established on the southern limits of the deposit adjacent to the crest of the existing open pit. Some backfilling of the pit has occurred with waste rock generated from the underground development.

Figure 1 Plan and aerial photograph showing the location of the Wonga deposit within the mining operations of Stawell Gold Mines.



2.5 Company Supplied Data

A series of data files were supplied to the author by Daniel Henry, Senior Mine Geologist Stawell Gold Mines. The files contained base data that was used to compile the geological model and subsequent resource estimate. The data summary is as follows:

- Wonga Minesight project with all the strings and preliminary geological solids as well as a copy of underground development and topographic files.
- Minesight data files PIT10.dat, PIT11.dat and PIT12.dat. The PIT11.dat and PIT12.dat are drill-hole files containing:
  - Drill-holes that are generally RC or Diamond Core
  - Drill hole location data - including collar and down-hole surveys as extracted from the mines operational Acquiretm database
- Assay and geological information in a range of selective fields.
- A series of strings made in cross sectional views representing ore envelopes and internal waste boundaries. All strings were "snapped" to drillholes. The lower "cut-off" for the ore zone has been based on the nominal grade of >0.50g Au/t above the 150m RL and at >1.00g

Au/t below the 150m RL. The strings overall were constructed from surface to an elevation of approximately -30m RL. Surface elevation at Wonga is approximately +250m RL.

- A number of interpretations of different porphyry dykes and of the Stawell Granite contact has been used and where necessary knowledge has been condensed for the purposes of generating overall representative lithological models.

Ravensgate has not completed detailed validation of the primary drillhole data other than being aware it been validated upon input into MineSight via standard validation routines. The geological interpretations have been checked for validity and details these processes are given in section 3 of this report.

3. WONGA GEOLOGICAL MODEL

3.1 Geological Setting

The Wonga Deposit is situated in the contact aureole of the Stawell Granite and one other significant granite body located on the southern margin of the township of Stawell.

The significant geological points pertinent to constructing a geological model and Mineral Resource estimate are related to the timing of mineralisation and intrusive events and the general nature of the mineralisation.

- The porphyry intrusives are pre-mineralisation and are generally cross cut by, although in some cases are parallel to the gold mineralizing structures. These porphyritic intrusive potentially host economic mineralisation.
- The mineralization event pre-dates the final emplacement of the Stawell Granite with intact blocks of mineralisation incorporated as xenoliths on the margin of the granite. The granite is observed to be intruding into the mineralised structures on several levels.
- Mineralisation is dominantly very fine grained arsenopyrite which has associated fine grained gold both within and along the arsenopyrite grain boundaries. The arsenopyrite is found as disseminations around the margins of single event quartz veins. The vein arrays which are related to significant faults structures are interpreted to have been active during the mineralisation event. Geological observation suggests the mineralisation event was not long lived or periodic in nature.
- A number of mineralised structures have been identified and previously exploited both in historical and recent times.

3.2 Mineralisation Domains

The interpretations supplied by SGM consisted of one single mineralised envelope fixed to the Hanging wall and Footwall of the overall mineralisation (Link - Hanging wall structures) and two internal waste domains.

A review of these shapes was made to ensure continuity of the envelopes and review the application of the cut-off strategies used by SGM personnel. During this process some minor changes were made to the southern limits of the mineralisation and waste envelopes.

The single mineralised domain supplied by SGM was clipped with the internal waste solids to generate the solids or domains that would be used to code the block model and limit the estimation. Further subdivisions of this solid were made based upon significant changes in strike and dip and to reflect the known structural domains. All resulting solids were checked for crossing triangles and openings.

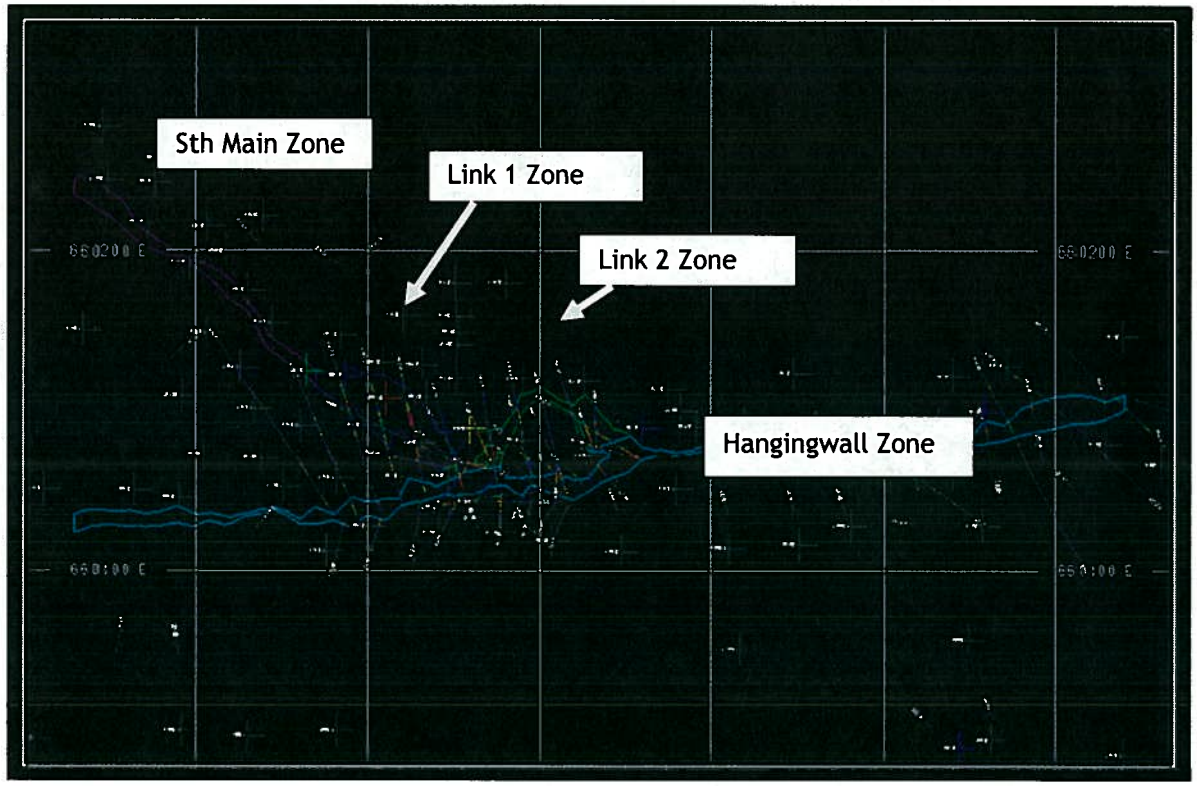
Note: The ore envelopes were clipped to the granite and top surface contacts but not the current topography.

Four major zones or domains were generated as shown in Figure 2:

- The south main mineralisation zone
- Link structure 1
- Link structure 2 (Dominantly within the mined pit shell)
- Hangingwall structure.

The domain numbers and structural orientations are detailed in Table 1.

Figure 2 Plan projection showing major mineralised domains constructed for controlling the estimation



In addition to these domains the granite contact in the vicinity of the mineralised structures was reviewed and where necessary adjusted to better reflect the geological information supplied in the drillhole set.

The porphyry intrusives are included in the MineSight model but have not been used as separate estimation domains given that the mineralisation does cross cut these structures. Adding these additional domains would not have added significantly to the value of the estimate.

It is recommended that additional geological interpretation of the porphyry structures is completed in respect of future potential for discovering additional mineralisation at depth. These may provide some indication with respect to the structural architecture and evolution of the mineralised system and also provide information helpful for the discovery of additional prospective corridors of mineralisation.

During the review of the mineralised domains it was apparent there is significant mineralisation within a zone between the Link 1 and hanging wall domains from -120 to -170m RL. This has not been included in current domain shapes as this area has not yet been comprehensively modelled in this review because the core has been selectively assayed and the orientation of the veining or mineralised structures is not reliably known. The level plans shown in Figures 3 and Figure 4 outline the approximate area of interest.

It is recommended that if possible additional large diameter diamond drill-holes be completed through this area with the aim of identifying the dominant orientations of the mineralised structures present. In addition all un-assayed core should be re-logged and assayed where possible.

Figure 3 Level Plan -120mRL - Area of Interest highlighted.

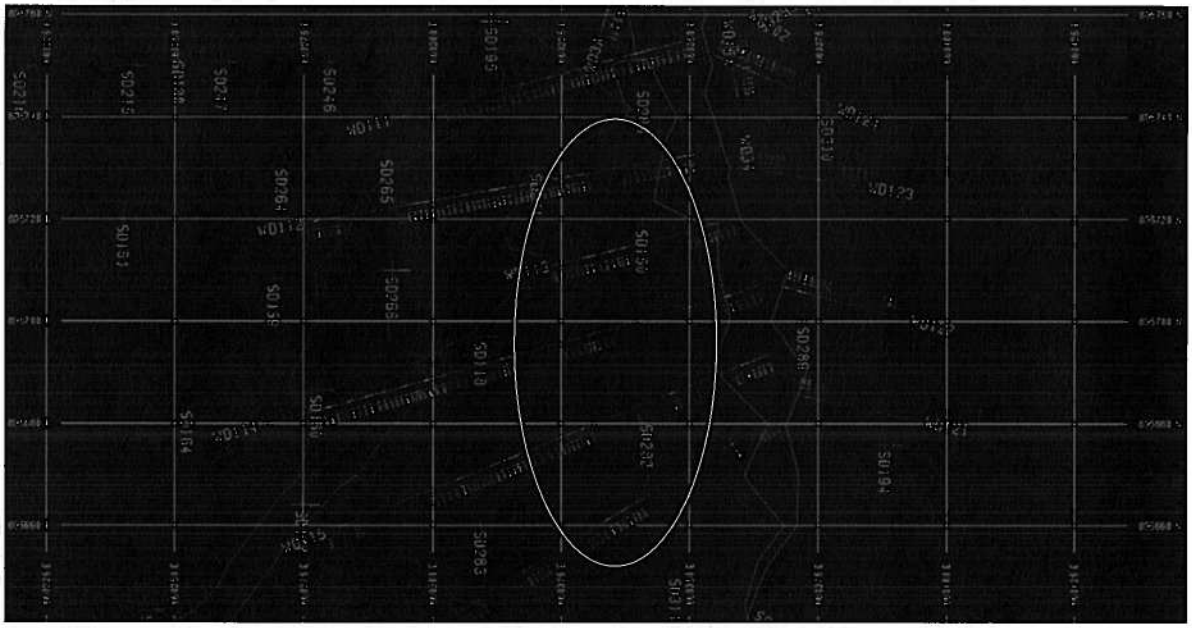


Figure 4 Level Plan -130mRL Area of Interest highlighted.

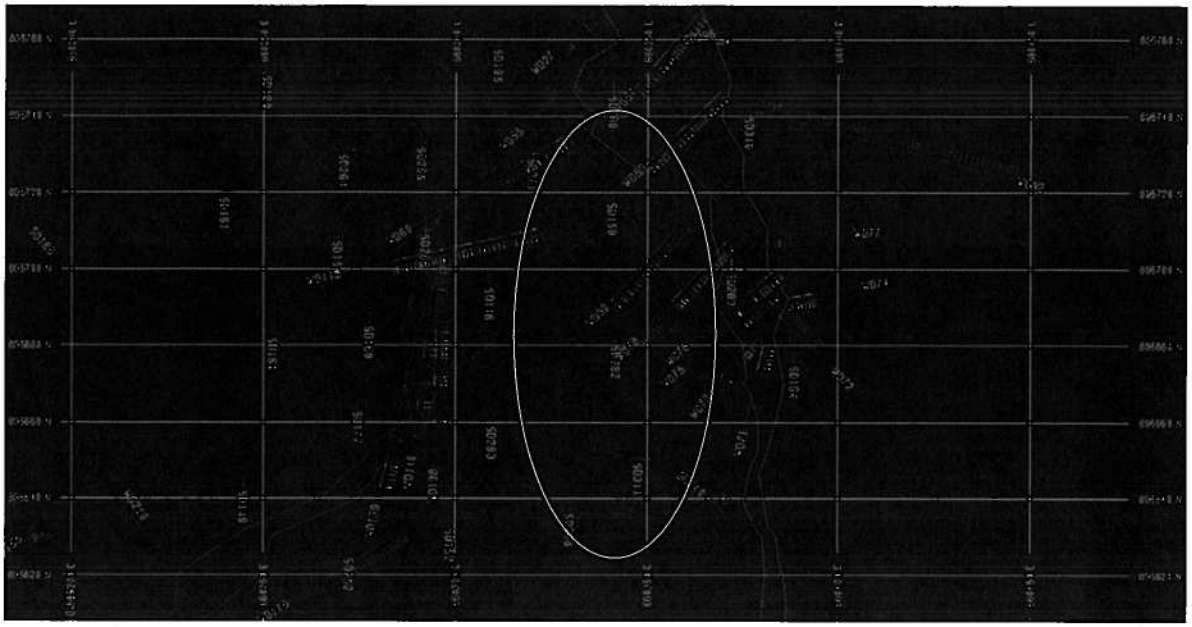


Table 1 Domain Structural Features and Domain Codes			
Domain	**Strike (degrees AMG)**	**Dip**	**Zone Code**
Southern Main Mineralisation	043-047	-42 to -48 degrees to SE	101
Link 1 Mineralisation	055	-45 degrees to SE	102
Link 2 Mineralisation	101-020	-28 degrees to ESE	103
Hanging-Wall Structure	350-000	-30 to -35 degrees to E	104
Granite			200
Waste Tip			300

3.3 Oxidation Domains and Bulk Density Modelling

A series of surfaces representing the base of the highly oxidised rock and the start of the fresh or un-oxidised material were supplied by SGM geologists whilst on site.

The objective of this "oxidation state" domain definition was to control the ore material bulk density that would be applied to the block model tonnage estimations. A number of density measurements were made by SGM geologists and supplied in a memo (Appendix C) with recommended values for both ore and waste within the model area. The recommended values are shown in Table 2.

Table 2 Recommended Densities by Weathering Domain as supplied by SGM	
Domain	**"Bulk Density" (t/m^3)**
Heavy oxidation	2.45
Moderate oxidation	2.65
Fresh rock	2.85

3.4 Drillhole Data Set, Coding and Compositing

The drillhole data set consists of several drill-hole types. The majority of assay data used for this study are derived from percussion RC samples and diamond core which has been collected on site during different drilling programs or phases and over quite a long period of time. No validation of this data set has been completed by Ravensgate and the data has been used as supplied SGM.

3.4.1 Drillhole Coding

Drillholes were coded based on the majority "contained in" envelope method and was coded according to the 50% within the appropriate solid method.

The "correctness" of the coding was checked visually on screen section by section. Some minor manual adjustments made to drillhole intervals based on cut-off grade criteria. Despite carefully snapping strings to many of the drill-holes, some intervals were not always effectively coded and required refinement. The wireframes in these instances were either adjusted to fit the drill-holes or sometimes manually coded in order to maintain the integrity of the initial interpretation and minimize any volume issues assuming they were considered to be small. Code Numbers applied were as per Table 1.

It is evident that there are significant number of drillhole intervals which have not been assayed by previous project operators and as such these are not included within the drillhole set. There are a number of these intervals that fall within the mineralised envelopes. A table showing these intervals is shown in Appendix A.

3.4.2 Compositing

Prior to selecting a composite interval a check of the raw assay sample interval lengths was completed. The statistics shown in Table 3 apply to assay intervals of the data set.

Table 3 Raw Assay Sample Interval Statistics			
Cut-off -Al- metres	**# Above Cut-off**	**% Above Cut-off**	**Mean (m) Above Cut-off**
0.00	5689	100.00	1.1
0.25	5675	99.75	1.1
0.50	5580	98.08	1.1
0.75	4987	87.66	1.2
1.00	4655	81.82	1.2
1.25	1452	25.52	1.7
1.50	1170	20.56	1.8
1.75	692	12.16	2.0
2.00	684	12.02	2.0
2.25	14	0.24	4.4
2.50	11	0.19	5.0
2.75	11	0.19	5.0
3.00	10	0.17	5.3

The sample interval less than 2.0m contains the last most significant number of samples and results in least splitting of existing sample intervals

Selecting an appropriate composite interval is generally a compromise between choosing an interval close to the block height (bench composite), maintaining uniform "sample support" and rationally reducing the variance of the data set, whilst simultaneously maintaining sufficient samples for a meaningful resource estimate. Sample splitting is not ideal as it will sometimes result in artificially lower sample or composite variances than if the longer samples were used in their un-modified form. In consideration of all these factors it was decided that the 2.0m composite be considered appropriate.

Compositing was carried out after coding of assay intervals and honoured the geological coding

Minesight compositing functions enable smaller composite intervals to be included in the preceding interval. The threshold for this was selected at 1.0m and as such composite intervals should range in length from 1.0 - 3.0 metres. There will however remain a small percentage of composites less than 1.0 metre in length which are a result of the un-assayed intervals being "left over" during the compositing process.

It is recommended when the preliminary future mining studies have been completed that any un-assayed core in the vicinity of potential mining areas be assayed (where possible) so that the modelling process can be continued and updated to reflect this additional information.

3.5 Domain Statistics

Domain statistics are shown in Table 4. These include the raw composite statistics as well as the declustered cell statistics for each domain. The Coefficient of Variation for all estimation domains are relatively low for a structural controlled gold deposits being between 1.00 - 1.23.

Declustered cell statistics were calculated at range of cell spacings. The lowest declustered mean grade is shown. This data may be used as a method of validating the final estimations.

A review of the populations for outliers or potential cut values was also completed. A log probability plot is shown in Figure 5. It is the author's opinion that top cutting should not be applied to the composite data set for the estimation. As such no top cuts have been applied to the data prior to estimation.

Table 4 Estimation Domain Statistics Calculated on Composite Intervals

Domain	Zone Code	# of Comps	Mean Grade (Raw)	CV	Mean Grade Declustered
Southern Main Mineralisation	101	490	2.41	1.16	2.45
Link 1 Mineralisation	102	578	2.84	1.09	2.66
Link 2 Mineralisation	103	231	2.00	1.00	2.02
Hangingwall Structure	104	1461	2.64	1.23	2.35

3.6 Variography and Neighbourhood Analysis

A number of different methods were used to estimate a suitable variogram model to control the estimation. All of this work was completed within the Minesight suite of programs on 2.0m composites flagged with the geological domains in which they fall.

3.6.1 Down Hole variogram models

The experimental down-hole variogram was calculated and modelled using all composites that fell within the mineralised structures. This approach provides more data pairs at the close spaced range to get a better overall estimate of the nugget that will be applied to latter variogram models.

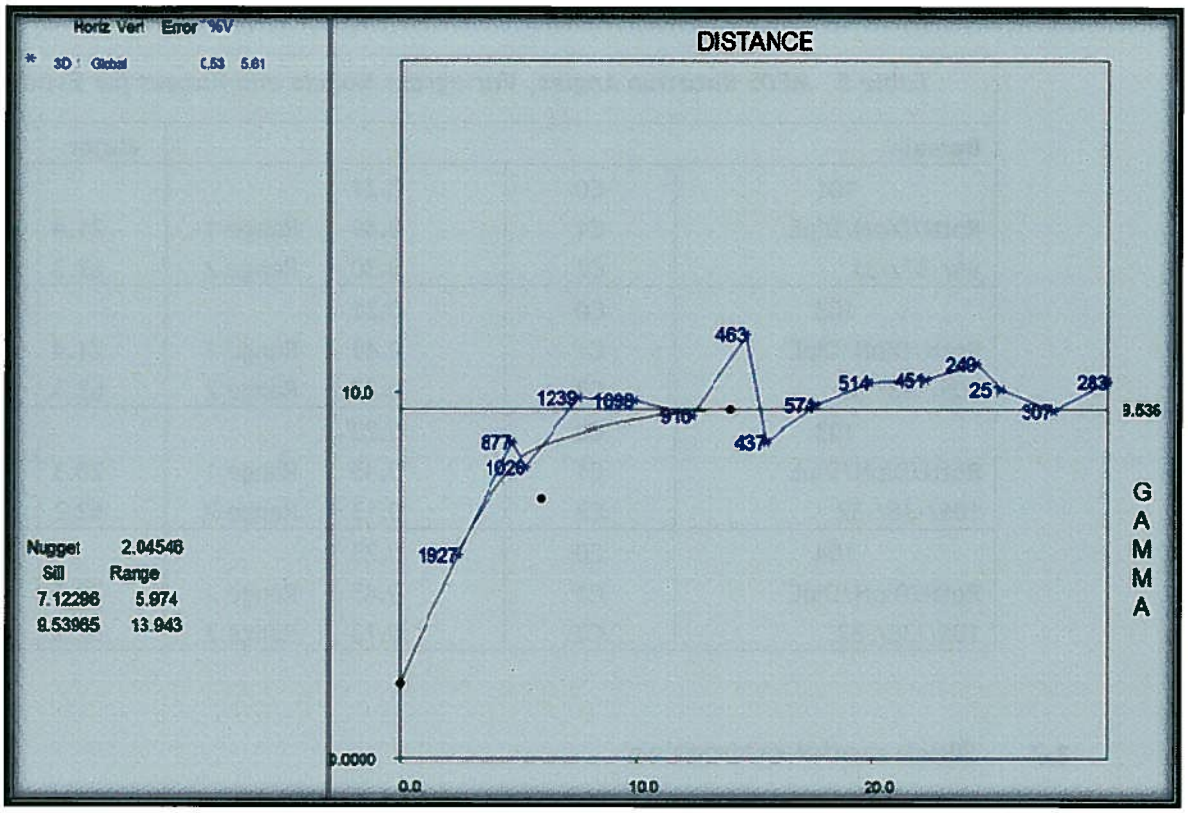
Domain Statistics indicate very little if any statistical variation between domains which provides further validation with respect to using the entire data set to estimate the model.

MineSight software enables the variogram to be calculated on the raw data (normal) or on transformed data such as Logs, Covariance, Relative, Pariwise relative or correlograms.

The downhole variograms were estimated on the normal data set which enables the direct calculation of the nugget as a percentage of the overall sill of the data.

The downhole variogram model is shown in Figure 5. The nugget is modelled at 21% of the sill as from what is a relatively well structured experimental variogram model.

Figure 5 Downhole Experimental Variogram on 2.0m composites for all mineralised domains and the resulting variogram model.



3.6.2 Directional Variograms

The variograms were modelled in the plane of the structure which is appropriate for lode or single mineralised structure observed for the main orebodies within the Wonga deposit. As such experimental variograms were calculated from along strike (0 degrees) around the lower hemisphere to along strike in the opposite direction (180 degrees). This enables analysis of the anisotropy or "plunge" analysis of the variogram trends within the structural domain.

The rotations (MEDS format, RotN, DipN, DipE) applied for all domains are shown in Table 5.

Domains 101 and 102 were modelled as a combined data set. The changes in direction are subtle and tests were completed on both data sets individually to check the validity of the models chosen.

Variography for domain 103 was not carried out as it is entirely within the mined Wonga Open pit and will not directly influence any of the current mining studies. The variogram models for domain 104 were used in the estimations and were rotated into the orientation of the domain.

Domain 104 did present some problems in obtaining well structured variograms. All transforms available were used, however a review of the data using level plans did not indicate strong trend directions. As such the resulting variogram model chosen is close to spherical within the plane of the structure.

The cross strike variogram used was the down-hole which although not entirely valid is a good estimate of the across strike structure as much of the drilling is oriented across the structural domains.

The directional experimental variograms are shown in Appendix B and the resulting variogram models are detailed in Table 5.

Table 5 MEDS Rotation Angles, Variogram Models and Ranges for Estimation Domain

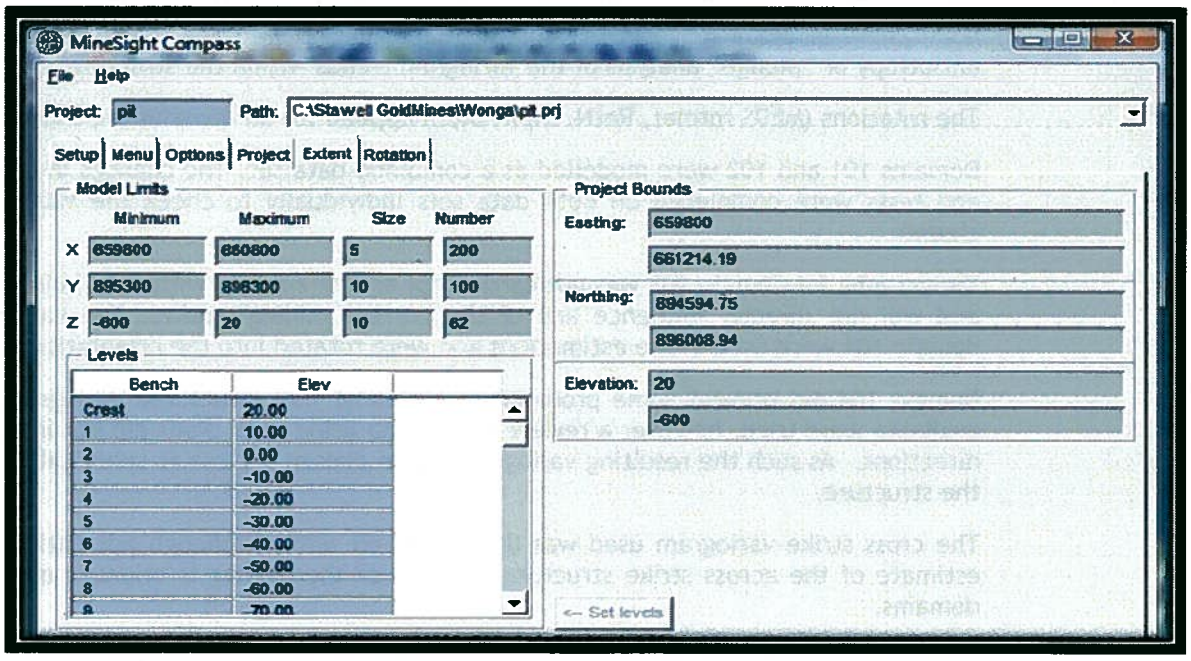
Domain				Major	Minor	Vertical
101	C0	0.21				
RotN/DipN/DipE	C1	0.49	Range 1	21.4	10.95	5.97
96/-37/-31	C2	0.30	Range 2	62.3	31.70	13.90
102	C0	0.21				
RotN/DipN/DipE	C1	0.49	Range 1	21.4	10.95	5.97
106/-38/-32	C2	0.30	Range 2	62.3	31.70	13.90
103	C0	0.23				
RotN/DipN/DipE	C1	0.45	Range 1	20.3	15.00	5.97
106/-38/-32	C2	0.13	Range 2	62.2	50.00	13.90
104	C0	0.23				
RotN/DipN/DipE	C1	0.45	Range 1	20.3	15.00	5.97
106/-38/-32	C2	0.13	Range 2	62.2	50.00	13.90

3.7 Block model estimation

3.7.1 Model Extents

The model extents and cell sizes are shown in Table 6.

Table 6 Wonga Model Extents and Cell Sizes.



The chosen model dimensions of 10 m X 5 m X 10 m are a compromise between adequate sample to block support geostatistics and reasonable selective mining unit (SMU) requirements. To optimise the kriging neighbourhood the block dimension would have ideally been closer to 20 m X 20 m in the Y and Z directions or perhaps larger again however it was felt a model of these dimensions could compromise the outcomes of a mining study with respect to overly large cell (SMU) dimensions.

3.7.2 Model Items

A Minesight block model with the items shown in Table 7 was created. The model file for use in studies is PIT15.fin.

Table 7 Block Model Items and Descriptions for PIT15.fin

PIT15.fin	Item Descriptors			
Label	**Minimum**	**Maximum**	**Prec.**	
TOPO	0	100	1	Topo% from QASCO DTM
ZONE	1	999	1	Zone as per zone table
ORE%	0	100	1	Percentage Ore/Zone per block
AUPPM	0	99	0.01	Kriged Au grade for block
AU1	0	99	0.01	NA - (not used)
AU2	0	99	0.01	NA - (not used)
HOLES	0	99	0.01	Number of holes used to estimate a block
SG	0	5	0.01	Assigned Bulk Density as per Oxide Domain
SLOPE	0	1	0.01	Slope of Regression of Estimate
DIST	0	50	1	Distance to nearest composite for estimation
CMPS	0	99	1	No of Composites used to estimate the block
OTYPE	0	10	1	NA - (not used)
WTYPE	0	10	1	NA - (not used)
OXID	0	10	1	Oxide 1=High Ox, 2=Mixed Ox, 3=Fresh Rock
VOID%	0	100	1	UG void %
RESCA	0	5	1	Resource Category : 1=Meas, 2=Ind, 3=Inf.
DEPLT	0	5	1	NA - (not used)

3.7.3 Model Coding

The model was coded by majority domain code with a minimum percentage inside the domain of 1%. The percentage inside the solid was stored as this will be required for resource volume and tonnage summaries.

The model is classified as an "Ore %" model and as such future calculations should be completed that way.

3.7.4 *Estimation Parameters*

The estimation method was ordinary kriging. The variogram parameters given in Table 8 were used as part of the primary inputs. In addition a series of kriging neighbourhood tests were completed for each domain to identify the optimal search strategies. These were conducted using the Minesight debug routine which outputs the following:

- The number of composites used to krige the block
- The local kriging weights or variances
- the slope of regression of the estimate

Several tests were conducted using various search strategies attempting to reduce the number of negative kriging weights and optimizing the slope of regression of the estimate so that it was as close as possible to 1.0. The biggest controlling factor in the slope of regression results of the estimate is the distance to the closest composite. As this distance increases there is a marked decrease in the slope of regression of the intercept. Both of these values are stored within the block model as part of the estimation process.

As stated previously, increasing of the block size (SMU) would have had significant impact on this parameter.

Following a number of these analyses the kriging parameters for each domain were chosen. These are shown in Table 8.

Table 8 Kriging Estimation Parameters by Estimation Domain

	Domain 101	**Domain 102**	**Domain 103**	**Domain 104**
Min # Composites	10	10	10	10
Max # composites	50	50	50	60
Min Distance to Nearest Composite *If no Composite within 30m do not estimate the block*	30	30	50	50
Search Type	Quadrant	Quadrant	Quadrant	Quadrant
Max # Per Quadrant	15	16	16	20
Max # per hole	N/A	N/A	N/A	N/A
Search Major	100	100	100	100
Search Minor	50	50	80	80
Search Vertical	25	25	20	20
RotN/DipN/DipE	96/-37/-31	106/-38/-32	106/-38/-32	106/-38/-32

The search quadrants were oriented with the search ellipse parameters or using the plane of the search rather than as a straight vertical search.

Each domain was estimated separately with the boundaries of coincident domains being used as "soft boundaries" allowing composites from neighbouring domains to be used in the estimate.

The Minesight run files used for interpolation of the main AUPPM Kriged gold item is shown in Appendix D.

3.8 Validation

The primary validation tools used were domain statistics as presented in Table 9. The mean estimated grades for all domains other than domain 104 compare very favourably with the declustered mean grade of the composites for each domain.

In addition "on screen" checks were completed particularly on the margins of the "higher grade shoots" as these were identified during or after the estimation process.

The model appears after review to be a valid resource estimate based on geological and mineralogical understanding of the deposit and is in line with the data supplied and the methods used.

Table 9 Model Domain Statistics Weighted by % Ore in the Block and Declustered Cell Statistics.

Domain	Zone Code	Mean Declustered Grade	Model Grade
Southern Main Mineralisation	101	2.45	2.45
Link 1 Mineralisation	102	2.66	2.61
Link 2 Mineralisation	103	2.02	2.23
Hangingwall Structure	104	2.35	2.10

3.9 Classification

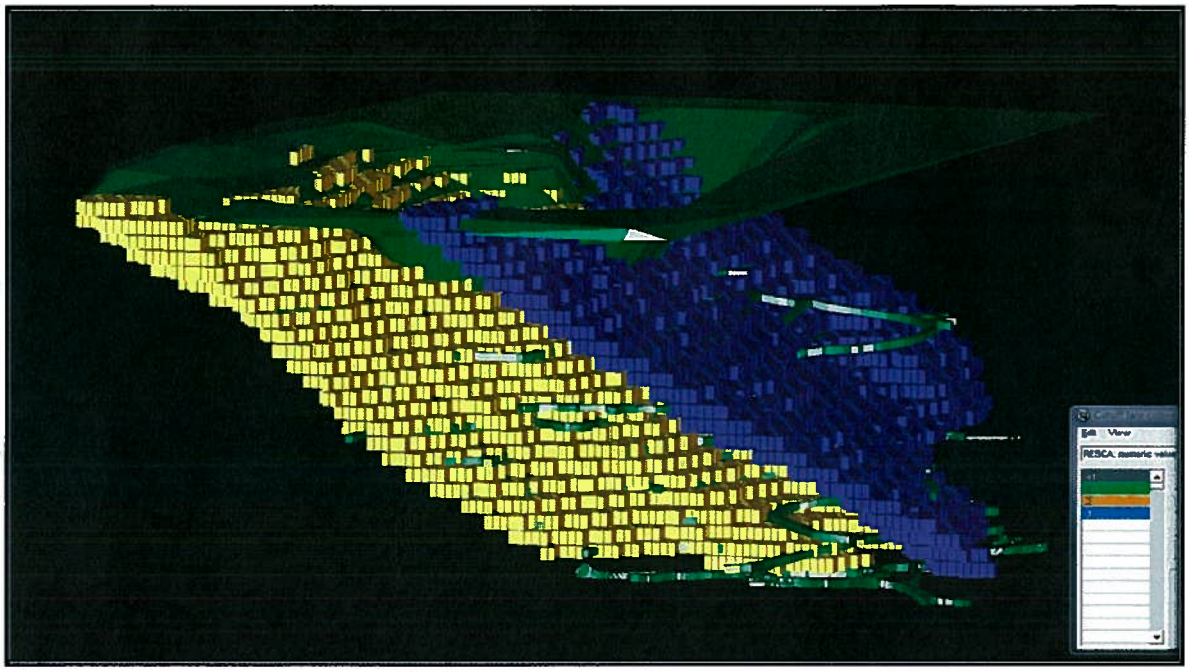
There are many methods used to classify Mineral Resources. For this model a review of the information density as measured by the distance to the nearest composite to a kriged block as well comprehensive visual reviews. The slope of regression of the estimate and some intuition based on the ease of obtaining variogram models and coherent interpretations the following classification scheme is proposed and has been loaded to the block model

Indicated Resource - Domains 101 and 102

Inferred Resources - Domains 103 and 104. This is a relatively low level of classification despite there being significant mining on parts of the hangingwall structure. If un-mined potentially economic mineralisation is identified then this classification can be further reviewed on a local basis using the above mentioned parameters.

The classified model is shown using a "block colour basis" in Figure 6.

Figure 6 Orthographic view of the Wonga block model with blocks coded for Resource Category - Gold



Rescat = 2 (Indicated Resource), Blue,

Rescat = 3 (Inferred Resource).

The existing open pit void and underground working are shown in green

3.10 Mining Constraints

The block model created was depleted for existing UG mining by utilising the voids supplied by SGM. These include the surveyed level outline solids, design stope shapes and CMS stope shapes (only 1 supplied within area of interest). The shapes were checked and where required adjusted to ensure they were valid and would code effectively.

For the Link Structure domain where it could reasonably be assumed that stoping had occurred or where it was intended to mine all of the mineralised structures then the mineralised domain was "cookie" cut with stope voids shapes. This was so as to not leave small zones of ore in the model where there is deemed to be no reasonable prospects of that ore ever being mined . This process still needs additional work by a qualified Mining Engineer to complete the mining potential component of the resource and reserve classification.

- A VOID% item was populated with the estimated percentage void (mining depletion) for underground blocks only.
- A TOPO item was also populated for the percentage of the block that is below the topographic surface.
- A combination of the two allows an estimate of the remaining mineralisation volumes and tonnages available for mining.

3.11 Mineral Resource Reporting

A Mineral Resource report has not been completed for this model as yet. As previously discussed this should await the results on an open pit optimisation study and further work on the impact of underground voids on being able to estimate the remaining "in-situ" resources.

It is recommended that a pit shell at an "inflated" gold price be used as a primary reporting tool for estimating the remaining "in-situ" resources.

The block model is immediately available as an ASCII text file for forwarding to a qualified Mining Engineer for the use in future pit optimisation studies. The details of the exported ASCII file and additional Minesight files supplied is given in Appendix E.

4. GLOSSARY

aeromagnetic	A survey undertaken by helicopter or fixed-wing aircraft for the purpose of recording magnetic characteristics of rocks by measuring deviations of the Earth's magnetic field.
airborne geophysical data	Data pertaining to the physical properties of the Earth's crust at or near surface and collected from an aircraft.
aircore	Drilling method employing a drill bit that yields sample material which is delivered to the surface inside the rod string by compressed air.
alluvial	Pertaining to silt, sand and gravel material, transported and deposited by a river.
alluvium	Clay silt, sand, gravel, or other rock materials transported by flowing water and deposited in comparatively recent geologic time as sorted or semi-sorted sediments in riverbeds, estuaries, and flood plains, on lakes, shores and in fans at the base of mountain slopes and estuaries.
alteration	The change in the mineral composition of a rock, commonly due to hydrothermal activity.
andesite	An intermediate volcanic rock composed of andesine and one or more mafic minerals.
anomalies	An area where exploration has revealed results higher than the local background level.
antiformal	An anticline-like structure.
Archaean	The oldest rocks of the Precambrian era, older than about 2,500 million years.
assayed	The testing and quantification metals of interest within a sample.
Au	Chemical symbol for gold.
Bedrock	Any solid rock underlying unconsolidated material.
Bulk Density	
carbonate	Rock of sedimentary or hydrothermal origin, composed primarily of calcium, magnesium or iron and CO_3. Essential component of limestones and marbles.
chert	Fine grained sedimentary rock composed of cryptocrystalline silica.
chlorite	A green coloured hydrated aluminium-iron-magnesium silicate mineral (mica) common in metamorphic rocks.
clastic	Pertaining to a rock made up of fragments or pebbles (clasts).
depletion	The lack of gold in the near-surface environment due to leaching processes during weathering.
dolerite	A medium grained mafic intrusive rock composed mostly of pyroxenes and sodium-calcium feldspar.
ductile	Deformation of rocks or rock structures involving stretching or bending in a plastic manner without breaking.
dykes	A tabular body of intrusive igneous rock, crosscutting the host strata at a high angle.
erosional	The group of physical and chemical processes by which earth or rock material is loosened or dissolved and removed from any part of the Earth's surface.
fault zone	A wide zone of structural dislocation and faulting.
feldspar	A group of rock forming minerals.

felsic	An adjective indicating that a rock contains abundant feldspar and silica.
foliated	Banded rocks, usually due to crystal differentiation as a result of metamorphic processes.
follow-up	A term used to describe more detailed exploration work over targets generated by regional exploration.
g/t	Grams per tonne, a standard volumetric unit for demonstrating the concentration of precious metals in a rock.
gabbro	A fine to coarse grained, dark coloured, igneous rock composed mainly of calcic plagioclase, clinopyroxene and sometimes olivine.
geochemical	Pertains to the concentration of an element.
geophysical	Pertains to the physical properties of a rock mass.
granite	A coarse-grained igneous rock containing mainly quartz and feldspar minerals and subordinate micas.
granodiorite	A coarse grained igneous rock composed of quartz, feldspar and hornblende and/or biotite.
greywackes	A sandstone like rock, with grains derived from a dominantly volcanic origin.
hydrothermal fluids	Pertaining to hot aqueous solutions, usually of magmatic origin, which may transport metals and minerals in solution.
igneous	Rocks that have solidified from a magma.
infill	Refers to sampling or drilling undertaken between pre-existing sample points.
intermediate	A rock unit which contains a mix of felsic and mafic minerals.
intrusions	A body of igneous rock which has forced itself into pre-existing rocks.
ironstone	A rock formed by cemented iron oxides.
joint venture	A business agreement between two or more commercial entities.
laterite	A cemented residuum of weathering, generally leached in silica with a high alumina and/or iron content.
lead	A metallic element, the heaviest and softest of the common metals.
magnetite	A mineral comprising iron and oxygen which commonly exhibits magnetic properties.
metamorphic	A rock that has been altered by physical and chemical processes involving heat, pressure and derived fluids.
Mt	Million Tonnes.
mylonite	A hard compact rock with a streaky or banded structure produced by extreme granulation of the original rock mass in a fault or thrust zone.
outcrops	Surface expression of underlying rocks.
pegmatite	A very coarse grained intrusive igneous rock which commonly occurs in dyke-like bodies containing lithium-boron-fluorine-rare earth bearing minerals.
porphyries	Felsic intrusive or sub-volcanic rock with larger crystals set in a fine groundmass.
ppb	Parts per billion; a measure of low level concentration.
Proterozoic	An era of geological time spanning the period from 2,500 million years to 570 million years before present.
regolith	The layer of unconsolidated material which overlies or covers in situ basement rock.
residual	Soil and regolith which has not been transported from its point or origin.

Resources	In situ mineral occurrence from which valuable or useful minerals may be recovered.
Resource Estimate	Methodology of estimation of the volume of tonnages of identified resources - normally summarized and stated according to the guidelines of the JORC (Dec 2004) reporting code.
rhyolite	Fine-grained felsic igneous rock containing high proportion of silica and felspar.
rock chip sampling	The collection of rock specimens for mineral analysis.
schist	A crystalline metamorphic rock having a foliated or parallel structure due to the recrystallisation of the constituent minerals.
scree	The rubble composed of rocks that have formed down the slope of a hill or mountain by physical erosion.
sedimentary	A term describing a rock formed from sediment.
sericite	A white or pale apple green potassium mica, very common as an alteration product in metamorphic and hydrothermally altered rocks.
shale	A fine grained, laminated sedimentary rock formed from clay, mud and silt.
sheared	A zone in which rocks have been deformed primarily in a ductile manner in response to applied stress.
sheet wash	Referring to sediment, usually sand size, deposited over broad areas characterised by sheet flood during storm or rain events. Superficial deposit formed by low temperature chemical processes associated with ground waters, and composed of fine grained, water-bearing minerals of silica.
silica	Dioxide of silicon, SiO_2, usually found as the various forms of quartz.
sills	Sheets of igneous rock which is flat lying or has intruded parallel to stratigraphy.
silts	Fine-grained sediments, with a grain size between those of sand and clay.
soil sampling	The collection of soil specimens for mineral analysis.
stocks	A small intrusive mass of igneous rock, usually possessing a circular or elliptical shape in plan view.
strata	Sedimentary rock layers.
stratigraphic	Composition, sequence and correlation of stratified rocks.
stream sediment sampling	The collection of samples of stream sediment with the intention of analysing them for trace elements.
strike	Horizontal direction or trend of a geological structure.
	Poorly exposed bedrock.
sulphide	A general term to cover minerals containing sulphur and commonly associated with mineralisation.
supergene	Process of mineral enrichment produced by the chemical remobilisation of metals in an oxidised or transitional environment.
tectonic	Pertaining to the forces involved in or the resulting structures of movement in the Earth's crust.
veins	A thin infill of a fissure or crack, commonly bearing quartz.
volcanics	Formed or derived from a volcano.
zinc	A lustrous, blueish-white metallic element used in many alloys including brass and bronze.

APPENDIX A

Un-sampled Assay intervals and associated domain codes

HOLE#	FROM	-TO-	-AI-	AU	Domain
SD107	107.35	113.5	6.15	-1	104
SD112	38.5	42.5	4	-1	104
SD115	39.5	43	3.5	-1	104
SD118	35	40.3	5.3	-1	104
SD118	42.3	43.3	1	-1	104
SD118	44.3	45.2	0.9	-1	104
SD118	80	80.1	0.1	-1	102
SD118	81.4	82.1	0.7	-1	102
SD118	86.8	96.1	9.3	-1	102
SD135	0	4	4	-1	101
SD135	9	15	6	-1	101
SD149	71	72.5	1.5	-1	101
SD152	21.1	21.6	0.5	-2	104
SD152	24.6	25.6	1	-2	104
SD158	55.5	55.6	0.1	-1	104
SD159	54.4	60.5	6.1	-1	102
SD163	141.4	143.7	2.3	-1	104
SD163	149.8	150.1	0.3	-1	104
SD164	0	2.9	2.9	-2	104
SD165	0	3.9	3.9	-1	104
SD165	69.8	72.1	2.3	-1	101
SD182	45.6	52.8	7.2	-1	104
SD184	39	43.3	4.3	-1	104
SD190	50	51.9	1.9	-1	101
SD196	44.5	60	15.5	-1	103
SD197	43.5	45	1.5	-1	103
SD201	36	37.6	1.6	-1	101
SD202	53.6	58	4.4	-1	101
SD210	4	6	2	-1	104
SD222	34	36	2	-1	104
SD234	18	20	2	-1	104
SD234	22	24	2	-1	104
SD266	60	64.5	4.5	-1	102
SD272	54.7	57.1	2.4	-1	104
SD280	59	68	9	-1	104
SD282	117	122.4	5.4	-1	102
SD287	149.1	158.7	9.6	-1	101
SD287	159.2	162	2.8	-1	101
SD288	146	148.4	2.4	-2	101
SD289	112.4	134	21.6	-1	102

HOLE#	FROM	-TO-	-AI-	AU	Domain
SD296	97	101.2	4.2	-1	104
SD298	96	97	1	-1	104
SD304	94.7	96	1.3	-1	104
SD304	99	104	5	-1	104
SD307	99.6	104.8	5.2	-1	104
SD308	86.6	89	2.4	-1	104
SD308	96.3	97	0.7	-1	104
SD310	98.8	101	2.2	-1	104
SD310	104	108	4	-1	104
SD311	133	137	4	-1	101
SD313	69	74	5	-1	101
SD314	108	110	2	-1	104
SD314	110.8	115.7	4.9	-1	104
SD329	81.9	82.6	0.7	-1	104
SD346	168.7	170.9	2.2	-1	101
SD346	170.9	171.4	0.5	-1	101
SD346	171.4	175.3	3.9	-1	101
SD352	161	170.5	9.5	-1	104
SD365	72.5	79.1	6.6	-1	104
SD366	61	64	3	-1	104
SP193	30	31	1	-1	101
SP193	31	32	1	-1	101
SP193	32	33	1	-1	101
SP193	33	34	1	-1	101
SP193	34	35	1	-1	101
SP193	35	36	1	-1	101
SP193	36	37	1	-1	101
WC1	7.93	15.85	7.925	-1	104
WD10	6.7	15	8.3	-1	104
WD105	32.2	37.1	4.9	-1	103
WD105	43.2	47.8	4.6	-1	103
WD115	87.8	92.5	4.7	-1	101
WD117	58.8	75.5	16.7	-1	101
WD129	3	7.5	4.5	-1	104
WD129	7.5	10.5	3	-1	104
WD129	12.1	17.4	5.3	-1	104
WD13	41.6	44	2.4	-1	103
WD13	45.8	50.2	4.4	-1	103
WD138	8	15	7	-1	102
WD138	32.5	40	7.5	-1	101
WD14	0	1	1	-1	104
WD140	45	50.5	5.5	-1	101
WD144	23.5	28	4.5	-1	102

HOLE#	FROM	-TO-	-AI-	AU	Domain
WD144	30	32.3	2.3	-1	102
WD144	50.4	55	4.6	-1	102
WD145	59.6	65.3	5.7	-1	102
WD148	59.1	64	4.9	-1	102
WD149	73.5	85.9	12.4	-1	102
WD15	1	8.55	7.55	-1	104
WD158	62.9	66	3.1	-1	101
WD158	71.5	75.5	4	-1	101
WD159	94	98.9	4.9	-2	101
WD159	99.8	113	13.2	-2	101
WD159	117	123.4	6.4	-2	101
WD159	128.8	135.5	6.7	-2	101
WD15A	0	3.8	3.8	-1	104
WD15A	51.5	55.7	4.2	-1	104
WD166	0	1.1	1.1	-1	104
WD167	15.1	19.6	4.5	-1	104
WD169	127.65	153.8	26.15	-1	101
WD17	33	33.5	0.5	-1	104
WD170	98	100.9	2.9	-1	101
WD170	101.5	109.6	8.1	-1	101
WD172	83.6	89.6	6	-1	101
WD174	48.5	50.6	2.1	-1	102
WD179	75.8	91.6	15.8	-1	101
WD18	24.6	31.5	6.9	-1	104
WD18	33.1	35.9	2.8	-1	104
WD18	38.2	40.4	2.2	-1	104
WD184	89.2	96	6.8	-1	101
WD188	44.7	48	3.3	-1	104
WD188	50.5	62.8	12.3	-1	104
WD191	73	78	5	-1	104
WD193	80.5	81	0.5	-1	101
WD193	83.8	85.1	1.3	-1	101
WD195	92.6	112	19.4	-1	104
WD198	23.5	28.8	5.3	-1	101
WD198	28.8	29.2	0.4	-1	101
WD198	29.2	38.9	9.7	-1	101
WD198	38.9	41.4	2.5	-1	101
WD198	41.4	47	5.6	-1	101
WD198	47	48	1	-1	101
WD198	48	64.9	16.9	-1	101
WD198	75	83.6	8.6	-1	101
WD201	38.7	56.5	17.8	-2	104
WD201	82.5	86.6	4.1	-2	104

HOLE#	FROM	-TO-	-Al-	AU	Domain
WD202	5.5	24.4	18.9	-1	104
WD202	52.9	82.9	30	-1	104
WD202	82.9	95.1	12.2	-1	104
WD202	97.5	102.8	5.3	-1	104
WD204	0	4.9	4.9	-1	101
WD204	4.9	5.1	0.2	-1	101
WD204	5.1	8.5	3.4	-1	101
WD204	8.5	8.6	0.1	-1	101
WD21	6	10.2	4.2	-1	104
WD21	12.6	14.1	1.5	-1	104
WD210	0	12	12	-1	101
WD214	62.6	65.4	2.8	-1	104
WD22	7.7	8.7	1	-1	104
WD22	16.9	18.9	2	-1	104
WD22	19.9	20.8	0.9	-1	104
WD222	34.3	37.5	3.2	-1	104
WD224	94.1	95.8	1.7	-1	104
WD224	105.4	113.7	8.3	-1	104
WD224	131.4	134.7	3.3	-1	104
WD224	144.5	174.5	30	-1	104
WD23	22.2	24	1.8	-1	104
WD233	57	70.8	13.8	-1	101
WD233	70.8	72.1	1.3	-1	101
WD233	72.1	73.4	1.3	-1	101
WD233	73.4	75.3	1.9	-1	101
WD233	75.3	79.1	3.8	-1	101
WD235	22	27.9	5.9	-1	102
WD235	58	59.8	1.8	-1	104
WD235	59.8	66.4	6.6	-1	104
WD236	12.6	15.63	3.03	-1	102
WD238	3	6	3	-1	102
WD24	18.2	21.2	3	-1	104
WD241	30	50.9	20.9	-1	102
WD249	0.9	2	1.1	-1	102
WD249	2	6.22	4.22	-1	102
WD250	0	2	2	-1	102
WD250	14.4	18.2	3.8	-1	102
WD250	18.2	21.3	3.1	-1	102
WD250	25.94	34.5	8.56	-1	102
WD252	29.9	30.7	0.8	-1	104
WD253	14	22.2	8.2	-1	104
WD253	34.55	40.75	6.2	-1	104
WD254	7.3	10.05	2.75	-2	101

HOLE#	FROM	-TO-	-AI-	AU	Domain
WD26	4.6	6.4	1.8	-1	104
WD26	7.5	13.5	6	-1	104
WD26	32.9	35.5	2.6	-1	104
WD263	50.95	62.1	11.15	-2	101
WD265	8	10.35	2.35	-1	104
WD266	0	5.3	5.3	-1	104
WD267	0	1	1	-1	104
WD267	1	1.1	0.1	-1	104
WD267	5	5.85	0.85	-1	104
WD27	0	0.5	0.5	-1	104
WD270	0	5.75	5.75	-1	104
WD271A	47.65	56.2	8.55	-1	101
WD272	76	77.2	1.2	-1	101
WD272	77.2	87	9.8	-1	101
WD272	90.42	91.1	0.68	-1	101
WD272	91.1	94.37	3.27	-1	101
WD274	33.4	34.6	1.2	-2	102
WD276	36.55	40.25	3.7	-1	101
WD277	40.6	44.9	4.3	-1	101
WD277	48.7	53.25	4.55	-1	101
WD278	33.23	42.25	9.02	-1	101
WD278	42.25	50.5	8.25	-1	101
WD279	29.35	35.1	5.75	-1	104
WD279	46	65.7	19.7	-1	104
WD279	85	103.45	18.45	-1	104
WD279	113.8	121	7.2	-1	104
WD28	34.3	36.9	2.6	-1	104
WD28	36.9	40.2	3.3	-1	104
WD28	41.5	45	3.5	-1	104
WD280	11.4	11.75	0.35	-1	104
WD280	11.75	16.3	4.55	-1	104
WD280	16.3	16.8	0.5	-1	104
WD280	16.8	18.25	1.45	-1	104
WD280	21	25.2	4.2	-1	104
WD280	25.2	39.4	14.2	-1	104
WD280	55.6	66.6	11	-1	104
WD280	66.6	66.9	0.3	-1	104
WD280	66.9	67	0.1	-1	104
WD280	69.2	73.2	4	-1	104
WD281	43.65	46.4	2.75	-1	104
WD283	11.65	12.3	0.65	-1	101
WD283	12.3	16	3.7	-1	101
WD289	46	48.13	2.13	-1	101

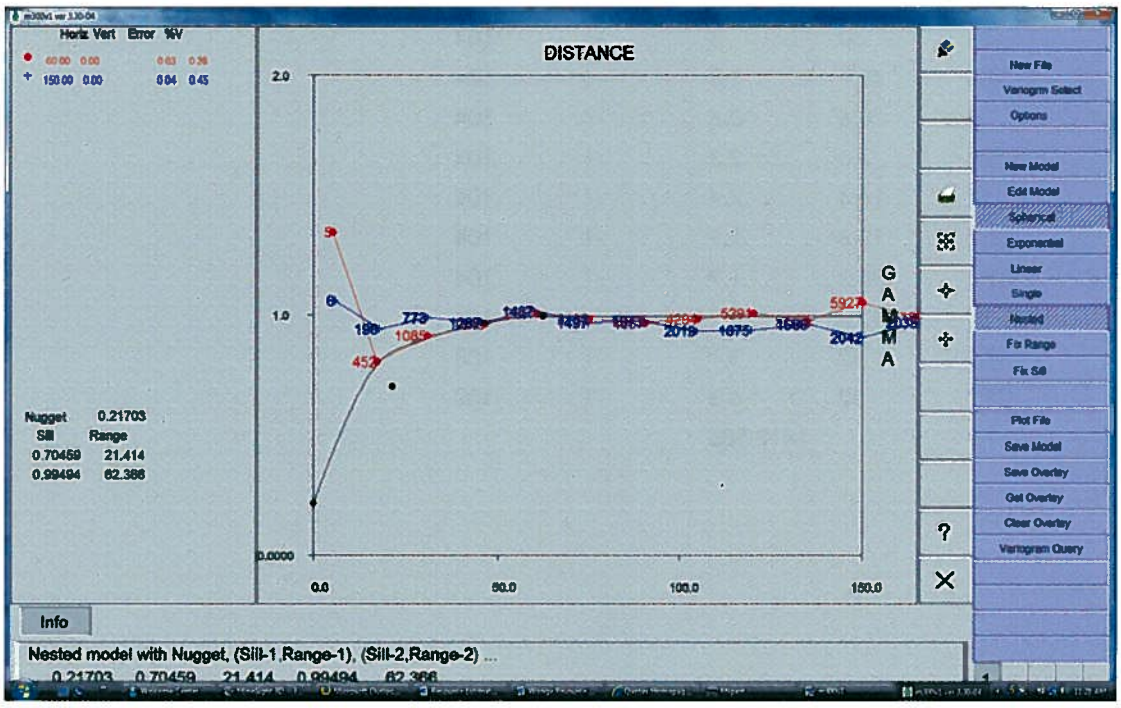
HOLE#	FROM	-TO-	-AI-	AU	Domain
WD289	48.13	51.15	3.02	-1	101
WD29	8.3	13.3	5	-1	104
WD29	44	47.1	3.1	-1	104
WD294	60	60.5	0.5	-2	101
WD295	118	122	4	-1	104
WD295	124	132	8	-1	104
WD295	188.45	197	8.55	-1	104
WD298	48.8	52	3.2	-1	104
WD299	59.35	78	18.65	-1	104
WD299	78	79.8	1.8	-1	104
WD299	79.8	86.15	6.35	-1	104
WD299	86.15	97.7	11.55	-1	104
WD3	3.2	10.9	7.7	-1	104
WD302	44.3	47.9	3.6	-1	104
WD302	47.9	49.2	1.3	-1	104
WD302	49.2	56	6.8	-1	104
WD303	49.36	59.3	9.94	-1	104
WD31	18	22.9	4.9	-1	104
WD33	0	2.5	2.5	-1	104
WD331	45.15	45.84	0.69	-1	104
WD34	1.7	23.7	22	-1	104
WD34	33.9	47.4	13.5	-1	104
WD35	0	0.6	0.6	-1	104
WD35	0.6	1.9	1.3	-1	104
WD35	7	11	4	-1	104
WD35	21.4	27.2	5.8	-1	104
WD35	42.4	51	8.6	-1	104
WD37	11.8	21.9	10.1	-1	104
WD37	23.5	27.4	3.9	-1	104
WD3A	7.5	12.9	5.4	-1	104
WD3A	17.4	26.7	9.3	-1	104
WD41	43.9	46.6	2.7	-2	104
WD42	41	46.4	5.4	-1	104
WD43	43	47.9	4.9	-1	102
WD43	72.7	80	7.3	-1	102
WD44	8.8	13.5	4.7	-2	104
WD45	97.2	104	6.8	-2	102
WD48	0	3.4	3.4	-1	104
WD5	0	3.4	3.4	-1	104
WD51	60.2	77.2	17	-1	104
WD51	79.7	88	8.3	-1	104
WD5A	0	4.7	4.7	-1	104
WD5A	13.4	20.3	6.9	-1	104

HOLE#	FROM	-TO-	-AI-	AU	Domain
WD62	0	8.7	8.7	-1	104
WD64	12.8	17.7	4.9	-1	104
WD67A	2.9	10.5	7.6	-1	104
WD67A	14.6	19	4.4	-1	104
WD67A	22	30	8	-1	104
WD68	39.9	49.7	9.8	-1	104
WD7	0	0.6	0.6	-1	104
WD7	5.3	7.6	2.3	-1	104
WD76	10	14.4	4.4	-1	104
WD8	16.7	19.6	2.9	-1	104
WD9	31.4	33.3	1.9	-1	104
WD9	39	43.7	4.7	-1	104
WD95	38.9	42	3.1	-1	103
WD97	25.7	32	6.3	-1	102
			1412.585		

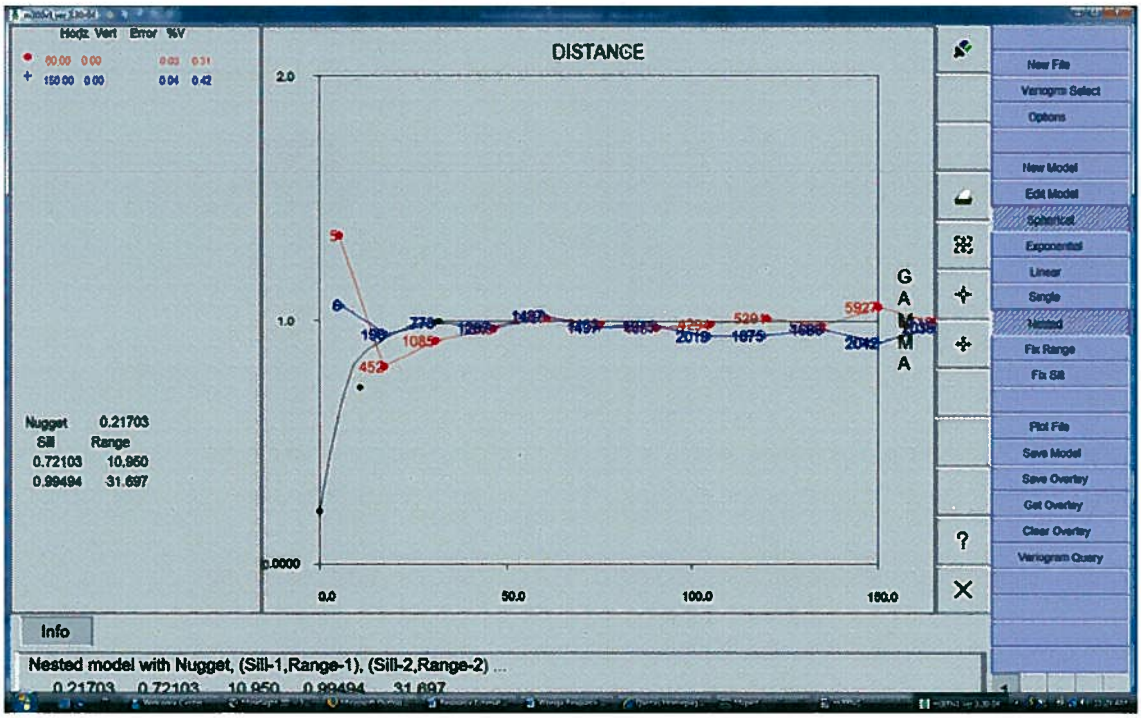
APPENDIX B

Directional Experimental Variograms

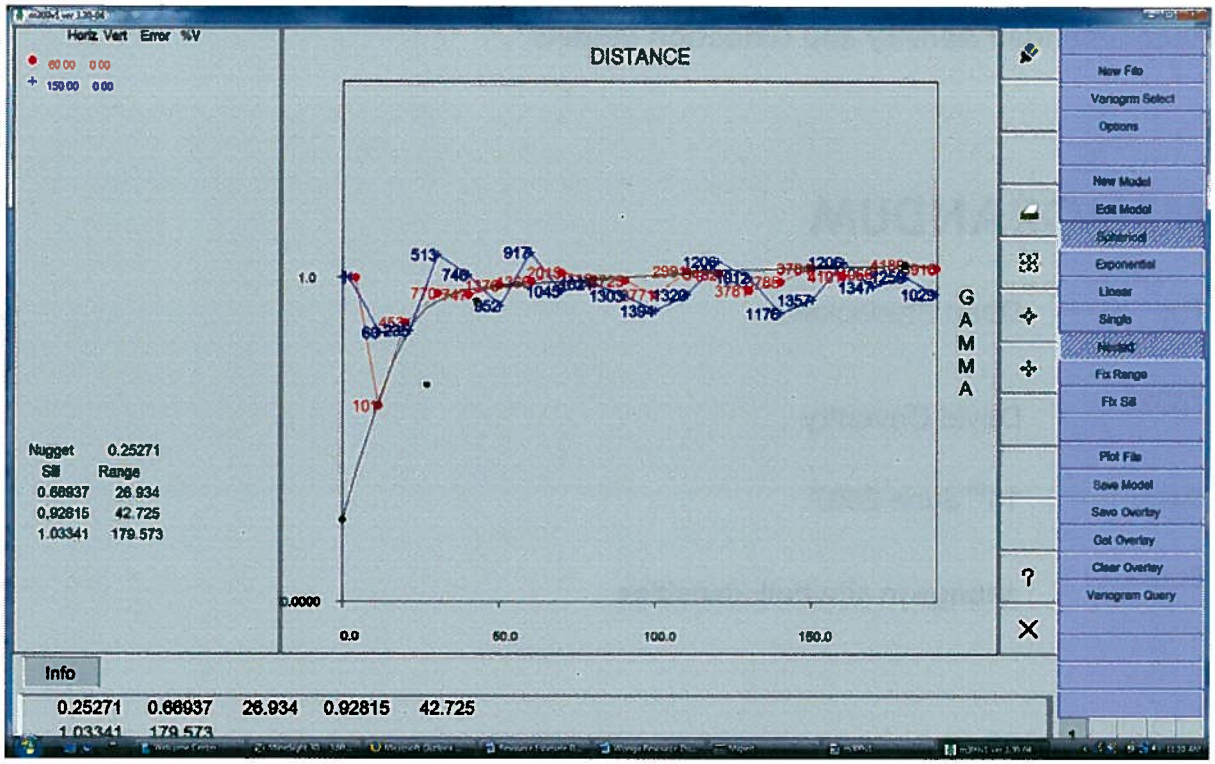
Domain 101



Domain 101 within plan of the structure - Major variogram model.



Domain 101 within plan of the structure - Major variogram model



Domain 104 within plan of the structure - Major variogram model. Note perpendicular to defined model indicates near spherical models

APPENDIX C

Recommended Density and Oxidation models

MEMORANDUM

TO: Dean Fredrickson

FROM: David Coventry

DATE: 14th June 2007

SUBJECT: **Wonga in situ bulk densities**

Over the period 4th – 13th June in situ bulk density tests were carried out for Wonga on the core from the diamond drill holes SD160, SD192 and SD281 as well as 19 grab samples through the oxidation profile of the existing pit wall. The in situ bulk density tests were carried out to validate the density of the rock that is to be moved during the proposed cut back of the Wonga open pit. It was noted that there were four weathering domains present within the resource so bulk density measurement needed to be carried out for each domain. The domains are extreme, heavy, moderately and fresh. The main rock types were Wonga schist, which was split into ore and waste zones through review of assays of the drill core, along with weathered saprolites.

The testing procedure to determine the dry bulk density was based a procedure outlined by I.T. Lipton in his 2001 paper "*Measurement of Bulk Density of Resource Estimation*". The core used was NQ and HQ in size and had to be greater than 10cm in length to be acceptable for the test. The core used was kept in storage racks adjacent to the Wonga open pit, due to the exposure to further oxidation over time since the core was drilled a series of grab samples were also used from the existing pit wall. There drill core was dried over night in the SGM lab ovens before being weighed. The steps taken to determine the in situ bulk density are as follows:

The down hole depth and the interval of the core to be tested was recorded along with the rock type and weathering domain.
The core sample was measured so that its length was known and was weighed dry to determine the dry mass (*M*).
The core sample was then coated in hot paraffin wax and then allowed to cool to room temp before being weighed to determine dry mass with wax (*M+wax*).
The wax coated sample was then immersed in water suspended from a balance to determine weight of the wax coated sample in water (*M+wax in water*)

The dry bulk density is calculated as the mass of dry sample divided by the volume of the sample, after accounting for the volume of wax. Hence the following formula was used

Bulk density = M/[(M+wax – M+wax in water) – {(M+wax – M)/Density of wax}]

The density of wax had to be measured as this value was not known. This was done by making three wax moulds of a ½ Cup measure which has a volume of 125ml. As density is determined by mass divided by

volume, the average mass of the three moulds was divided by 125ml and the density of the wax was calculated to be 0.75t/m3.

Potential errors in the experiment are mainly due to two problems arising during the process. The first was that the scales used to determine all the measurements only measured to the nearest 5 grams. But as the lightest measurement taken was 415g in water, a rounding of 2.5g by the scales produces a total weight error of only 0.6%. This small weight error is deemed minimal and isn't significant to the final averaged in situ bulk density as the percentage error would be reduced further by the averaging of 15 samples in the same weathering domain.

The second potential error is that the wax was cracking slightly when the coated sample was being immersed in the water to be weighed. As the wax coating was used to keep the water out of the sample so that the dry mass of the core and wax could be measured this presented a problem. To test how quickly the water absorbed into the core a coated sample was immersed in the water and its weight was recorded straight away, then it was left to sit in the water. After two minutes the weight of the sample had increased by between 5-10 grams. This small weight gain after two minutes of immersion was perceived as a small problem but to further reduce its impact the mass of the coated sampled was recorded as soon as it was immersed in the water. By doing this it was deemed that the recorded dry mass of the core plus wax in water would have minimal to no water absorption interference.

The results of the in situ dry bulk density test are shown below:

S.G. (t/m3)	**extreme**	**heavy**	**mod**	**fresh**
Sapro	2.11	2.45		
Waste			2.63	2.86
Ore			2.70	2.81
Total	**2.11**	**2.45**	**2.65**	**2.84**

Number of samples	**extreme**	**heavy**	**mod**	**fresh**
Sapro	10	10		
Ore			10	8
Waste			20	22
Total	**10**	**10**	**30**	**30**

Overall 80 samples where tested and it was thought that 10 samples per weathering domain would be sufficient.

These new S.G. values for the tested weathering zones are concluded from the experiment taken:

Domain	Specific Gravity (t/m3)
Extreme oxidation	2.1
Heavy oxidation	2.45
Moderate oxidation	2.65
Fresh rock	2.85

Dave Coventry
Exploration Geologist

APPENDIX D

Minesight estimation run files

```
Minesight Run file Domain 101
MEDS-624V1  10=pit10.dat  9=pit09.dat  15=pit15.dat;
MEDS-624V1   3=rpt624.101  30=
** KRIGING of 3-D block values for AUPPM **
RUN = RESET
USR =   /  Thu Jun 7, 2007 2:31:13 PM AUS Eastern Standard Time

IOP2 = 15 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 50 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 200. / Primary X-search distance
PAR2 = 200. / Primary Y-search distance
PAR3 = 200. / Primary Z-search distance
PAR4 = 200. / Limiting search distance
PAR7 = 30. / Max distance to closest point
PAR8 = 30. / Max distance to project single composite
PAR27= 2 4 4 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH 100. 50. 25. 91. -40. -36.
CMD = NUG 0.21
CMD = SPH 0.49 21.4 10.95 5.97 91. -40. -36.
CMD = SPH 0.3 62.3 31.7 13.9 91. -40. -36.

ITM1 = AUPPM AU CALC KRIGE
ITM2 = SLOPE Block error
ITM3 = ZONE Block limit
ITM4 = GEOCD RANGE 101. 103.
ITM5 = DIST Block calc rings
ITM6 = CMPS Block calc #comp
ITM7 = AU3 Block calc #comp
CMD = Block limit ZONE 101
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

```
Minesight Runfile Domain 102
MEDS-624V1  10=pit10.dat  9=pit09.dat  15=pit15.dat;
MEDS-624V1   3=rpt624.102  30=
** KRIGING of 3-D block values for AUPPM **

RUN = OMIT

USR =   /  Thu Jun 7, 2007 3:52:52 PM AUS Eastern Standard Time

IOP2 = 15 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 50 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 200. / Primary X-search distance
PAR2 = 200. / Primary Y-search distance
PAR3 = 200. / Primary Z-search distance
PAR4 = 200. / Limiting search distance
PAR7 = 30. / Max distance to closest point
PAR8 = 30. / Max distance to project single composite
PAR27= 2 4 4 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH 100. 50. 25. 106. -38. -32.
CMD = NUG 0.21
CMD = SPH 0.49 21.4 10.95 5.97 106. -38. -32.
CMD = SPH 0.3 62.3 31.7 13.9 106. -40. -32.

ITM1 = AUPPM AU CALC KRIGE
ITM2 = SLOPE Block error
ITM3 = ZONE Block limit
ITM4 = GEOCD RANGE 101. 103.
ITM5 = DIST Block calc rings
ITM6 = CMPS Block calc #comp
ITM7 = AU3 Block calc #comp
CMD = Block limit ZONE 102
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

```
Minesight Runfile Domain 103
MEDS-624V1  10=pit10.dat  9=pit09.dat  15=pit15.fin;
MEDS-624V1   3=rpt624.103  30=
** KRIGING of 3-D block values for AUPPM **

RUN = OMIT

USR =   /  Fri Jun 15, 2007 9:02:43 AM AUS Eastern Standard Time

IOP2 = 20 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 60 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 200. / Primary X-search distance
PAR2 = 200. / Primary Y-search distance
PAR3 = 200. / Primary Z-search distance
PAR4 = 200. / Limiting search distance
PAR7 = 30. / Max distance to closest point
PAR8 = 30. / Max distance to project single composite
PAR27= 2 4 4 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH 100. 50. 25. 72. -24. -16.
CMD = NUG 0.23
CMD = SPH 0.45 21.4 10.95 5.97 72. -24. -16.
CMD = SPH 0.13 62.3 31.7 13.9 72. -24. -16.

ITM1 = AUPPM AU CALC KRIGE
ITM2 = SLOPE Block error
ITM3 = ZONE Block limit
ITM4 = GEOCD RANGE 102. 104.
ITM5 = DIST Block calc rings
ITM6 = CMPS Block calc #comp
ITM7 = HOLES Block calc #comp
CMD = Block limit ZONE 103
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Minesight Runfile Domain 104

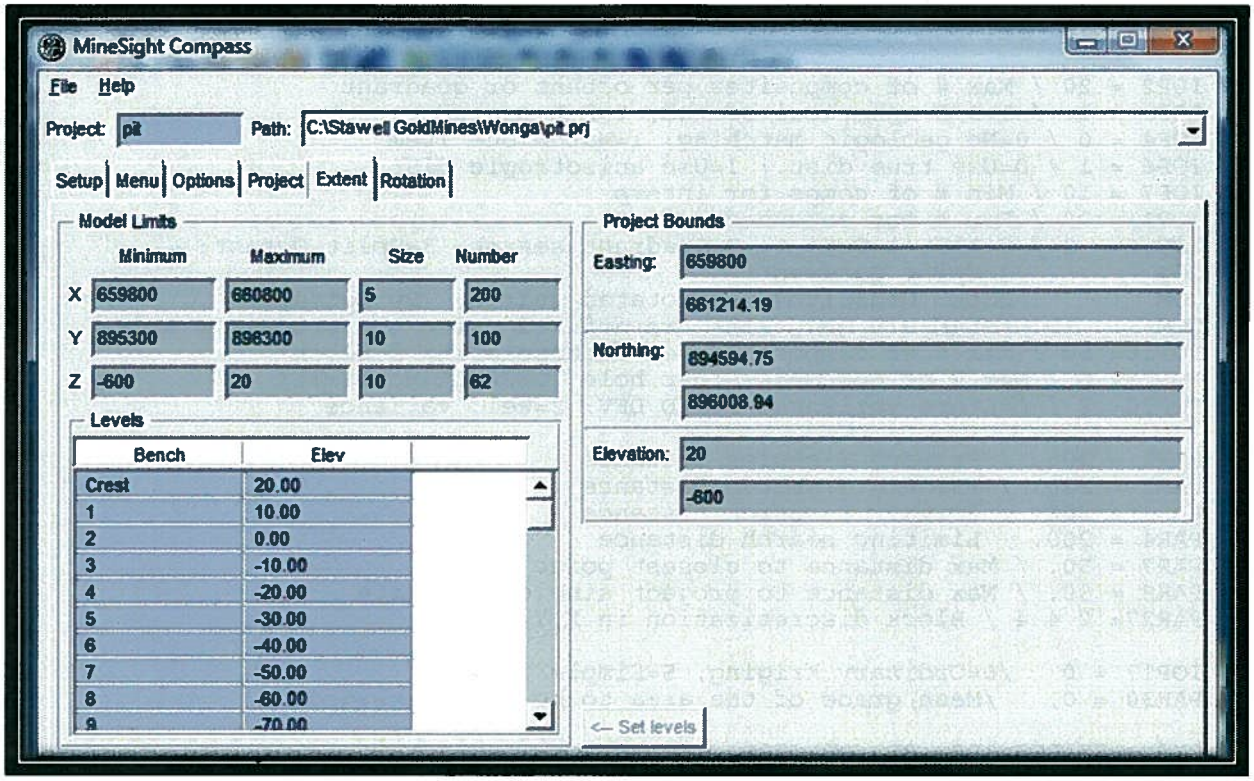
```
MEDS-624V1  10=pit10.dat  9=pit09.dat  15=pit15.dat;
MEDS-624V1   3=rpt624.104  30=
** KRIGING of 3-D block values for AUPPM **

RUN = OMIT

USR =   /  Fri Jun 8, 2007 11:36:03 AM AUS Eastern Standard Time

IOP2 = 20 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 60 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 200. / Primary X-search distance
PAR2 = 200. / Primary Y-search distance
PAR3 = 200. / Primary Z-search distance
PAR4 = 200. / Limiting search distance
PAR7 = 50. / Max distance to closest point
PAR8 = 50. / Max distance to project single composite
PAR27= 2 4 4 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH 100. 80. 20. 329. 16. -30.
CMD = NUG 0.23
CMD = SPH 0.45 20.3 15. 5.97 329. 16. -30.
CMD = SPH 0.13 62.2 55. 13.9 329. 16. -30.

ITM1 = AUPPM AU CALC KRIGE
ITM2 = SLOPE Block error
ITM3 = ZONE Block limit
ITM4 = GEOCD RANGE 102. 104.
ITM5 = DIST Block calc rings
ITM6 = CMPS Block calc #comp
ITM7 = AU3 Block calc #comp
CMD = Block limit ZONE 104
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

APPENDIX E

Model Export File Descriptors

Wonga - June 2007 Resource Model

Model Limits



Model Files - Minesight

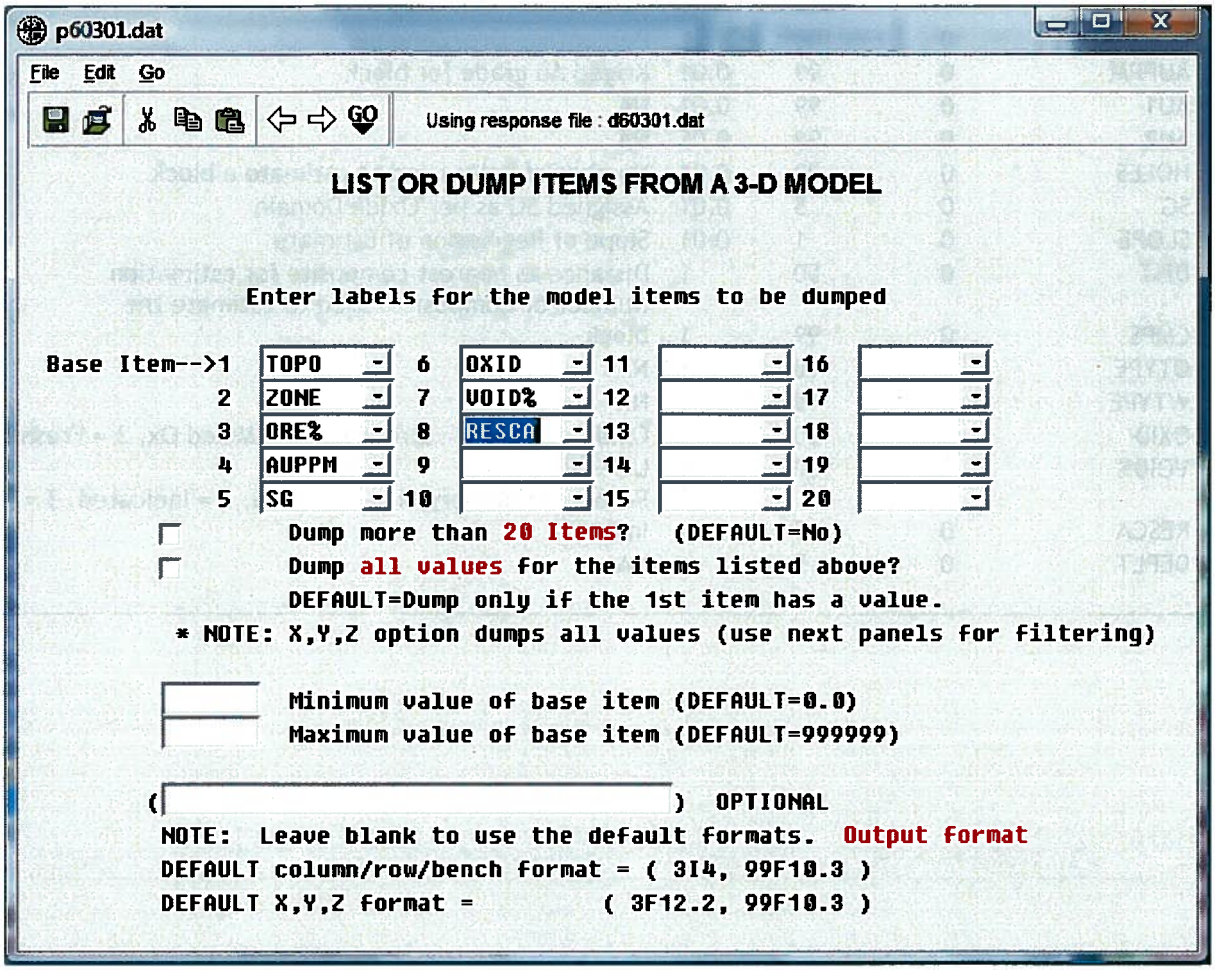
- Project Control File – Pit10.dat
- Drillhole Files – Pit11.dat, Pit12.dat
- Composite File – Pit09.dat
- Block Model File – Pit15.fin
- Surface Model File – Pit13.pit

PIT15.fin Item Descriptors

Label	Minimum	Maximum	Prec.	
TOPO	0	100	1	Topo% from QASCO DTM
ZONE	1	999	1	Zone as per zone table
ORE%	0	100	1	Percentage Ore/Zone per block
AUPPM	0	99	0.01	Kriged Au grade for block
AU1	0	99	0.01	NA
AU2	0	99	0.01	NA
HOLES	0	99	0.01	Number of holes used to estimate a block
SG	0	5	0.01	Assigned SG as per Oxide Domain
SLOPE	0	1	0.01	Slope of Regression of Estimate
DIST	0	50	1	Distance to nearest composite for estimation
CMPS	0	99	1	Number of Composites used to estimate the block
OTYPE	0	10	1	NA
WTYPE	0	10	1	NA
OXID	0	10	1	Oxide domain 1 = Strong Ox, 2 = Mixed Ox, 3 = Fresh rock
VOID%	0	100	1	UG void %
RESCA	0	5	1	Resource Category, 1 = Measured, 2 = Indicated, 3 = Inferred.
DEPLT	0	5	1	NA

Model Export to Ascii

- The model has been exported to an ASCII file Wonga.out
- Fields are X, Y, Z then in the order as shown below



TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX B

Magdala Surface Resource Report

December 1998

Executive Summary

A resource estimate for Big Hill (BH1998) was completed on 31st December 1998.

This estimate was based on a synthesis of all available data, including:
All surface drilling up to October 1998
Re-logging and re-assaying of underground diamond core
Surface and underground mapping
Underground sample drilling

During 1997/98, 217 reverse circulation drill holes were drilled for 19,500m, and 30 cored diamond holes for 2,369m were also drilled. Drill hole spacing is nominally 20m North by 25m RL.

Geological interpretation was carried out on cross sections, then a three dimensional wire frame model of the geology was built using Minesight software. Geostatistical analysis and block modelling was completed using MEDSYSTEM software.

The resource estimate is based on blocks of 5m East * 10m North * 5m vertically. Interpolation was by Ordinary Kriging. Cut values were applied to individual geological domains on the basis of population statistics.

The global resource estimated to –195m RL is as follows:

Big Hill 1998 Resource Estimate - global resource to -195m RL

	> 0 g/t cutoff									> 0.8g/t cutoff								
	Constrained			Unconstrained			Total			Constrained			Unconstrained			Total		
Material	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz
Oxide	1,748	2.28	128	3,352	0.40	43	5,100	1.05	171	1,647	2.39	126	341	1.31	14	1,988	2.20	141
Trans	402	2.61	34	1,308	0.57	24	1,710	1.05	58	398	2.63	34	286	1.49	14	685	2.16	47
Sulph	1,261	2.98	121	7,059	0.73	165	8,320	1.07	286	1,257	2.99	121	2,240	1.59	115	3,497	2.10	236
Total	3,411	**2.58**	283	#####	**0.62**	232	#####	**1.06**	515	3,302	**2.65**	281	2,867	**1.55**	143	6,169	**2.14**	424

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PCF limits

	Minimum	Maximum	Cell size	Number blocks
X direction	4500	5000	5	100
Y direction	4400	6000	10	160
Z direction	-200	20	5	44

Figure 1: PCF limits and block dimensions

Solid modeling and block modeling was stopped at 4930mN (247 Section). This was the limit of sectional drilling during the current round of exploration.

The project is currently stored in directory I:\medsys\bh1998

Data sources

Drilling

	Big Hill Drilling Summary					
	Current Programme		Pre May 19978		Total	
	# holes	metres	# holes	metres	# holes	Metres
Surface Percussion	217	19,537	63	5,916	280	25,516
Surface Diamond	18	1,972	81	*	99	20,370
Diamond Tails	12	397			12	397
U/G diamond 145-180 levels			167	14,920	167	15,087
-212 upholes			38	2,242	38	2,280
						63,650

* Although 18,300m of older diamond core drilling falls within the PCF limits, most of this was targeted at deeper sections of the orebody and lies within mine schist west of the ore zone.

Figure 2: Summary of drilling

Reverse Circulation drilling

RC drilling was conducted in three programmes over a period of years.

Holes SP1-SP81 were drilled by Western Mining Corporation between 1978 and 1993. Rig type is unknown but it is guessed that they were probably drilled using a 4.5" cross-over sub hammer. The sampling protocol for these holes is unknown, but probably only consisted of grab or spear samples. The assays should be treated with caution.

Holes SP137-148 were drilled by SGM during Mar/April 1997, using a 5 ¼" face sampling hammer on a track mounted drill rig.

Holes SP180-447 have been drilled by SGM during 1997/8. All of this drilling was carried out with two similar UDR 1000 drill rigs, using a 5 ¼ " face sampling hammer.

RC Sampling protocol:

All SGM RC sampling was carried out using the following protocol

Samples collected at 1m sampling interval, bit pulled back and flushed between intervals
Samples discharged into tightly fitting plastic sample bag from cyclone
Sample transferred to plastic fish crate
Sample tipped into three tier splitter from crate to ensure equal quantities available to all vanes
1/12 subsplit (nominally 3kg) collected in calico sample bag, tied and placed in 5's in plastic bag

Residue sample collected in original sample bag and transported to bag farm for storage
Splitter cleaned by shaking/banging/brush/air compressor as necessary between samples
Cyclone cleaned at regular intervals by banging/checked by hand/arm
Every 20th sample resplit to give a 2nd 3kg sample, field splits dispatched along with first splits
Samples dispatched to Aminya Laboratories (Ballarat) for analysis by Fire Assay

Around 99.5% of drilling was conducted dry. Big Hill has been very effectively de-watered by the underground mining operation. In a handful of holes, samples were damp after a rod change. Where a sample could not be effectively riffle split it was spear sampled using a PVC spear. Sample volumes were not routinely recorded. An initial programme in which samples were weighed showed little variation in recovered volume by depth or geology and it was decided that routine weighing was unneccesary.

Exploration DDH

Surface (Hole prefix SD):
Surface diamond holes are of two generations.

- Pre SD500 were drilled by Western Mining (and previously in some cases). In many cases these were rotary pre-collared to around 100m. Core was generally ½ core (split or sawn) sampled. Assay method was mostly by AAS.
- Post SD500 were drilled by SGM during the course of this exploration programme. All of this core was HQ, full core sampled and fire assayed.

Underground (Hole prefix MD):
Underground diamond holes can be broken into two sets also

- Holes after MD1000 were full core sampled and assayed by Fire Assay, with the entire volcanogenic zone sampled. In addition holes over MD 1300 were logged in Datcol and all data entered to database.
- Holes drilled before MD1000were mostly ½ core sampled (split or sawn) and assayed using Aqua Regia digest. Only selected sections of the volcanogenic zone were sampled. Assays but no geological logging were entered into databases

Relogging:

A programme of relogging carried out in 1997 had shown a very poor correlation to exist between original Aqua Regia assays and subsequent Fire Assays. In addition, assaying of previously unsampled volcano-genics showed significant mineralisation. In addition all original logs were paper based, and had been logged to an earlier WMC logging scheme.

Consequently it was decided to embark upon a substantial programme of relogging/reassaying of old core. The bulk of drill holes collared on the -180 and -145mRL levels were re-logged/assayed by G Masur with assistance from G McDermott. If the previous interval had been re-split, the remaining ½ core was assayed. If it had not been previously sampled the remaining full core was sampled. Resampling did not mix ½ and full core samples.

Diamond Core Relogging Programme, 1998							
Surface Diamond				6	1,209	6	1,215
U/G diamond	145-180 levels			71	5,922	71	5,993
	-212 upholes			14	837	14	851
							8,059

Figure 3: Diamond core relogging programme

A scattergram of original (aqua regia) vs new (fire) assays, composited to equivalent 1m intervals, again shows a very poor correlation. Check assaying of lab duplicates during the current programme shows good repeatability of fire assay grades.

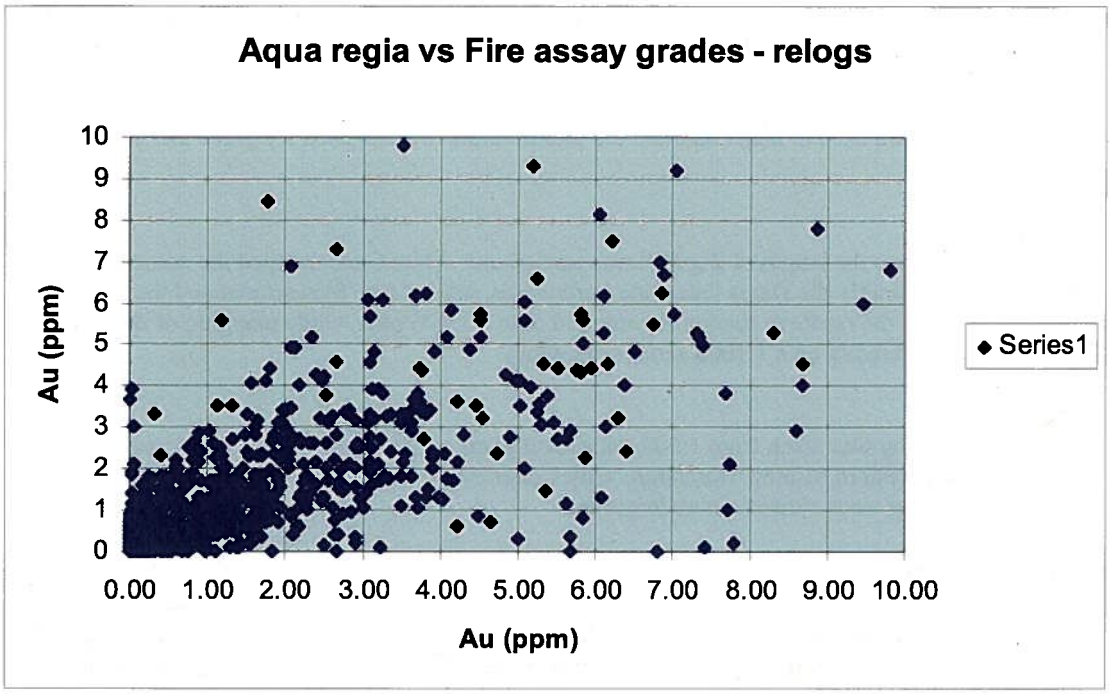


Figure 4: Aqua Regia vs Fire Assay grades for relogged core

A second assaying problem became apparent during this programme, Around 20 holes in the low MD300's show a problem, probably an incorrect dilution error. The original assays show long runs of elevated values in the range 0.5-1.2g/t. Re-assays of these holes show that these intervals actually have background grades in the range 0.1-0.15g/t. The problem can be clearly identified visually on section and is clearly shown in the attached scattergram.

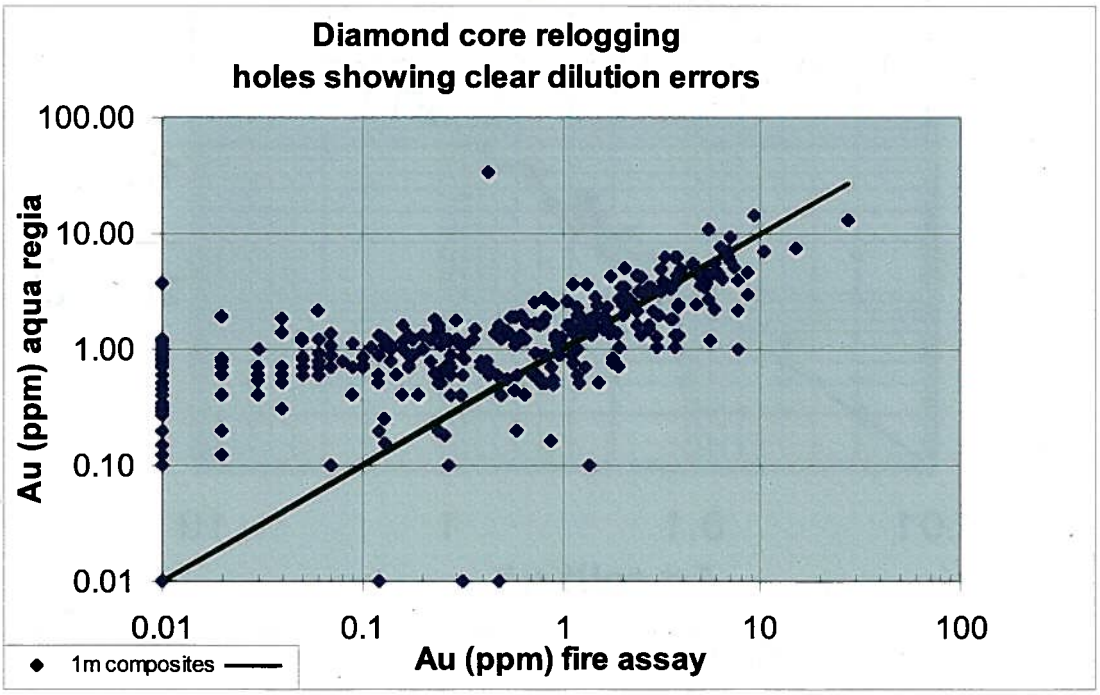


Figure 5: Aqua regia vs Fire assay grades – dilution errors

Solos:
Recent and old longholes were imported to a drillhole set and used to help condition the interpretation but were not used for grade interpolation.

Channels:
as above, imported and used to help condition the interpretation but not used for grade interpolation.

Mapping:
Surface:
Surface mapping along the length of Big Hill was carried out. Mike Gane mapped the entire northern section of the hill (roughly the Davis fence line northwards, while Mike Stewart mapped the southern section. Mapping of the northern section is compiled onto a 1:500 plan, while mapping of the southern section is compiled directly on a 1:1000 aerial orthophoto.

Underground:
Underground mapping was taken from 1:500 scale level plans and sections. Much of this mapping was carried out during Western Mining ownership. Quick checking of some features was undertaken but no significant remapping waqs carried out underground.

Assay Validation

Repeatability of field sample splitting was tested by re-splitting every20th sample and submitting this for assay using consecutive sample numbering. Lab was not aware of this practice. In general repeatability was excellent.
In addition lab duplicates and repeats of all sample types were tracked.

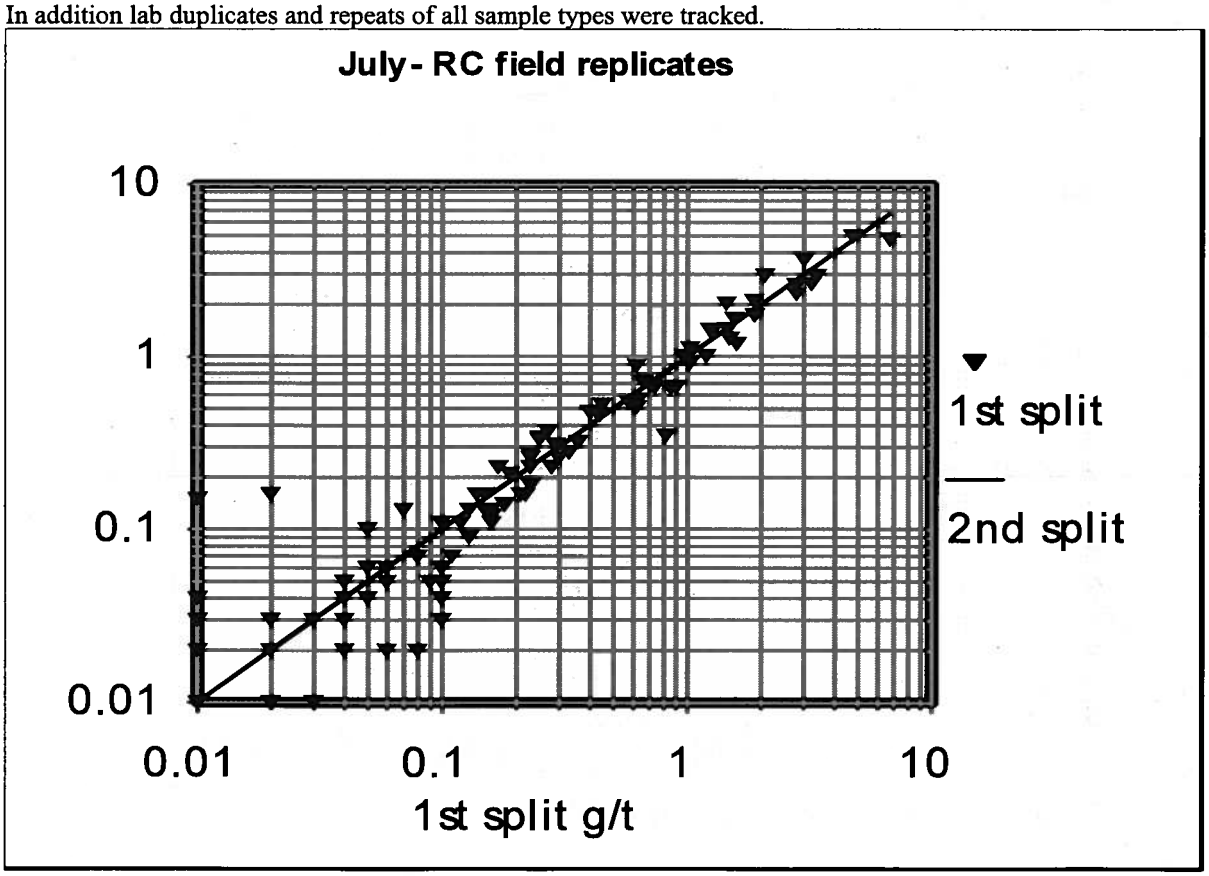


Figure 6: Assay Validation – Field replicates

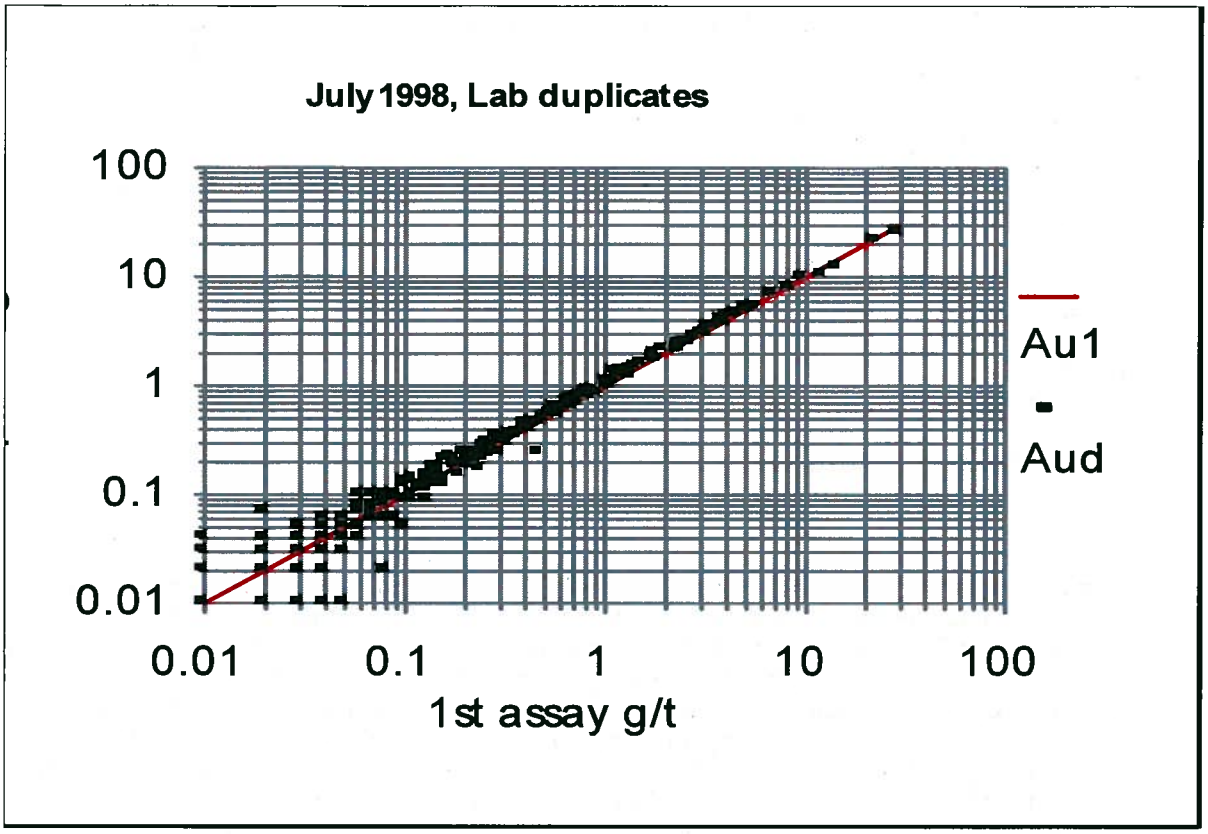


Figure 7: Assay Validation – Lab Duplicates July 1998

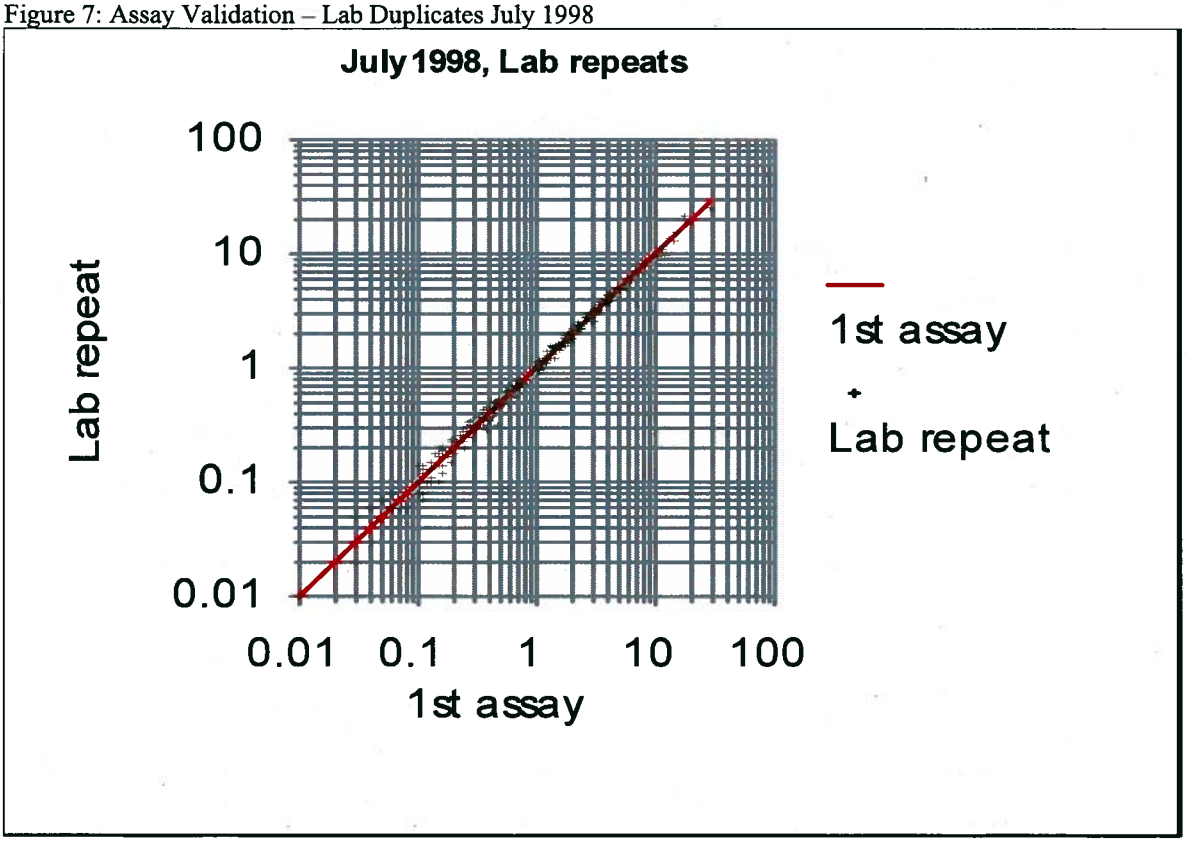


Figure 8: Assay Validation – Lab repeats July 1998

While only July data is shown here, this data was tracked monthly and showed consistently high repeatability.

Geological interpretation

No effort was made during interpretation to follow or match the existing nomenclature. Instead all geology was interpreted as seen.

Big Hill can be broken into 4 different geological ore domains:
Mariners
Allen's
Iron Duke
Magdala Flank

All except Mariner's and Allen's are separated by faults.

Mariners'

Mariner's is the least problematic geological unit to define. It has a consistent width and orientation, averaging around 14m thick and dipping to the northwest (mine grid). It constitutes a shear package, with massive quartz veining, A spur (domain 103) has been modelled on the upper surface (sections 289/290N). This is only defined by 2 drill holes and the geology in this area will be more complex than modelled. An offset extension of the Mariner's shear has also been modelled down dip (domain 102). This is offset by a fault (404) striking at 145 degrees (Mine Grid) and dipping at around 35-40 to the NE. It has an apparent reverse displacement of between 5 and 25m, decreasing northwards. The offset section of the ore body is then truncated at depth by the Cross Course fault zone. Evidence for both of these faults will be discussed later.

Diamond core intercepts include

SD540	36-45m	sheared to massive vein quartz
SD549	29-35m	massive to sheared vein quartz + sulphides
SD541	12-17.5m	clay altered siliceous volcs, strong FeO cementation, ? sheared
	17.5-33m	sheared quartz + boxwork after sulphide
SD550	47-71m	strongly sheared qrtz lode + volcs
SD544	12-25m	quartz lode,
	25-30m	siliceous volcanogenics, sheared, smokey vein quartz
	30-36.5m	siliceous volcs, pelitic, foliated, haem alteration
SD552	0-6m	Massive quartz lode

Beneath the 404 fault generally definition of geology is hampered by lack of diamond core. The principal reason for modelling a fault in this location is to explain the lack of semi-planar continuity of the otherwise predictable Mariner's structure. Without a fault, it would be necessary to invoke a 30-40° change in the orientation of the shear. Note that the geological confidence of the interp around these faults is lower than the more planar section of Mariner's, because of the greater degree of structural complexity and uncertainty in the location of the faults.

Allen's

Interpretation of the Allen's zone is somewhat problematic and the interpretation of geology in this zone is of lower confidence than other sections of the ore body. Exposures in Allen's open pit show lithological layering (S0 and later S2/3 foliation), as well as massive veining, to be very steeply dipping to vertical. Diamond core shows the same. Two massive veins exposed in the Allen's Adit strike approximately N-S mine grid and dip vertically. Some stockwork style veining is seen near the top of the zone (immediately beneath the Mariner's structure), but much of the geology consists of well bedded, foliated volcanogenic

sediments which are unpredictably mineralised. Much of the gold occurs in these sediments. Because of the highly oxidised state of these rocks the form of sulphide mineralisation is largely obliterated.

There is a poor correlation between quartz percentage and gold mineralisation. Accordingly the wire-framed structures which have been interpreted in the Allen's zone should not be regarded as having hard boundaries for ore/waste selection. Rather they have been used to constrain a tonnage that should reflect the tonnage of ore grade material adequately. They have dimensions that reflect the attitude of mappable geology, which in turn will probably exert the main influence on gold mineralisation. However the locations of the wire-frames are controlled as much by distribution of gold grades as by definable geological contacts/zones.

It is worth noting that variography using quartz percentage (based on an indicator set to 1 above and 0 below 65% estimated quartz by volume) showed a strong continuity towards mine grid north (plunge of 20°, range of 80m to sill), while the principal direction identified for gold grades was towards 345° (65m to sill) i.e. both gold grade and geology show similar orientations of maximum continuity. Because of the angle of drilling insufficient data is available to identify the ranges in the vertical direction.

It is suspected that a degree of remobilisation of gold within the weathering profile may exist, although no distinct enrichment horizons could be identified. The geology is of such variety that it is very difficult to separate any secondary effects from primary variation.

The relationship between the Mariner's and Allen's zones is also still problematic. Little overlap exists beneath the main flatter section of Mariner's and the Allen's zone, which dies out rapidly northwards beneath Mariner's. No clear evidence is seen of the timing relationships between these structures, the Allen's mineralisation appears to hang as a pendant of dilational veining beneath a change in orientation of the Mariner's shear structure.

Iron Duke

The Iron Duke zone is a wedge of geology occurring between the Scotsmans Fault and the Lower Cross Course Fault. It is the up-dip extension of the volcanogenic package/shear zone which has been offset from the main Magdala shear system by reverse movement on the Lower Cross Course fault. . It is truncated above by the Scotsmans fault zone, which again displaces the orebody westward. Interpretation of the geology in this zone is hampered by the strongly oxidised nature of the rocks. It has been interpreted as a series of constrained stockwork style ore zones (same as Allen's), where the envelopes have been used to constrain a tonnage, but which in reality will have "fuzzy" boundaries.

Magdala Flanks

"Magdala flanks" is the term used to describe the package of volcanogenic rocks lying to the west of, and continuing up dip from the nose of, the basalt antiform. The western margin of this zone is the contact between the volcanogenics and the mine schist, which often plays host to the Hanging Wall shear. The eastern margin is generally the contact with the basalts, but above the basalt noses the eastern margin is a transitional boundary to siliceous eastern schists, usually marked by a clear grade boundary. The Hanging Wall shear is currently being airleg mined from the –180 and –145 levels (and from sub-levels up to –109), between 269 and 278 N sections. Significant old stoping on this shear was encountered up dip of the current airleg workings.

Scotchmans Fault:
The Scotsman's Fault zone is a major structure which lies across the top of the basalt antiform and displaces the Magdala shear zone. It has been proven in underground development driven northwards at the –290, -?? and -?? levels. It's position at surface is pinned down by an exposure at 4730mE, 5500mN (Mine Grid). The general position of the fault can be clearly identified by the upwards termination of anomalous and ore grade structures. Below the base of transition zone the fault can generally be easily identified in both RC chips and diamond core by the presence of graphite and sheared textures. The logged intercepts do not fall onto a single, neat plane but are dispersed over a zone. In the weathered profile

identification of the fault is difficult. As is seen underground there is some significant mineralisation associated with the Scotsmans Fault but this is of a highly variable and poddy nature. Hole SP298 encountered 12m@5.5g/t before hitting a stope. However other holes drilled to establish the continuity of this mineralisation failed to encounter any significant mineralisation.

Cross Course Fault:
Four lines of evidence support the presence of the Cross Course fault zone:
1/ the Mariner's shear structure is truncated at depth.
2/ Surface mapping picked up evidence of sheared textures with an appropriate strike and dip at two locations, which project to the interpreted position of faulting.
4/ The Hanging Wall lodes (Scotsmans Vertical and Cross Vertical) are offset by a fault – shown on old sections. This is where the name "Cross Course" fault is derived. Offsets by faults dipping in this orientation are also shown in a 1900 section of the Mariner's and Sloan's shafts. These also project to approximately the correct position.
4/ Two of four diamond holes show good evidence of fault disruption and core loss which the upper and lower boundaries of the fault zone have been pushed through. The other two cored holes show little evidence of faulting, although zones of weaker core and core loss are found at the appropriate positions.

Lower Cross Course Fault:
The Lower Cross Course Fault is a newly recognised fault. It was interpreted independantly of the Cross Course Fault and it wasn't until the two were viewed together that it was realised that they are co-planar. Again no definitive plane of faulting can be identified but the evidence of offset of the ore zones is compelling. It has only been modelled as a single plane rather than as a zone, principally because of lack of drilling control – but also because the zone appears more discrete.

The timing relationship of the Cross Course to the Scotsmans is unknown. Below –80mRL there is a dogleg in the Scotsmans fault zone coincident with the intersection of the Cross Course, which suggests the Cross Course may in fact post-date the Scotsman.

Basalts:
As with previous interpretations basalts were interpreted as a series of thrust stacked antiformal noses. Detached noses floating above the main termination of the basalt are also present. These occur primarily within the weathered zone so deciphering the complex shearing around these noses is difficult.

Foliation and shearing
Two dominant foliation fabrics are recognised. S2 is a pervasive schistosity accompanying tight to isoclinal folding. It is generally north-south striking and steep to vertical in orienatation, although variation does occur. Dips generally flatten slightly away from the basalt nose/ shear zone area. S3 is a less pervasively developed crenulation cleavage. Regionally it has a shallow to moderate dip to the north or north east (Mine grid) although locally dip is to the north west or west. In the Davis pit area the dominant foliation is S2, dipping vertically to steeply east.
On the east wall of Davis Pit a set of D4 shears dip at a shallow angle to the northeast sub-parallel to the plunge of the basalt noses. A shear of this orientation appears to have been involved in the documented failure which occurred during the life of the Davis Pit. Shears of similar orientation have been interpreted in the deeper sections of the ore body.

Solid modelling - Minesight

Geological solids
The package of Magdala volcanogenics as a whole plays host to the bulk of mineralisation, Where continuous mineralised structures can be defined with a reasonable degree of confidence, these structures have been constrained with in wire-frame solids. Where it has not been possible to confidently define the geological control to mineralisation this material has been left unconstrained. There are significant volumes of the volcanogenic package and significant numbers of assays that are unconstrained.

Geology is coded using a 3 digit code, the first digit being the "family" code, while the second 2 arejust incremental numbers. The following family codes have been used:

Family Code	Description
1xx	Mariner's, Iron Duke and Magdala constrained structures
2xx	Allen's constrained structures
3xx	Mine Schist/Eastern Schist waste
4xx	Faults
5xx	Basalts
7xx	Unconstrained volcanogenics
8xx	Stopes

Figure 9: Geological code types

The following solids have been used in construction of the 3D solid model, and for coding of geology to the 3D Block Model:

Domain	Primary domains on basis of geology	Sub-domained on basis of orientation	Description
Mariner's constrained structures	101	121	Main Mariner's shear structure
		122	"
	102		Offset of Mariner's beneath fault 404
	103		Spur sitting above Mariner's shear at steeper angle. Outcrops
Magdala Flanks constrained structures	105	123	Southern extension of Hanging Wall shear
		124	"
		125	"
	106	126	Main Hanging Wall shear
		127	"
		128	"
	107	129	Equivalent of Central Lode and 110
		130	"
	108		Stkwk envelope, partly stoped
	109		Shortcut shear cutting off shoulder of 106 (HW) below SLF
	110		Central lode + stkwk mixed
	111		? Extension of Central lode, west of detached basalt nose
	112		HW type shear, developed on 145/180levels, inside HW proper
	113		Flat shear, beneath detached basalt nose
	114		Splay from underside Hanging Wall (Davis area)
	115		Flat shear splaying from Hanging Wall, dips north, nth end Davis Pit
	116		Linking shear, HW -> Central
Iron Duke constrained structures	117	131	Equivalent of HW/Mariner's shear in Iron Duke zone, i.e forms western boundary of package
		132	
	118		Stockwork style grade envelopes
	119		"
	120		"
Allen's constrained structures	201		Stockwork style grade envelopes, boundaries based primarily on grade but dimensions and orientation reflect geological controls
	203		
	204		
	205		
	206		
	207		

	208		
	209		
	210		
	401sol		Scotchmans fault solid
	405sol		Cross Course fault solid
Mariner's volc package	700		Package of volcanogenic rocks. Western boundary defined by base of Mariner's shear, eastern boundary by grade transition to siliceous Eastern Schist
Allen's volc package	701		Volcanogenic package within which Allens stockwork envelopes occur, bounded above by Mariner's shear, below west by Cross course Fault, below east by grade transition to E Schist
Iron Duke volcs	702		Volcangeinc package within which Iron Duke lodes occur, bounded above by Scotsmans fault, below west by Lower Cross Course fault, and to east by grade transition to E-schist
Magdala flank volcs	703		Volcanogenic package within which Magdala lodes occur, bounded to west by contact with Mine Schist (often coincident with Hanging wall shear), to east by basalt contact (or East schist)
Basalts	50,502,503,505, 507,508,509,513		Basalt noses

Figure 10: Geological domains

Underground workings:

Known underground workings were imported from MS-STOPES minesight project:
Data imported included:

Airlegs:	current sub-level development and rises and airleg stoping on Hanging Wall shear
Oldstopes:	Stoping outlines estimated from stope designs for development between 1981-1993 (WMC)
Levels:	All decline and level development
Old workings:	Principally pre-1925 development levels digitised from old plans, source generally unknown, positional accuracy unknown.

Old stopes, previously unrecorded, were also encountered during drilling. An estimate of the possible stope shapes has been made by joining stopes between intercepts, and projecting ½ drill spacing along interpreted structures. The resultant solids, when compared with the current airleg stope shapes are almost certainly too large.

Modelling of other aspects

Weathering model

A model of weathering state was constructed using the following criteria:

Base of total oxidation was defined as the point downhole at which weathering is not completely pervasive of the rock mass and where remnant fresh rock first appears.

Base of transition zone was defined as the point downhole at which oxidised material becomes an insignificant volume component (?<1% by volume).

In diamond core, these boundaries were identified from logging and core photographs. In RC chips identification was principally made from colour changes. Many of the holes were relogged specifically for identification of these boundaries.

Both surfaces are highly variable in reality due to uneven fracture density and fluid flows, and holes may pass in and out of totally weathered, partially weathered and fresh material. The surfaces are therefore necessarily smoothed.

Dig-ability model

Assignment of dig-ability is based primarily on the estimation of rock mass strength from diamond drill core, and the likely quartz content of a geological domain.

All cored diamond holes drilled since May1997 (and up to Dec 1998) have been logged by SGM staff to a semi-quantitative logging scheme based on the guidelines of the IRMS. These holes were not treated in any special manner after drilling and in most cases spent a significant period of time on the core racks. All have since been destroyed by full core assaying.

A further set of 8 diamond core holes was drilled specifically to look at geotechnical characteristics in the region of likely final pit wall positions. These holes were sealed in plastic immediately after being brought to surface and only unwrapped immediately prior to logging. The rock strengths of lithologies in these holes were significantly lower than the strengths of similar lithologies in core that had been exposed to air. Therefore strengths estimated from holes exposed to air will be significantly higher than in-situ rock strengths, particularly in clay rich lithologies with higher initial moisture contents. In general strengths appeared to be approximately 1-2 categories higher than in-situ.

All holes were plotted in section with strength being plotted as a coloured bar, and quartz percent as a wiggle trace. Strength is strongly related to quartz content. Constrained shear structures generally have elevated quartz contents also.

Because the diamond holes are widely spaced and represent a series of semi-random snapshots of the geotechnical conditions no interpolation of values is possible and they can really only be used for assessing trends. Even then the data must be treated with caution.

Following input from Kevin Rosengren, and discussions with Karl Guilfoyle and Jeff Moncrieff the following dig-ability categories were applied to the model:

Free dig (1)
Rip (2)
Drill and Blast (3)

Geol code	DIG	
	0 to -40m below surface	> -40m below surface
300,301 (waste = Mine schist and Eastern schist)	Free dig	Drill and Blast
4xx (Fault zones)	Free dig	Drill and Blast
5xx (Basalts)	Free dig	Drill and Blast
700,701,702 (Mariner's, Allens, Iron Duke volcanogenics)	Free dig	Drill and Blast
703 (Magdala unconstrained) - >0.8g.t cutoff	Drill and Blast	Drill and Blast
< 0.8g/t cutoff	Free dig	Drill and Blast
Mariner's constrained	Drill and Blast	Drill and Blast
Allens – Inside Allens_dig solid	Drill and Blast	Drill and Blast
Iron Duke – Inside ID_dig solid	Drill and Blast	Drill and Blast

Figure 11: Dig-ability coding

There is a general trend that below approximately 40m all material within the pits is likely to require blasting to be dug efficiently. Ripping or free dig may still be possible below this depth but will be increasingly inefficient.

Because of their generally elevated quartz contents, all constrained structures are drill and blast.

An attempt was made to krige a quartz percentage to the model for the Allens and Iron Duke zones, where there is poor correlation between gold grade and quartz percentage (although both gold and geology show the same spatial continuity). The resulting distribution of quartz looks geologically sensible but it was decided against using this to set a dig-ability value. Instead an envelope which encloses all likely ore in

these zones was constructed around the individual constrained structures, and all of this zone assigned to Drill and Blast.

The Magdala volcanogenics were problematic. It was decided that all blocks interpolated as "ore" (ie with a grade greater than 0.8g/t) would be drill and blast while all sub-ore blocks would be free dig (down to 40m). Although this will be wrong in detail, the relative volumes of drill/blast cf free dig should be approximately correct.

Bulk Density

Specific gravity (dry bulk density) was calculated on the basis of averaged core weights.

- All core weighed to +/- 10gm prior to despatch, core weights recorded on sample sheets
- Core weights entered to database
- Specific Gravity values calculated as follows:
  SG = measured weight/volume (=length*area*recovery%)

- SG values imported to file 11 (bh9811.den) and coded with geology

- Mean SG values by geological domain and weathering type reported excluding obvious outlier values (both high and low). Stat reports are appended.

- Core weights were of air dried core. Test work shows that in-situ moisture contents vary between 2 and 20%, but that these have uniformly dropped to 0.5-2.5% after air drying.

Mean values used in the model are as follow:

Domain		Oxide	Transition	Sulphide
Mariner's		2.3	2.5	2.85
Allen's	Constrained	2.15	2.3	2.85
	Volcangenics	2.1	2.3	2.85
Iron Duke	Constrained	2.15	2.3	2.85
	Volcangenics	2.1	2.3	2.85
Magdala	Constrained	2.1	2.3	2.7
	Volcangenics	2.0	2.3	2.85

Figure 12: Bulk Densities of ore

This method was deemed to be the most effective way of accounting for voids and fractures which cannot be accounted for in wrapped core determinations, and which provide the largest variable component in oxidised core.

Moisture determinations:

Moisture determinations performed on RC chips immediately upon delivery to surface showed in-situ moisture contents to vary between 2 and 20%. After air-drying for two days (average exposure of core to air before sampling?) moisture contents varied from 0.5-2.5%. In-situ moisture is directly related to clay percentage (and therefore parent lithology) and is not directly linked to depth.

Recovery

Metallurgical recoveries were loaded to the database, imported to a file 11, and coded with geology. Insufficient data is available to attempt any meaningful interpolation. The following mean values were applied to all ore zones

Oxide	92%
Transition	90%
Sulphide	84%

Topography
Surface DTM at 1m contour resolution based on aerial photography flown Jan 1998. Includes backfilled area of Davis OC. Davis pit built as solid from sections digitised from old sections. (Set Davispit, Member Solid -> exported as Davispit.dxf/davispitsurf.dxf)

Drill Holes

Loading

Final extraction date for data from Paradox databases was 16th December 1998. All surface Diamond and RC drilling was included in this extraction. There were still some outstanding underground relogs at this date which were not included. A tabulation of all drillholes used in this estimate is included at Appendix 1.

Data was loaded using the SGM grade control procedure (with some modifications). Load procedure was interrupted after extraction of text files from Paradox databases and the text files edited (using a macro in Textpad) to remove any holes which did not have an SD, SP, MD or Y (partial original sections of relogged holes) prefix. Extraction was restricted by RL (collars between 20 to –250m RL). No holes drilled below –250mRL penetrate up to above.–200mRL.

BH9811.exp	Exploration drillholes, au coded by geology
BH9811.geo	Exploration drillholes, geology alpha item rock_type converted to integer
BH9811.exg	Exploration drillholes, geology
BH9811.dat	Combined (exp,lon,old and chn files) – not used as only used expl holes for interpolation
BH9811.lon	Longholes
BH9811.old	Old solo holes
BH9811.rqd	Exploration geotechnical logging coded by geology, strength estimates (alpha) converted to numeric. Used to examine changes in RQD, strength, recovery, fracture density and quartz percentage by depth downhole.
BH9811.qtz	Quartz percentage and logged geology (alpha item rock_type converted to integer) coded by geology, examine quartz content by geological domain
BH9811.den	Sample weights and calculated SG values, core recovery

Figure 13: Drill hole file names

A list of items and what is in them is appended.

**NB original Paradox database contains negative values to represent specific sample information

Value in AU1_num field in database	Description
-6	< 0.01
-5	Lost Core
-4	Sample missing
-3	ENA
-2	Not sampled
-1	Below level of detection
0	Since ?1993 values below level of detection have been entered as 0.0

Figure 14: Non-numeric codes in gold field of database

Values in database of -5,-4,-3,-2, are be loaded to MEDS file 11 as –1
Values in database of -6 and -1 should be loaded to MEDS file 11 as 0.01 but are actually loaded as –1 (only affects 152 old values from database)

Values in database of 0.0 are loaded to MEDS file 11 as 0.0 – To avoid problems with stats, LOG plots and the kriging matrix these were converted to a value of 0.01.

Drillhole coding

Drill hole File BH9811.exp was coded using pspear.met from DXF/SHL files of geological solids. Sequence of coding was as follows:

	DXF solid	Code	Comments
Run 1	E-schist.dxf	301	Set Default code of 300
Run 2	700.dxf	700	Mariner's volcanogenics
	701.dxf	701	Allen's volcanogenics
	702.dxf	702	Iron Duke volcanogenic
	703.dxf	703	Magdala Flank volcs
	401.dxf	401	Scotchmans Fault Zone
	405.dxf	405	Cross Course fault zone
Run3	1xx	1xx	All constrained 1xx structures
	2xx	2xx	All constrained 2xx structures
Run4	5xx	5xx	All basalts, overprint all before

Figure 15: Sequence for coding geology to drillholes

All coding was checked against solids in Minesight, minor manual recoding was necessary (about 2 holes for each of solids 208 and 116 required fixing)

Composite statistics

Raw composite statistics, (no. comps, min and max values, mean, median and declustered statistics, cut values applied and cut means (all >0.0g/t cutoff)are tabulated in spreadsheet BH1998.xls.

Cell declustering shows that overall there is a slight clustering of high grade. In three domains (127,128,130) declustered grades were actually higher than raw grades, but in all other grades decreased.

Cut values have not been uniformly applied. Cut values for individual domains or groups of domains were assessed from log probability plots. Top assay values were listed and used to help decide what if any cut value was appropriate. Where a cut was deemed necessary, a value was applied which reduced the cut mean to around the level of the cell declustered mean.

Lognormal probability plots are included at Appendix 2.

Variography

Analysis of variography was carried out for all domains (or grouped domains) with sufficient data.

For the constrained structures variograms were rotated into the plane of the structure. Run parameters were varied for each domain but generally utilised a lag of around 20m, windowing angles of 15+/- 10° in increments of 15° horizontally and 10° vertically, and bandwidths of 10-20m. Pairwise relative variograms generally gave the best results, with correlograms also producing usable results.

Variogram parameters, and associated kriging paramters, are tabulated in the spreadsheet krig_parm.xls.

All runfiles, report files and data files are stored in directory I\medsys\bh1998\varios\

Modelled variograms were based on the following experimental variograms:

Summary of variography used for BH1998					
File extension for DAT/RPT/RUN files	Based on	Applied To	Orientation of modelled variogram (rotated)		True trend principal axis within plane of structure (Mine Grid)
			Principal Axis	Principal axis	
.128	128	128	20/0	2D global	004
.130	129,130	111,129,130	60/0	140/10	348
.ts1	126,127,128 below – 100m RL	109,112,123,124,125,126,127	140/10	60/-10	334
.stk	108,110	108,110,113,116	80/-20	= principal	11
.a64	201-210	201-210	2D global	2D global	345
.id	118,119,120, 131,132	118,119,120,131,132	120/10	20/-10	328
.m6a	121,122	102,103,121,122	150/-10	60/0	330
.700	700	700	0/0	100/0	10/-18
.701	701	701	160/0		345/-20
.702	702	702	160/0	80/0	344/-24
.7t1 and 7t2	703	703	150/-10	80/0	348/-20

Figure 16: Summary of variography

Modelled variograms are included at Appendix 3

In general principal axes trend N to NNW,at shallow to moderate plunges (roughly parallel the plunge of the basalt noses).

Where modelled variogram parameters were transferred between domains the following method was used.

1/Identify azimuth and plunge (= MEDS Rotation to new N and DIP N) of principal axis of modelled variogram from stereonet.
2/Maintaining azimuth, obtain new DIP N and DIP E to ensure variogram lies within plan of structure.
3/Apply rotation to kriging runfile, use no rotation within variogram parameter file

Block Modelling

Block Model name: BH9815.new

Following items are in model

TOPO	0.	100.	0.1	/ Block % below TOPOG
mcat	0.	99.	1.	/ Material category for block (not set)
class	0.	4.	1.	/ Classification (all interpolated blocks currently set to CLASS =2,(indicated)
weath	0.	4.	1.	/ Weathering (0=air, 1=oxide, 2=transition, 3=sulphide)
rec%	0.	100.	0.1	/ Recovery percentage (oxide=92, transition=90, sulphide=82)
prf	0.	5.	0.1	/ Preg Rob Factor (NOTSET)
sg	0.	10.	0.01	/ Specific gravity (based on geology and weathering)
geol	0.	999.	1.	/ Whole block geology item (majority code geol1/geol3)
aubk	0.	128.	0.01	/ Whole block gold item (accumulated above nominal cutoff grade)
auik	0.	128.	0.01	/ Whole block indicator kriged gold item (NOT SET)
ore%	0.	100.	0.1	/ Whole block ore % (volume % of block greater than cutoff grade)
netv	0.	250000.	1.	/ Net value (NOT SET)
hard	0.	10.	1.	/ Hardness (being used as FLAG item for assigning block Au)
dig type)	0.	3.	1.	/ Diggability (1=free dig, 2= rip, 3= drill and blast, crudely set on weathering and ore
kerr	0.	999.	0.001	/ Kriging error (NOT SET)
ncmp	0.	99.	1.	/ No. comps (being used as FLAG item for assigning block Au)
dist	0.	1000.	0.1	/ Distance to nearest composite (not set)
stope	0.	10.	1.	/ Stope item (being used as FLAG item for assigning block Au)
stop%	0.	100.	0.1	/ Stope percentage

Partial group 1 (constrained ore structures, codes 100's, 200's)

geol1	0.	999.	1.	/ Partial group 1 - geology code
geo1%	0.	100.	0.1	/ Partial group 1 - geology %
clas1	0.	4.	1.	/ Partial group 1 - classification
mcat1	0.	99.	1.	/ Partial group 1 - material category
ncmp1	0.	99.	1.	/ Partial group 1 - # composites
dist1	0.	100.	1.	/ Partial group 1 - distance to nearest comp
aupl1	0.	999.	0.01	/ Partial group 1 - polygonal gold grade
auok1	0.	99.	0.01	/ Partial group 1 - Ordinary kriged gold
auik1	0.	99.	0.01	/ Partial group 1 - indicator kriged gold

Partial group 2 (constrained ore structures, codes 100's, 200's, = minority portion of block where more than 1 ore structure in a block)

geol2	0.	999.	1.	/ Partial group 2 - geology code
geo2%	0.	100.	0.1	/ Partial group 2 - geology %
clas2	0.	4.	1.	/ Partial group 2 - classification
mcat2	0.	99.	1.	/ Partial group 2 - material category
ncmp2	0.	99.	1.	/ Partial group 2 - # composites
dist2	0.	100.	1.	/ Partial group 2 - distance to nearest comp
aupl2	0.	999.	0.01	/ Partial group 2 - polygonal gold grade
auok2	0.	99.	0.01	/ Partial group 2 - Ordinary kriged gold
auik2	0.	99.	0.01	/ Partial group 2 - indicator kriged gold

Partial group 3		(unconstrained.		Waste codes: 300's =HW and eastern schist waste, 400's=fault zones, 500's=basalts,)
		(unconstrained		Ore codes: 700's= volcanogenic package,)
geol3	0.	999.	1.	/ Partial group 3 - geology code
geo3%	0.	100.	0.1	/ Partial group 3 - geology %
clas3	0.	4.	1.	/ Partial group 3 - classification
mcat3	0.	99.	1.	/ Partial group 3 - material category
ncmp3	0.	99.	1.	/ Partial group 3 - # composites
dist3	0.	100.	1.	/ Partial group 3 - distance to nearest comp
aupl3	0.	999.	0.01	/ Partial group 3 - polygonal gold grade
auok3	0.	99.	0.01	/ Partial group 3 - Ordinary kriged gold
auik3	0.	99.	0.01	/ Partial group 3 - indicator kriged gold

xtra1	0.	100.	1.	/ Flag item for limiting interpolation with expanded search parameters
xtra2	0.	100.	1.	/
xtra5	0.	1000.	1.	/ Used for composite weighting item for 608 reports
xtra3	0.	100.	0.01	/
xtra4	0.	1000.	0.01	/

Modelling method

Three "partial groups" have been used to account for up to two constrained solids and 1 unconstrained rock type to occur within a single block. For each block one of these partials represents the items associated with one geology type – geology code, % of the block, kriged grade, material catogory, classification etc.

Coding geology

Because the unconstrained geology has not been constructed as a solid, (i.e a negative or inverse solid of the constrained solids) this cannot be coded to the model from Minesight in the same manner as the constrained solid. The unconstrained geology is therefore always coded to partial group 3, while the constrained solids are coded to partial group 1 and 2 (i/e constrained solid which is largest portion of the block to group1, constrained solid which is smaller portion to code 2)

Partial groups 1 and 2 are coded from Minesight using "update model" to code geol1, geo1%, geol2, geo2% (all 1xx and 2xx solids)
Partial group 3 is coded from Minesight using update model to code integer item only (50% threshold) to geol3 in the following sequence:

E_schist
Volcanogenics and fault zones
Basalts

Geo%3 is defined as 100-(geo1% + geo2%) – i.e 100% where no constrained solids, remainder of block where constrained structures exist.

Topography

Item topo coded using 50% threshold above/below surface (also coded to file13)

Specific Gravity

Coded with M612rp using AssignSG.bat

Recovery

Coded with M612rp using AssignREC.bat

Classification

The entire model was classified as Indicated (2). The deposit is evenly drilled out at approximately 20*25m centres. The majority of blocks were filled using tight search parameters. Refinement of classification may be carried out at a later date.

Dig-ability

The following sequence was used to code ITEM DIG

- Built solids of the main "ore zones" of Allen's and Iron Duke zones (i.e solid enclosing the constrained solids, but leaving volcanogenics on margins)
- Copied PCFtopo to PCFtopo-40, moved entire topography down by 40m. Edited PCFtopo-40 to remove existing pits, exported surface to DXF
- Used PDTMCD.met to code model above/below surface. Code 0 to –40m DIG=1, >-40m DIG=3
- From Minesight update model with Allens_dig and ID_dig (DIG = 3)
- Use M621RP to code DIG = 3 where Geol = Mariner's(102,103,121,122) and Magdala (108-116, 123-130)
- Use M612RP to code DIG = 3 where GEOL = 703 and AUBK > 0.8g/t

Interpolation of grade

General

Ordinary kriging was used for interpolation. The general search strategy was to limit selection of composites to the first two rings of samples (where drillhole spacing approximately 20*25m) and keep the maximum number of composites used to 15 composites. The estimate is thus fairly local.

Any blocks that were not filled on the first pass were flagged, then the flagged blocks only kriged using an expanded search environment. Any blocks still not filled were then left un-interpolated

File naming convention:

R6241.xxx	First partial group, normal search parameters
R62412.xxx	First partial group, expanded search parameters
R6242.xxx	Second partial group, normal search paramters
R62422.xxx	Second partial group, expanded search parameters

Kriging parameters used for each domain are tabulated in the attached spreadsheet. An example of runfiles for the initial and expanded search parameters are also included in Appendix 4.

Validation

Interpolation was validated firstly by visual assessment in plan and section. In addition mean global kriged grades were compared with mean input grades by domain (see Spreadsheet XX)

A grade tonnage plot for the whole block accumulation within the January 1999 pit design and the relative contribution to that from unconstrained and constrained structures is shown below:

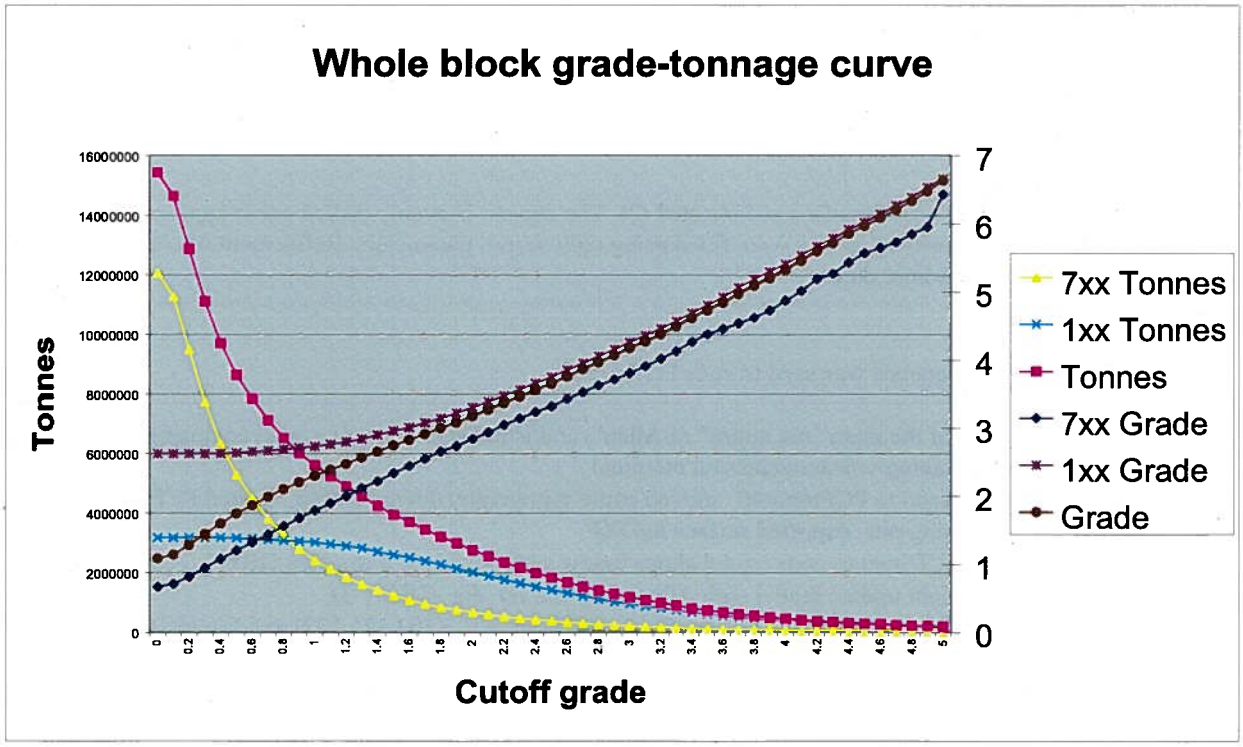


Figure 17: Grade tonnage curves, within Jan 1999 pit design

A grade tonnage curve for the low grades is also shown.

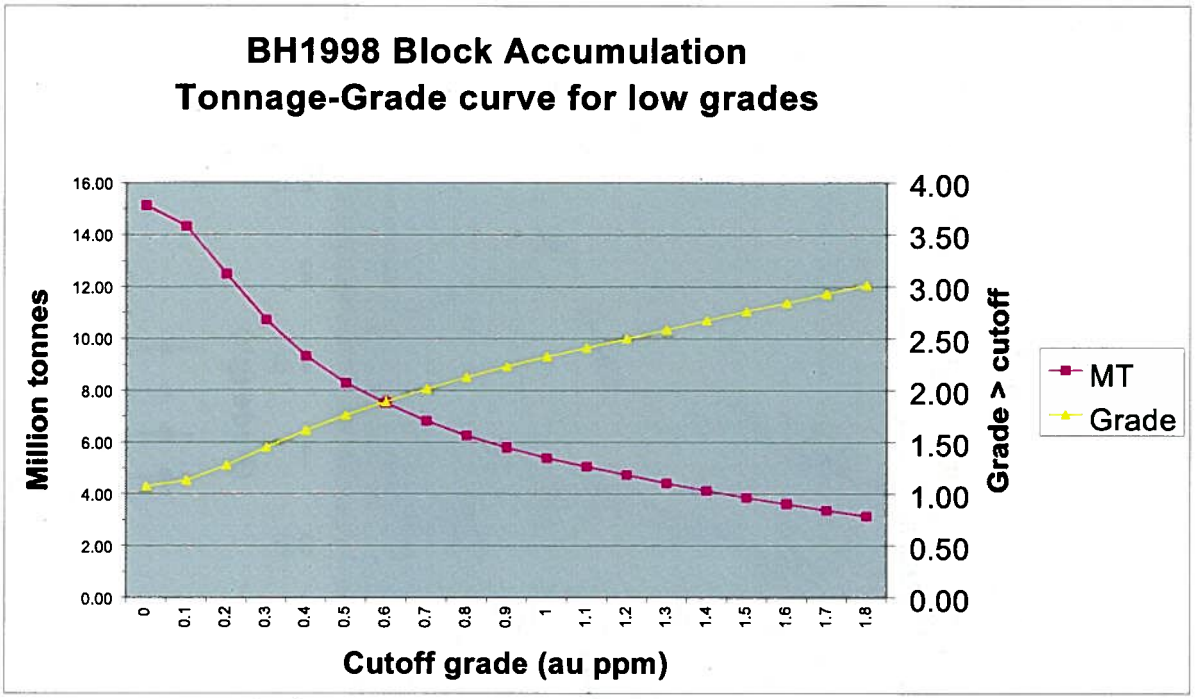


Figure 18: Whole block grade tonnage curve, low grades

Change of support was modelled for Domain 703, using Herco.exe. Theoretical vs actual grade tonnage curves for this domain are also shown below:

Figure 19: Change of support modelling, Domain 703 (Magdala volc's)

Input grades vs mean modelled grades by bench for both constrained structures

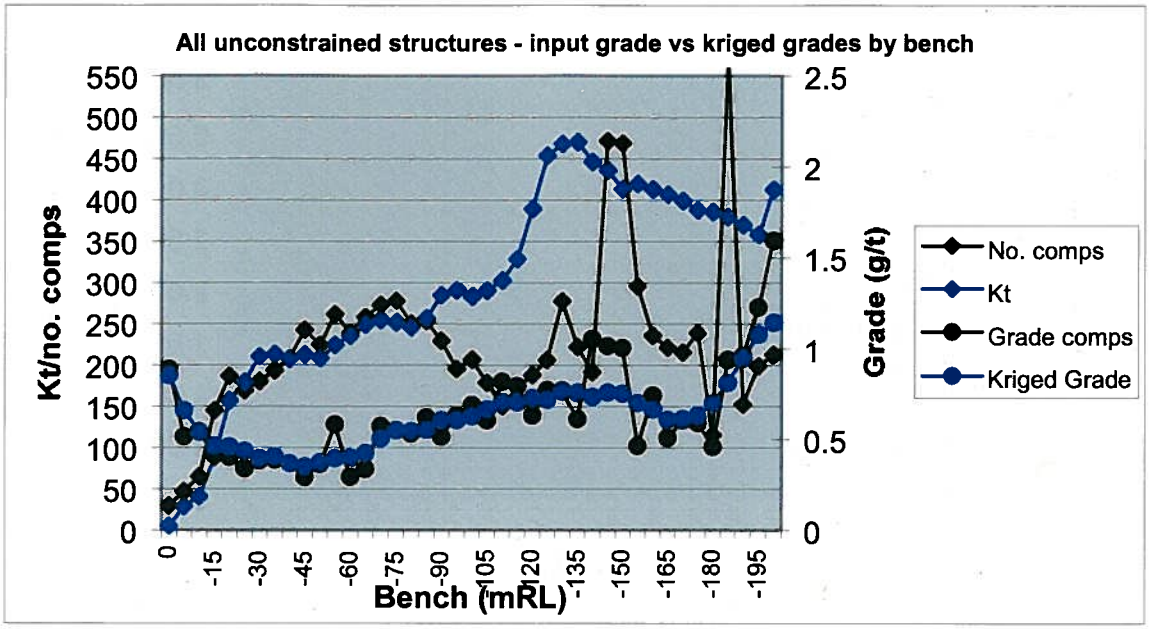


Figure 20: Input grades vs kriged output grades by bench. Constrained structures

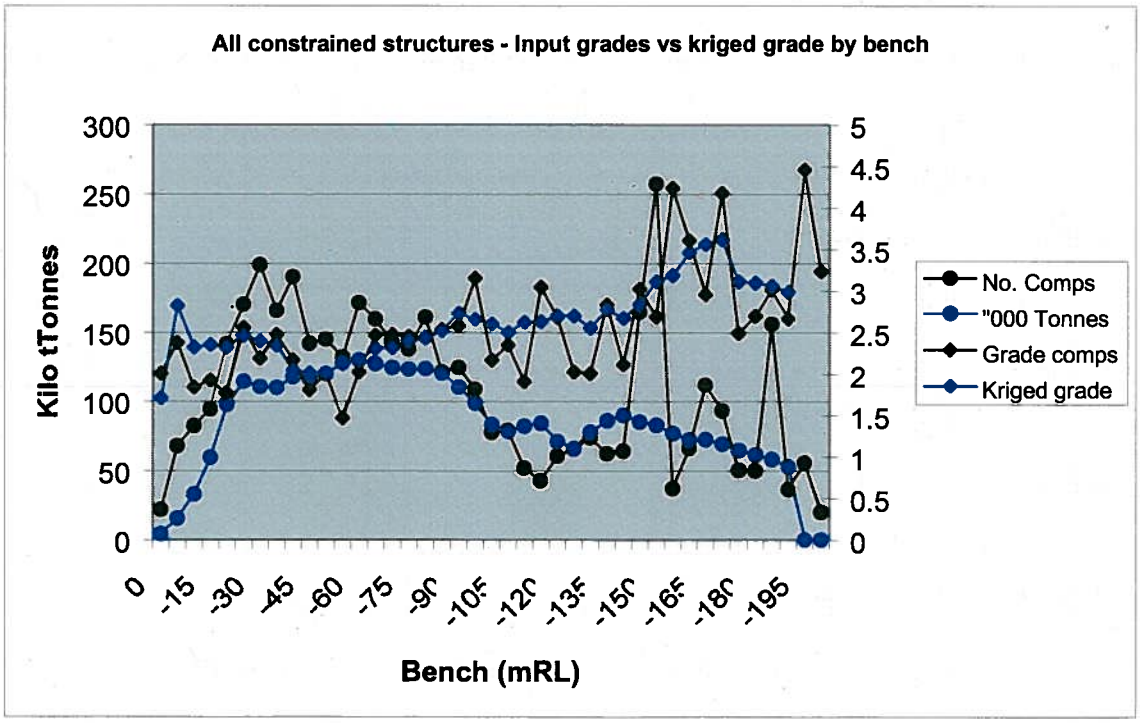


Figure 21: Input grades vs kriged output grades by bench. Un-constrained structures

Resource Table

The global resource estimated to –195m RL is as follows:

Big Hill 1998 Resource Estimate - global resource to -195m RL																		
	> 0 g/t cutoff									> 0.8g/t cutoff								
	Constrained			Unconstrained			Total			Constrained			Unconstrained			Total		
Material	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz	Kt	Grade	K Oz
Oxide	1,748	2.28	128	3,352	0.40	43	5,100	1.05	171	1,647	2.39	126	341	1.31	14	1,988	2.20	141
Trans	402	2.61	34	1,308	0.57	24	1,710	1.05	58	398	2.63	34	286	1.49	14	685	2.16	47
Sulph	1,261	2.98	121	7,059	0.73	165	8,320	1.07	286	1,257	2.99	121	2,240	1.59	115	3,497	2.10	236
Total	3,411	**2.58**	283	#####	**0.62**	232	#####	**1.06**	515	3,302	**2.65**	281	2,867	**1.55**	143	6,169	**2.14**	424

Figure 22: Global Resource to –195mRL

At present no account of of old stope volumes is made within the model. The estimated volume of old stoping in the south pit area is approximately 14,800 m^3 or 35,000t. This is liable to be an overestimate. Stope volumes were estimated by joining stope intercepts between drillholes and projecting ½ drillhole spacing. The volume is insignificant in terms of the total open pit volume.

No estimate has yet been made of the potential underground resource remaining above –195mRL. Old stoping has not yet been properly examined and it is not yet possible to account for the volume removed in an estimate of the resource. This work will be undertaken as part of the Big Hill potential underground resource study.

Further Work

Indicator kriging of Allen's, Iron Duke, Magdala volcs?
Coding stopes to model (reporting grades within stopes)

Preparation of model for Pit Optimisation

Whole Block Accumulation

Item GEOL is based on majority proportion of block
ie if geo1% > geo3%, GEOL = geol1, else GEOL = geol3
set using accumGEOL.bat,
(using st-geo612 -f run612.geo gives same result)

Accumulation of both ore% and geology assumes a nominal cutoff grade. This has initially been chosen at 0.8g/t

ORE% is the sum of those partial volume percentages of the block, with a grade greater than the COG. It represents the % volume of the block greater than COG (not the probability above cutoff).

AUBK is the weighted average grade of all partial volumes of the block with a grade greater than COG.

Accumulation has been carried out using the batch file assignBLOCKAU.bat
First set three flag items for each partial group based on CUTOFF grade

i.e FLAG1 (item HARD) = 1 in range AUOK1 0.8->999
FLAG2 (item NCMP) = 1 in range AUOK2 0.8->999
FLAG3 (item STOPE) = 1 in range AUOK3 0.8->999

then accumulate using RP's ranged by the following flags

FLAG1	1	-1	1	1	-1	-1	1	-1
FLAG2	1	1	-1	1	-1	1	-1	-1
FLAG3	1	1	1	-1	1	-1	-1	-1
RUN612.??	bal	b23	B13	B12	B3	B2	B1	B0

A fortran subroutine has also been written to achieve the same result
executable = st-accum612
run files = run612.geo

Cutoff is set by **PAR1.** Change PAR1 to accumulate at different cutoffs

Accumulation of a single block grade, tonnage, and ore% leads to a shift of material between constrained and unconstrained in the block accumulation. This is because the GEOL code is set on the basis of majority code (i.e always given the code of the largest component of the block), while the grade and ore% are accumulated wherever grade is greater than the specified cut-off. This mostly affects blocks on the margins of constrained structures, as the grade of constrained structures is more likely to be above cut-off than the unconstrained. Material where the grade is contributed by the constrained but the ore% of the constrained is a minority (ie ore1%<50) will be re-classified as unconstrained in the whole block accumulation. The reverse is possible but is less common.

The following shifts of tonnage occurred:

From	**To**	**Tonnage ('000 tonnes)**
Constrained (geol1=1xx,2xx;auok1>= COG	Waste (geol = 3xx,4xx,5xx)	168
Constrained (geol1=1xx,2xx;auok1>= COG	Unconstrained (geol=700-703)	743
Unconstrained (geol3 = 700-703, auok3>=COG	Constrained (geol=1xx-2xx)	134

The tonnage shifted to waste was effectively lost to the model. To prevent this occurring the following manual re-coding of block geology was carried out:

Where 1/ constrained volume is a minority
And 2/ the constrained volume has a grade >0.01g/t
And 3/ the unconstrained portion of the block has a waste code

Then GEOL = GEOL1

After correction to the above problem there is a nett shift of 609Kt from Constrained to Unconstrained. This makes no difference to pit optimisation.

There is no significant change to the global tonnage or grade. There is a 3% difference in the global tonnage reported using full block partials compared to the block accumulation and no difference in grade.

Appendix 1

Drill hole collars

Appendix 2

Log normal probability plots

Appendix 3

Modelled variograms

Appendix 4

Kriging Runfiles

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX C

C7 Resource Report

November 2005

Resource Summary Sheets

Stawell November 05 C7 Resource Model

Model Name :	C7 - Geological resource model
Prepared by :	Mine Geology Group - Stawell
Drillhole Data To:	November 2005
Development Survey date :	N/A No tonnes depleted.
Date Model Completed :	7th November 2005
Date Report Completed :	7th November 2005
Interpretations By :	Jane Murray
Resource Modelling and Grade Estimation By :	Chris De-Vitry
Report Title:	Stawell November 2005 C7 Resource Model
Compiled by :	Chris De-Vitry
Approved By :	Vincent O'Brien Senior Mine Geologist
Audited by :	To be audited by QG
Accepted by :	Andrew McDougall Senior Mining Engineer - Operations
Report Distribution :	Geoff McDermott Vincent O'Brien Stawell site geologists Andrew McDougall

MODELLING ENVIRONMENT		
	Project Directory :	S:\Mining-Geology\North Mine\MineSight\Geology\Current Model
	Model Identification :	C7
	Modelling Software :	Minesight and Isatis
	Operating System :	Windows XP
MODEL DESCRIPTION :	*(narrative, purpose, zones, relation to other models, etc):*	This model covers the area from 2200-2570mN (70° grid) and although the interpretation covers the area above 960 RL this area was not interpreted with the same diligence as the area below 960RL. The area south of 2200 mN and above 960 RL are not areas intended for future mining. Grade and waste material modelled by geological domain (GEOCD). The reported resource has been classified into Indicated and Inferred resources (using wireframes selection).
GEOLOGY/ORE DESCRIPTION :	*(structure, stratigraphy, dip, strikes, size, ore type etc)*	Central lode is a sheared quartz rich volcanogenic with a hanging-wall composed of mine schist. The FW is weak mineralised volcanogenic that is less sheared and has lower quartz content. In the footwall of the weak volcanogenic domain is the Dukes Lode which lies directly against a Basalt contact.
ELEMENTS MODELLED :		Au
PROPERTIES MODELLED :		DENSITY (DEFAULTS)
GRID USED (Local, AMG etc)		AMG

DRILLHOLE DATA :	Directory :	S:\Mining-Geology\North Mine\Minesight\Geology\Current Model_msresources\Drillholes		
Sample data extent				70° Mine Grid
Easting :	658550 mE	to	658900 mE	
Northing :	897760 mN	to	898060 mN	2200 mN – 2570mN
RL:	-870 RL	to	-1080 RL	
Drillhole Spacing:	15m * 15m to 60 * 60m – Staggered pattern			
Raw drillhole data	Project-nth10.dat, nth11.ddh,nth12.ddh. Modal length of 1m. Flagged by GEOCOD using wireframes. Flagging was visually and statistically checked and when required wireframes were modified and the drillholes manually coded.			
Composited drillhole data	1m comps. assay – nth09.ddh (Composites broken at GEOCD)			
STRUCTURAL INTERPRETATION :				
No. of Sections :	37			
Section Interval:	10m (70° rotated grid)			
Sectional Interpretation:	S:\Mining-Geology\North Mine\Minesight\Geology\Current Model_msresources\Geology\Dynamic Strings			
Comments :	Cross-sectional strings for stratigraphy validated in 1m plan slices. Strings are snapped to drillholes.			
WIREFRAMES :		No. of Wireframes :	a total of 6	
	1 fault 2 Waste solids 3 Volc solids			
Construction of the Wireframes :	In Minesight by Jane Murray			
Date Wireframes Completed :	October 2005			
SAMPLE CODING :	Flagged by GEOCD using wireframes. Flagging was visually and statistically checked and when required drillholes manually coded.			
Composited drillhole data	1m comps – nth09.ddh (Composites broken at GEOCD)		2798 samples (Volcanogenic only)	

BLOCK MODEL PARAMETERS :

Model Limits and Parameters :	Origin		Rotation		Range (metres)	Number Of Cells	Cell Dimensions Parent	
Easting : (along strike)	657770		1	347	825	55	15	
Northing : (down dip)	898200		2	0	825	55	15	
RL : (perpendicular to lode)	-1350		3	57	150	75	2	

GRADE INTERPOLATION :

Estimation Technique : Ordinary kriging for Mineralised volcanogenics (including accumulation and true width for Dukes Lode.

Grade Estimation Files : Estimated in Isatis

Default Grades : The Basalt and Mine Schist domains were assigned default grades.

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Slope of Regression	.dxf filename	Inferred Resource	Indicated Resource
Basalt	WGEO=400	W%	WAUD	WSG		bas		
Mine Schist	EGEO=1	E%	EAUD	ESG		ms		
Dukes (Au)	BGEO=200	B%	BAUD	BSG	BSLRD	duk		
Dukes (Accumulation)			BAUBC		BSLRA	duk		
Dukes (True Width)			BAUS		CSLRA	duk		
Dukes (Au from accum)			BACCM					
Central Lode	CGEO=100	C%	CAUD	CSG	CSLRD	cent	None	CCAT=2
Stockworks	VGEO=600	V%	VAUD	VSG	SSLRD	volc		

DOMAIN	Default Au	DRY DENSITY (t/m^3)
Basalt	0.01	2.8
Mine Schist	0.01	2.7
Dukes	N/A	2.85
Central Lode	N/A	2.85
Stockworks	N/A	2.85

Resource at a 4g/t cutoff (Calculated from MineSight detailed reserves)

Domain	Description	Resource Classification	Tonnes	Grade	Ounces
100	Central Lode	Indicated	222,000	6.36	45,430

Grade Wireframes

The Minesight subblocking routine was utilised to generate grade wireframes for 3, 4, and 7g/t cutoffs (see S:\Mining-Geology\North Mine\Minesight\Geology\Current Model_msresources\Block Model)

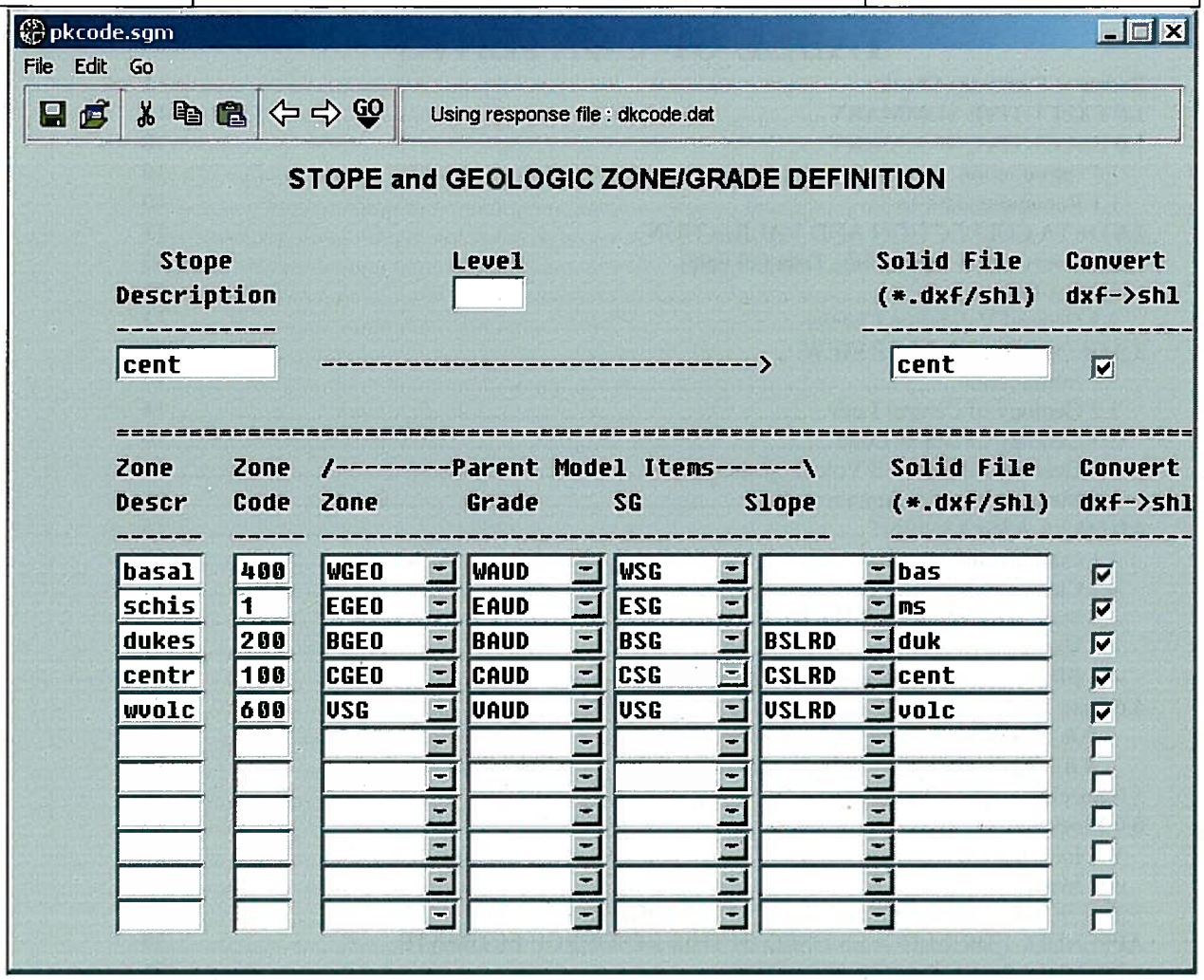
SUMMARY SHEETS HANDED TO ENGINEERS

Location - **S:\Mining-Geology\North Mine\Minesight\Geology\Current Model_msresources\Block Model\September 2005**
Model – nth15.dat
Coded Drillholes – nth11.ddh

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Slope of Regression	.dxf filename	Inferred Resource	Indicated Resource
Basalt	WGEO=400	W%	WAUD	WSG		bas		
Mine Schist	EGEO=1	E%	EAUD	ESG		ms		
Dukes (Au)	BGEO=200	B%	BAUD	BSG	BSLRD	duk		
Dukes (Accumulation)			BAUBC		BSLRA	duk		
Dukes (True Width)			BAUS		CSLRA	duk		
Dukes (Au from accum)			BACCM					
Central Lode	CGEO=100	C%	CAUD	CSG	CSLRD	cent	N/A	CCAT=2
Stockworks	VGEO=600	V%	VAUD	VSG	VSLRD	volc		

All else in the model is mullock

The main Panel in the sub-block routine should look like this


pkcode.sgm
File Edit Go
Using response file : dkcode.dat
STOPE and GEOLOGIC ZONE/GRADE DEFINITION
Stope Description
Level
Solid File (*.dxf/shl)
Convert dxf->shl
cent
cent
Zone Descr
Zone Code
Parent Model Items
Zone
Grade
SG
Slope
Solid File (*.dxf/shl)
Convert dxf->shl
basal 400 WGEO WAUD WSG bas
schis 1 EGEO EAUD ESG ms
dukes 200 BGEO BAUD BSG BSLRD duk
centr 100 CGEO CAUD CSG CSLRD cent
wvolc 600 VSG VAUD VSG VSLRD volc

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

This C7 resource model covers Central and Dukes lodes between sections 2200-2570mN (70° grid) and it represents the unmined down-plunge extensions of these lodes. The model incorporates 14 new holes drilled in 2004 and 19 new holes drilled in 2005. Central Lode is now drilled out to approximately 30*30m and is all classified as an Indicated Resource. Mineralisation exists in Dukes lode and the stockworks between Central Lode and Dukes Lode, however at present this mineralisation is not considered to be of sufficient width and grade to be classified as a resource. As presented in Table 1 numerous people have been involved in this project.

The current resource model contains 222,000 tonnes at 6.4 g/t (Table 2). It is not possible to directly compare the current estimate with the previous resource model as the previous model stopped at 2430mN and the current model extends to 2570mN. When the models are compared over the same area they contain a similar number of total ounces (Table 3).

Table 1. Identification of the project team.

Resource	Role
Justin Cheyne	Drillhole data input
Carol Durant	Database management and validation of drillhole data
Catherine Smith	Logging
Lachlan Macdonald	Logging
Jane Murray	Logging and creation of geological domains.
Chris De Vitry	Preparing final solids, resource model, preparation of this report
Vincent O'Brien	Part of the geological section of this report. Project review

Table 2. Indicated resource above a 4g/t cutoff for Central Lode below the 950RL.

Domain	Description	Resource Classification	Tonnes	Grade	Ounces
100	Central Lode	Indicated	222,000	6.4	45,430

Table 3. Comparison between the previous and current resource models below 950 RL and south of section 2430mN.

Domain	Description	Tonnes	Grade	Ounces
100	Central Lode (current model)	158,600	6.5	32,950
100	Central Lode (previous estimate)	144,300	6.9	31,835

1.1 Recommendations

The C7 resource model is now drilled to Indicated status, however some additional drilling (approximately 5 holes) is required to better define the northern and southern extents of the mineralisation directly above South Fault.

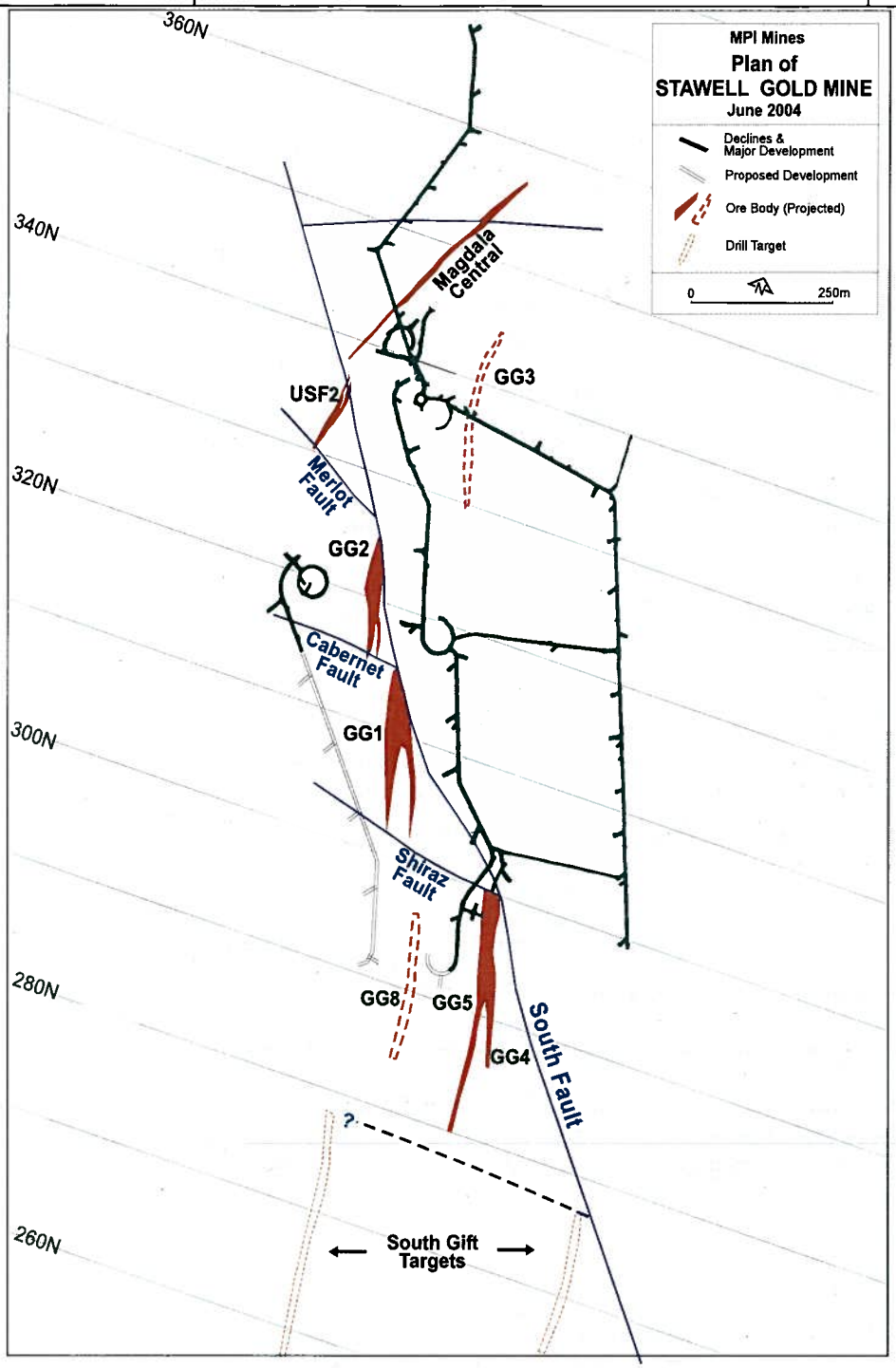


Figure 1. Plan view of the Golden Gift orebodies.

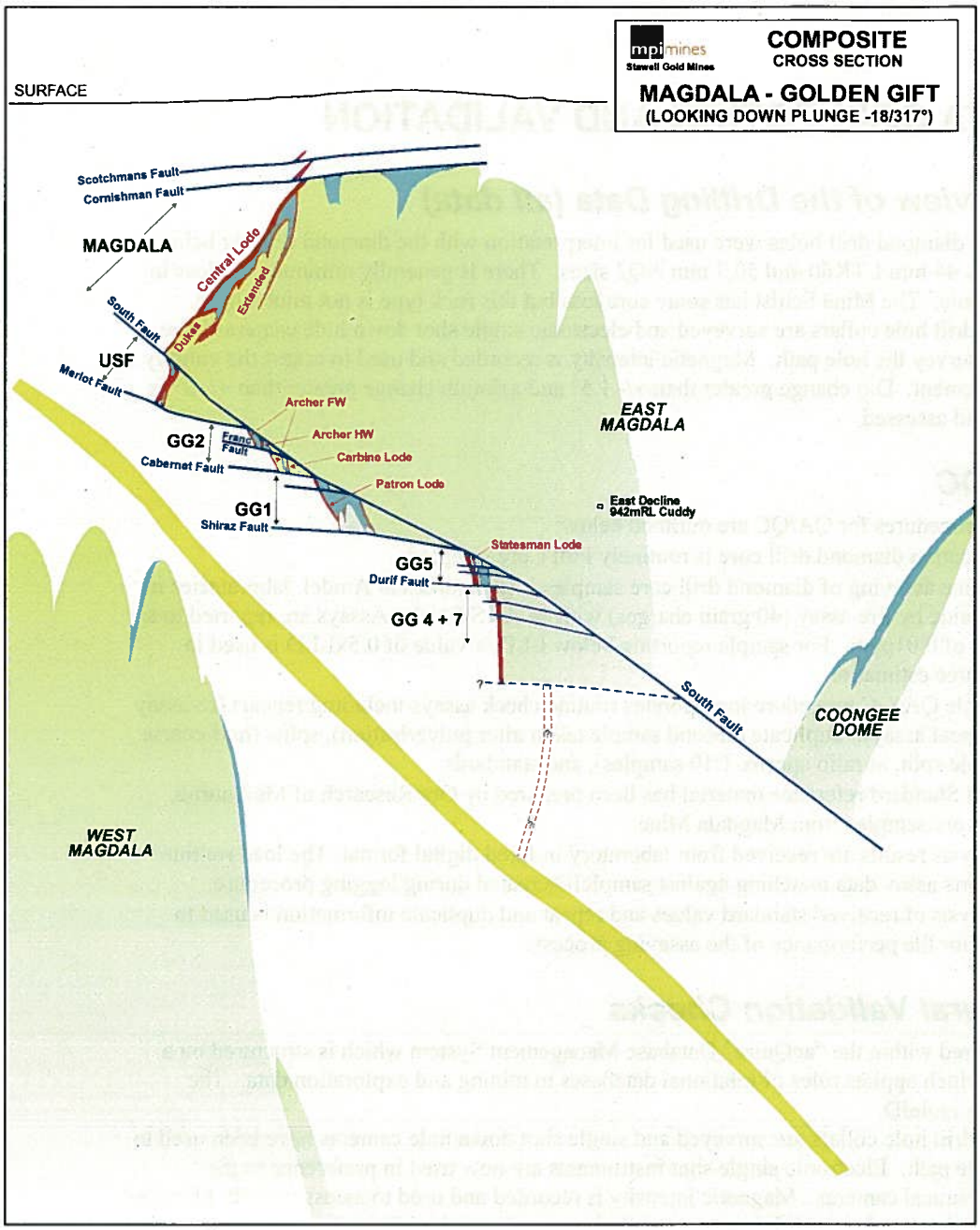


Figure 2. Down plunge cross section of the Magdala and Golden Gift orebodies.

2.0 DATA COLLECTION AND VALIDATION

2.1 Overview of the Drilling Data (all data)

A total of 116 diamond drill holes were used for interpretation with the diamond drilling being predominantly 44 mm LTK60 and 50.7 mm NQ2 sizes. There is generally minimal core loss in the volcanogenic. The Mine Schist has some core loss but this rock type is not mineralised.

Diamond drill hole collars are surveyed and electronic single shot down hole cameras have been used to survey the hole path. Magnetic intensity is recorded and used to assess the validity of the measurement. Dip change greater than +/- 1.5° and azimuth change greater than +/- 3° is highlighted and assessed.

2.2 QA/QC

The general procedures for QA/QC are outlined below:

- Production diamond drill core is routinely Full Core sampled.
- Routine assaying of diamond drill core samples is undertaken at Amdel laboratories in Adelaide by fire assay (40 gram charges) with an AAS finish. Assays are reported to an LLD of 0.01ppm. For sample reporting below LLD, a value of 0.5xLLD is used in resource estimation.
- Sample QA/QC procedure incorporates routine check assays including repeats (re-assay or repeat assay), duplicate (second sample taken after pulverisation), splits (half coarse sample split, at ratio approx 1:10 samples), and standards.
- SGM Standard reference material has been prepared by Ore Research of Melbourne, from ore samples from Magdala Mine.
- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sampleID created during logging procedure.
- Analysis of received standard values and repeat and duplicate information is used to monitor the performance of the assaying process.

2.3 General Validation Checks

All data is stored within the "acQuire" Database Management System which is structured on a Data Model which applies rules of relational databases to mining and exploration data. The primary key is HoleID.

Diamond drill hole collars are surveyed and single shot down hole cameras have been used to survey the hole path. Electronic single shot instruments are now used in preference to the Eastman mechanical cameras. Magnetic intensity is recorded and used to assess validity of record. Dip +/- 1.5° and azimuth +/- 3° (between consecutive samples) is highlighted and assessed.

All logging of diamond drill core is undertaken using "Datcol" software, to a defined logging scheme. It is a barcode system, eliminating errors involved in keying. Logging is both quantitative and qualitative but in both cases only valid codes or values can be recorded. Additional validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth
- Checks are performed for overlapping samples

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (eg Minesight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted. After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

3.0 GEOLOGICAL OVERVIEW

3.1 Introduction

Geological interpretation was undertaken on 10m sections between 2200 and 2570N (70° Grid), and utilised backs mapping, development face sketches, diamond drilling and sludge drilling information. Solids are constructed on screen by producing interpretation strings in cross section, snapping strings to drill holes in 3D space, then linking strings to form wireframe solids. Solids are overlapped, then intersected against each other to form common surfaces. The following primary geological domains have been interpreted. Domain boundaries are based primarily on lithological or structural contacts.

Table 4. Domains interpreted for this resource model

Central Lode
Dukes Lode
Dukes Basalt
Weak Volcanogenics
Mine Schist

3.2 Geology of Central Lode

Central lode shear zone, hanging wall to shear generally well defined, frequently along the Mine Schist/ Volcanogenic contact but may also occur within volcanogenic package. Footwall generally marked by shear surface also, but at times gradational into stockworked volcanogenics.

Variation in Central lode thickness is closely related to the geometry of the underlying basalt noses. In particular, Central lode broadens markedly in the zone immediately above the Dukes basalt nose – This is attributed to the de-focussing/dispersal of strain in the pressure shadow caused when the buttressing effect of the basalt is removed.

Variations in metal distribution are more complex. As well as the primary control exerted by the basalts, high grades are concentrated into shoots, which overall plunge steeply northwards. Controls on shoot boundaries are still as yet poorly understood. Probable candidates include foot and hanging wall stratigraphy and structure, initial whole rock chemistry, local geometry etc.

Central Lode varies in thickness from 0.8 to 20m in width. Mean (and median) true width thickness is 5.6m. Average downhole intercept length is 7.6m,

Diamond drillhole intercepts of Central lode are drilled from both foot and hanging wall platforms. The majority of holes test the full width of the structure. Because of the dip of the orebody holes drilled from the hanging wall are generally close to orthogonal to the structure, while footwall holes tend to be more oblique.

3.3 Geology of Dukes Lode

Dukes Basalt contact lode mineralization comprises dark green chloritic volcanogenics, host to quartz sulphide tension veining. The hanging wall to the Dukes structure is generally defined by a quartz pyrrhotite shear structure, while the footwall is defined by the contact with Dukes Basalt. In places Central lode has sheared out part or all of the Dukes lode and forms the hanging wall. In the North Mine area there are no basalts developed west of Dukes basalt, so the Dukes lode all occurs on an "open flank". The Dukes lode continues above the crest of Dukes Basalt as a shear structure.

3.3 Geology of Footwall Volcanogenics

The footwall volcanogenics domain is the volcanogenic lithologies between the Central and Dukes lodes which are not wireframed into lode structures. This domain contains isolated areas of stockwork mineralization of economic tenor however these areas are generally not large and consistent enough to wireframe.

3.4 Methodology for domain coding

The diamond drillholes were coded based on which of the domains they passed through. All drillholes were visually checked and corrected to guarantee that the correct code related to each interval (Table 5).

Table 5. Domains used for coding.

Description (Codes used during grade interpolation)	Numeric Code	% block coded
Basalt	WGEO=400	W%
Mine Schist	EGEO=1	E%
Dukes	BGEO=200	B%
Central Lode	CGEO=100	C%
Stockworks	VGEO=600	V%

4.0 DATA ASSESSMENT

4.1 Compositing

There was no length or density weighting during the formation of composites and compositing was within the geological domains. All composites are of equal length except for the last (deepest downhole) composite in each domain which is variable in length. Missing Au values (i.e. unassayed core) were set to 0.01 before compositing. If this is not done errors occur in some composite intervals. In addition, there are intervals of non-sampled internal basalt that must be assigned default grades.

The 1m composite files were validated by comparing them to the uncomposited data to insure that all of the data was composited. The composites were also checked to insure that they are of the correct length and that composites are within geological domains. Removing composites less than 0.5m in length had very little affect on the total length and total Au metal (Table 6).

Table 6. Basic statistics showing the affect of compositing samples to 1m.

UNCOMPOSITED								
Dom	# samp	Mean Length	Std Length	Mean Au	Min Au	Max Au	Total Length	Total Metal
100	916	0.90	0.27	3.48	0.01	364	821.2	2853.4
200	364	0.93	0.24	3.08	0.01	33.5	339.2	1046.1
600	1521	0.98	0.46	1.04	0.01	55.2	1488.4	1551.0

1m COMPOSITES												
Dom	# samp	Mean Length	Std Length	Mean Au	Min Au	Max Au	Std Au	Coeff. Var	Total Length	Total Metal	% Length kept	% Metal kept
100	822	0.98	0.07	3.51	0.01	257.1	10.94	3.12	808.4	2822.4	98.4%	98.9%
200	342	0.96	0.11	3.14	0.01	33.5	4.66	1.48	328.8	1023.8	96.9%	97.9%
600	1492	0.99	0.06	1.04	0.01	33.5	2.19	2.11	1477.2	1536.2	99.2%	99.0%

4.2 Cutoff Analysis

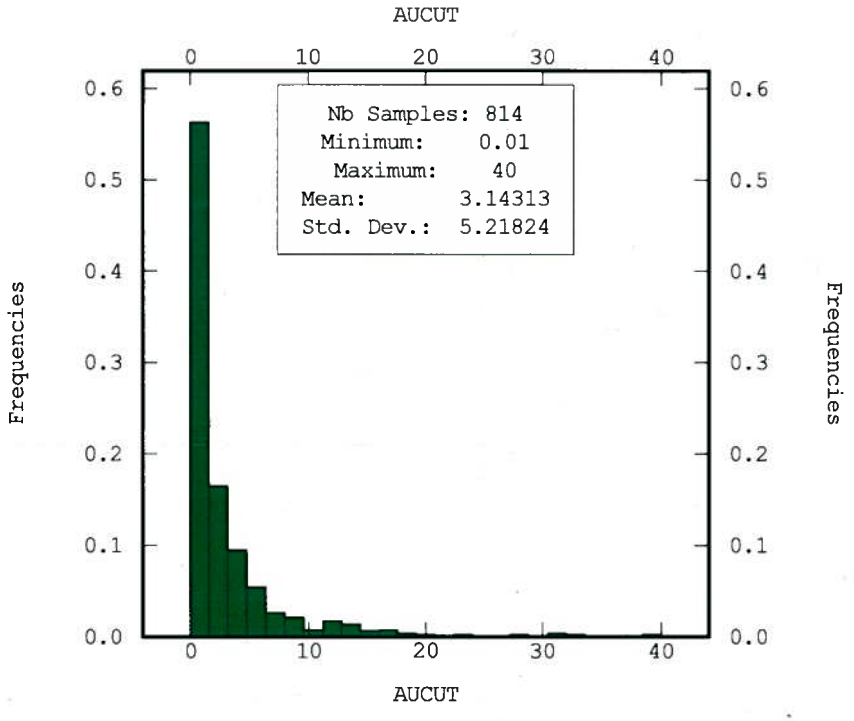
Only Central Lode had top cut applied. Top cutting of the remaining domains was not required. Due to an extreme gold grade of 257g/t in Central lode cutting to 40g/t resulted in 11% of the metal being lost.

Table 7. Statistics for domains after high grade cutting.

Cutting 1m composites							
Dom	Cut Value	# samples cut	Mean Au Cut	Std Au Cut	Coeff. Var Cut	Total Metal	% Metal Kept
100	40	2	3.15	5.23	1.66	2523	89%
200	N/A						
600	N/A						

4.3 Statistical Analysis of Domains

The data used in this section has the Au top cuts and minimum sample lengths discussed in the previous sections. Figure 3 to Figure 5 represent histograms with basic statistics of the volcanogenic domains.



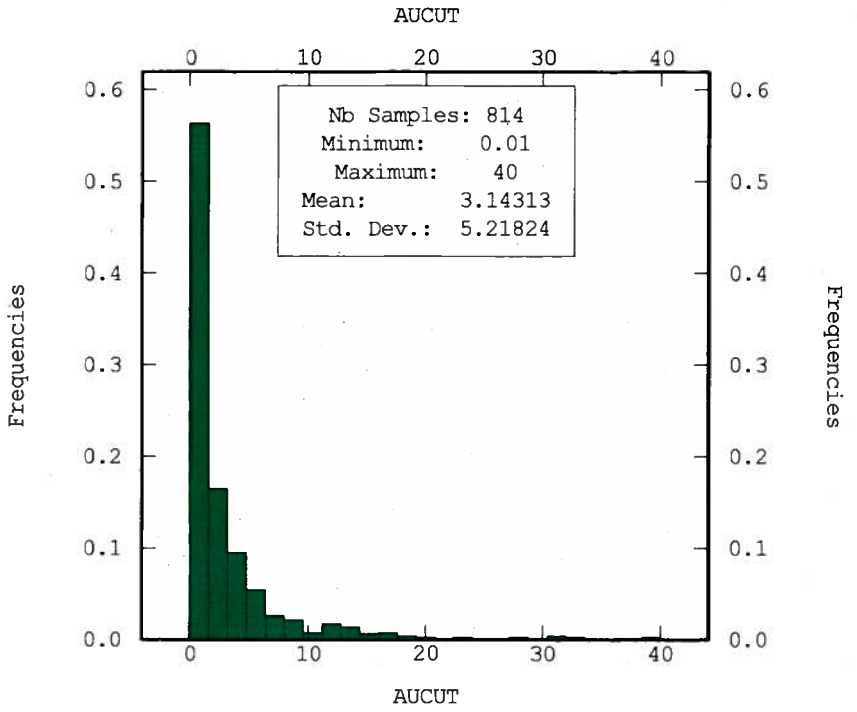


Figure 3. Histogram of gold grades for domain 100.

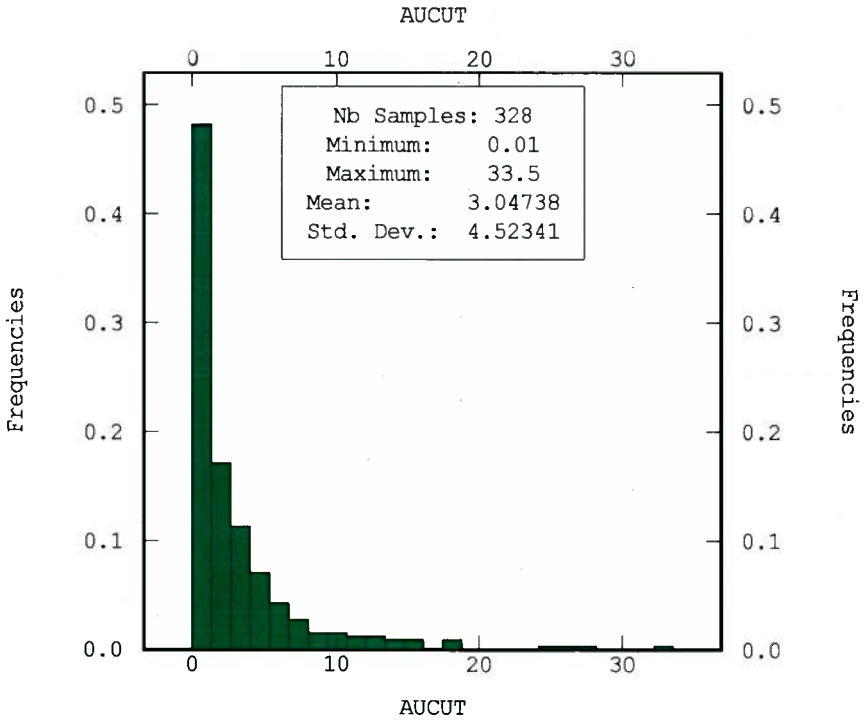


Figure 4. Histogram of gold grades for domain 200.

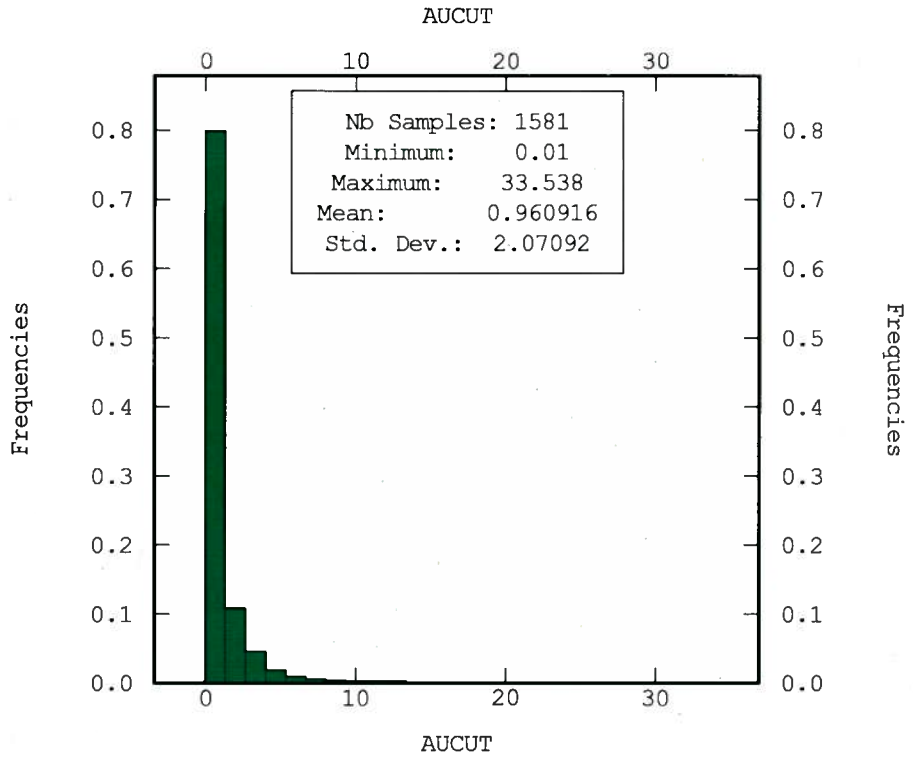


Figure 5. Histogram of gold grades for domain 600.

4.4 Variography

The orientations for variogram fans were generated in the plane of average strike and dip of each domain. Several lag separations, angular tolerances and slicing heights were trialled to produce the best variograms. Gaussian transformation of the Au data improved the variograms significantly and these were utilised. The variograms and the variogram models are presented in Figure 6 to Figure 11 and Table 8. Maximum directions of continuity were interpreted as steeply to moderately north dipping within the plane of strike 167° and dip 57° to the west. These directions of continuity are consistent with the known shoot controls for the Magdala lodes.

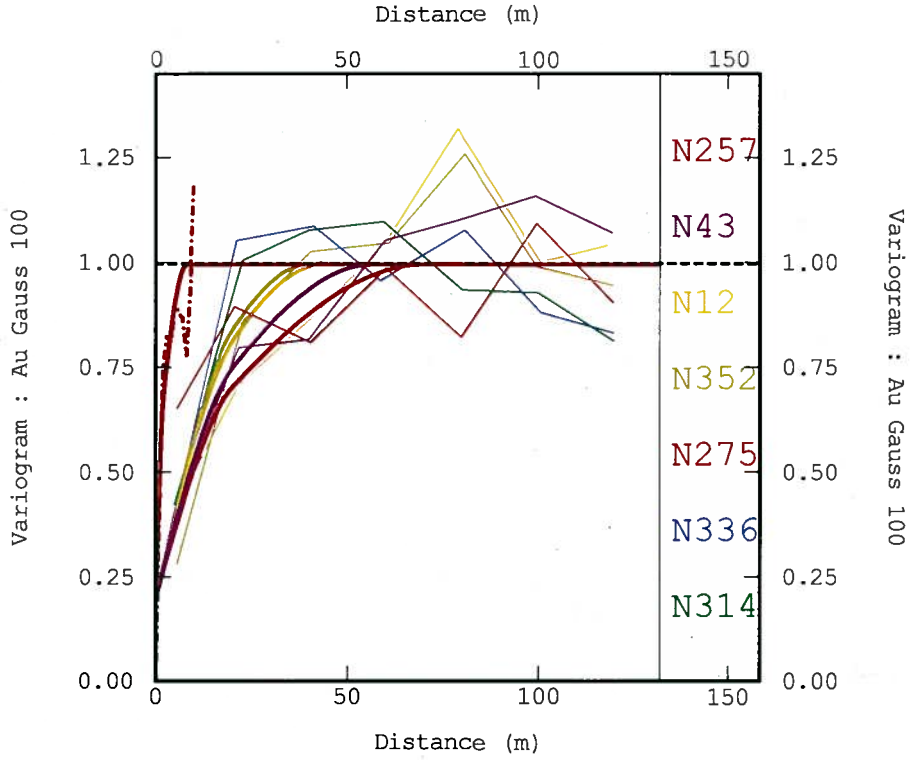


Figure 6. Normal directional variograms of gaussian transformed Au for domain 100. The variogram fan is generated in a plane of strike 167° and dip 57° to the West. The directions N130, N160 etc represent the strike of variograms within the abovementioned plane. The variogram N257 is orthogonal to the abovementioned variogram fan.

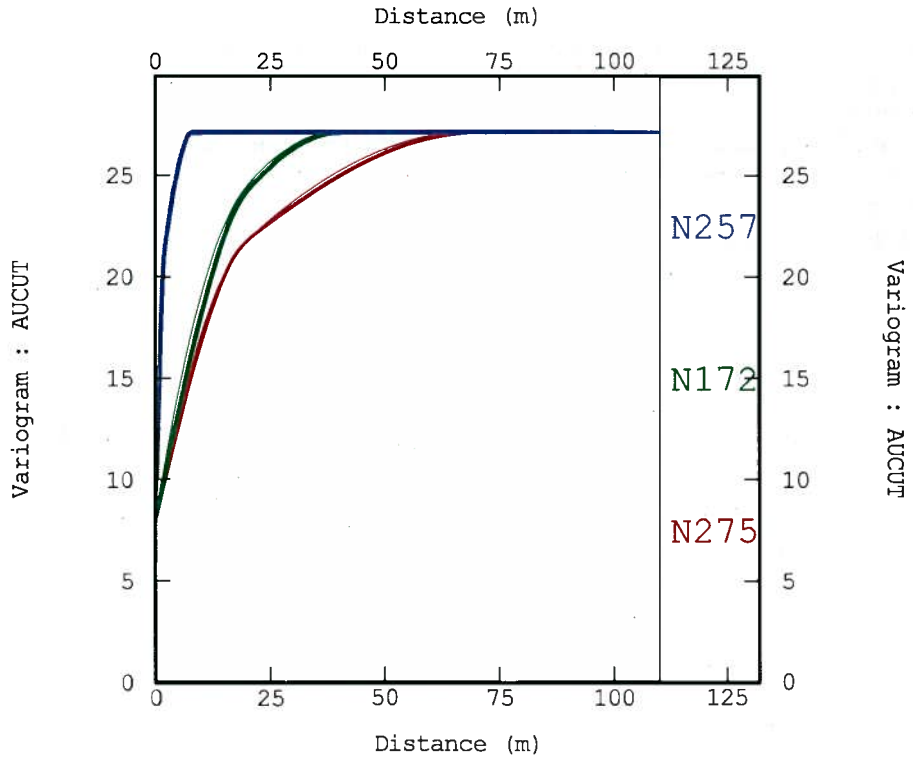


Figure 7. Normal directional variogram models created from the backtransformation of the gaussian variogram model for domain 100. The variogram fan is generated in a plane of strike 167° and dip 57° to the West. The directions N130, N160 etc represent the strike of

variograms within the abovementioned plane. The variogram N257 is orthogonal to the abovementioned variogram fan.

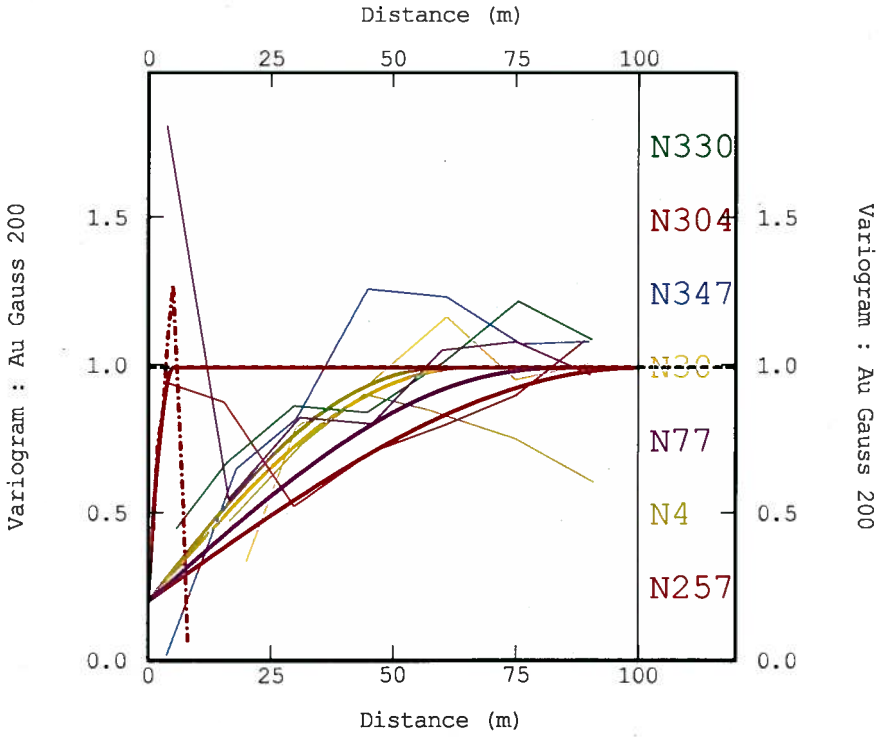
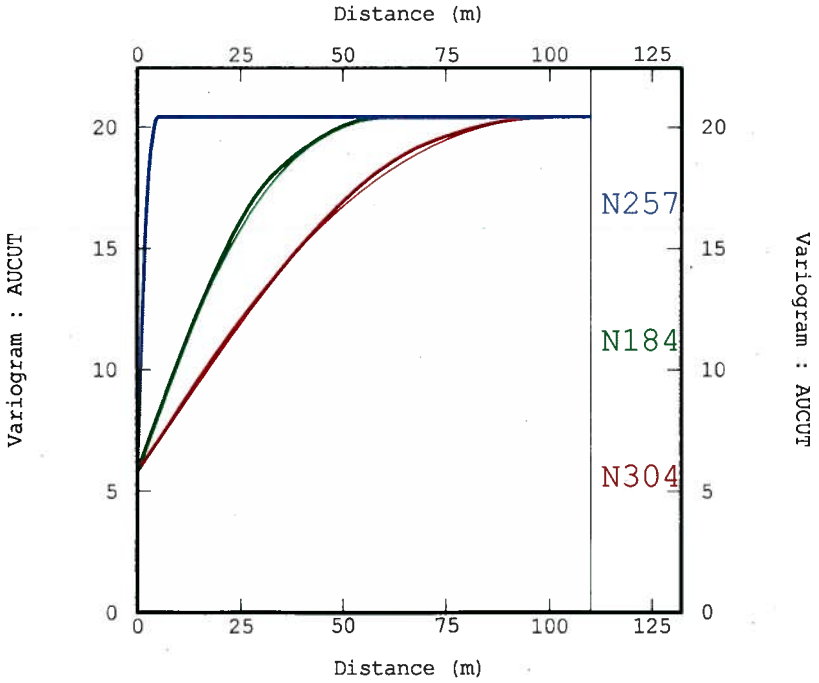


Figure 8. Normal directional variograms of gaussian transformed Au for domain 200. The variogram fan is generated in a plane of strike 167° and dip 57° to the West. The directions N130, N160 etc represent the strike of variograms within the abovementioned plane. The variogram N257 is orthogonal to the abovementioned variogram fan.



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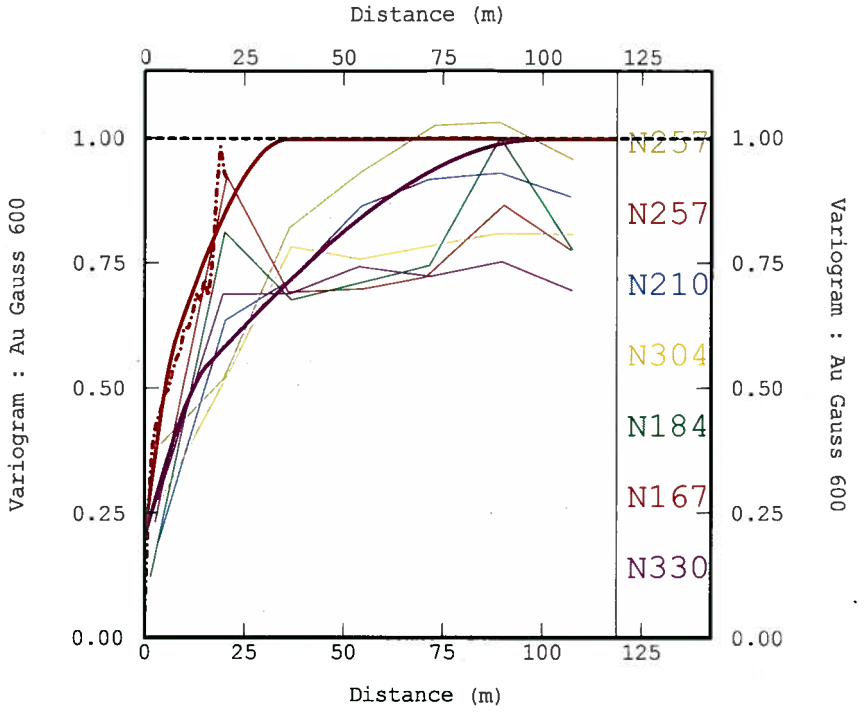

Figure 10. Normal directional variograms of gaussian transformed Au for domain 600. The variogram fan is generated in a plane of strike 167° and dip 57° to the West. The directions N130, N160 etc represent the strike of variograms within the abovementioned plane. The variogram N257 is orthogonal to the abovementioned variogram fan.

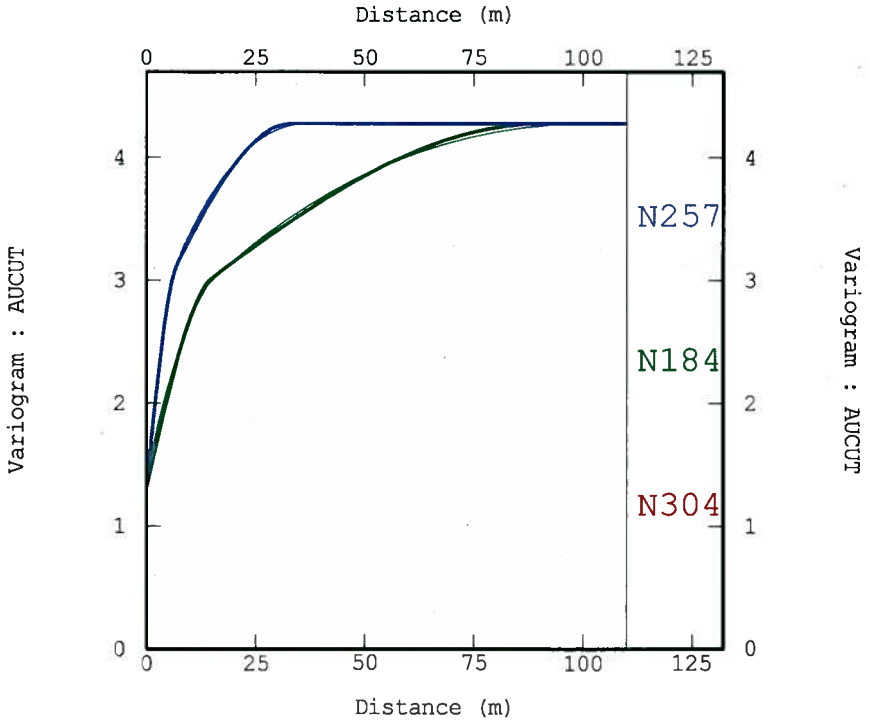

Figure 11. Normal directional variogram models created from the backtransformation of the gaussian variogram model for domain 600. The variogram fan is generated in a plane of strike 167° and dip 57° to the West. The directions N130, N160 etc represent the strike of variograms within the abovementioned plane. The variogram N257 is orthogonal to the abovementioned variogram fan.

Table 8. Variogram model parameters utilised for interpolation of Au.

Domain	Rotation (Geologist)	Nugget	Sill 1	Range Horizontal Along Domain	Range Down Dip	Range Across Strike	Sill 2	Range Horizontal Along Domain	Range Down Dip	Range Across Strike
100	167,57,100	8	10	20	20	2	9.1	70	40	8
200	167,57,120	5.8	5.5	70	34	3	9.1	100	60	5
600	167,57,0	1.3	1.3	15	15	7	1.67	90	90	33

4.5 Block size selection and search ellipse optimisation

The block size selected for this resource estimate was increased from 7.5*7.5*2.5m (minimum width is across strike) to 15*15*2m. This new block size is approximately equal to about half the average drillhole spacing. The search parameters have been optimised utilising MineSight and selecting several representative blocks (Table 9 and Table 10). The parameters for the weak volcanogenic domain have not been optimised.

The MineSight anisotropic distance option was utilised and there is no minimum distance criteria limiting block interpolation. Sample searching is in quadrants rotated to the average strike and dip of the orebody.

Table 9. Parameters utilised to optimise search ellipse parameters.

Dom	Search ellipse size	Search ellipse orientation (Minsight GSLIB)	MinSight Block ref. L,R,C	Block Mean Au	# -gtive weigths	Slope	Min # samples	# Samples per quad	Total # samples
100	140,80,16	275,-55,14	24,23,27 Well informed	7.40	5	0.91	**10**	**10**	**40**
100	140,80,16	275,-55,14	24,23,27 Well informed	7.46	15	0.93	10	15	60
100	140,80,16	275,-55,14	24,23,27 Well informed	6.80	2	0.90	10	8	32
100	140,80,16	275,-55,14	25,29,20 Poor informed	1.91	0	0.28	10	8	32
100	140,80,16	275,-55,14	25,29,20 Poor informed	1.86	0	0.33	10	15	60
100	140,80,16	275,-55,14	25,29,20 Poor informed	2.35	0	0.29	**10**	**10**	**40**
200	200,120,20	314,-47,14	28,19,23 Well informed	3.01	9	0.98	10	8	32
200	200,120,20	314,-47,14	28,19,23 Well informed	3.00	13	0.98	**10**	**10**	**40**
200	200,120,20	314,-47,14	28,19,23 Well informed	3.07	22	0.98	10	15	60
200	200,120,20	314,-47,14	28,19,23 Well informed	2.97	4	0.98	10	6	24
200	200,120,20	314,-47,14	36,33,11 Poor informed	8.22	0	0.60	10	6	24
200	200,120,20	314,-47,14	36,33,11 Poor informed	7.77	0	0.62	10	8	32
200	200,120,20	314,-47,14	36,33,11 Poor informed	7.49	0	0.65	**10**	**10**	**40**
200	200,120,20	314,-47,14	36,33,11 Poor informed	7.17	2	0.68	10	15	60

Table 10. Search parameters utilised for grade interpolation.

Domain	Rotation (GSLIB))	**Search**	**Min no samples**	**Maximum no. samples**	**Max samples per quadrant**
100	275,-55,14	252*144*30	10	40	10
200	314,-47,14	300*180*50	10	40	10
600	257,-57,0	180*180*66	10	40	10

4.6 Data Assessment for the Dukes Lode Accumulation

4.6.1 Introduction and basic statistics

Due to the average width of 2m for Dukes lode, estimating total width and the product of total width and Au (accumulation) is preferred to the traditional kriging of Au directly into blocks. At Stawell the accumulation and width are estimated directly into the standard block model rather then a girded seam model or a 2D model.

The MinSight composite and drillhole files for this methodology are nth09.acc, nth11.acc and nth12.acc. The composites have their full true width estimated by the CMPTRU.dat program which utilises the strike and dip of each drillhole and the average dip of the Dukes lode. This approximation was checked visually on screen and found to be acceptable.

Figure 12 and Figure 13 are histograms of accumulation and true width. The histogram of true width has a break at a width of about 3m. Widths of greater than 3m are not typical of Dukes Lode it is possible that these greater widths represent a combination of both Dukes lode and stockwork mineralisation. More domaining may be required in these areas. For the estimation of accumulation and true width, Au and true width should not be correlated. As represented in Figure 14 these variables are not correlated.

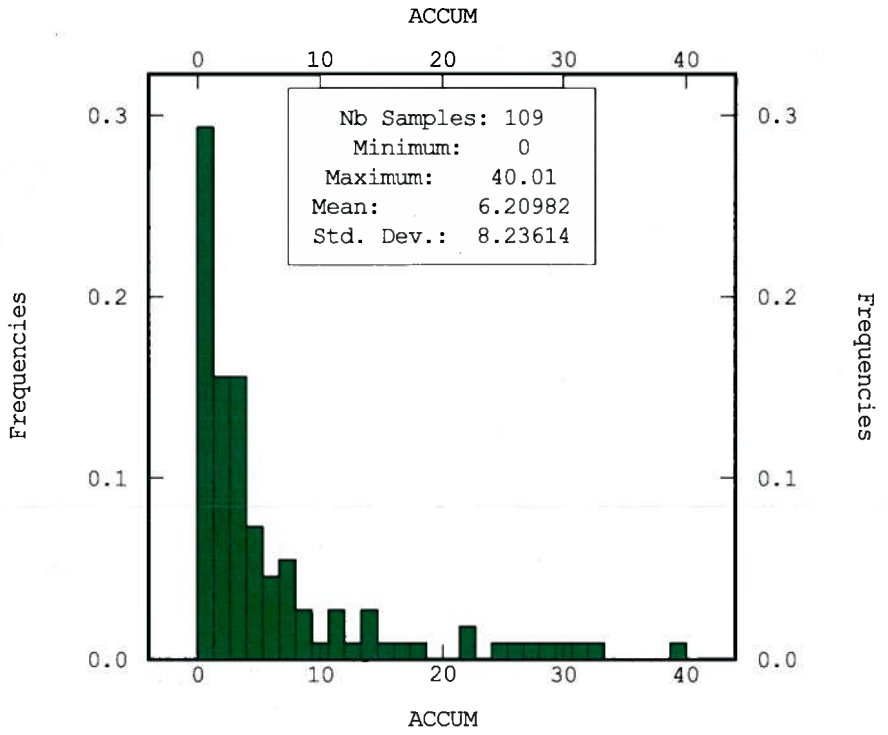

Figure 12. Histogram of accumulation for the Dukes domain.

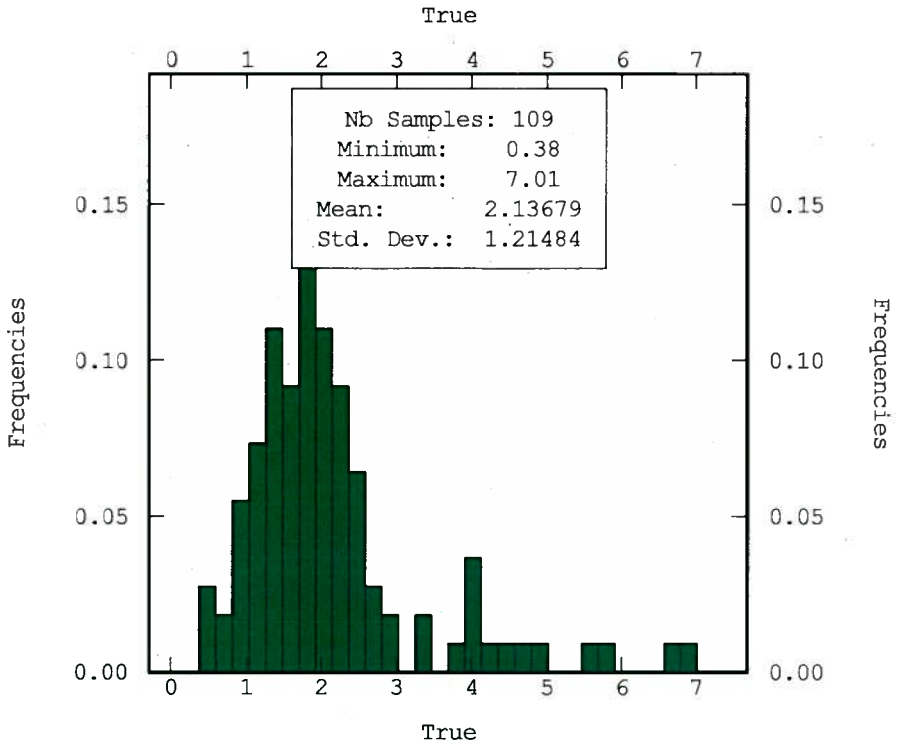

Figure 13. Histogram of accumulation for the Dukes domain.

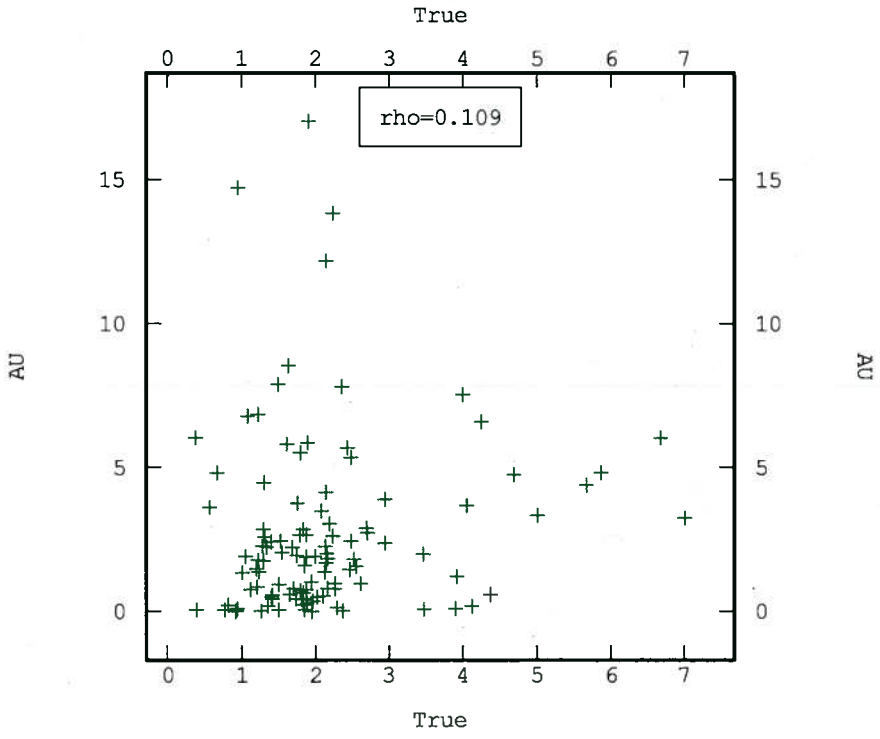

Figure 14. Scatterplot of Au versus true width.

4.6.2 Variography

Figure 15 and Figure 16 represent variograms and variogram models of accumulation and true width. The maximum direction of continuity for the accumulation remains the same as Au i.e. 30° to the north (in the plane of the Dukes Lode). The true width did not have an obvious continuity direction and it was modelled as anisotropic within the plane of Dukes Lode. Because at Stawell the accumulation and true width are modelled in 3D not 2D a variogram perpendicular to the Dukes Lode must be assumed. The minimum direction of continuity within the plane of Dukes Lode will be used to represent the continuity direction perpendicular to Dukes Lode. It would have been preferable to model accumulation and true width in 2D but insufficient time was available to utilise this approach. Table 11 contains the variogram model parameters utilised for interpolation.

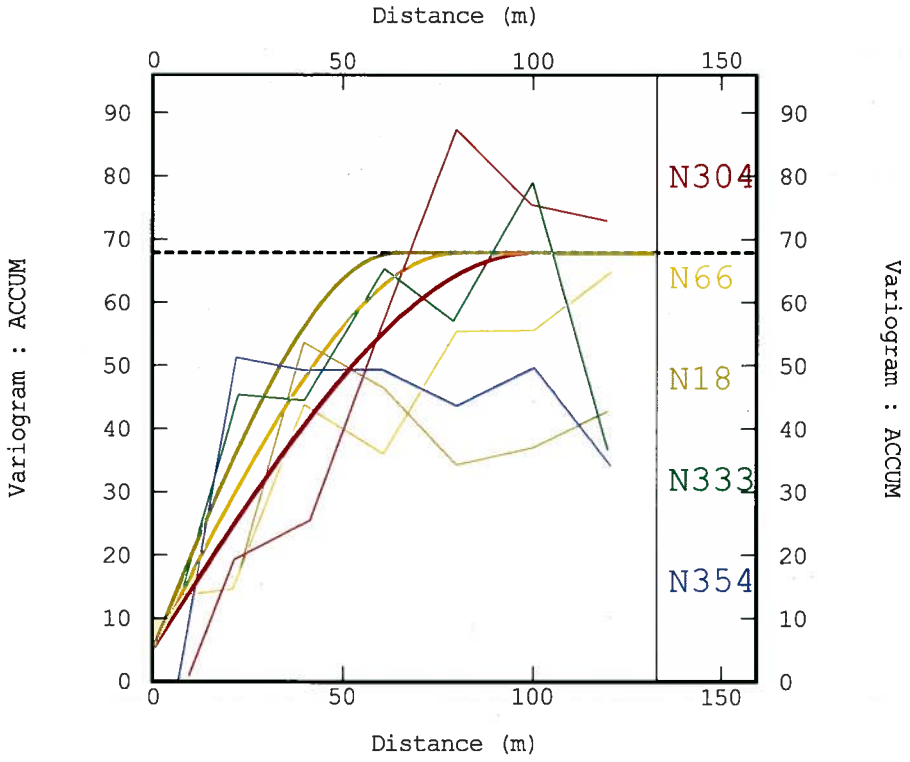

Figure 15. Normal directional variograms of accumulation for the Dukes Lode. The variogram fan is generated in a plane of strike 167° and dip 57° to the West. The directions N130, N160 etc represent the strike of variograms within the abovementioned plane.

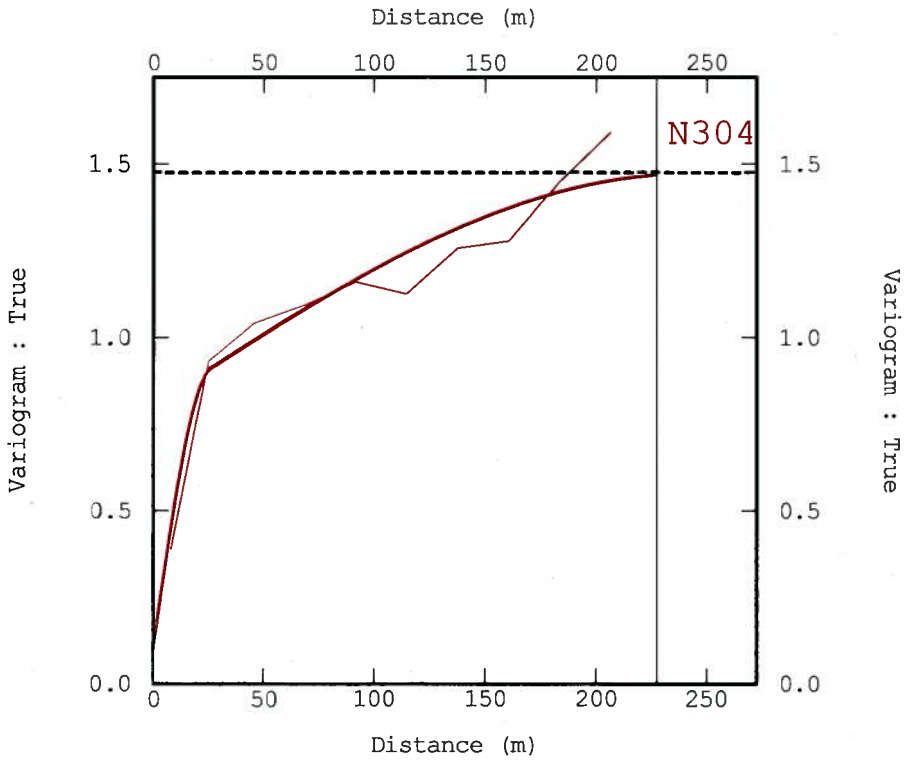

Figure 16. Normal directional variograms of true width for the Dukes Lode. The variogram fan is generated in a plane of strike 167° and dip 57° to the West. The directions N130, N160 etc represent the strike of variograms within the abovementioned plane.

Table 11. Variogram model parameters utilised for interpolation of accumulation and true width.

Variable	Rotation (Geologist)	Nugget	Sill 1	Range Horizontal Along Domain	Range Down Dip	Range Across Strike	Sill 2	Range Horizontal Along Domain	Range Down Dip	Range Across Strike
Accum	167,57,120	5	42.8	100	63	63	20	100	63	63
T Width	167,57,0	0.1	0.7	26	26	26	0.67	240	240	240

5.0 Specific Gravity and Default Au

The default densities and default gold grades are those traditionally utilised for the North Mine models (Table 12).

Table 12. Default densities and default Au utilised in the resource estimate.

DOMAIN	Default Au	DRY DENSITY (t/m^3)
Basalt	0.01	2.8
Mine Schist	0.01	2.7
Dukes	N/A	2.85
Central Lode	N/A	2.85
Stockworks	N/A	2.85

6.0 Block Model

6.1 Introduction

Table 13 contains the parameters utilised to create the rotated Minsight block model. The block size and model rotation were checked before moving onto domain coding. Domain wireframes were utilised to code the percentage of each block within each domain (Table 14). This was validated on screen to insure correct coding. A single pass ordinary kriging is used to interpolate Au for all the mineralised volcanogenic domains. All domain boundaries of the domains listed in Table 14 are treated as hard. In the Duke Lode the interpolation for accumulation and true width utilise the same parameters as those for Au. Insufficient time was available to re-optimise the kriging neighbourhood parameters.

Table 13. Block model parameters.

	Origin	Rotation	Range (m)	Number of Cells	Cell Dimensions
Easting	657770	1 is 347	825	55	15
Northing	898200	2 is 0	825	55	15
RL	-1350	3 is 57	150	75	2

Table 14. Block model domain coding.

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Slope of Regression	.dxf filename	Inferred Resource	Indicated Resource
Basalt	WGEO=400	W%	WAUD	WSG		bas		
Mine Schist	EGEO=1	E%	EAUD	ESG		ms		
Dukes (Au)	BGEO=200	B%	BAUD	BSG	BSLRD	duk		
Dukes (Accumulation)			BAUBC		BSLRA	duk		
Dukes (True Width)			BAUS		CSLRA	duk		
Dukes (Au from accum)			BACCM					
Central Lode	CGEO=100	C%	CAUD	CSG	CSLRD	cent	N/A	CCAT=2
Stockworks	VGEO=600	V%	VAUD	VSG	VSLRD	volc		

6.2 Model Validation

The model was validated by visually comparing the diamond drilling sample grades for each element to block grades on sections and in 3D. The block size is approximately ½ the size of the drilling grid and parent cell estimation is used. This factor means that individual blocks are not similar to individual holes rather a block grade is most affected by the nearest drillholes. Despite this the comparison between composites and the blocks was considered appropriate for all domains. Table 15 is a comparison of the kriged Au to the declustered Au from 1m composites. Four declustered estimates of Au have been obtained for each domain by changing the declustering block size. The ordinary kriging estimates are acceptably close to the declustered statistics.

Table 15. Comparison of block model Au to declustered Au (1m composites).

				Decluster block size (X,Y,Z)			
Domain	1m comps Clustered	OK Mean	No Blocks	1000*15*15	1000*25*25	1000*30*30	1000*45*45
100	3.15	2.57	2568	2.87	2.89	2.75	2.56
200	3.14	3.08	1,997	3.03	2.98	3.04	2.95
600	1.04	0.93	4,717	1.28	1.16	1.22	1.09

7.0 RESOURCE CLASSIFICATION AND REPORTING

Table 16 contains the Central lode Indicated resource calculated from the Minesight detailed reserves routine. The model blocks above a 4 gt cutoff are generally drilled out to 30*30m or better. A small number of these blocks are isolated or are considered to be the result of over-smoothing. These blocks are not considered as indicated resources. Following economic analysis by AMC small areas of stockwork and Dukes Lode mineralisation may be included in the Indicated resource (at these stage these resources may not be economic). If this is the case this report will be modified to include these resources.

Table 16. Resource at a 4g/t cutoff .

Domain	Description	Resource Classification	Tonnes	Grade	Ounces
100	Central Lode	Indicated	222,000	6.36	45,430

APPENDIX 1 DRILLHOLES USED IN THIS RESOURCE ESTIMATE

Note these statistics are from accumulations over full width of the domains 100, 200 AND 600.

BHID	TO	LENGTH	AU	GEOCODE
MD2393	275.45	15.45	0.08	600
MD2397	260	1.95	0.69	100
MD2397	280	20	0.1	600
MD2469	276.25	1.25	0.02	600
MD2469	277.3	1.05	0.51	100
MD2469	295.1	17.8	0.08	600
MD2469W1	290	20	0.01	600
MD2469W1	302.15	12.15	0.26	600
MD2498	304.6	9	3.19	100
MD2498	323.1	18.5	0.45	600
MD2508	46	2	5.11	100
MD2508	64	18	2.19	600
MD2509	43.1	5.3	0.65	100
MD2689	172	8.9	0.84	100
MD2689	186	14	0.22	600
MD2689	188.75	2.75	0.76	200
MD2728	130.9	13	1.25	100
MD2728	141.2	10.3	0.64	600
MD2728	147.4	6.2	4.79	200
MD2729	119.3	2.9	6.24	100
MD2729	124.25	4.95	0.06	600
MD2729	126.7	2.45	5.65	200
MD2730A	138.35	8.45	3.21	100
MD2730A	147.1	8.75	0.47	600
MD2730A	152.2	5.1	6.55	200
MD2731	128.6	3.75	3.41	100
MD2731	134	5.4	1.37	600
MD2731	137.25	3.25	2.36	200
MD2733	180.5	7.85	0.3	100
MD2739	157.8	8.4	2.8	100
MD2739	170	12.2	0.32	600
MD2739	172	2	0.06	200
MD2743	135.05	8.85	1.04	100
MD2743	136.9	1.85	1.94	200
MD2744	120.3	4.05	0.55	100
MD2744	123	2.7	0.25	600
MD2744	125.35	2.35	3.03	200
MD2745	140.1	5.55	4.54	100
MD2745	157	16.9	0.11	600
MD2745	159.25	2.25	0.44	200
MD2746	174.9	8.6	2.14	100
MD2746	194.9	20	0.9	600
MD2746	198	3.1	2.77	600
MD2746	204	6	3.66	200
MD2770	158.1	10.2	2.3	100
MD2770	166.3	8.2	0.24	600
MD2770	168.95	2.65	1.37	200

MD2778	147	5.9	2.63	100
MD2778	167	20	0.19	600
MD2778	176.95	9.95	0.56	600
MD2780	125.2	3.95	1.37	100
MD2780	131.15	5.95	0.84	600
MD2780	133.5	2.35	12.16	200
MD2781	138	5.5	1.27	100
MD2781	150.45	12.45	0.63	600
MD2781	152.8	2.35	1.03	200
MD2782	144.7	5.7	1.19	100
MD2782	158.5	13.8	0.16	600
MD2782	159.7	1.2	14.7	200
MD2783	127.4	8.4	1.37	100
MD2783	128	0.6	3.59	200
MD2794	134.7	1.3	0.37	100
MD2794	153.2	18.5	0.32	600
MD2794	155.05	1.85	0.44	200
MD2795	126	6.5	1.21	100
MD2795	130	4	9.49	600
MD2795	132.3	2.3	0.13	200
MD2796	137.15	6.55	0.77	100
MD2796	139	1.85	0.54	600
MD2796	140.95	1.95	0.79	200
MD2797	125	6.95	0.35	100
MD2797	145	20	0.17	600
MD2797	165	20	0.79	600
MD2797	171	6	0.12	600
MD2807	168.1	4.9	0.27	100
MD2807	180	11.9	0.66	600
MD2807	182.7	2.7	0.5	200
MD2808	185.7	15.1	2.4	100
MD2808	199	13.3	0.28	600
MD2808	201.05	2.05	7.84	200
MD2809	136.8	2.8	6.46	100
MD2809	144	7.2	0.68	600
MD2809	146.2	2.2	3.47	200
MD2810	162	8.6	3.44	100
MD2810	176.6	14.6	1	600
MD2810	178.8	2.2	0.72	200
MD2811	164	9.4	1.05	100
MD2811	178	14	0.24	600
MD2811	179	1	0.06	200
MD2812	116.15	6.15	0.56	100
MD2812	117.2	1.05	1.39	600
MD2812	119	1.8	5.49	200
MD2813	127.4	7	1.25	100
MD2813	134.65	7.25	0.09	600
MD2813	136.35	1.7	0.6	200
MD2814	127.3	8.3	2.98	100
MD2814	129.8	2.5	1.45	200
MD2924	122.3	3.15	1.59	100
MD2924	126.55	4.25	0.37	600
MD2924	129.4	2.85	2.71	200

MD2925A	151.75	8.15	3.61	100
MD2925A	163	11.25	0.47	600
MD2925A	167.2	4.2	1.98	200
MD2926	141	12	0.96	100
MD2926	146	5	1.46	600
MD2926	147.95	1.95	0.65	200
MD2927	156.9	8.95	2.51	100
MD2927	171.3	14.4	0.22	600
MD2927	171.8	0.5	0.06	200
MD2928	127.3	7.9	1.2	100
MD2928	146.5	19.2	0.69	600
MD2928	150	3.5	0.08	200
MD2929	148.1	5.8	71.61	100
MD2929	155.85	7.75	0.79	600
MD2929	156.8	0.95	0.22	200
MD2930	199	6.9	5.12	100
MD2930	209.6	10.6	1.52	600
MD2930	211.65	2.05	2.26	200
MD2954	139	10	0.65	100
MD2954	159	20	0.1	600
MD2954	165.45	6.45	0.03	600
MD2955	190.15	11.8	3.82	100
MD2955	194.7	4.55	1.1	600
MD2955	196.5	1.8	1.36	200
MD2956	128.5	3.4	0.75	100
MD2956	132.05	3.55	0.29	600
MD2956	134.25	2.2	2.24	200
MD2957	148	3.4	4.08	100
MD2957	155.1	7.1	1.23	600
MD2957	157.6	2.5	2.6	200
MD2958	166	5	1.47	100
MD2958	186	20	0.96	600
MD2958	189	3	0.75	600
MD2958	191.45	2.45	0.54	200
MD2959A	210.4	5.4	1.64	100
MD2959A	217.3	6.9	2	600
MD2959A	220.35	3.05	1.68	200
MD2982	136.4	11.2	0.71	600
MD3049A	187	8.65	6.23	100
MD3049A	199.3	12.3	0.9	600
MD3049A	201.05	1.75	1.77	200
MD3059	186	9.25	2.12	100
MD3059	193.1	7.1	1.91	600
MD3059	195.2	2.1	0.94	200
MD3060A	142.15	4.75	0.39	100
MD3060A	162.15	20	0.75	600
MD3060A	182.15	20	0.1	600
MD3060A	184	1.85	0.31	600
MD3060A	186.1	2.1	0.3	200
MD3061	212.5	9.45	6.17	100
MD3061	218.6	6.1	0.02	600
MD3061	221.45	2.85	2.64	200
MD3067	279.8	7.8	6.58	100

MD3067	292.65	12.85	2.73	600
MD3067	297.5	4.85	1.21	200
MD3091	267.3	1.3	0.02	200
MD3092	328.3	4.45	7.21	100
MD3092	348.15	19.85	2.24	600
MD3092	357.25	9.1	3.22	200
MD3092W1	317	4.65	0.37	100
MD3092W1	324.25	7.25	2.32	600
MD3092W1	332	7.75	5.99	200
MD3093	384.1	1.75	0.19	200
MD3112	258.85	6.15	2.65	100
MD3112	267	8.15	0.38	600
MD3112	269.95	2.95	0.96	200
MD3113	264.4	5.05	2.37	100
MD3113	269.3	4.9	1.59	600
MD3113	272.25	2.95	1.56	200
MD3226	211.6	3.95	0.93	100
MD3226	213.65	2.05	2.18	600
MD3226	216.25	2.6	4.11	200
MD3235	188.35	2.1	0.35	100
MD3235	200.8	12.45	0.92	600
MD3235	202.7	1.9	2.22	200
MD3237	170.2	7.36	0.78	100
MD3237	177.8	7.6	0.76	600
MD3237	180.35	2.55	5.32	200
MD3242	220	8.6	6.06	100
MD3242	224	4	4.93	600
MD3242	228.05	4.05	3.87	200
MD3243	253.35	12.35	4.76	100
MD3243	262.15	8.8	1.42	600
MD3243	265	2.85	17.01	200
MD3258	224.55	2.25	0.82	100
MD3258	240.95	14.55	0.45	600
MD3258	248.7	7.75	4.36	200
MD3281	147.25	19.2	0.71	100
MD3281	167.25	20	0.18	600
MD3281	183	15.75	0.01	600
MD3283	146.4	2.85	12.23	100
MD3283	166.4	20	1.55	600
MD3283	174.9	8.5	2.44	600
MD3302	170.5	6.85	0.43	100
MD3343	136.4	6.7	4.21	100
MD3343	154.35	17.95	0.56	600
MD3343	156.3	1.95	0.23	200
MD3407	172	5.6	1.5	100
MD3407	192	20	0.13	600
MD3407	205.5	13.5	0.1	600
MD3418	124.1	3.3	0.52	100
MD3418	131.2	7.1	1.43	600
MD3418	132.2	1	0.01	200
MD3420	160.5	8.8	3.37	100
MD3420	174.6	14.1	1.43	600
MD3420	177.85	3.25	7.77	200

MD3439	65	2.9	0.4	100
MD3442	119.5	1.5	0.5	100
MD3442	126.75	7.25	0.6	600
MD3442	128.4	1.65	8.49	200
MD3443	128.9	7.2	0.53	100
MD3443	132.75	3.85	2.51	600
MD3443	135.3	2.55	1.8	200
MD3444	130.8	6.6	1.81	100
MD3444	134.6	3.8	0.31	600
MD3444	136.6	2	0.36	200
MD3445	137.7	5.1	1.65	100
MD3445	141.5	3.8	0.1	600
MD3445	144.2	2.7	2.43	200
MD3446	140	1.95	1.68	100
MD3446	160	20	0.47	600
MD3446	168.7	8.7	0.45	600
MD3703	65.2	2.4	4.78	200
MD3703	84.4	19.2	1.46	600
MD3703	102.6	18.2	1.34	100
MD3705	66	6.9	1.58	100
MD3706	94.4	2.8	6	200
MD3706	108.9	14.5	2.59	600
MD3707	46.2	2.9	4.45	200
MD3707	66.2	20	1.04	600
MD3707	69.1	2.9	0.98	600
MD3707	77	7.9	2.34	100
MD4024	69.5	2.2	0.43	200
MD4024	80	10.5	0.06	600
MD4024	87.95	7.95	8.78	100
MD4025	53.4	1.35	0.85	200
MD4025	59.9	6.5	0.69	600
MD4025	69.3	9.4	3.04	100
MD4026	45.6	2	2.84	200
MD4026	52.8	7.2	1.37	600
MD4026	54.7	1.9	0.8	100
MD4027	51	1.6	2.04	200
MD4027	55.35	4.35	4.37	600
MD4027	62.1	6.75	0.27	100
MD4028	45.4	1.9	1.88	200
MD4028	52.1	6.7	0.42	600
MD4028	56.35	4.25	2.97	100
MD4029	44.15	2.1	1.59	200
MD4029	55.5	11.35	2.1	600
MD4029	60.9	5.4	1.74	100
MD4030	78.6	1.9	1.9	200
MD4030	90.3	11.7	2.13	600
MD4030	102.6	12.3	2.24	100
MD4031	55.35	2.65	1.9	200
MD4031	66.25	10.9	0.75	600
MD4031	75.6	9.35	2.73	100
MD4032	48.25	1.3	0.76	200
MD4032	59.6	11.35	0.33	600
MD4032	68.1	8.5	3.88	100

MD4033	36.3	1.85	0.42	200
MD4033	50.3	14	0.62	600
MD4033	57.3	7	4.96	100
MD4034	70.1	2.15	2.57	200
MD4034	78.35	8.25	1.38	600
MD4034	92.5	14.15	2.59	100
MD4035	68.3	3.6	0.97	200
MD4035	77.1	8.8	2.23	600
MD4035	87.1	10	1.61	100
MD4036	58.3	1.8	1.75	200
MD4036	67.8	9.5	0.5	600
MD4036	78.65	10.85	6.5	100
MD4042	45	4.2	0.1	200
MD4042	52.3	7.3	0.31	600
MD4042	57.6	5.3	1.03	100
MD4043	37	1.45	0.57	200
MD4043	50.6	13.6	0.32	600
MD4043	58.5	7.9	5.18	100
MD4044	39.6	1.95	0.06	200
MD4044	51.45	11.85	0.21	600
MD4044	56.55	5.1	5.53	100
MD4045	38.3	2.4	0.02	200
MD4045	47	8.7	0.27	600
MD4045	54.5	7.5	5.05	100
MD4046	45.8	4.4	0.19	200
MD4046	50.45	4.65	0.11	600
MD4046	56.1	5.65	5.18	100
MD4047	43.85	2.2	0.8	200
MD4047	50.4	6.55	0.06	600
MD4047	56.4	6	8.86	100
MD4048	44.45	1	0.04	200
MD4048	52.5	8.05	0.29	600
MD4048	56	3.5	1.7	100
MD4401	123.45	3	6.74	200
MD4401	137.3	13.85	1.9	600
MD4401	140.05	2.75	13.88	100
MD4402	109.95	4.4	3.74	200
MD4402	119.9	9.95	3.08	600
MD4402	122.9	3	5.7	100
MD4403	71.8	3.1	5.83	200
MD4403	83.15	11.35	1.24	600
MD4403	93	9.85	4.88	100
MD4404A	138.35	7.5	13.83	200
MD4404A	158.35	20	0.91	600
MD4404A	158.45	0.1	0.09	600
MD4404A	164.1	5.65	10.44	100
MD4405	124	6.8	2.87	200
MD4405	139	15	2.63	600
MD4405	143.1	4.1	14.17	100
MD4406	85	9.3	3.31	200
MD4406	100.75	15.75	1.23	600
MD4406	107.1	6.35	6.47	100
MD4407	106.7	2.35	1.34	200

MD4407		120	13.3	3.15	600
MD4407		123.3	3.3	0.65	100
MD4408		94.55	2.8	2.22	200
MD4408		105.5	10.95	4.16	600
MD4408		111.55	6.05	2.4	100
MD4409A		136.7	4	6.8	200
MD4409A		156.7	20	6.5	600
MD4409A		166.7	10	1.91	600
MD4409A		173.2	6.5	2.18	100
MD4410		108.1	3.3	2.4	200
MD4410		128.1	20	1.87	600
MD4410		135.15	7.05	1.39	600
MD4410		141.05	5.9	12.69	100
MD4411		88	3	2.43	200
MD4411		96.1	8.1	0.09	600
MD4411		110.4	14.3	5.2	100
MD4412		183.5	5.8	2.83	200
MD4412		188	4.5	0.51	600
MD4574A		197.05	5.5	2	200
MD4574A		212.7	15.65	1.27	600
MD4574A		217	4.3	7.75	100
MD4575		234.5	5.85	1.47	200
MD4575		244.6	10.1	3.66	600
MD4575		251.8	7.2	6.06	100
MD4576		211.05	3.2	0.1	200
MD4576		223.25	12.2	2.59	600
MD4576		232	8.75	0.87	100
MD4577		162.75	10.6	0.55	200
MD4577		168.9	6.15	0.12	600
MD4577		184.9	16	2.37	100
MD4581		183.4	6	0.78	200
MD4581		198.6	15.2	1.63	600
MD4581		206.95	8.35	2.28	100
MD4582		208.35	5.3	5.77	200
MD4582		217.3	8.95	0.99	600
MD4582		223.25	5.95	1.79	100
MD4583		233.75	7.65	1.82	200
MD4583		253.75	20	0.25	600
MD4583		267.1	13.35	0.3	600
MD4583		272.55	5.45	1.48	100
MD4604		203.65	4.05	0.5	200
MD4604		210.9	7.25	0.91	600
MD4604		217.1	6.2	2.17	100
SD598C	1	76.1	8.55	5.96	100
SD598C	1	82	5.9	6.17	600
SD598C	1	87.15	5.15	4.71	200
SD598CW1	1	68	5	0.11	100
SD598CW1	1	79	11	0.5	600
SD598CW1	1	81	2	2.66	200
SD598CW2	1	183.45	5.95	1.33	100
SD598CW2	1	187	3.55	0.11	600
SD598CW2	1	191.6	4.6	7.48	200
SD606	1	163.3	2	0.01	200

APPENDIX 2 MINESIGHT RUN FILES

Central Lode

```
MEDS-624V1  10=nth10.dat  9=nth09.ddh  15=nth15.dat;
MEDS-624V1  3=rpt624.dbg  31=comps.xyz 30=100.var
** KRIGING of 3-D block values for CAUD

RUN = RESET

USR =  / C7 Central lode estimate

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 29 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octants;
COM       4=Split Quadrants
COM      Note: Negative #=use rotated search. By Default,
COM       octants & quadrants are defined using the
COM       project's orthogonal axes).
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 252. / Primary X-search distance
PAR2 = 144. / Primary Y-search distance
PAR3 = 30. / Primary Z-search distance
PAR4 = 252. / Limiting search distance
PAR7 = 0. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 8 8 2 / Block discretization in X,Y,Z

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 252. 144. 30. 275. -55. 14.

ITM1 = CAUD AUCUT CALC KRIGE
ITM2 = CSLRD Block error
ITM3 = CGEO Block limit
ITM4 = GEOCD RANGE 99. 101.
CMD = Block limit codes 100
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Dukes Lode Au

```
MEDS-624V1  10=nth10.dat  9=nth09.ddh  15=nth15.dat;
MEDS-624V1  3=rpt624.dbg  31=comps.xyz 30=200.var
** KRIGING of 3-D block values for BAUD

RUN = RESET

USR =  / Kriging for Dukes Load

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 33 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octants;
COM       4=Split Quadrants
COM       Note: Negative #=use rotated search. By Default,
COM       octants & quadrants are defined using the
COM       project's orthogonal axes).
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 300. / Primary X-search distance
PAR2 = 180. / Primary Y-search distance
PAR3 = 50. / Primary Z-search distance
PAR4 = 300. / Limiting search distance
PAR7 = 0. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 8 8 2 / Block discretization in X,Y,Z

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 300. 180. 50. 310. -47. 45.

ITM1 = BAUD AUCUT CALC KRIGE
ITM2 = BSLRD Block error
ITM3 = BGEO Block limit
ITM4 = GEOCD RANGE 199. 201.
CMD = Block limit codes 200
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Weak Volcanogenic

```
MEDS-624V1  10=nth10.dat  9=nth09.ddh  15=nth15.dat;
MEDS-624V1  3=rpt624.dbg  31=comps.xyz 30=600.var
** KRIGING of 3-D block values for VAUD

RUN = RESET

USR =  / Stockworks ordinary kriging

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 23 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octants;
COM       4=Split Quadrants
COM       Note: Negative #=use rotated search. By Default,
COM       octants & quadrants are defined using the
COM       project's orthogonal axes).
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 180. / Primary X-search distance
PAR2 = 180. / Primary Y-search distance
PAR3 = 66. / Primary Z-search distance
PAR4 = 180. / Limiting search distance
PAR7 = 0. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 8 8 2 / Block discretization in X,Y,Z

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 180. 180. 66. 257. -57. 0.

ITM1 = VAUD AUCUT CALC KRIGE
ITM2 = VSLRD Block error
ITM3 = VGEO Block limit
ITM4 = GEOCD RANGE 599. 601.
CMD = Block limit codes 600
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Dukes True Width

```
MEDS-624V1  10=nth10.dat  9=nth09.acc  15=nth15.dat;
MEDS-624V1  3=rpt624.dbg  31=comps.xyz 30=200tw.var
** KRIGING of 3-D block values for BAUS

RUN = RESET

USR =  / Kriging for Dukes Load true width

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 33 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octants;
COM       4=Split Quadrants
COM       Note: Negative #=use rotated search. By Default,
COM       octants & quadrants are defined using the
COM       project's orthogonal axes).
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 300. / Primary X-search distance
PAR2 = 180. / Primary Y-search distance
PAR3 = 50. / Primary Z-search distance
PAR4 = 300. / Limiting search distance
PAR7 = 0. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 8 8 2 / Block discretization in X,Y,Z

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 300. 180. 50. 257. -57. 0.

ITM1 = BAUS TRUE CALC KRIGE
ITM2 = CSLRA Block error
ITM3 = BGEO Block limit
ITM4 = GEOCD RANGE 199. 201.
CMD = Block limit codes 200
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Dukes Accumulation

```
MEDS-624V1  10=nth10.dat  9=nth09.acc  15=nth15.dat;
MEDS-624V1   3=rpt624.dbg  31=comps.xyz 30=200ac.var
** KRIGING of 3-D block values for BACCM

RUN = RESET

USR =   /  Kriging for Dukes Load true width

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 33 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octants;
COM        4=Split Quadrants
COM       Note: Negative #=use rotated search. By Default,
COM        octants & quadrants are defined using the
COM        project's orthogonal axes).
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 300. / Primary X-search distance
PAR2 = 180. / Primary Y-search distance
PAR3 = 50. / Primary Z-search distance
PAR4 = 300. / Limiting search distance
PAR7 = 0. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 8 8 2 / Block discretization in X,Y,Z

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 300. 180. 50. 310. -47. 45.

ITM1 = BACCM ACCUM CALC KRIGE
ITM2 = BSLRA Block error
ITM3 = BGEO Block limit
ITM4 = GEOCD RANGE 199. 201.
CMD = Block limit codes 200
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX D

Upper South Fault Resource Report

(U3)

November 2004

Resource Summary Sheets

Stawell Upper South Fault Resource Model

Model Name :	Usf415.dat
Prepared by :	Mine Geology Group - Stawell
Drillhole Data To:	20 October 2004
Development Survey date :	18 November 2004
Date Model Completed :	18 November 2004
Date Report Completed :	31 November 2004
Interpretations By :	Lachlan Macdonald
Resource Modelling and Grade Estimation By :	Chris De-Vitry
Report Title:	Upper South Fault - geological resource model
Compiled by :	Chris De-Vitry
Approved By :	Vincent O'Brien Senior Mine Geologist
Audited by :	Not Audited yet
Accepted by :	Roddy Ormonde Senior Mining Engineer - Stawell
Report Distribution :	Geoff McDermott Vincent O'Brien Stawell site geologists Stawell site engineers

MODELLING ENVIRONMENT		
	Project Directory :	S:\Mining-Geology\Upper South Fault\Minesight\Geology\Current Model
	Model Identification :	USF
	Modelling Software :	Minesight and Isatis
	Operating System :	Windows XP
MODEL DESCRIPTION :		Updating the previous model generated in March 2004.
	(narrative, purpose, zones, relation to other models, etc):	The new information includes 10 new drillholes and 205 previous drillholes for a total of 215 drillholes and mapping. Grade and waste material modelled by geological domain (GEOCOD). The reported resource has been classified into Indicated and Inferred resources.
GEOLOGY/ORE DESCRIPTION :		
	(structure, stratigraphy, dip, strikes, size, ore type etc)	The Upper South Fault orebody is contained within a Volcanogenic package which is defined by Dukes Basalt to the East and Mine Schist to the West, with an internal basalt (Oriental Basalt). This ore body is a faulted block, with the upper reaches constrained by South Fault and lower by the Merlot Fault (both faults are flat). The average strike of these formations is 333° and dip approximately 64° to the West. The Upper South Fault area is still open at depth. There are three major ore formations within the Volcanogenic package. These include i) Dukes Lode, which is a basalt contact mineralisation on the Western flank of the Dukes Basalt, ii) Oriental Stockworks, which is a basalt contact mineralisation on the western flank of the Oriental Basalt combined with minor stockworks over the top of this nose, and iii) Central Lode, which is a quartz rich, sheared lode. While i) and ii) are proximal to associated basalt formations and commonly contain siliceous sediments, Central Lode is typically constrained to the West by Mine Schist and to the East by the Oriental Basalt (when present)and is typically laminated quartz dominated. The strongest and most constant lode is Central lode, ranging from 0.5m to 6m wide in parts. Shallowly dipping ore shoots within Central Lode appear to dip to the South. The prominent sulphides are arsenopyrite, pyrite and pyrrhotite with visible gold found on occasion.
ELEMENTS MODELLED :		Au
PROPERTIES MODELLED :		DENSITY (DEFAULTS)
GRID USED (Local, AMG etc)		AMG

DRILLHOLE DATA :	Directory :	S:\Mining-Geology\Upper South Fault\Minesight\Geology\Current Model _msresources\Drillholes		
Sample data extent				70° Mine Grid
Easting :	657792 mE	to	658123 mE	
Northing :	898023 mN	to	898449 mN	1790 mN – 2270mN
RL:	-1170 RL	to	-823 RL	
Drillhole Spacing:	At best approximately 20*20 (along strike and down dip)			
Raw drillhole data	Project-usf210.dat, usf211.dat, usf212.dat. Flagged by GEOCOD using wireframes. Flagging was visually and statistically checked and when required wireframes were modified and the drillholes manually coded.			
Composited drillhole data	Usf208.ddh (Composites broken at GEOCOD) 8822 samples.			
STRUCTURAL INTERPRETATION :				
No. of Sections :	54			
Section Interval:	10m (70 rotated grid)			
Sectional Interpretation:	S:\Mining-Geology\Upper South Fault\Minesight\Geology\Current Model_msresources\Geology\Dynamic Strings			
Comments :	Cross-sectional strings for stratigraphy validated in 1m plan slices. Strings are snapped to drillholes. A total of 215 diamond drill holes were used.			
WIREFRAMES :		No. of Wireframes :	a total of 12	
	4 volcanogenic solids, 2 basalt solids 2 Fault surfaces. 3 Wireframes defining indicated resources. 1 Solid defining material mined (dec1004_solid)			
Construction of the Wireframes :	In Minesight by Lachlan Macdonald			
Date Wireframes Completed :	November 2004			
SAMPLE CODING :	Flagged by GEOCOD using wireframes. Flagging was visually and statistically checked and when required wireframes were modified and the drillholes manually coded.			
Composited drillhole data	1m comps – usf208.ddh view USF1M comps (Composites broken at GEOCOD)		8822 1m composites	

BLOCK MODEL PARAMETERS :

Model Limits and Parameters :	Origin		Rotation		Range (metres)	Number Of Cells	Cell Dimensions Parent	
Easting : (along strike)	658215.19		1	329	520	26	20	
Northing : (down dip)	897770		2	0	980	49	20	
RL : (perpendicular to lode)	-1270		3	70	150	75	2	

GRADE INTERPOLATION :

Estimation Technique : Ordinary kriging for volcanogenics.

Grade Estimation Files : Rpt624.501, Rpt624.600, Rpt624.100, Rpt624.200.

Default Grades All waste was assigned the average Au grade of 0.001.

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Kriging standard deviation
Central Lode	CGEO=100	C%	CAUD	CSG	CSLRD
Dukes Lode	BGEO=200	B%	BAUD	BSG	BSLRD
Oriental Stockworks	SGEO=600	S%	SAUD	SSG	SSLRD
Weak Volcanogenic	VGEO=501	V%	VAUD	VSG	WSLRD
Waste was not coded					

DOMAIN	DRY DENSITY (t/m^3)
Central Lode	2.77
Dukes Lode	2.94
Oriental Stockworks	2.94
Weak Volcanogenic	2.81
Waste was not coded	2.8

Table 1. Resource at a 4g/t cutoff below the 1004 level (Derived from Minesight detailed reserves panel).

Domain	Resource Classification	Tonnes	Au g/t	Ounces
Central Lode	Indicated	63,500	5.6	11,470
Dukes Lode	Indicated	10,200	5.2	1,710
Oriental Stockworks	Indicated	4,400	4.6	650
Central Lode	Inferred	63,200	5.8	11,820
Dukes Lode	Inferred	5,600	4.8	870
Oriental Stockworks	Inferred	40,900	5.4	7,120
Central Lode	Inferred + Indicated	126,700	5.7	23,260
Dukes Lode	Inferred + Indicated	15,900	5.1	2,620
Oriental Stockworks	Inferred + Indicated	45,300	5.3	7,750
Total	Indicated	78,100	5.5	13,860
Total	Inferred	109,700	5.6	19,820

Table 2. Resource at a 4g/t cutoff before mining i.e. above and below the 1004 level (Derived from Minesight detailed reserves panel).

Domain	Resource Classification	Tonnes	Au g/t	Ounces
Central Lode	All classifications	349,900	6.1	68,850
Dukes Lode	All classifications	137,800	7.8	34,670
Oriental Stockworks	All classifications	75,500	6.6	16,070

Resource Classification Wireframes

The wireframes defining the Indicated Resources are "Central Indicated" (CCAT =1) and "Dukes Indicated" (BCAT=1) and "Stockwork Indicated" (SCAT = 1). Material outside these wirframes is Inferred (CCAT=2 or BCAT=2 or SCAT = 2). The confidence wirframes are in S:\Mining-Geology\Upper South Fault\Minesight\Geology\Current Model_msresources\Geology\Final Solids).

Grade Wireframes

The Minesight subblocking routine was utilised to generate grade wireframes for 3, 4, 5 and 7g/t cutoffs (see S:\Mining-Geology\Upper South Fault\Minesight\Geology\Current Model_msresources\Block Model)

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

The Upper South Fault area below the 1004 level is undeveloped and completely mined out above the 1004 level. This model will be used to justify developing below the 1004 level. The Upper South Fault is cut and bounded by the South Fault above and the Merlot Fault below and the orebody is still open at depth (Figure 1 and Figure 2). As presented in Table 3 numerous people have been involved in this project.

Table 3. Identification of the project team.

Resource	Company	Role
Lachlan Macdonald	MPI	Mapping, logging and creation of geological domains. Writing the geological sections of this report.
Chris De Vitry	MPI	Resource model, this report.
Roddy Ormonde	MPI	Project review and approval.
Geoff McDermott	MPI	Project review and approval.
Vince O'Brien	MPI	Project review and approval.

1.2 Summary and Conclusions

There have been 10 new drillholes and significant improvements to the geological interpretation. The amount of Indicated ounces (13,860) is in line with expectations for the new drilling.

Due to a lack of stationarity in domains, a lack of drillholes and the smoothing of the ordinary kriging the quality of the resource estimate is comprised. If the decision to proceed with developing below the 1004 level is marginal this resource estimate should be redone with improved domaining and conditional simulation.

Table 4. Resource at a 4g/t cutoff below the 1004 level (Derived from Minesight detailed reserves panel).

Domain	Resource Classification	Tonnes	Au g/t	Ounces
Central Lode	Indicated	63,500	5.6	11,470
Dukes Lode	Indicated	10,200	5.2	1,710
Oriental Stockworks	Indicated	4,400	4.6	650
Central Lode	Inferred	63,200	5.8	11,820
Dukes Lode	Inferred	5,600	4.8	870
Oriental Stockworks	Inferred	40,900	5.4	7,120
Central Lode	Inferred + Indicated	126,700	5.7	23,260
Dukes Lode	Inferred + Indicated	15,900	5.1	2,620
Oriental Stockworks	Inferred + Indicated	45,300	5.3	7,750
Total	Indicated	78,100	5.5	13,860
Total	Inferred	109,700	5.6	19,820

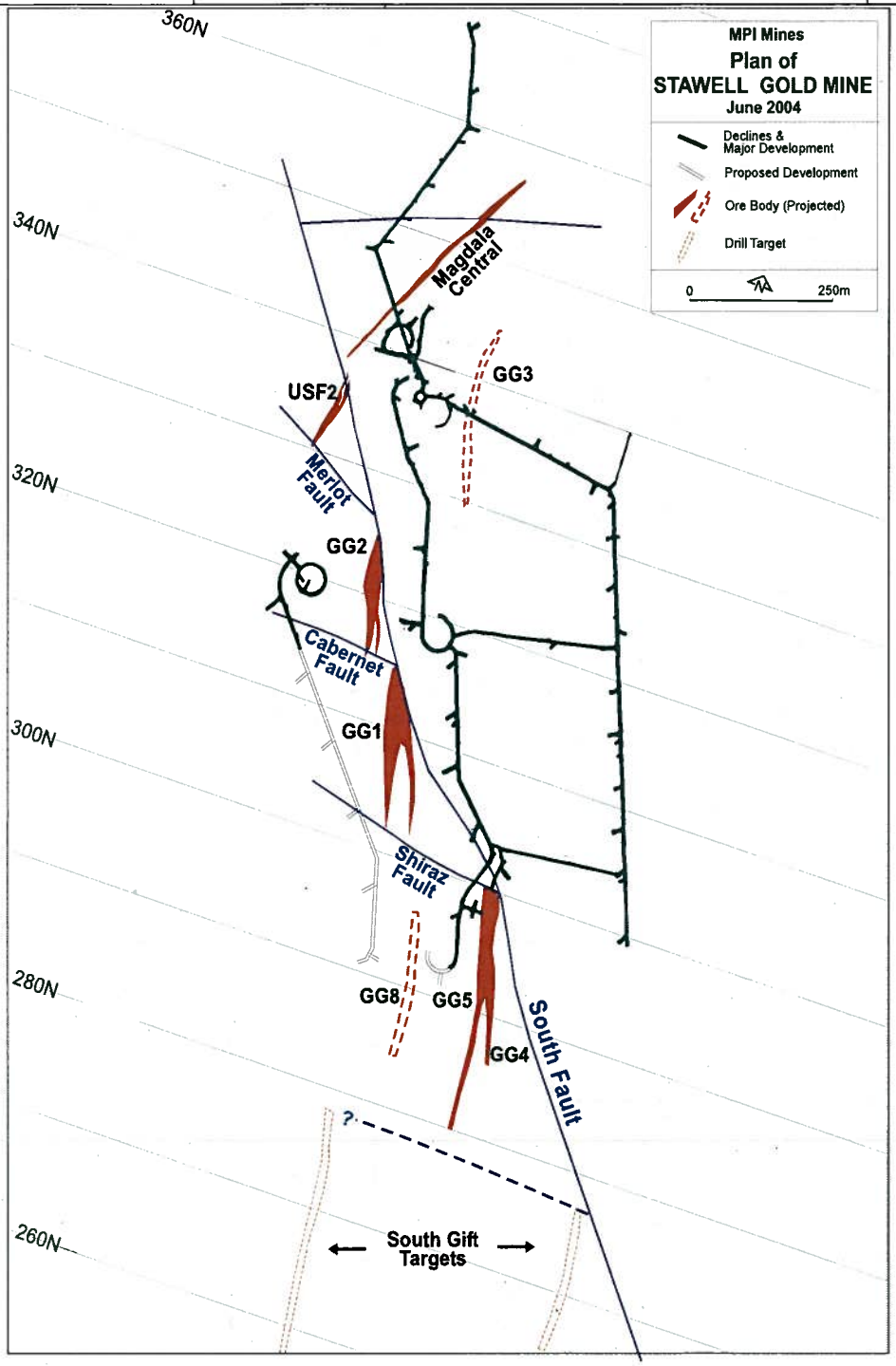

Figure 1. Plan view of the golden gift and Upper South Fault orebodies.

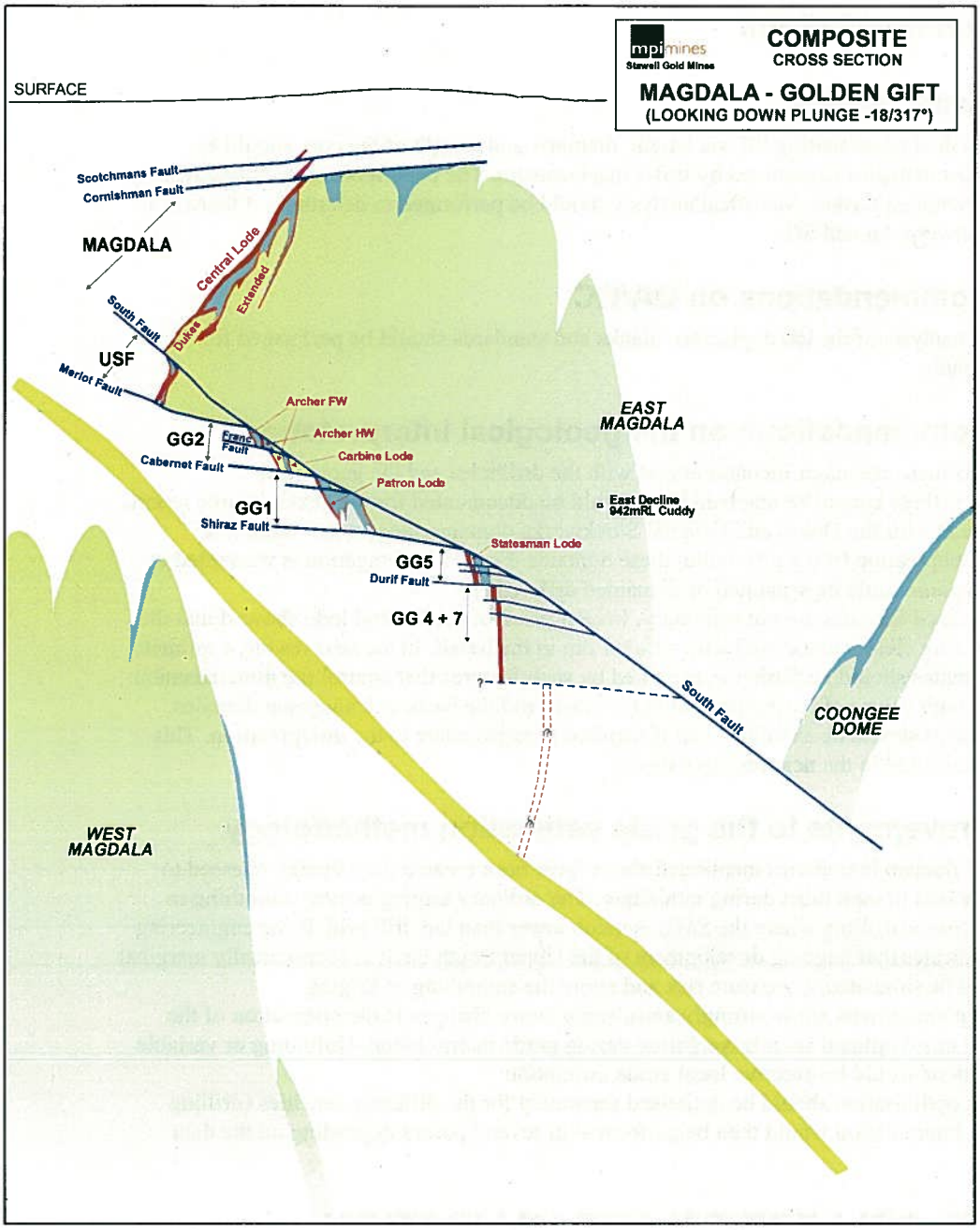

Figure 2. Down plunge cross section of the Magdala and Golden Gift orebodies.

1.4 Recommendations

1.4.1 Specific gravity

Our current method of measuring SG via length, diameter and weight of the core should be validated against densities determined by water displacement. The benefit of interpolating density should be investigated further. Statistical analysis should be performed to determine if there is a relationship between Au and SG.

1.4.2 Recommendations on QA/QC

More detailed analysis of the lab duplicates, blanks and standards should be performed for the Upper South Fault.

1.4.3 Recommendations on the geological interpretations

In several areas there are major inconsistencies with the drillholes and the geological interpretation. If these cannot be resolved they should be documented for the next resource report.

The histograms for the Dukes and Oriental Stockworks domains suggest that there is an unmineralised population (< 0.5 g/t) within these domains. Further investigation is warranted to see if these domains could be separated or domained differently.

The mineralised domains are not stationary. Investigation of the Central lode showed that the higher grades were clearly associated with a flatter dip in the basalt. In the next resource estimate the current domains should be further constrained by such features that control the mineralisation.

Unusually high values of Au are present in the waste and the weak volcanogenic domains. These occurrences should be examined and if required changes made to the interpretation. This should be documented in the next resource report.

1.4.5 Improvements to the grade estimation methodology

When the new domain boundaries mentioned above have been created they should assessed to determine how best to treat them during modelling. The ordinary kriging is over smoothing in areas of wide spaced drilling where the SMU is much lower than the drill grid. If the engineering department indicates that ongoing development of the Upper South Fault is economically marginal the area should be simulated to measure risk and avoid the smoothing of kriging.

Because the variograms are so strongly anisotropic minor changes in the orientation of the domains result in suboptimal sample weighting during grade interpolation. Unfolding or variable search orientations would be improve local grade estimation

The search optimisation should be optimised separately for the different densities (drilling grids) of data. Interpolation would then be performed in several passes depending on the data density.

2.0 DATA COLLECTION AND VALIDATION

2.1 Overview of the Drilling Data

A total of 215 diamond drill holes were used for interpretation with the diamond drilling being predominantly 44 mm LTK60 and 50.7 mm NQ2 sizes.

The core was generally of a consistent nature with minimal core loss. The softer, more broken core of the Mine Schist was where some core loss was encountered.

Diamond drill hole collars are surveyed and electronic single shot down hole cameras have been used to survey the hole path. Magnetic intensity is recorded and used to assess validity of record. Dip +/- 1.5° and azimuth +/- 3° is highlighted and assessed.

2.2 QA/QC

The general procedures for QA/QC are outlined below:

- Production diamond drill core is routinely Full Core sampled.
- Routine assaying of diamond drill core samples is undertaken at Aminya laboratories in Ballarat by fire assay (50 gram charges) with an AAS finish. Assays are reported to an LLD of 0.01ppm. For sample reporting below LLD, a value of 0.5xLLD is used in resource estimation.
- Sample QA/QC procedure incorporates routine check assays including repeats (re-assay or repeat assay), duplicate (second sample taken after pulverisation), splits (half coarse sample split, at ratio approx 1:10 samples), and standards.
- SGM Standard reference material has been prepared by Ore Research of Melbourne, from ore samples from Magdala Mine.
- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sample ID created during logging procedure.
- Analysis of received standard values and repeat and duplicate information is used to monitor the performance of the assaying process.

2.3 General Validation Checks

All data is stored within the "acQuire" Database Management System which is structured on a Data Model which applies rules of relational databases to mining and exploration data. The primary key is HoleID.

Diamond drill hole collars are surveyed and single shot down hole cameras have been used to survey the hole path. Electronic single shot instruments are now used in preference to the Eastman mechanical cameras. Magnetic intensity is recorded and used to assess validity of record. Dip +/- 1.5° and azimuth +/- 3° is highlighted and assessed.

All logging of diamond drill core is undertaken using "Datcol" software, to a defined logging scheme. It is a barcode system, eliminating errors involved in keying. Logging is both quantitative and qualitative but in both cases only valid codes or values can be recorded. Additional validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth
- Checks are performed for overlapping samples

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (eg Minesight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted. After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

3.0 GEOLOGICAL OVERVIEW

3.1 Geological Setting

The Upper South Fault orebody is contained within a Volcanogenic package which is defined by Dukes Basalt to the East and Mine Schist to the West, with a internal basalt (Oriental Basalt). This ore body is a faulted block, with the upper reaches constrained by South Fault and lower by the Merlot Fault (both faults are flat). The average strike of these formations is 333° and dip approximately 64° to the West. The Upper South Fault area is still open at depth. Where units are close to South Fault and Merlot Fault, these units (and ore bodies) display some warping into these faults, which give a general Hangingwall thrust West over Footwall (reverse) sense of movement along each of the faults.

There are three major ore formations within the Volcanogenic package. These include:

i) Dukes Lode, which is a basalt contact mineralisation on the Western flank of the Dukes Basalt,

ii) Oriental Stockworks (Lode), which is a basalt contact mineralisation on the western flank of the Oriental Basalt combined with minor stockworks over the top of this nose, and

iii) Central Lode, which is a laminated quartz rich, sheared lode.

While i) and ii) are proximal to associated basalt formations and commonly contain siliceous sediments, Central Lode is typically constrained to the West by Mine Schist and to the East by the Oriental Basalt (when present) and is typically laminated quartz dominated. The strongest and most constant lode is Central lode, ranging from 0.5m to 6m wide in parts. Shallowly dipping ore shoots within Central Lode appear to dip to the South. The prominent sulphides are arsenopyrite, pyrite and pyrrhotite with visible gold found on occasion.

3.2 Methodology for the Creation of Geological Domains

Several shapes were created and then clipped against each other to produce the final geology shapes. The South Fault and Merlot Fault created using information from diamond core and pickups from underground development. These surfaces were created by digitising a line on 45° grid sections where there was some information. The lines were then joined together to form a surface and later made into a solid to assist the clipping process.

Firstly the basalts were constrained and modelled (Dukes followed by the Oriental Basalt). Central Lode, being the following dominant feature, was modelled then followed by Oriental and Dukes Lodes.

Information from diamond core and development were used in the interpretation of these contacts. For these shapes strings and polygons were digitised on 10m sections of the 70° grid and joined together to form solids.

The solids that were created were then clipped against each other as follows. The Dukes Basalt was subtracted from the Oriental Basalt. The Basalts were then subtracted from Central Lode, then Central and the Basalts subtracted from Dukes Lode, Dukes and Central Lodes and Basalts subtracted from Oriental Lode. A Volcanogenics solid was created and from it, Central, Duke and Oriental Lodes and Basalts were subtracted. This created a waste Volcanogenic block. These shapes representing the main rock types were then clipped against the faults to form the final shapes.

3.3 Description of the domains

The Volcanogenic package was broken into 4 domains to assist with the modelling process. The domains were the Central Lode, Dukes Lode, Oriental Stockworks and Weak Volcanogenics. The Eastern flank of the Oriental Basalt was not mineralised where distal from Dukes Lode.

To create the domains polygons were digitised on 10m sections of the 45° grid and joined together to form solids. These solids were then clipped against the final shape of the Volcanogenics to produce the final domains.

Table 5. Domains interpreted for this resource estimate. The numeric codes are utilised to select domains for modelling and statistical interpretation.

Description (Codes used during grade interpolation)	Numeric Code
Central Lode	CGEO=100
Dukes Lode	BGEO=200
Oriental Stockworks	SGEO=600
Weak Volcanogenic	VGEO=501
Waste was not coded	

3.3 Methodology for domain coding

The diamond drillholes were coded based on which of the domains they passed through. This coding was done manually on sections in Minsight. All drillholes were visually checked and

corrected to guarantee that the correct code related to each interval. A small number of drillholes that were considered to have incorrect collars or surveys were not coded (the hole numbers were not recorded but this should be done before the next resource report.).

4.0 DATA ASSESSMENT

4.1 Compositing

4.1.1 Introduction

In each domain the model lengths are 1m thus compositing less than 1m is not advisable as too many sample intervals would be broken. The domains are narrow with true widths of 3-5m common. For this reason a composite length of 1m was selected.

4.1.2 Methodology for producing composites

Any missing gold values were reset to 0.001. This avoids errors occurring in the Minesight compositing process. There was no length or density weighting during the formation of composites and compositing was within the geological domains. All composites are of equal length except for the last (deepest downhole) composite in each domain which is variable in length. The composite files were validated by comparing them to the uncomposited data to insure that all of the data was composited. The composites were also checked to insure that they are of the correct length and that composites are within geological domains.

The 1m compostes were imported into Isatis for statistical analysis and variography. The Minsight sample statistics were identical to those calculated in Isatis confirming a correct import.

4.1.3 Analysis of core loss for various composite lengths

Table 6 contains the statistics comparing length and metal lost for 1m composites. At most the metal lost after compositing was 3.8%. This is acceptable.

Table 6. Comparison of length and metal lost during compositing to 1m sample lengths. The gold grades for this section do not have a top cut thus there is no risk of underestimating the amount of metal lost in the small composite lengths.

Length Cutoff	Geocode	Mean length	Mean Au	No. Samples	Total Metal	Total Length	% Metal compared to uncomposited	% Length compared to uncomposited
0.5	100	0.985	4.483	1880	8294.0	1851.4	99.3	98.6
0.5	200	0.964	7.338	399	2837.3	384.7	97.3	96.7
0.5	600	0.963	6.679	293	1889.6	282.3	96.2	97.9
0.5	501	0.984	0.865	1830	1561.0	1800.7	98.5	98.6

4.2 Cutoff Analysis

Top cuts for Au were assessed by examining histograms in conjunction with the tabulated and sorted data. The results of cutting a very small number of unrealistically high samples are presented in Table 7. For the three ore domains the top cutting reduced the Au metal by 1.2-2.1%. The metal loss in the weak volcanogenic domain was 6.4%. This relatively high level of cutting is probably due to incorrect domaining i.e. ore being in the weak volcanogenic domain.

Table 7. Analysis of metal loss after applying top cuts.

Domain	Top Cut	# of samples	% Loss of Au Metal	Mean Au
100	45	7	-1.23	4.43
200	55	3	-1.43	7.23
600	45	3	-2.12	16.54
501	10	15	-6.37	0.81

4.3 Statistical Analysis of Domains

The data used in this section has the Au top cuts and minimum sample lengths discussed in the previous sections. Figure 3 to Figure 20 represent histograms, log Q-Q plots, cutoff versus tonnage and cutoff versus grade for each domain. These grade tonnage curves are from the sample data with no change of support.

The histograms for the Dukes and Oriental Stockworks domains suggest that there is an unmineralised population (< 0.5 g/t) within these domains.

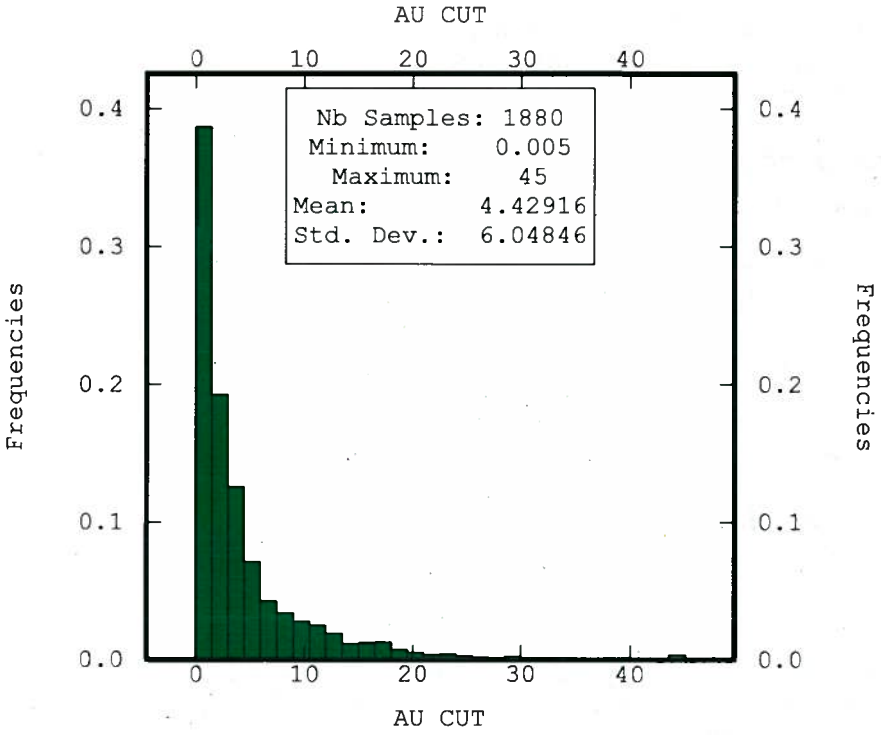

Figure 3. Histogram of gold grades for the Central Lode domain.

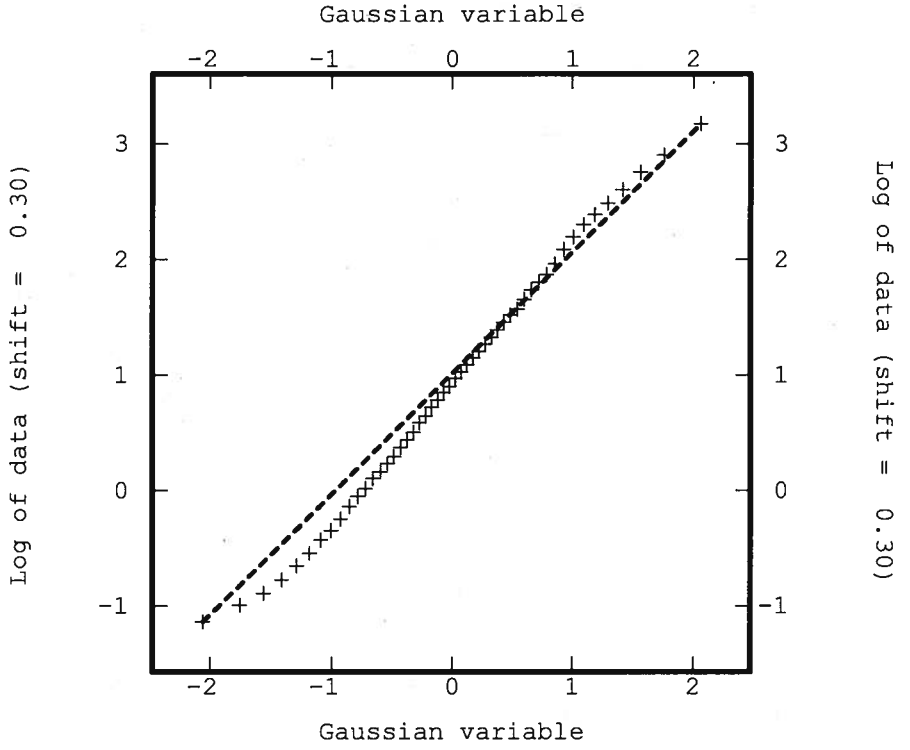

Figure 4. Log Q-Q plot of gold grades for the Central Lode domain.

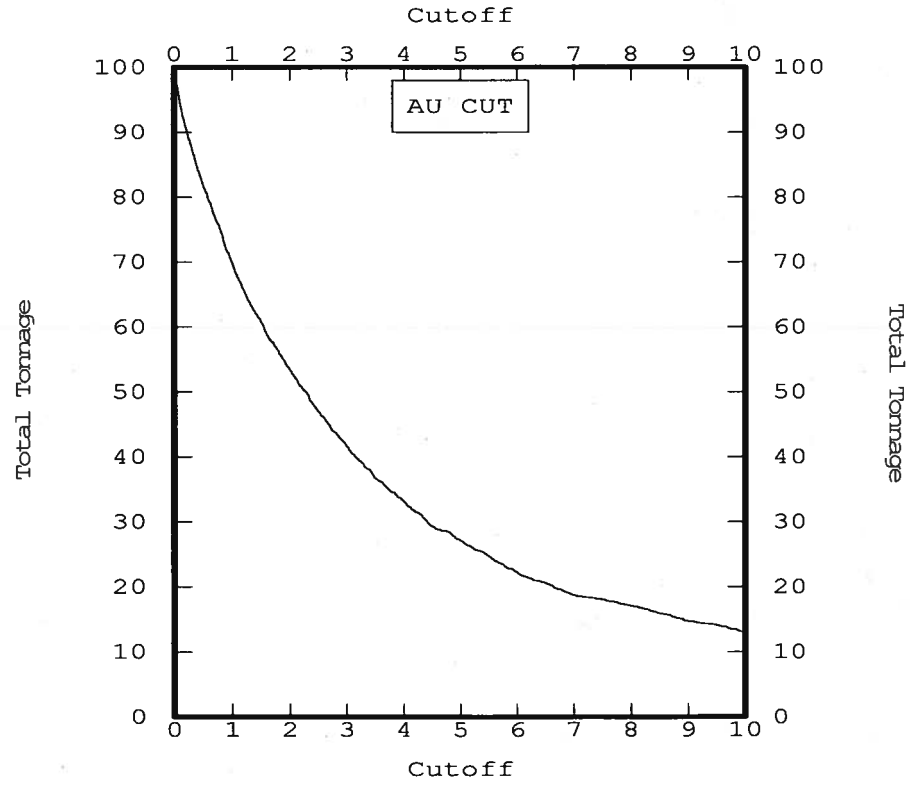

Figure 5. Cutoff versus tonnage for the Central Lode domain.

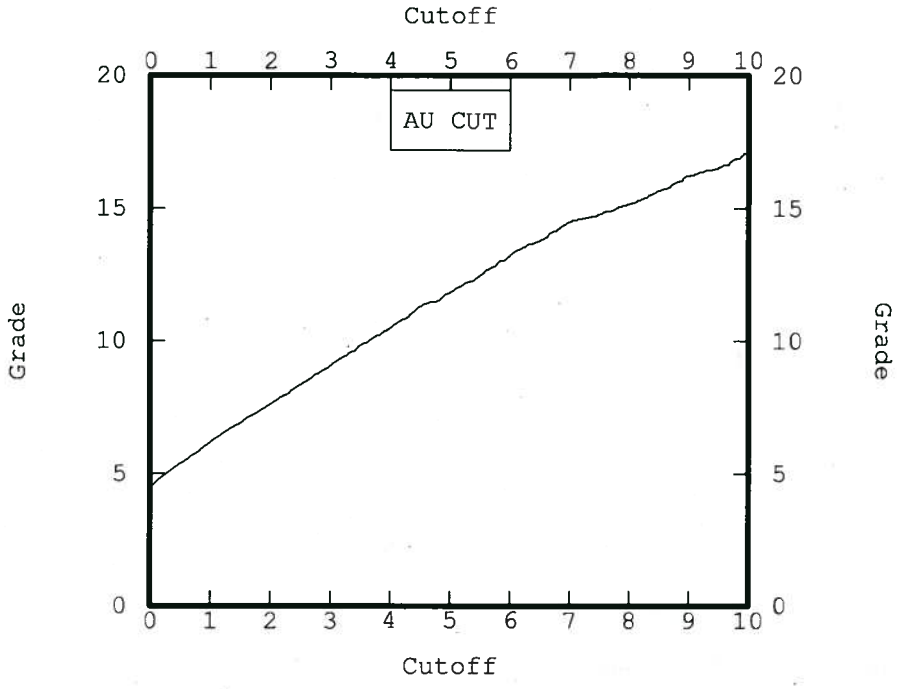

Figure 6. Cutoff versus grade for the Central Lode domain.

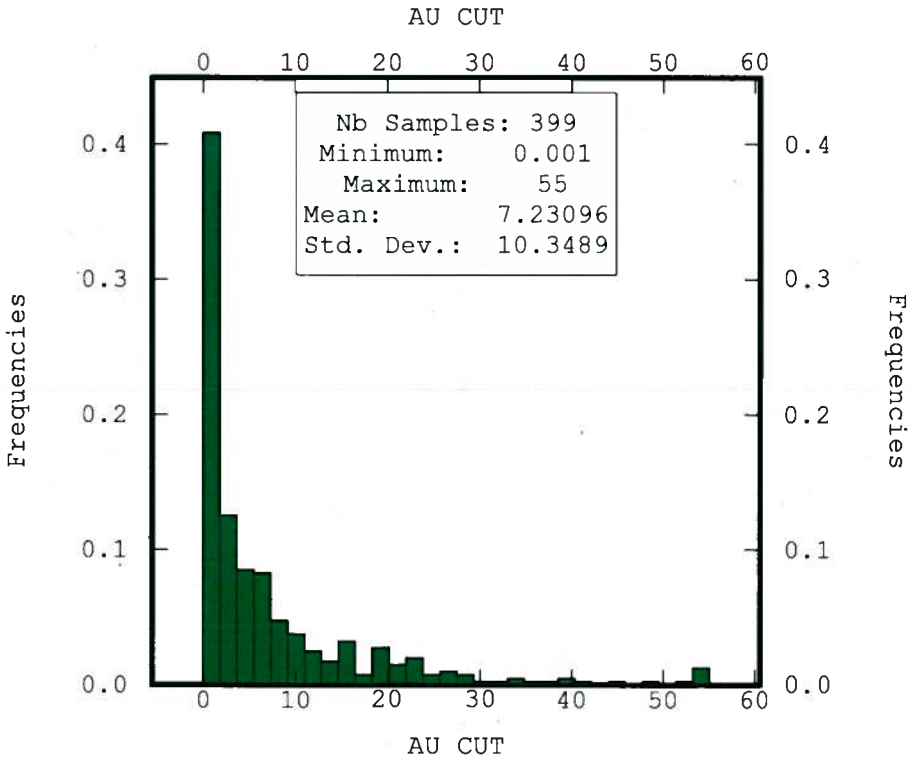

Figure 7. Histogram of gold grades for the Dukes domain.

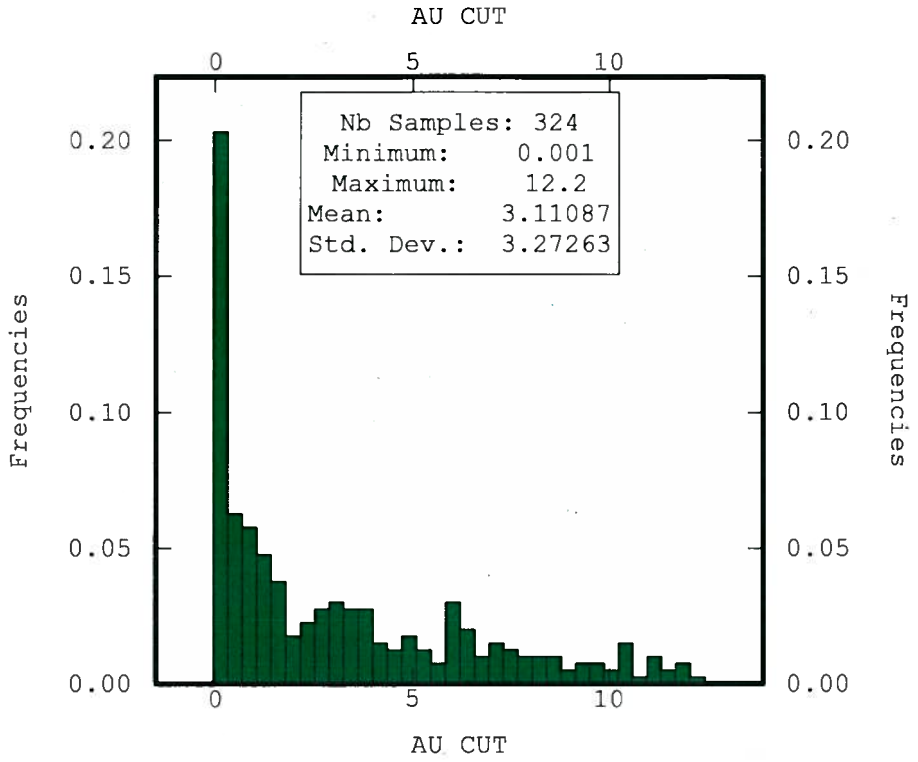

Figure 8. Histogram of gold grades for the Dukes domain (12.2 g/t upper cut applied).

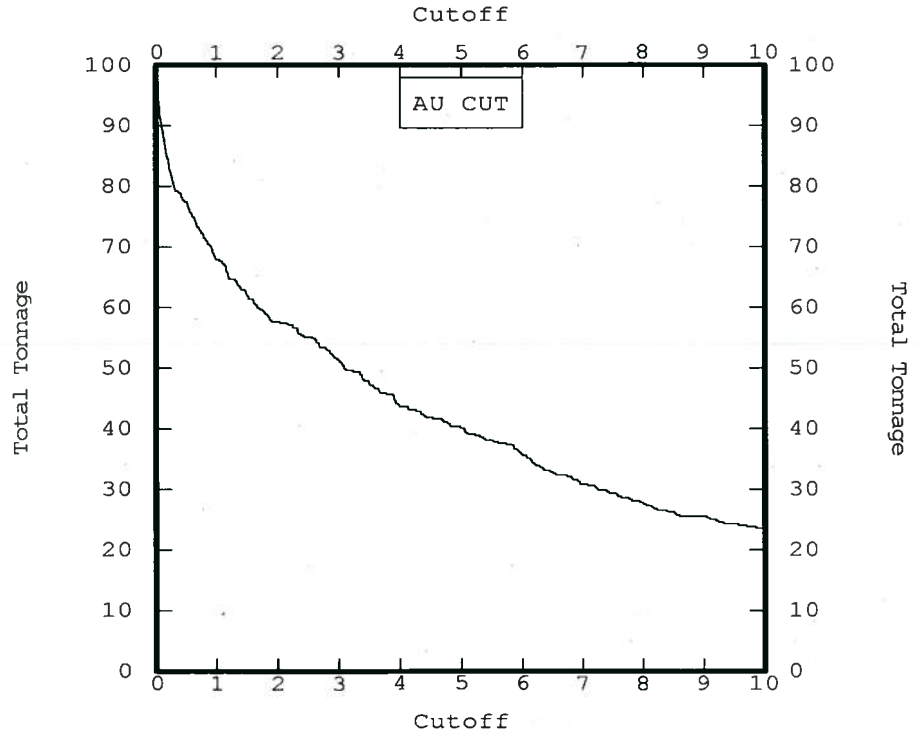

Figure 9. Cutoff versus tonnage for the Dukes domain.

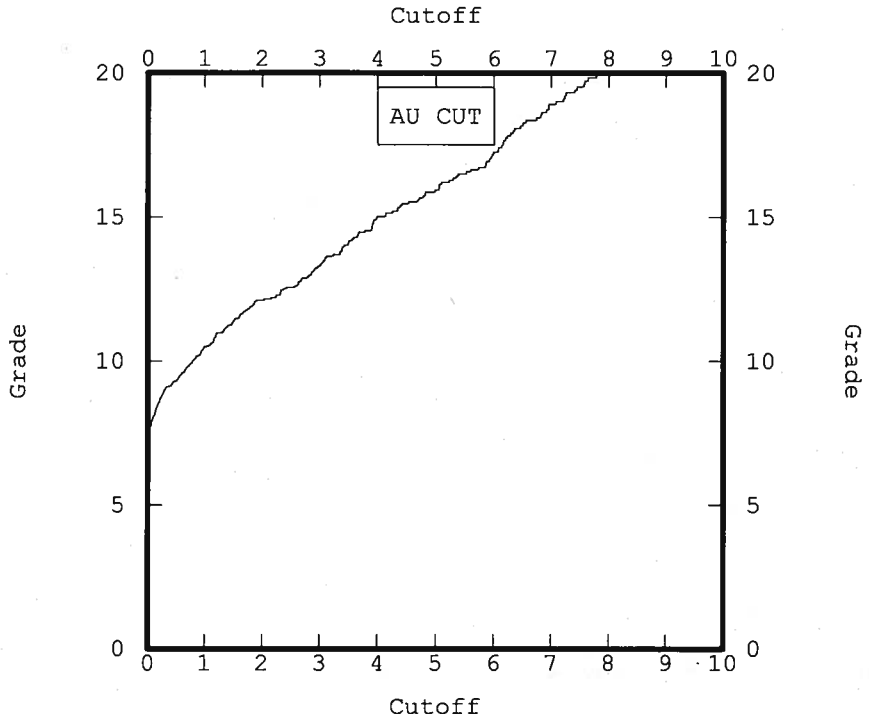

Figure 10. Cutoff versus grade for the Dukes domain.

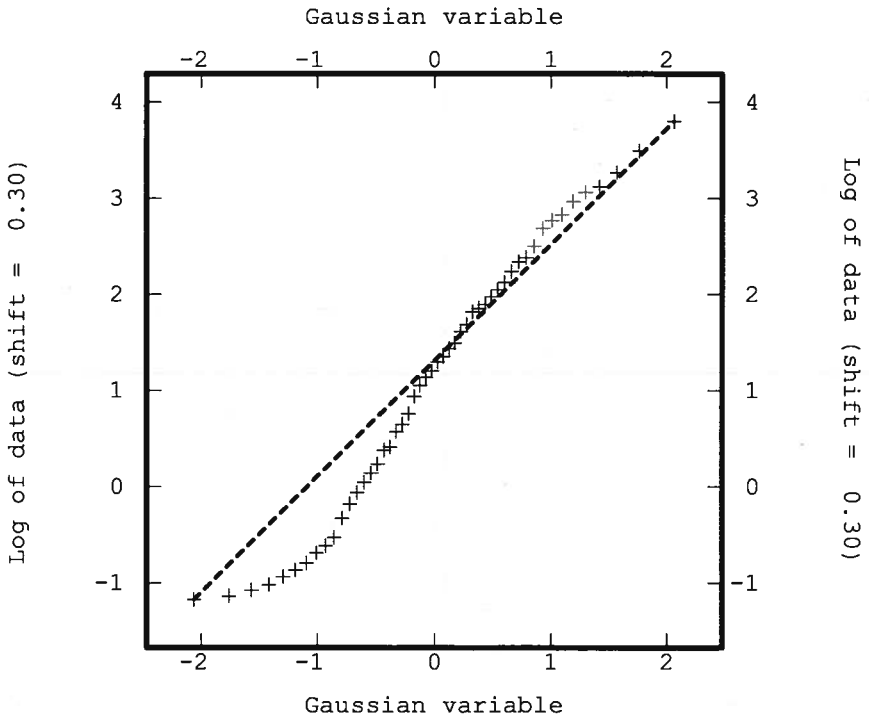

Figure 11. Log Q-Q plot of gold grades for the Dukes domain.

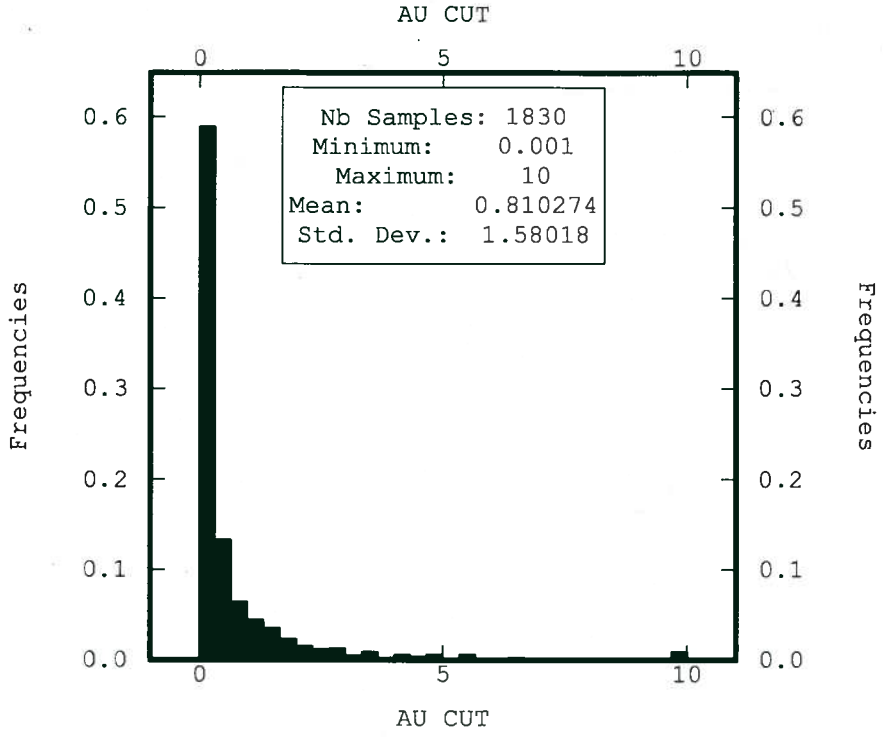

Figure 12. Histogram of gold grades for the Weak Volcanogenic domain.

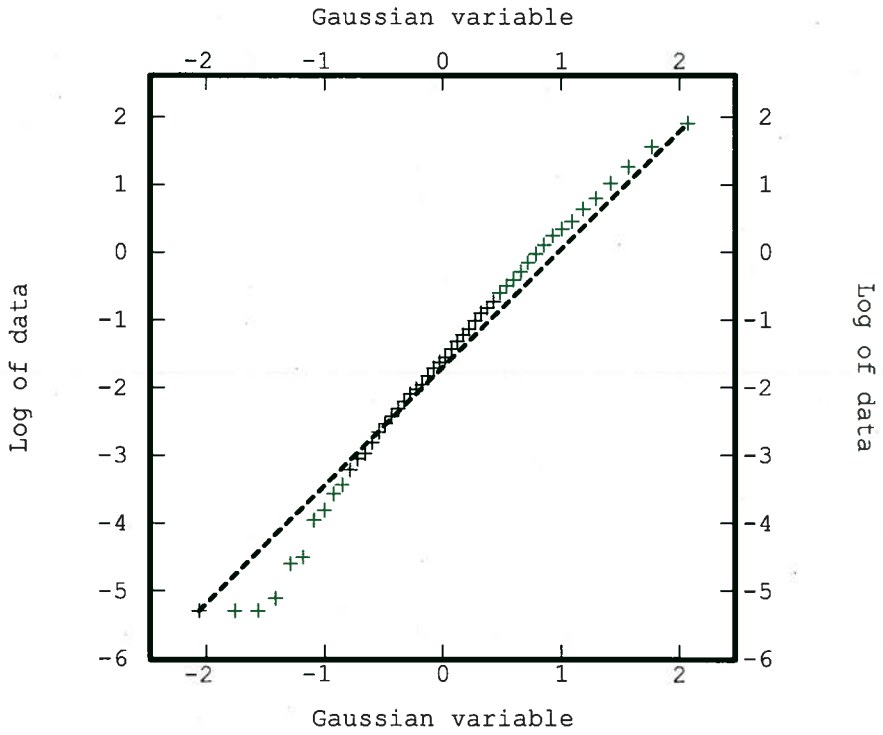

Figure 13. Log Q-Q plot of gold grades for the Weak Volcanogenic domain.

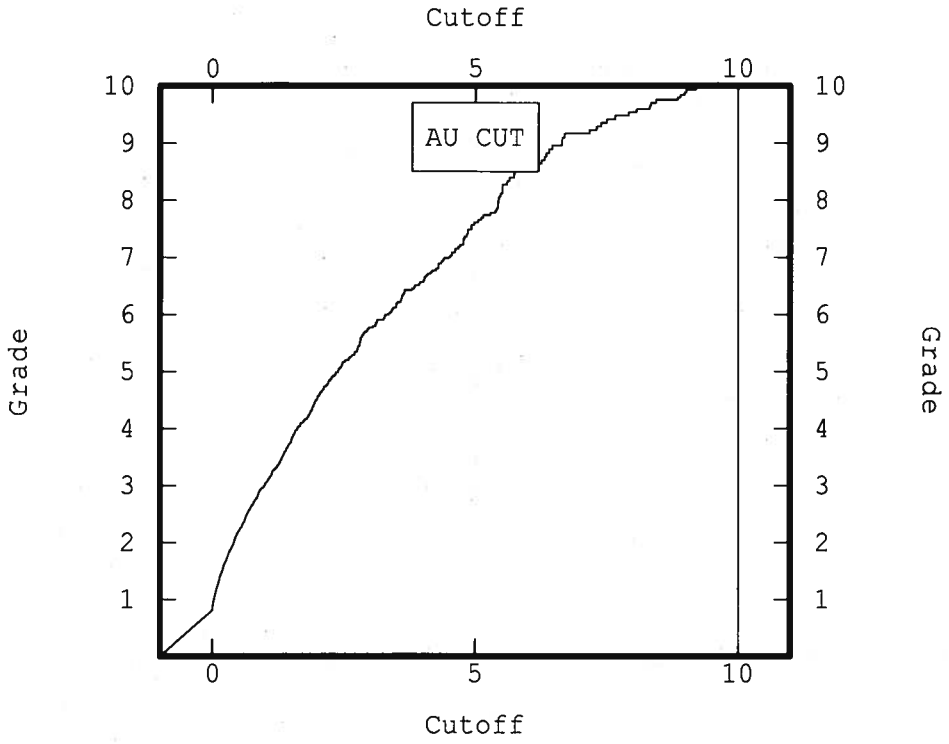

Figure 14. Cutoff versus grade for the Weak Volcanogenic domain.

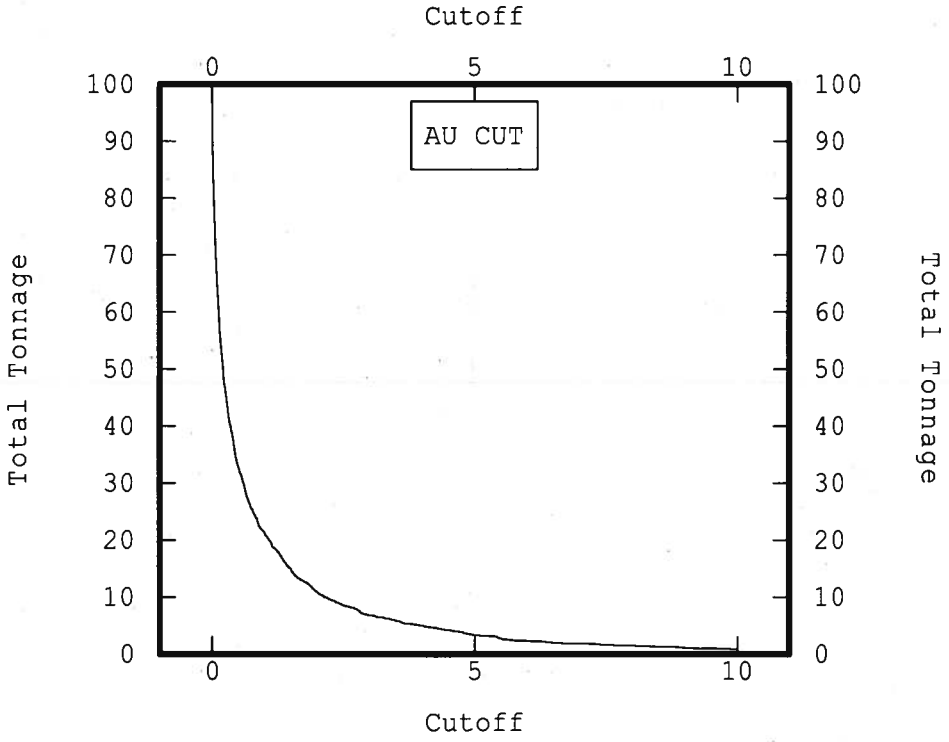

Figure 15. Cutoff versus tonnage for the Weak Volcanogenic domain.

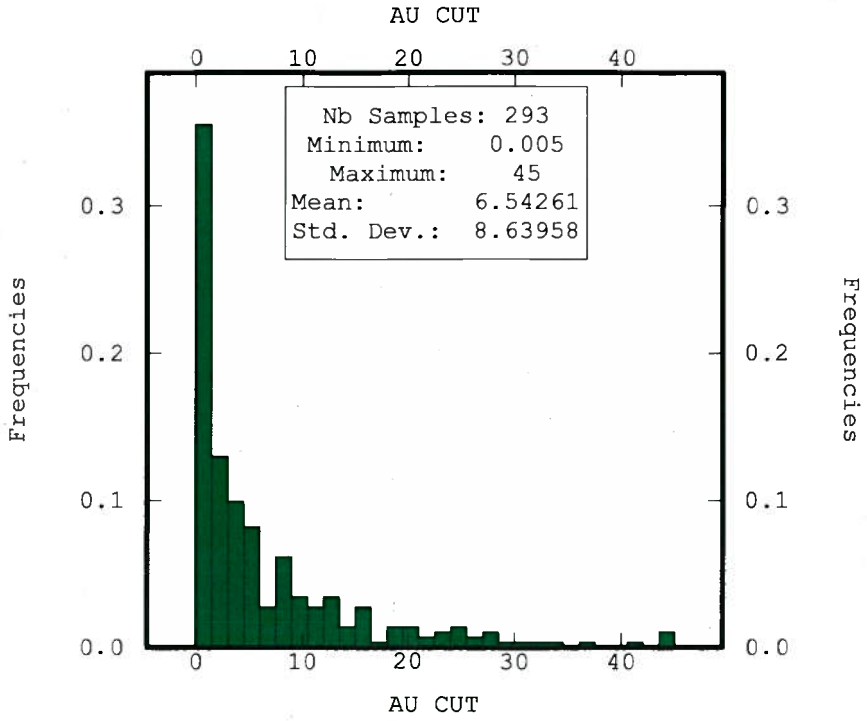

Figure 16. Histogram of gold grades for the Oriental Stockworks domain.

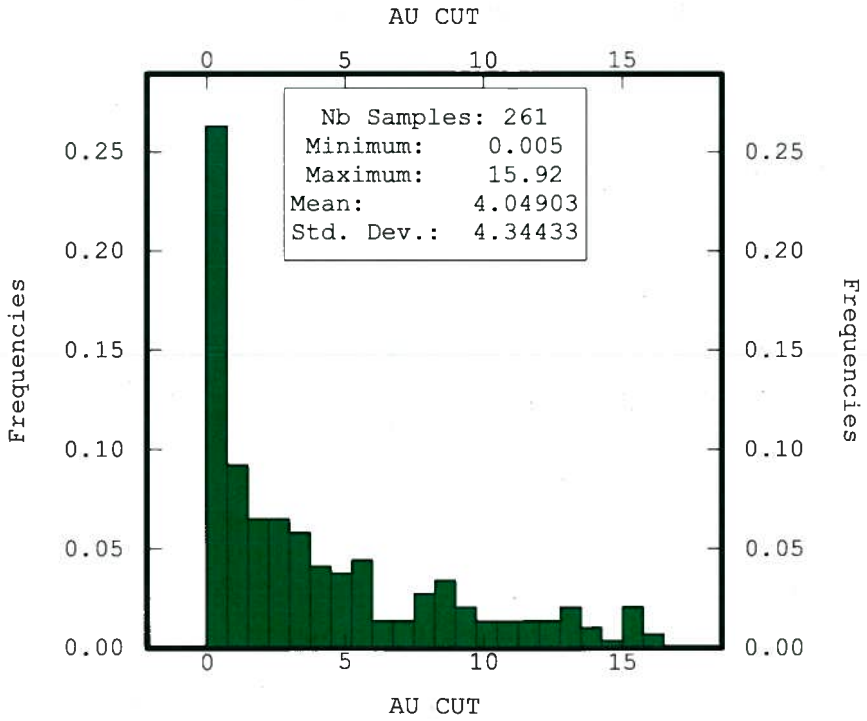

Figure 17. Histogram of gold grades for the Oriental Stockworks domain with Au < 15.9 g/t.

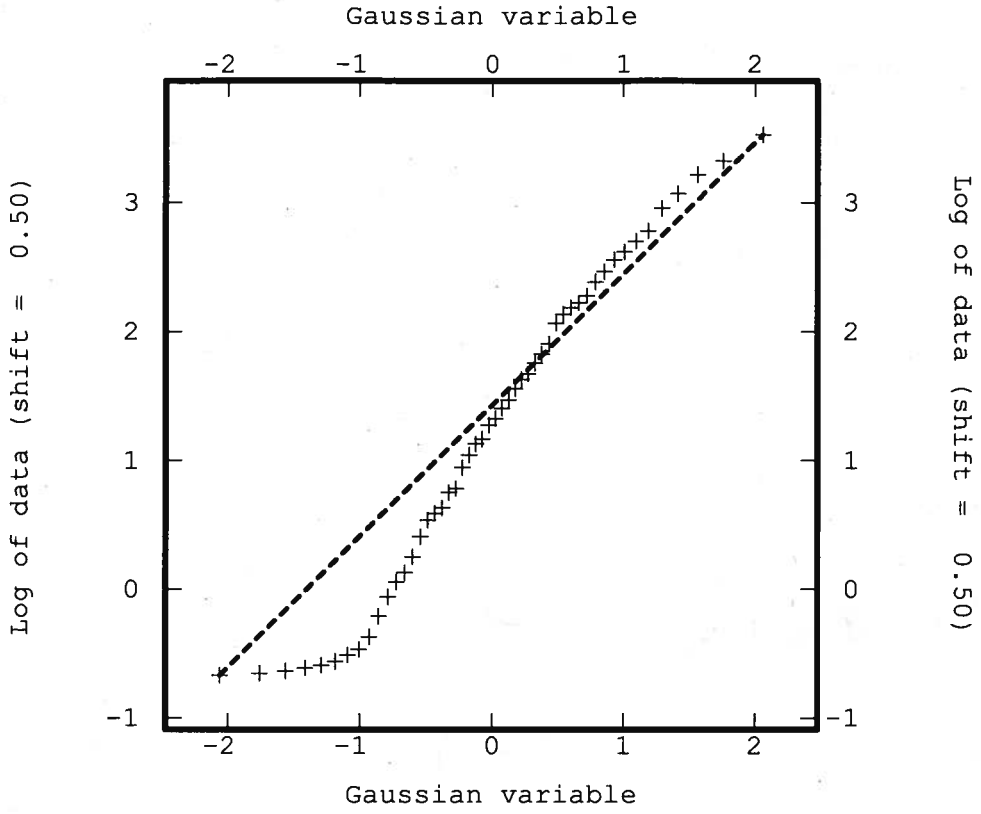

Figure 18. Log Q-Q plot of gold grades for the Oriental Stockworks domain.

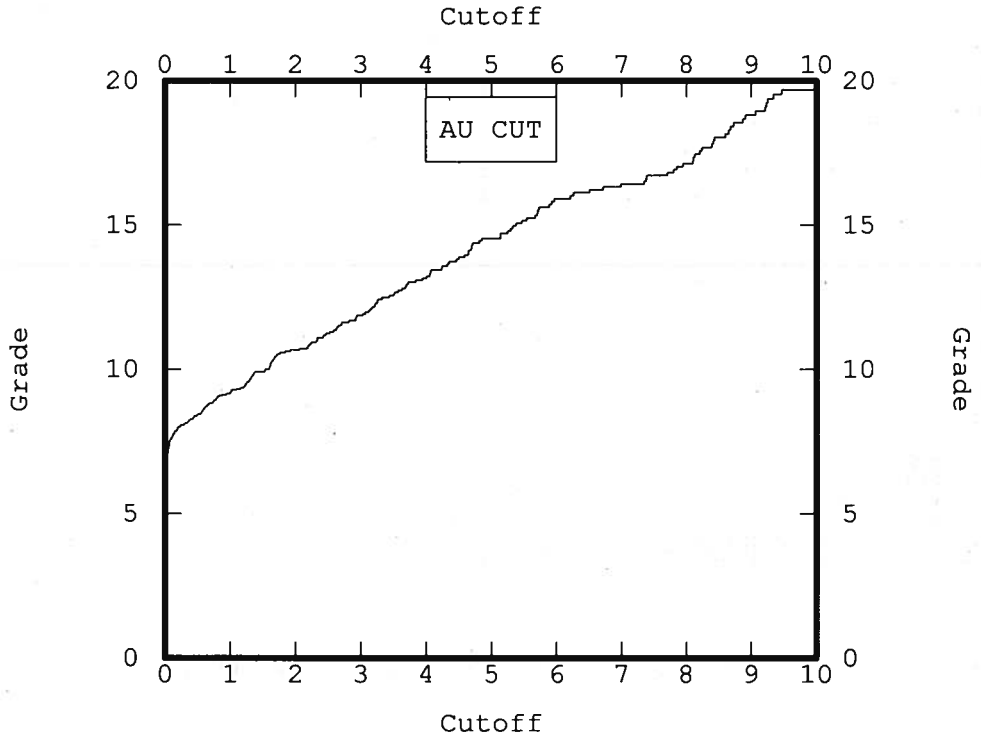

Figure 19. Cutoff versus grade for the Oriental Stockworks domain.

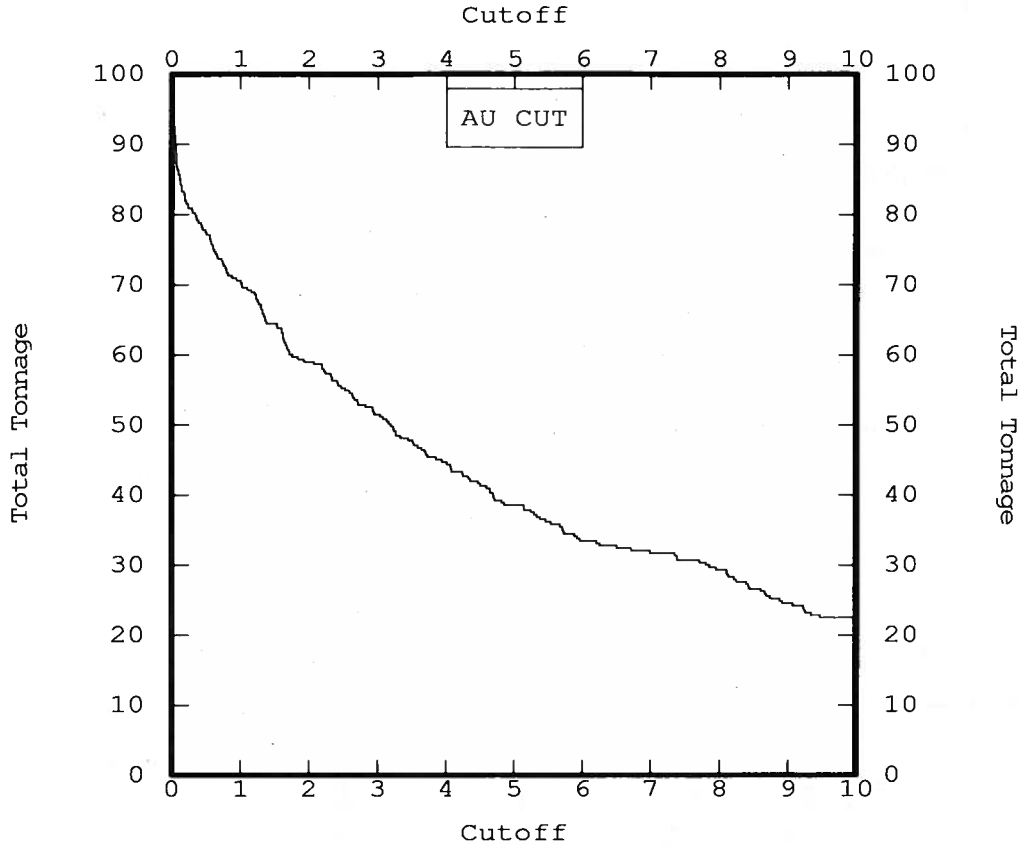

Figure 20. Cutoff versus tonnage for the Oriental Stockworks domain.

4.4 Moving Window Statistics of Domains

Figure 21 represents a longitudinal plot of the average Au grade (with Au > 0.3 g/t) for all domains combined. Figure 22 to Figure 25 are longitudinal plots of the average Au grade per domain. Figure 26 is a longitudinal plot of the number of samples with Au greater than 15 g/t. Each block within these plots represents the average Au grade of the 1m composites that fall within each block.

These plots show that there are continuous areas of high and low grade within shoots. This lack of stationarity within domains will reduce the quality of variograms and the quality of the final estimate. This will be worse in areas of less drilling were there will be an undesirable amount of smoothing.

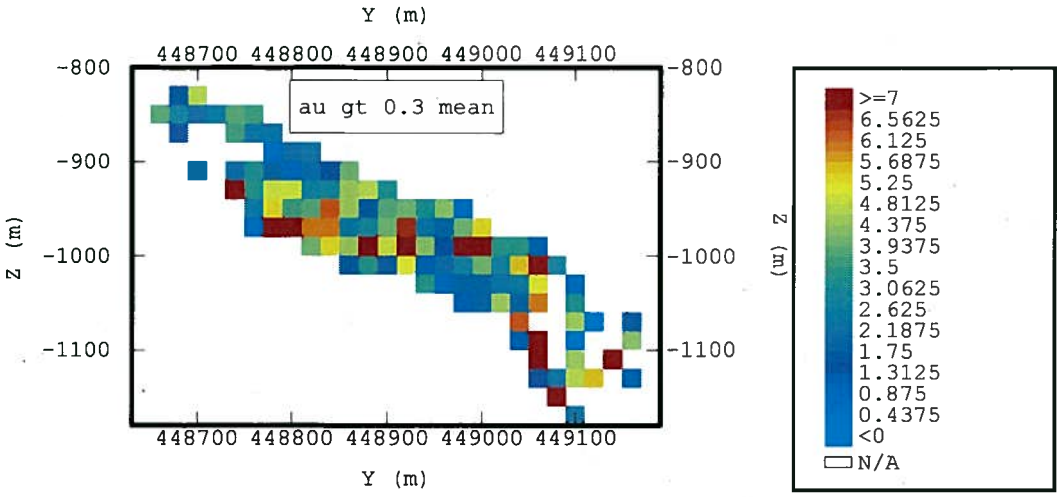

Figure 21. Longitudtional plot of mean Au for all domains combined with Au greater than 0.3.

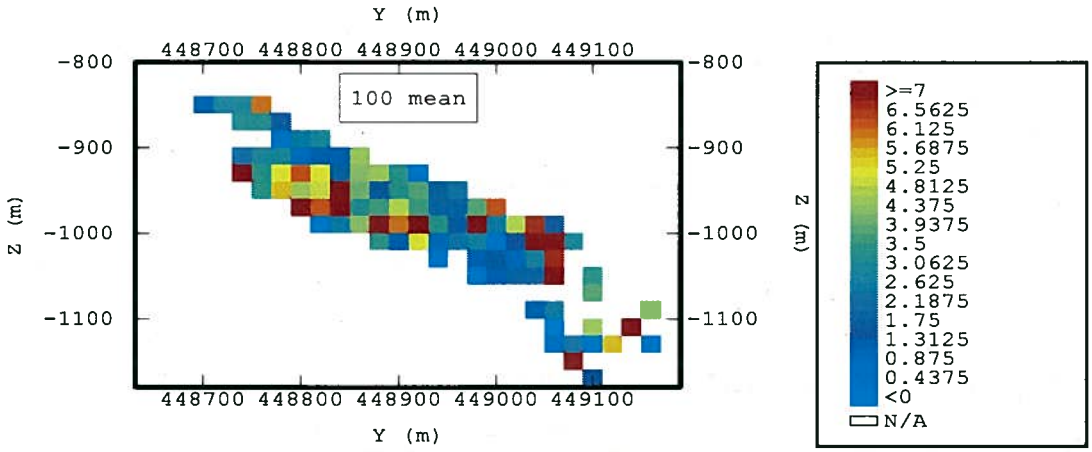

Figure 22. Longitudtional plot of mean Au for the Central Lode domain.

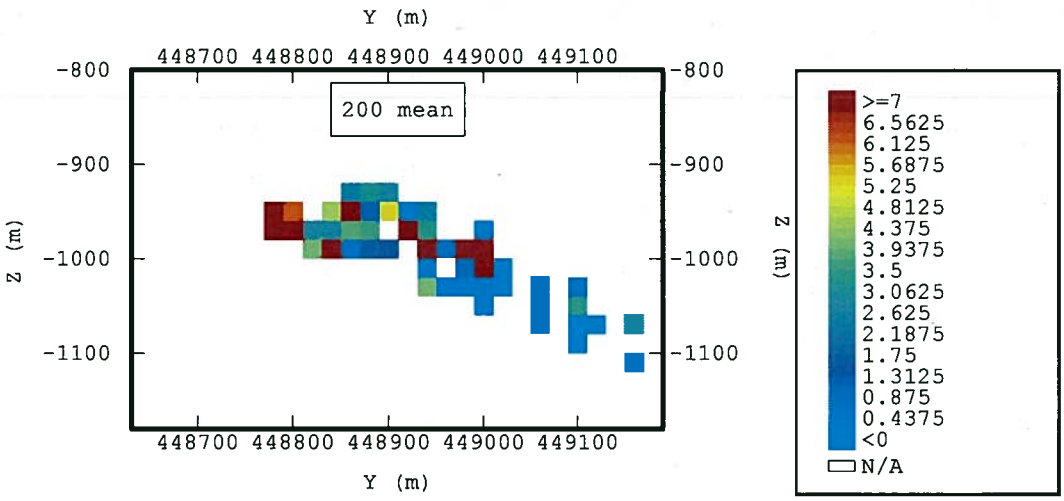

Figure 23. Longitudtional plot of mean Au for the Dukes domain.

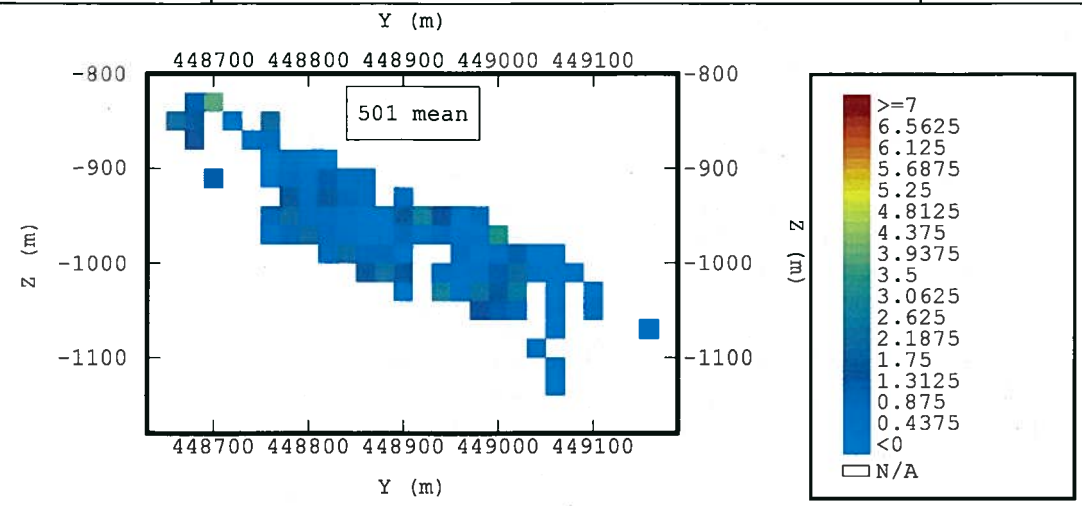

Figure 24. Longitudtional plot of mean Au for the Weak Volcanogenic domain.

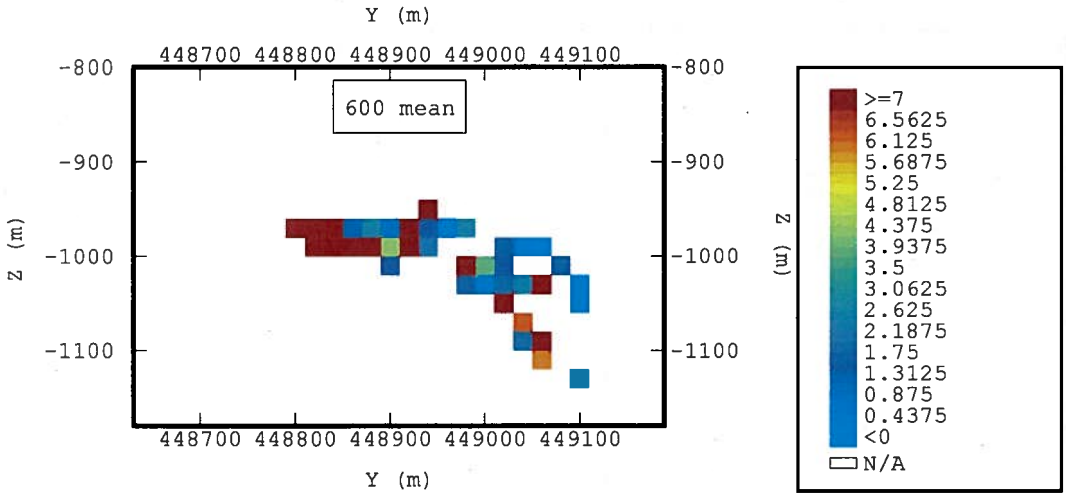

Figure 25. Longitudtional plot of mean Au for the Oriental Stockworks domain

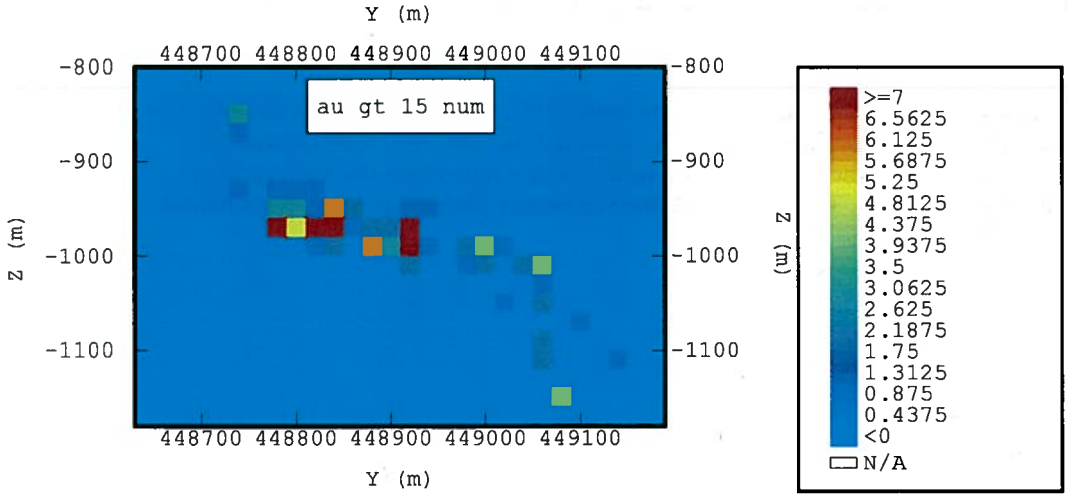

Figure 26. Longitudtional plot of the number of samples with Au gt 15 for all the combined domains.

4.5 Variography

The data from the variography includes all the information from the Upper South Fault area even though we are only interested in the model below the 1004 level. All the data was utilised in an effort to obtain improved variograms due to the closer spaced data above the 1004 level.

The orientations for variogram fans were generated in the plane of average strike and dip of each domain. Numerous lag separations, angular tolerances and slicing heights were trialled to produce the best variograms. The directional variograms (both regular and guassian transformed) are unstable and difficult to interpret with minor changes in parameters significantly changing directions of continuity. The northerly plunge to the mineralisation expected from geological observations could not be confirmed by variography. For these reasons an omnidirectional variogram in the plane of the lodes and a variogram perpendicular to this plane are modelled (Table 8). Approximately ¾ of the variability occurs in the first 5m's. This will result in a high degree of smoothing in the estimate.

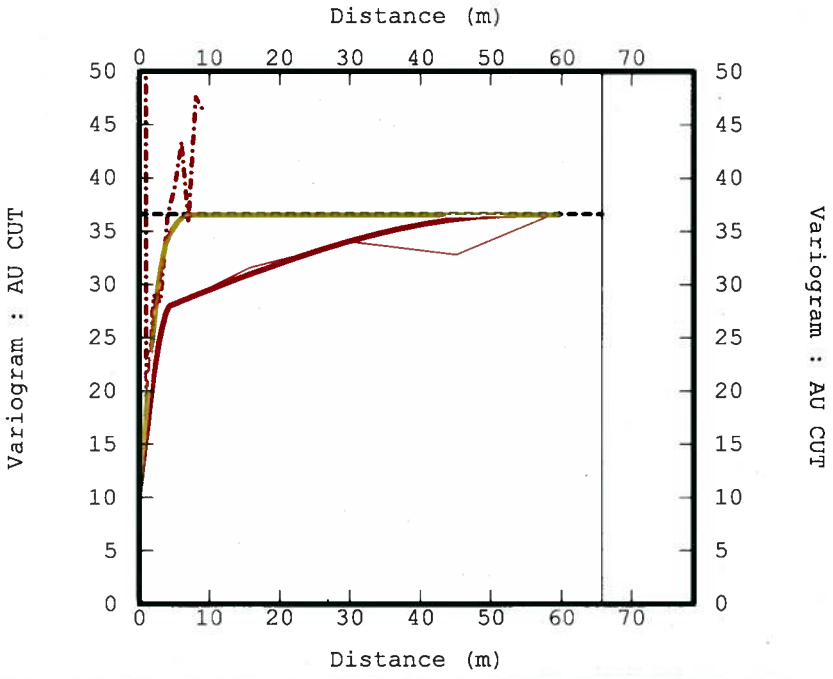

Figure 27. Directional variograms and variogram models of Au for the Central domain. The variogram fan is generated in a plane formed by rotating East -60° clockwise around RL and then -70° (down) around the East axis. The solid red line represents the variogram in the plane of the mineralisation. The dotted red line represents the variogram perpendicular to the plane of mineralisation.

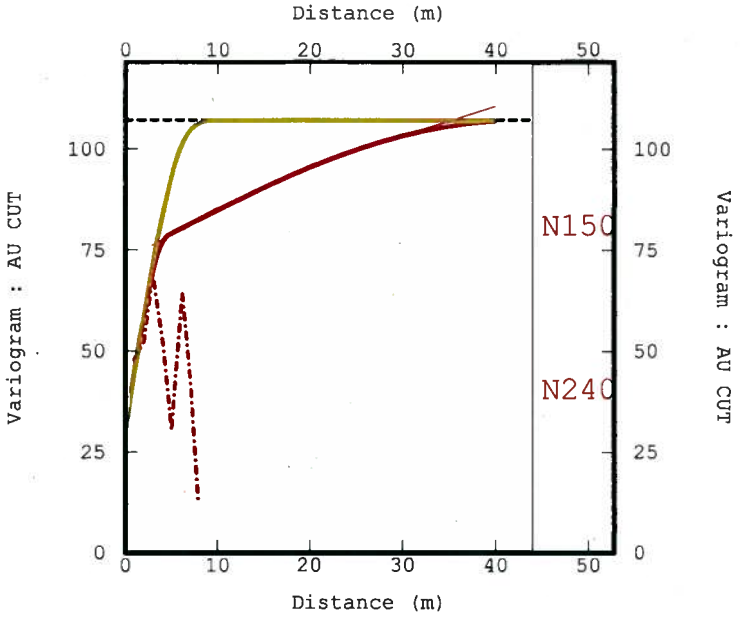

Figure 28. Directional variograms and variogram models of Au for the Dukes domain. The variogram fan is generated in a plane formed by rotating East -60° clockwise around RL and then -70° (down) around the East axis. The solid red line represents the variogram in the plane of the mineralisation. The dotted red line represents the variogram perpendicular to the plane of mineralisation.

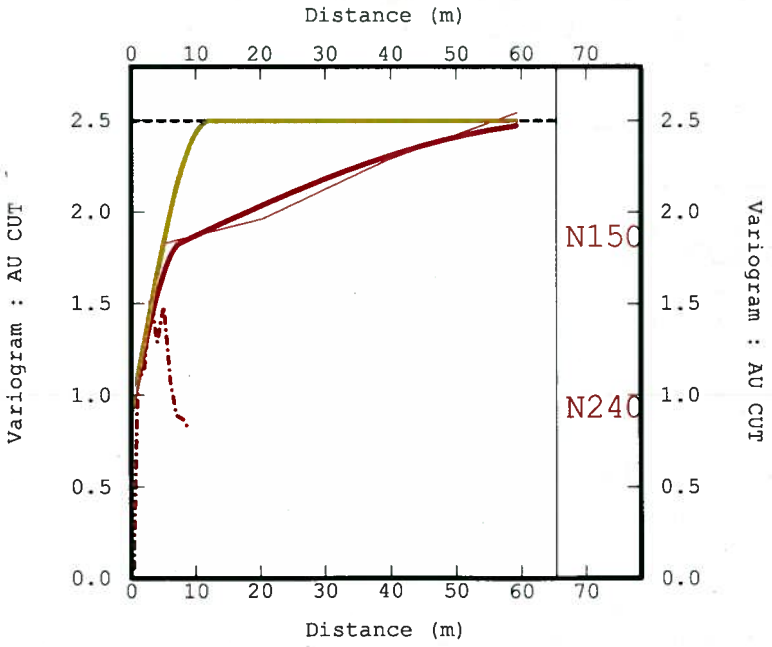

Figure 29. Directional variograms and variogram models of Au for the Weak Volcanogenic domain. The variogram fan is generated in a plane formed by rotating East -60° clockwise around RL and then -70° (down) around the East axis. The solid red line represents the variogram in the plane of the mineralisation. The dotted red line represents the variogram perpendicular to the plane of mineralisation.

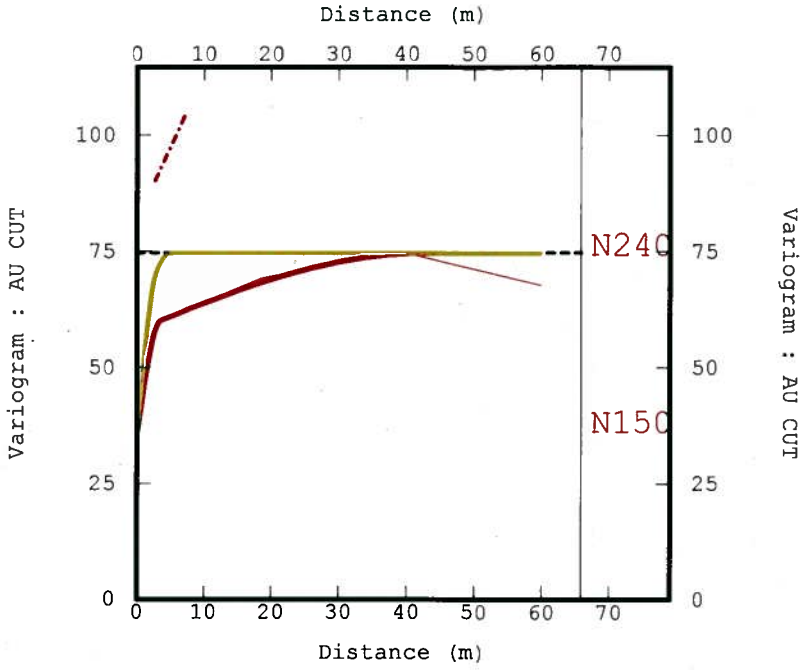

Figure 30. Directional variograms and variogram models of Au for the Oriental Stockworks domain. The variogram fan is generated in a plane formed by rotating East -60° clockwise around RL and then -70° (down) around the East axis. The solid red line represents the variogram in the plane of the mineralisation. The dotted red line represents the variogram perpendicular to the plane of mineralisation.

Table 8. Variogram model parameters from variograms based on the full Upper South Fault dataset.

Domain	Nugget	Sill 1	Range in plane of mineralisation	Range Across Strike	Sill 2	Range in plane of mineralisation	Range Across Strike
100	10.2	16.6	4.5	4	9.7	54	7
200	30	43	4.5	4	34	42	9
501	0.9	0.8	7.5	12	0.795	70	12
600	35	23	3.5	3	16.5	42	5

4.6 Block size selection and search ellipse optimisation

Immediately below the 1004 level the drill grid at best is approximately 20*20m in the plane of mineralisation. The remainder below the 1004 level is drilled at approximately 40*40m. A block size of 20*20*2m was selected.

To optimise the search ellipse for the volcanogenic domains Isatis was utilised. Rather than selecting representative blocks for test work, Isatis calculates the search ellipse parameters for all blocks and averages of these parameters are used to optimise the search strategy. All data above and below the 1004 level was utilised to optimise the search strategy. It was envisaged that the optimal interpolation would be imported back into Minesight and thus all data was utilised in search ellipse. Insufficient time was available to do a further 2 search optimisations based on the higher and lower drill densities below the 1004 level.

The search ellipse sizes selected were slightly smaller than optimal. This was to lessen the averaging of grades from distant samples at or near the variogram sill. This averaging is extreme in Upper South Fault because there is a lack of stationarity and the variogram is very short range. This solution is not optimal and further domaining is required.

Table 9. Test kriging results utilised to optimise the kriging search ellipses for the volcanogenic domains. Octant searching is used. The bold text refers to the optimal search tested.

	Mean	Std	Max positive weights	Slope	# of blocks	# samples
Central	4.31	2.31	1.00	0.57	965	20
Search is 200*200*50m	**4.39**	**2.18**	**1.00**	**0.62**	**965**	**28**
	4.45	2.11	1.01	0.66	965	36
	4.48	2.06	1.01	0.68	965	44
	4.51	2.03	1.02	0.70	965	52
	4.51	2.01	1.02	0.72	965	60
	4.52	1.99	1.02	0.73	965	68
Central	4.31	2.33	1.00	0.57	965	20
Search is 200*200*50m	4.41	2.19	1.00	0.62	965	28
	4.48	2.11	1.01	0.66	965	36
	4.52	2.07	1.01	0.68	965	44
Central Search is 60*60*25	4.43	2.14	1.01	0.66	957	44
Dukes	5.24	4.67	1.00	0.47	632	20
Search is 200*200*50m	5.17	4.48	1.00	0.50	632	28
	5.11	**4.39**	**1.00**	**0.53**	**632**	**36**
	5.13	4.33	1.01	0.54	632	44
	5.21	4.29	1.01	0.56	632	52
	5.27	4.26	1.02	0.57	632	60
Weak Volcanogenic Search is 200*200*50m	0.80	2.35	1.04	0.72	1443	36
Search is 250*250*50m	0.81	2.35	1.01	0.69	1446	28
Search is 350*350*100m	**0.80**	**2.35**	**1.01**	**0.68**	**1448**	**28**
Oriental Stockworks	5.38	3.58	1.00	0.34	285	20
Search is 200*200*50m	5.41	3.36	1.00	0.37	285	28
	5.25	3.24	1.00	0.40	285	36
	5.17	3.17	1.00	0.41	285	44
	5.08	3.11	1.00	0.43	285	52
	4.97	**3.07**	**1.00**	**0.44**	**285**	**60**
	5.00	3.03	1.01	0.45	285	68

5.0 Bulk Density

The composite lengths (and minimum composite lengths) and top cuts for Au used in this section are identical to those used in interpolation. Mean densities per domain have been obtained by removing extreme values from histograms of specific gravity (Table 10). Bottom and top cuts were 2 and 4 respectively. This reduced the number of samples with SG from 4671 to 4612.

The different specific gravity of the domains (Dukes and Oriental stockworks versus the Central, Weak volcanogenic and waste domains) are considered to be significant and reflect the known mineralogy's of the domains.

Table 10. Default densities utilised in the resource estimate.

DOMAIN	DRY DENSITY (t/m^3)	Number of samples
Central	2.77	1669
Dukes	2.94	366
Volcanogenic - weakly mineralised	2.81	1624
Oriental Stockworks	2.94	267
Waste	2.80	N/A

6.0 Block Model

6.1 Creation and validation of the block model

Table 11 contains the parameters utilised to create the rotated Minsight block model. The block size and model rotation were checked before moving onto domain coding. Domain wireframes were utilised to code the percentage of each block within each domain (Table 12). This was validated on screen to insure correct coding.

Table 11. Block model parameters.

Model Limits and Parameters :	Origin	Rotation		Range (metres)	Number Of Cells	Parent
Easting : (along strike)	658215.19	1	329	520	26	20
Northing : (down dip)	897770	2	0	980	49	20
RL : (perpendicular to lode)	-1270	3	70	150	75	2

Table 12. Block model domain coding.

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Kriging standard deviation
Central Lode	CGEO=100	C%	CAUD	CSG	CSLRD
Dukes Lode	BGEO=200	B%	BAUD	BSG	BSLRD
Oriental Stockworks	SGEO=600	S%	SAUD	SSG	SSLRD
Weak Volcanogenic	VGEO=501	V%	VAUD	VSG	WSLRD
Waste was not coded					

6.3 Estimation methodology

Ordinary kriging is used to interpolate Au for all the volcanogenic domains. The parameter files used for interpolation are outlined in Table 13.Blocks were all estimated in the first pass. Waste was not modelled and thus does not have a default Au grade. All boundaries were treated as hard during interpolation.

Table 13. Parameter files used for resource estimation.

Domain	Variogram Parameter File	Run File	# blocks estimated
Central	100.vp	Run624.100	965
Dukes	200.vp	Run624.200	632
Weak Volcanogenic	501.vp	Run624.501	1448
Oriental Stockworks	600.vp	Run624.600	285

6.4 Model Validation

6.4.1 Visual Validation

The model was validated by visually comparing the diamond drilling sample grades for each element to block grades on sections and in 3D. Due to smoothing, individual block grades are not similar to individual holes. Despite this the comparison between composites and the blocks was considered appropriate for all domains.

6.4.2 Statistical Comparison Between the Block Model and the 1m Composites

Table 14 is a comparison of the kriged Au to the declustered Au from 1m composites. Declustered estimates of Au have been obtained for each domain by changing the declustering block size. The declustered grades do not closely reflect the kriged estimates. Despite this both the Isatis and Minsight estimates are very similar (the Minsight estimate is reported and the Isatis estimate was not imported into Minesight). They are not exactly the same because Isatis cannot exactly duplicate the way Minsight searches for samples. The estimates for Isatis and Minsight are valid, the difference between the estimates and the declustered grades relates to the extreme smoothing of the kriging due to the very short range variability (75% of variability within 5m). In addition a reliable declustered grade is difficult to obtain due to the extreme clustering in the data with declustering cell sizes significantly changing the declustered grades.

Table 14. Comparison of block model Au to declustered Au (1m composites).

Declustering cell size. In plane * across strike	Mean Au - Central	Mean Au - Dukes	Mean Au - Oriental Stockworks	Mean Au - Weak Volcanogenic
15*500	3.99	6.76	5.92	
20*500	4.03	5.95	5.48	
25*500	4.09	5.54	5.57	
30*500	4.17	5.27	5.88	
35*500	4.16	5.24	5.85	
40*500	4.13	5.00	5.97	
15*2	4.38	7.06	6.40	
20*2	4.34	6.82	6.13	
25*2	4.26	6.45	5.88	
30*2	4.30	6.26	5.85	
35*2	4.30	6.21	5.85	
40*2	4.23	5.86	5.92	
15*4	4.35	6.96	6.22	
20*4	4.25	6.60	5.96	
25*4	4.21	6.09	5.92	
30*4	4.23	5.84	5.98	
35*4	4.25	5.74	5.96	
40*4	4.20	5.51	6.14	
Clustered composites	4.43	7.23	0.81	6.54
Isatis	4.39	5.11	4.97	0.80
Minesight	4.28	5.17	5.02	0.79

7.0 RESOURCE CLASSIFICATION AND REPORTING

It was not within the scope of this report to do significant work on QA/QC for gold assays. The QA/QC is an integral part of resource classification in the JORC code. The resource classification described below assumes that the QA/QC is of a high standard.

None of the resource is defined with sufficient confidence to classify as Measured. Utilising the kriging slope of regression, geological judgement and the drill hole geometry wireframes were created around the areas considered to meet Indicated resource status. This classification was performed in long section view with only complete blocks being classified across the full width of a domain. As a guide, blocks with a slope of regression of 0.7 or greater were considered as Indicated however depending on the abovementioned factors blocks with lower slopes of regression are also included. The Weak volcanogenic domain contains no ore and was not classified. The resource estimate is tabulated in Table 15 and Table 16.

Table 15. Resource at a 4g/t cutoff below the 1004 level (Derived from Minesight detailed reserves panel).

Domain	Resource Classification	Tonnes	Au g/t	Ounces
Central Lode	Indicated	63,500	5.6	11,470
Dukes Lode	Indicated	10,200	5.2	1,710
Oriental Stockworks	Indicated	4,400	4.6	650
Central Lode	Inferred	63,200	5.8	11,820
Dukes Lode	Inferred	5,600	4.8	870
Oriental Stockworks	Inferred	40,900	5.4	7,120
Central Lode	Inferred + Indicated	126,700	5.7	23,260
Dukes Lode	Inferred + Indicated	15,900	5.1	2,620
Oriental Stockworks	Inferred + Indicated	45,300	5.3	7,750
Total	Indicated	78,100	5.5	13,860
Total	Inferred	109,700	5.6	19,820

Table 16. Resource at a 4g/t cutoff before mining i.e. above and below the 1004 level (Derived from Minesight detailed reserves panel).

Domain	Resource Classification	Tonnes	Au g/t	Ounces
Central Lode	All classifications	349,900	6.1	68,850
Dukes Lode	All classifications	137,800	7.8	34,670
Oriental Stockworks	All classifications	75,500	6.6	16,070

APPENDIX 1 DRILLHOLES USED IN THIS RESOURCE ESTIMATE–BHID, FROM AND GEOCODE

MD2063A	242.000	501.000
MD2063A	268.650	501.000
MD2063A	269.800	200.000
MD2063A	360.000	
MD2124	233.600	100.000
MD2124	239.700	100.000
MD2124	270.000	100.000
MD2124	273.000	501.000
MD2124	276.300	100.000
MD2124	280.850	501.000
MD2124	294.000	200.000
MD2124	300.050	200.000
MD2124	351.100	
MD2262	131.000	501.000
MD2262	150.400	501.000
MD2262	230.800	
MD2263	129.800	200.000
MD2263	132.000	501.000
MD2263	144.000	501.000
MD2263	180.500	
MD2270	211.400	
MD2282	157.100	
MD2283	125.100	200.000
MD2283	129.000	501.000
MD2283	169.300	
MD2449	177.700	200.000
MD2449	181.800	501.000
MD2449	198.800	
MD2450	146.300	200.000
MD2450	147.000	200.000
MD2450	150.050	100.000
MD2450	156.000	501.000
MD2450	176.000	100.000
MD2450	177.000	100.000
MD2450	215.000	
MD2451	153.000	501.000
MD2451	260.500	
MD2452	155.800	200.000
MD2452	161.550	100.000
MD2452	164.600	100.000
MD2452	181.000	501.000
MD2452	185.000	100.000
MD2452	200.400	
MD2453A	187.600	501.000
MD2453A	192.200	501.000
MD2453A	202.000	100.000
MD2453A	215.000	501.000
MD2453A	221.700	501.000
MD2453A	294.800	
MD2454	153.150	100.000
MD2454	165.000	100.000
MD2454	180.000	

MD2457	169.650	200.000
MD2457	170.100	100.000
MD2457	175.000	501.000
MD2457	176.500	100.000
MD2457	193.000	501.000
MD2457	195.000	501.000
MD2457	213.000	
MD2459	168.300	100.000
MD2459	171.750	501.000
MD2459	184.900	
MD2461	175.000	100.000
MD2461	201.000	
MD2463	214.500	
MD2612	199.000	100.000
MD2612	218.000	100.000
MD2612	227.900	501.000
MD2612	237.100	501.000
MD2612	282.000	
MD2623	239.950	100.000
MD2623	241.650	501.000
MD2623	348.000	
MD2637	233.000	200.000
MD2637	237.200	501.000
MD2637	240.000	501.000
MD2637	262.600	403.000
MD2637	265.000	403.000
MD2637	278.300	600.000
MD2637	283.900	403.000
MD2637	297.000	403.000
MD2637	297.900	100.000
MD2637	330.400	
MD2643	201.100	501.000
MD2643	201.900	501.000
MD2643	220.900	403.000
MD2643	222.000	403.000
MD2643	243.400	501.000
MD2643	245.700	403.000
MD2643	248.900	501.000
MD2643	250.000	501.000
MD2643	476.700	
MD2737	265.900	200.000
MD2737	270.200	501.000
MD2737	273.500	501.000
MD2737	292.300	
MD2738	231.850	200.000
MD2738	234.000	501.000
MD2738	241.000	501.000
MD2738	255.400	403.000
MD2738	271.000	403.000
MD2738	271.600	600.000
MD2738	278.100	403.000
MD2738	288.450	100.000
MD2738	301.300	100.000
MD2738	306.500	501.000
MD2738	319.400	
MD2779	186.700	200.000
MD2779	190.000	100.000
MD2779	200.000	100.000
MD2779	211.850	501.000
MD2779	215.000	100.000
MD2779	218.600	100.000
MD2779	237.000	100.000

MD2779 239.100 501.000
MD2779 282.600

MD2831 211.250 200.000
MD2831 215.000 100.000
MD2831 219.000 100.000
MD2831 249.000 100.000
MD2831 302.000

MD2918 181.750 200.000
MD2918 189.000 501.000
MD2918 191.000 100.000
MD2918 210.000 100.000
MD2918 247.000

MD2922 190.900 200.000
MD2922 191.900 100.000
MD2922 195.050 501.000
MD2922 199.900 501.000
MD2922 200.300 100.000
MD2922 253.300

MD2923 214.300 200.000
MD2923 215.200 100.000
MD2923 219.250 100.000
MD2923 249.200 100.000
MD2923 265.500

MD2931A 206.050 200.000
MD2931A 214.000 501.000
MD2931A 223.000 501.000
MD2931A 237.600

MD2932 267.700 200.000
MD2932 270.450 501.000
MD2932 277.100 403.000
MD2932 294.800

MD2932W1 234.850 200.000
MD2932W1 236.900 501.000
MD2932W1 247.000 600.000
MD2932W1 258.300 600.000
MD2932W1 262.100 403.000
MD2932W1 263.650 100.000
MD2932W1 274.800

MD2933A 248.350 200.000
MD2933A 261.800 501.000
MD2933A 275.200 403.000
MD2933A 276.000 403.000
MD2933A 284.050 600.000
MD2933A 292.000 100.000
MD2933A 306.000 100.000
MD2933A 329.800

MD2996 104.700

MD2997 88.800 501.000
MD2997 92.500 501.000
MD2997 110.450 100.000
MD2997 122.100

MD2999 125.400

MD3000 120.400

MD3004 91.800

MD3005A 73.000 501.000
MD3005A 109.000

MD3009 71.650 100.000

MD3009 74.300 501.000
MD3009 80.000 100.000
MD3009 135.800

MD3013A 94.100

MD3064 586.500 403.000
MD3064 602.000 403.000
MD3064 607.100 100.000
MD3064 641.500

MD3064W1 590.000 403.000
MD3064W1 604.800 100.000
MD3064W1 629.500

MD3065 577.950 200.000
MD3065 579.100 403.000
MD3065 605.000 403.000
MD3065 610.250 100.000
MD3065 653.000

MD3097 164.700 100.000
MD3097 184.500

MD3098 182.700 200.000
MD3098 186.000 600.000
MD3098 192.000 600.000
MD3098 194.950 403.000
MD3098 200.850 600.000
MD3098 204.000 100.000
MD3098 257.800

MD3099 185.950 501.000
MD3099 186.350 200.000
MD3099 187.850 501.000
MD3099 192.000 501.000
MD3099 199.200 403.000
MD3099 200.200 600.000
MD3099 212.650 100.000
MD3099 222.150 100.000
MD3099 237.500

MD3100 252.400 200.000
MD3100 258.300 501.000
MD3100 262.500 403.000
MD3100 270.000 403.000
MD3100 337.000

MD3101 230.000 501.000
MD3101 241.400 100.000
MD3101 245.550 100.000
MD3101 258.300

MD3102 245.100 200.000
MD3102 246.100 200.000
MD3102 251.400 501.000
MD3102 266.000 403.000
MD3102 268.750 501.000
MD3102 276.000 501.000
MD3102 277.250 403.000
MD3102 307.000 403.000
MD3102 327.500 501.000
MD3102 330.600 100.000
MD3102 337.050 100.000
MD3102 349.300

MD3103 250.600 200.000
MD3103 253.600 501.000
MD3103 263.500 403.000
MD3103 274.000 403.000
MD3103 304.000 403.000
MD3103 334.000 403.000

MD3103	341.400	600.000
MD3103	345.150	501.000
MD3103	356.000	100.000
MD3103	366.100	
MD3104	288.700	200.000
MD3104	293.500	403.000
MD3104	306.000	403.000
MD3104	336.000	403.000
MD3104	375.550	600.000
MD3104	378.150	501.000
MD3104	392.000	100.000
MD3104	396.000	100.000
MD3104	436.700	
MD3115	558.700	200.000
MD3115	560.400	501.000
MD3115	564.700	403.000
MD3115	591.000	403.000
MD3115	593.000	100.000
MD3115	620.300	
MD3137	162.150	200.000
MD3137	165.100	200.000
MD3137	166.200	501.000
MD3137	184.000	600.000
MD3137	189.900	100.000
MD3137	194.950	100.000
MD3137	220.400	
MD3138	198.900	200.000
MD3138	202.000	501.000
MD3138	211.750	403.000
MD3138	223.000	403.000
MD3138	234.150	100.000
MD3138	281.800	
MD3139	161.000	200.000
MD3139	164.550	200.000
MD3139	166.350	501.000
MD3139	191.700	
MD3140	171.550	200.000
MD3140	174.000	501.000
MD3140	194.650	403.000
MD3140	195.800	403.000
MD3140	207.350	600.000
MD3140	209.900	403.000
MD3140	216.150	100.000
MD3140	229.000	
MD3141	213.230	200.000
MD3141	216.800	501.000
MD3141	227.500	403.000
MD3141	228.300	403.000
MD3141	248.100	501.000
MD3141	252.100	100.000
MD3141	258.600	100.000
MD3141	275.700	
MD3142	192.850	200.000
MD3142	194.000	501.000
MD3142	202.600	403.000
MD3142	204.650	100.000
MD3142	242.800	
MD3144	201.850	200.000
MD3144	204.000	501.000
MD3144	212.950	403.000
MD3144	230.000	403.000
MD3144	241.000	100.000
MD3144	263.600	

MD3145	121.600	200.000
MD3145	122.300	100.000
MD3145	128.550	501.000
MD3145	138.050	100.000
MD3145	141.950	100.000
MD3145	143.200	501.000
MD3145	175.300	
MD3147	222.900	403.000
MD3147	225.000	100.000
MD3147	230.000	100.000
MD3147	245.800	
MD3148	237.150	501.000
MD3148	245.000	600.000
MD3148	248.600	100.000
MD3148	268.900	
MD3149	172.000	200.000
MD3149	175.000	501.000
MD3149	186.900	403.000
MD3149	196.000	403.000
MD3149	204.850	600.000
MD3149	209.900	403.000
MD3149	221.300	100.000
MD3149	226.000	100.000
MD3149	232.200	
MD3150	195.500	200.000
MD3150	199.000	501.000
MD3150	210.200	403.000
MD3150	222.000	403.000
MD3150	223.850	600.000
MD3150	224.900	403.000
MD3150	232.000	501.000
MD3150	243.000	
MD3174	222.300	200.000
MD3174	226.550	501.000
MD3174	227.900	403.000
MD3174	233.000	501.000
MD3174	236.750	403.000
MD3174	242.700	403.000
MD3174	262.700	600.000
MD3174	265.400	501.000
MD3174	272.600	501.000
MD3174	275.550	100.000
MD3174	284.900	
MD3178	212.200	501.000
MD3178	234.000	403.000
MD3178	236.000	403.000
MD3178	243.900	100.000
MD3178	266.500	
MD3179	177.150	200.000
MD3179	182.200	501.000
MD3179	183.650	501.000
MD3179	188.850	100.000
MD3179	206.400	
MD3245	23.000	501.000
MD3245	27.300	100.000
MD3245	30.200	100.000
MD3245	46.700	501.000
MD3245	50.200	100.000
MD3245	53.200	501.000
MD3245	54.000	200.000
MD3245	69.600	
MD3246	5.600	100.000

MD3246 30.250 100.000
MD3246 40.000 501.000
MD3246 45.000 100.000
MD3246 48.700 200.000
MD3246 60.800

MD3247 1.050 100.000
MD3247 36.200

MD3248 0.000 100.000
MD3248 37.200

MD3250 13.710 501.000
MD3250 57.500

MD3251 5.950 100.000
MD3251 21.000 501.000
MD3251 30.250 501.000
MD3251 35.400 200.000
MD3251 55.500

MD3252 8.100 100.000
MD3252 15.250 501.000
MD3252 23.600 200.000
MD3252 48.200

MD3253 11.900 501.000
MD3253 16.000 100.000
MD3253 23.800 501.000
MD3253 30.000 501.000
MD3253 53.800 200.000
MD3253 69.100

MD3254 12.500 100.000
MD3254 21.100 501.000
MD3254 30.400 501.000
MD3254 44.950 200.000
MD3254 52.300 501.000
MD3254 56.200 200.000
MD3254 64.500

MD3282 29.450 100.000
MD3282 42.050 200.000
MD3282 68.100

MD3311 89.300 501.000
MD3311 89.850 501.000
MD3311 119.900 501.000
MD3311 125.700 200.000
MD3311 139.200

MD3312AW1 100.700 501.000
MD3312AW1 118.000 501.000
MD3312AW1 145.200

MD3313 90.500 100.000
MD3313 93.900 501.000
MD3313 112.800 200.000
MD3313 124.500

MD3314 92.950 100.000
MD3314 97.950 501.000
MD3314 115.450 200.000
MD3314 128.400

MD3315 87.300 100.000
MD3315 89.000 501.000
MD3315 89.700 501.000
MD3315 105.000 100.000
MD3315 105.900 200.000
MD3315 113.600

MD3316	97.100	100.000
MD3316	100.000	501.000
MD3316	109.700	100.000
MD3316	110.200	501.000
MD3316	116.500	200.000
MD3316	152.700	
MD3317	88.100	100.000
MD3317	91.500	501.000
MD3317	97.800	501.000
MD3317	100.700	100.000
MD3317	103.600	501.000
MD3317	108.300	200.000
MD3317	119.800	
MD3318	96.000	100.000
MD3318	100.000	501.000
MD3318	104.000	100.000
MD3318	106.550	501.000
MD3318	114.700	200.000
MD3318	157.100	
MD3319	99.200	100.000
MD3319	103.000	501.000
MD3319	104.000	501.000
MD3319	105.000	100.000
MD3319	108.000	501.000
MD3319	119.000	200.000
MD3319	157.000	
MD3320	85.800	100.000
MD3320	89.850	100.000
MD3320	95.050	501.000
MD3320	96.700	100.000
MD3320	101.100	200.000
MD3320	113.800	
MD3321	109.550	100.000
MD3321	122.200	501.000
MD3321	133.000	200.000
MD3321	137.000	200.000
MD3321	170.700	
MD3322	91.650	501.000
MD3322	95.000	100.000
MD3322	99.000	501.000
MD3322	102.000	100.000
MD3322	107.000	501.000
MD3322	110.000	200.000
MD3322	128.500	
MD3323	85.150	100.000
MD3323	87.450	100.000
MD3323	96.100	501.000
MD3323	98.300	100.000
MD3323	100.450	200.000
MD3323	113.800	
MD3324	58.900	100.000
MD3324	63.700	501.000
MD3324	79.000	100.000
MD3324	86.000	100.000
MD3324	104.700	
MD3325	94.500	501.000
MD3325	95.500	100.000
MD3325	104.750	501.000
MD3325	107.000	100.000
MD3325	109.000	100.000
MD3325	110.000	200.000
MD3325	119.100	

MD3326	99.800	501.000
MD3326	103.400	100.000
MD3326	111.000	501.000
MD3326	113.100	100.000
MD3326	118.400	501.000
MD3326	120.300	200.000
MD3326	131.400	
MD3327	99.700	501.000
MD3327	100.200	100.000
MD3327	103.900	501.000
MD3327	107.300	100.000
MD3327	110.200	100.000
MD3327	111.200	501.000
MD3327	118.500	200.000
MD3327	134.500	
MD3328	95.900	100.000
MD3328	110.200	501.000
MD3328	111.000	100.000
MD3328	118.000	200.000
MD3328	123.000	
MD3329	79.100	501.000
MD3329	90.100	501.000
MD3329	109.400	
MD3330	115.350	100.000
MD3330	127.100	501.000
MD3330	136.350	501.000
MD3330	138.650	200.000
MD3330	174.500	
MD3331	105.550	501.000
MD3331	110.650	100.000
MD3331	114.000	100.000
MD3331	117.600	501.000
MD3331	122.000	100.000
MD3331	129.100	200.000
MD3331	140.000	
MD3332	135.150	501.000
MD3332	141.400	100.000
MD3332	149.900	100.000
MD3332	152.750	501.000
MD3332	157.000	200.000
MD3332	167.500	
MD3333	100.250	501.000
MD3333	109.150	100.000
MD3333	112.100	501.000
MD3333	112.850	100.000
MD3333	123.400	200.000
MD3333	139.100	
MD3334	141.000	501.000
MD3334	147.150	100.000
MD3334	149.900	100.000
MD3334	156.800	600.000
MD3334	160.000	501.000
MD3334	172.600	200.000
MD3334	182.700	
MD3335	137.200	501.000
MD3335	142.200	100.000
MD3335	150.000	100.000
MD3335	153.000	600.000
MD3335	156.000	501.000
MD3335	167.400	200.000
MD3335	176.200	
MD3336	113.500	501.000

MD3336 121.100 100.000
MD3336 133.350 501.000
MD3336 137.400 200.000
MD3336 149.500

MD3338 123.100 501.000
MD3338 129.600 100.000
MD3338 135.000 100.000
MD3338 138.700 501.000
MD3338 151.000 200.000
MD3338 163.600

MD3339 114.100 501.000
MD3339 117.900 100.000
MD3339 132.000 501.000
MD3339 145.300 200.000
MD3339 157.900

MD3340 91.200 501.000
MD3340 93.600 100.000
MD3340 99.500 501.000
MD3340 101.000 100.000
MD3340 104.900 200.000
MD3340 116.100

MD3341 120.550 501.000
MD3341 123.800 100.000
MD3341 126.500 100.000
MD3341 137.400 501.000
MD3341 153.050 200.000
MD3341 156.200 200.000
MD3341 172.300

MD3342 99.500 501.000
MD3342 100.500 100.000
MD3342 107.500 501.000
MD3342 124.700 200.000
MD3342 134.500

MD3361 51.500 100.000
MD3361 66.800

MD3368 52.150 501.000
MD3368 74.300

MD3369 54.800 200.000
MD3369 57.150 501.000
MD3369 59.000 100.000
MD3369 73.000 100.000
MD3369 95.600

MD3371 87.700

MD3373 51.850 200.000
MD3373 54.400 501.000
MD3373 63.000 100.000
MD3373 74.000 100.000
MD3373 100.600

MD3374 69.300 200.000
MD3374 71.000 501.000
MD3374 73.000 501.000
MD3374 77.000 100.000
MD3374 115.700

MD3375 53.200 200.000
MD3375 56.500 501.000
MD3375 69.800 403.000
MD3375 70.900 100.000
MD3375 77.000 100.000
MD3375 111.100

MD3376	47.100	200.000
MD3376	48.150	501.000
MD3376	49.800	100.000
MD3376	80.300	
MD3395	90.400	200.000
MD3395	97.300	501.000
MD3395	102.400	100.000
MD3395	114.600	100.000
MD3395	116.900	501.000
MD3395	129.300	
MD3396	71.700	200.000
MD3396	74.600	501.000
MD3396	82.500	100.000
MD3396	90.000	100.000
MD3396	111.000	
MD3397A	69.850	200.000
MD3397A	72.450	501.000
MD3397A	82.950	100.000
MD3397A	95.600	100.000
MD3397A	98.050	501.000
MD3397A	109.500	
MD3398	59.700	200.000
MD3398	61.800	100.000
MD3398	98.800	
MD3399	64.600	100.000
MD3399	87.500	
MD3400	60.600	200.000
MD3400	61.500	200.000
MD3400	62.600	100.000
MD3400	68.100	501.000
MD3400	69.000	100.000
MD3400	96.000	
MD3401	59.950	100.000
MD3401	63.100	501.000
MD3401	65.000	100.000
MD3401	84.200	
MD3402	56.300	200.000
MD3402	57.150	100.000
MD3402	60.000	100.000
MD3402	60.600	501.000
MD3402	61.500	100.000
MD3402	76.100	
MD3404	73.650	501.000
MD3404	74.750	100.000
MD3404	95.400	
MD3413	95.650	501.000
MD3413	165.000	
MD3424	86.500	100.000
MD3424	89.800	100.000
MD3424	104.000	
MD3485	31.900	501.000
MD3485	42.000	100.000
MD3485	74.400	
MD3486	40.200	501.000
MD3486	41.400	50.000
MD3486	42.400	501.000
MD3486	63.900	
MD3487	24.650	100.000

MD3487 29.750 100.000
MD3487 61.600

MD3488 20.000 100.000
MD3488 59.400

MD3489 30.100 100.000
MD3489 37.400

MD3490 0.000 501.000
MD3490 11.100 100.000
MD3490 12.900 501.000
MD3490 21.800 100.000
MD3490 26.000 501.000
MD3490 29.900 501.000
MD3490 36.400

MD3491 0.000 100.000
MD3491 3.050 501.000
MD3491 7.000 100.000
MD3491 25.900

MD3492 0.000 100.000
MD3492 1.200 501.000
MD3492 3.500 100.000
MD3492 15.400

MD3493 3.150 200.000
MD3493 6.000 100.000
MD3493 8.850 501.000
MD3493 17.200 100.000
MD3493 28.100 501.000
MD3493 29.600 501.000
MD3493 50.000

MD3494 5.150 200.000
MD3494 5.900 100.000
MD3494 8.300 501.000
MD3494 15.000 100.000
MD3494 17.450 501.000
MD3494 20.000

MD3495 22.200 100.000
MD3495 30.000

MD3496 10.200 200.000
MD3496 11.100 100.000
MD3496 17.900 501.000
MD3496 23.200 100.000
MD3496 29.600 100.000
MD3496 39.200 501.000
MD3496 60.500

MD3497 27.900 100.000
MD3497 28.350 501.000
MD3497 29.700 501.000
MD3497 35.150 100.000
MD3497 46.800

MD3498 29.600 501.000
MD3498 34.800 100.000
MD3498 36.700 501.000
MD3498 50.100

MD3499 34.200 100.000
MD3499 35.950 501.000
MD3499 45.300

MD3500 37.250 100.000
MD3500 37.500 501.000
MD3500 66.600

MD3593	42.000	100.000
MD3593	47.000	600.000
MD3593	56.300	
MD3594	42.600	100.000
MD3594	53.600	
MD3595	40.600	403.000
MD3595	44.800	403.000
MD3595	60.100	501.000
MD3595	66.700	501.000
MD3595	79.400	200.000
MD3595	89.600	
MD3596	37.600	501.000
MD3596	40.800	100.000
MD3596	44.900	600.000
MD3596	55.700	501.000
MD3596	59.900	501.000
MD3596	63.000	200.000
MD3596	131.400	
MD3597	29.700	100.000
MD3597	34.000	600.000
MD3597	39.700	403.000
MD3597	45.200	501.000
MD3597	51.700	200.000
MD3597	59.100	
MD3598	56.300	100.000
MD3598	60.600	100.000
MD3598	77.600	
MD3599	32.000	501.000
MD3599	37.600	600.000
MD3599	40.600	100.000
MD3599	56.400	
MD3601	34.500	501.000
MD3601	39.900	600.000
MD3601	42.700	403.000
MD3601	44.200	100.000
MD3601	56.300	
MD3602	37.900	501.000
MD3602	42.300	100.000
MD3602	54.400	
MD3686	0.000	501.000
MD3686	9.000	600.000
MD3686	11.900	100.000
MD3686	22.700	501.000
MD3686	30.000	
MD3687	0.000	501.000
MD3687	9.100	600.000
MD3687	20.500	100.000
MD3687	29.700	100.000
MD3687	39.600	
MD3688	0.000	501.000
MD3688	6.300	600.000
MD3688	9.100	100.000
MD3688	15.300	100.000
MD3688	16.300	501.000
MD3688	29.600	
MD3689	0.000	501.000
MD3689	18.000	403.000
MD3689	18.900	600.000
MD3689	25.500	100.000
MD3689	30.200	100.000

MD3689 49.600

MD3690 0.000 501.000
MD3690 8.600 100.000
MD3690 13.100 501.000
MD3690 21.600

MD3691 0.000 501.000
MD3691 11.080 600.000
MD3691 13.550 100.000
MD3691 43.800

MD3692 0.000 501.000
MD3692 15.000 403.000
MD3692 20.700 600.000
MD3692 22.700 100.000
MD3692 29.700 100.000
MD3692 45.100

MD3693 0.000 501.000
MD3693 5.800 403.000
MD3693 22.350 600.000
MD3693 28.700 100.000
MD3693 30.000 100.000
MD3693 53.000

MD3694 0.000 501.000
MD3694 2.600 403.000
MD3694 11.600 600.000
MD3694 13.000 100.000
MD3694 29.500

MD3695 0.000 501.000
MD3695 2.500 403.000
MD3695 7.700 100.000
MD3695 32.200

MD3696 0.000 501.000
MD3696 4.800 403.000
MD3696 18.900 600.000
MD3696 20.700 100.000
MD3696 29.900 100.000
MD3696 41.100

MD3697 0.000 600.000
MD3697 6.000 403.000
MD3697 14.600 600.000
MD3697 15.600 403.000
MD3697 17.400 600.000
MD3697 18.800 403.000
MD3697 22.700 600.000
MD3697 26.200 100.000
MD3697 30.000 100.000
MD3697 60.000

MD3698 0.000 600.000
MD3698 3.400 100.000
MD3698 20.100

MD3699 0.000 501.000
MD3699 10.300 100.000
MD3699 27.800

MD3700 0.000 100.000
MD3700 1.000 600.000
MD3700 4.800 501.000
MD3700 7.600 200.000
MD3700 20.200

MD3701 0.000 100.000
MD3701 0.900 501.000
MD3701 10.300 200.000

MD3701 20.000

MD3702 0.000 501.000
MD3702 11.100 200.000
MD3702 20.200

MD3704 54.600 100.000
MD3704 68.500

MD3705 54.300 600.000
MD3705 59.100 100.000
MD3705 60.100 100.000
MD3705 80.000

MD3745 0.000 501.000
MD3745 1.000 100.000
MD3745 36.000

MD3746 0.000 100.000
MD3746 28.500

MD3747 0.000 100.000
MD3747 26.600

MD3748 0.000 100.000
MD3748 0.600 501.000
MD3748 3.000 100.000
MD3748 24.400

MD3872 0.000 501.000
MD3872 7.000 600.000
MD3872 12.600 100.000
MD3872 16.000 501.000
MD3872 21.300

MD3873 0.000 501.000
MD3873 8.100 600.000
MD3873 11.800 100.000
MD3873 15.500 501.000
MD3873 20.000

MD3874 17.400

MD3875 0.000 501.000
MD3875 6.400 600.000
MD3875 12.700 100.000
MD3875 16.600 501.000
MD3875 20.700

MD3876 0.000 501.000
MD3876 6.150 600.000
MD3876 11.050 100.000
MD3876 15.800 501.000
MD3876 23.000

MD3877 0.000 501.000
MD3877 5.200 600.000
MD3877 11.950 100.000
MD3877 27.000

MD3878 0.000 501.000
MD3878 1.850 600.000
MD3878 5.400 100.000
MD3878 10.800 501.000
MD3878 15.800

MD3879 0.000 501.000
MD3879 3.500 600.000
MD3879 9.400 100.000
MD3879 15.000 100.000
MD3879 29.100

MD3880 0.000 600.000
MD3880 0.600 403.000
MD3880 4.300 501.000
MD3880 7.500 200.000
MD3880 15.900

MD3881 0.000 600.000
MD3881 5.900 501.000
MD3881 14.700 200.000
MD3881 19.500

MD3882 0.000 600.000
MD3882 4.400 100.000
MD3882 17.200

MD3883 0.000 600.000
MD3883 2.000 100.000
MD3883 12.700

MD3884 0.000 501.000
MD3884 1.000 600.000
MD3884 1.900 100.000
MD3884 10.200 501.000
MD3884 13.900

MD3885 0.000 501.000
MD3885 0.700 100.000
MD3885 10.000

MD3886 0.000 501.000
MD3886 4.500 600.000
MD3886 6.200 100.000
MD3886 17.300

MD3887 0.000 501.000
MD3887 2.500 600.000
MD3887 4.900 100.000
MD3887 12.200

MD3888 0.000 501.000
MD3888 5.400 600.000
MD3888 6.300 100.000
MD3888 11.800

MD3910 0.000 501.000
MD3910 2.000 403.000
MD3910 6.250 600.000
MD3910 8.300 100.000
MD3910 18.100

MD3911 0.000 501.000
MD3911 4.550 600.000
MD3911 6.300 100.000
MD3911 17.100

MD3912 43.500 200.000
MD3912 44.600 501.000
MD3912 52.350 403.000
MD3912 54.500 403.000
MD3912 59.900 100.000
MD3912 67.600

MD3913 51.200 200.000
MD3913 53.150 501.000
MD3913 61.100 403.000
MD3913 78.800 600.000
MD3913 82.000 100.000
MD3913 82.700 100.000
MD3913 92.200

MD3914 61.500 200.000
MD3914 63.750 501.000

MD3914	72.200	403.000
MD3914	87.400	403.000
MD3914	89.600	600.000
MD3914	93.150	100.000
MD3914	104.200	
MD3915	49.700	200.000
MD3915	53.050	501.000
MD3915	54.200	501.000
MD3915	62.000	403.000
MD3915	69.900	600.000
MD3915	71.100	100.000
MD3915	78.900	
MD3916	51.200	200.000
MD3916	53.400	501.000
MD3916	62.500	501.000
MD3916	63.600	403.000
MD3916	78.100	100.000
MD3916	89.500	
MD3919	55.300	200.000
MD3919	56.650	501.000
MD3919	61.300	403.000
MD3919	89.100	600.000
MD3919	91.300	600.000
MD3919	92.800	100.000
MD3919	102.300	
MD3920	58.300	200.000
MD3920	59.500	501.000
MD3920	64.000	403.000
MD3920	89.800	403.000
MD3920	91.600	600.000
MD3920	95.100	501.000
MD3920	97.000	100.000
MD3920	107.200	
MD3921	60.400	200.000
MD3921	64.450	501.000
MD3921	70.900	403.000
MD3921	87.400	600.000
MD3921	88.700	600.000
MD3921	92.750	100.000
MD3921	107.400	
MD3922	61.100	200.000
MD3922	62.450	501.000
MD3922	70.400	403.000
MD3922	95.100	403.000
MD3922	98.000	501.000
MD3922	107.700	100.000
MD3922	110.600	
MD3924	67.350	200.000
MD3924	75.800	501.000
MD3924	78.650	403.000
MD3924	91.500	403.000
MD3924	102.250	600.000
MD3924	112.200	100.000
MD3924	119.600	
MD3925	65.700	200.000
MD3925	66.600	501.000
MD3925	73.650	403.000
MD3925	98.800	403.000
MD3925	106.150	501.000
MD3925	115.000	100.000
MD3925	118.100	
MD4120	104.400	200.000
MD4120	107.250	501.000

MD4120	110.200	403.000
MD4120	119.100	501.000
MD4120	121.250	403.000
MD4120	122.200	403.000
MD4120	138.750	600.000
MD4120	139.600	100.000
MD4120	151.600	
MD4121	109.800	200.000
MD4121	113.000	501.000
MD4121	115.400	403.000
MD4121	121.000	403.000
MD4121	128.700	600.000
MD4121	134.600	100.000
MD4121	144.500	
MD4122	139.200	200.000
MD4122	140.000	403.000
MD4122	170.400	403.000
MD4122	186.000	600.000
MD4122	190.500	501.000
MD4122	196.200	100.000
MD4122	199.750	100.000
MD4122	224.400	
MD4123	123.300	200.000
MD4123	126.850	501.000
MD4123	129.300	403.000
MD4123	150.650	403.000
MD4123	171.100	600.000
MD4123	178.100	100.000
MD4123	181.000	100.000
MD4123	189.500	
MD4124	165.150	403.000
MD4124	184.850	403.000
MD4124	211.400	403.000
MD4124	230.300	100.000
MD4124	247.200	
MD4125	139.300	200.000
MD4125	140.350	200.000
MD4125	143.550	403.000
MD4125	183.800	403.000
MD4125	200.400	600.000
MD4125	204.100	100.000
MD4125	215.500	
MD4126	137.500	403.000
MD4126	167.300	403.000
MD4126	190.300	100.000
MD4126	210.600	
MD4178	116.900	200.000
MD4178	120.800	501.000
MD4178	124.100	403.000
MD4178	139.250	403.000
MD4178	160.500	100.000
MD4178	173.800	
MD4179	146.050	200.000
MD4179	147.250	403.000
MD4179	152.200	403.000
MD4179	182.200	403.000
MD4179	192.100	100.000
MD4179	206.200	

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX E

GG1 Resource Model

September 2005

Resource Summary Sheets

Stawell Sep 05 GG1 Resource Model

Model Name :	GG1 - Geological resource model
Prepared by :	Mine Geology Group - Stawell
Drillhole Data To:	August 2005
Development Survey date :	N/A No tonnes depleted.
Date Model Completed :	16th September 2005
Date Report Completed :	27th September 2005
Interpretations By :	Jane Murray
Resource Modelling and Grade Estimation By :	Chris De-Vitry
Report Title:	Stawell September 2005 GG1 Resource Model
Compiled by :	Chris De-Vitry
Approved By :	Vincent O'Brien Senior Mine Geologist
Audited by :	To be audited by QG
Accepted by :	Andrew McDougall Senior Mining Engineer - Operations
Report Distribution :	Geoff McDermott Vincent O'Brien Stawell site geologists Andrew McDougall

MODELLING ENVIRONMENT		
	Project Directory :	S:\Mining-Geology\Golden Gift 1\Minesight\Geology\Current Model
	Model Identification :	GG1
	Modelling Software :	Minesight and Isatis
	Operating System :	Windows XP
MODEL DESCRIPTION :	*(narrative, purpose, zones, relation to other models, etc):*	New geological interpretation whereby East and West Patron are subdivided by sub-horizontal faults. New data from recent diamond drilling above the 960mRL is incorporated in this model. Only south of 310 section is included in this model. North of 310 is mined out and for this area refer to the August 04 model. Grade and waste material modelled by geological domain (GEOCD). The reported resource has been classified into Indicated and Inferred resources (using wireframes selection). This model adjoins the GG1.5 and GG2 models which are above GG1.
GEOLOGY/ORE DESCRIPTION :	*(structure, stratigraphy, dip, strikes, size, ore type etc)*	The GG1 orebody contains a large internal Basalt separating two Volcanogenic packages, the East Patron and the West Patron. Above the Basalt nose the lodes join together in a stockwork zone which generally contains the best gold mineralisation. To the east of the lodes is vesicular Basalt and the South Fault, while to the west is the Mine Schist and discontinuous felsic porphyry bodies. Reinterpretation and recent drilling identified numerous flat faults which divide GG1 into 6 blocks, all offset bottom-block-east along these faults. The Shiraz Fault and Cabernet Fault define the base and top of GG1 respectively. Internal faults typically dip 0-20° north although one fault appears to dip at a similar angle (0-20°) to the south. Small scale dip reversals occur on these faults. The Pinot Fault forms the northern boundary of the orebody. GG1 remains open up-plunge to the south. For the current model, GG1 has only been reinterpreted/ remodelled south of Section 310N (45° Mine Grid) as the orebody is completely mined-out north of this section. North of Section 310N, East Patron was the weaker of the two lodes with significant mineralisation only found on the basalt contacts with a weak volcanogenic package in between. The tenor of East Patron has increased towards the southern end of GG1 whereas the West Patron limb has thinned and is largely faulted out. The West Patron waterloo persists to the south with significant width and patchy to high grade gold mineralisation. The mined-out northern end of West Patron had more continuous mineralisation both along strike and down dip but incorporated several sheared internal basalts. Several high grade stockwork zones focused above these internal basalts and a very large high grade zone was mined above the main basalt nose. Mineralisation was not consistent across strike with weakly mineralised volcanogenics separating the higher grades. Several flat north-east dipping thrust faults offset the orebodies for 1-2m in an east over west sense of direction.
ELEMENTS MODELLED :		Au
PROPERTIES MODELLED :		DENSITY (DEFAULTS)
GRID USED (Local, AMG etc)		AMG

DRILLHOLE DATA :	Directory :	S:\Mining-Geology\Golden Gift 1\Minesight\Geology\Current Model_msresources\Drillholes\July04		
Sample data extent				45° Mine Grid
Easting :	658550 mE	to	658900 mE	
Northing :	897760 mN	to	898060 mN	298 mN – 310mN
RL:	-870 RL	to	-1080 RL	
Drillhole Spacing:	15m East, 20m RL – Staggered pattern			
Raw drillhole data	Project-gg110.dat, gg111.ddh, gg112.ddh. Modal length of 1m. Flagged by GEOCOD using wireframes. Flagging was visually and statistically checked and when required wireframes were modified and the drillholes manually coded.			
Composited drillhole data	1m comps. assay – gg108.1m (Composites broken at GEOCD)			
STRUCTURAL INTERPRETATION :				
No. of Sections :	22			
Section Interval:	10m (45 rotated grid)			
Sectional Interpretation:	S:\Mining-Geology\Golden Gift 1\Minesight\Geology\Current Model_msresources\Geology\Dynamic Strings			
Comments :	Cross-sectional strings for stratigraphy validated in 1m plan slices. Strings are snapped to drillholes. A total of 120 diamond drill holes were used.			
WIREFRAMES :	No. of Wireframes :		a total of 30	
	6 fault blocks each with volcanogenic solids, basalt solids and mine schist solids. 8 Fault surfaces. 6 Wireframe solids defining indicated resources.			
Construction of the Wireframes :	In Minesight by Jane Murray			
Date Wireframes Completed :	September 2005			
SAMPLE CODING :	Flagged by GEOCD using wireframes. Flagging was visually and statistically checked and when required drillholes manually coded.			
Composited drillhole data	1m comps – gg108.1m (Composites broken at GEOCD)		5487 samples (Volcanogenic only)	

BLOCK MODEL PARAMETERS :

Model Limits and Parameters :	Origin		Rotation		Range (metres)	Number Of Cells	Cell Dimensions Parent	
Easting : (along strike)	658300		1	30	560	56	10	
Northing : (down dip)	897600		2	-60	450	30	15	
RL : (perpendicular to lode)	-1270		3	0	300	150	2	

GRADE INTERPOLATION :

Estimation Technique : Ordinary kriging for Mineralised volcanogenics.

Grade Estimation Files : Estimated in Isatis

Default Grades The low grade volcanogenic domains were assigned the average grade of the sample composites in these domains.

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Slope of Regression	.dxf filename	Inferred Resource	Indicated Resource
Mid weak volcanogenic	VGEO=502	V%	VAUD	VSG		MEWkp		
Upper weak volcanogenic	UGEO=500	U%	UAUD	USG		UEWkp		
Upper West Patron	WGEO=600	W%	WAUD	WSG	WSLRD	UWestp	WCAT=3	WCAT=1
Mid West Patron	XGEO=602	X%	XAUD	XSG	XSLRD	MWestp	N/A	XCAT=1
Lower West Patron	YGEO=604	Y%	YAUD	YSG	YSLRD	LWestp	N/A	YCAT=1
Upper East Patron	EGEO=606	E%	EAUD	ESG	ESLRD	UEestp	N/A	ECAT=1
Mid East Patron	FGEO=608	F%	FAUD	FSG	FSLRD	MEastp	N/A	FCAT=1
Lower East Patron	GGEO=610	G%	GAUD	GSG	GSLRD	LEastp	N/A	GCAT=1

DOMAIN	Default Au	DRY DENSITY (t/m^3)
Mine Schist	0.01	2.67
Basalt	0.01	2.79
Mineralised Volcanogenics	N/A	2.85
Mid weak volcanogenic	0.34	2.85
Upper weak volcanogenic	0.36	2.85

Resource at a 4g/t cutoff (Calculated from Minesight detailed reserves)

Domain	Description	Resource Classification	Tonnes	Grade	Grams	Ounces
600	Upper West Patron	Indicated	427,551	5.7	2,424,214	77,940
		Inferred	55,558	5.0	278,346	8,949
		Total Indicated and Inferred	483,109	5.6	2,702,560	86,889
602	Mid West Patron	Indicated	71,427	5.9	418,562	13,457
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	71,427	5.9	418,562	13,457
604	Lower West Patron	Indicated	33,200	6.5	216,132	6,949
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	33,200	6.5	216,132	6,949
	West Patron	Indicated	532,178	5.7	3,058,908	98,346
		Inferred	55,558	5.0	278,346	8,949
		Total Indicated and Inferred	587,736	5.7	3,337,254	107,295
606	Upper East Patron	Indicated	19,041	5.6	105,678	3,398
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	19,041	5.6	105,678	3,398
608	Mid East Patron	Indicated	105,481	6.8	715,161	22,993
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	105,481	6.8	715,161	22,993
610	Lower East Patron	Indicated	54,942	5.5	299,983	9,645
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	54,942	5.5	299,983	9,645
	East Patron	Indicated	179,464	6.2	1,120,822	36,035
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	179,464	6.2	1,120,822	36,035
	Total	Indicated	711,642	5.9	4,179,730	134,381
		Inferred	55,558	5.0	278,346	8,949
		Total Indicated and Inferred	767,200	5.8	4,458,076	143,330

Grade Wireframes

The Minesight subblocking routine was utilised to generate grade wireframes for 3, 4, and 7g/t cutoffs (see S:\Mining-Geology\Golden Gift 1\Minesight\Geology\Current Model_msresources\Block Model)

SUMMARY SHEETS HANDED TO ENGINEERS

Location - **S:\Mining-Geology\Golden Gift 1\Minesight\Geology\Current Model_msresources\Block Model\September 2005**
Model – gg115.jun
Coded Drillholes – gg111.ddh
Coded 1m composites - 1m_comps.ddh

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Slope of Regression	.dxf filename	Inferred Resource	Indicated Resource
Mid weak volcanogenic	VGEO=502	V%	VAUD	VSG		MEWkp		
Upper weak volcanogenic	UGEO=500	U%	UAUD	USG		UEWkp		
Upper West Patron	WGEO=600	W%	WAUD	WSG	WSLRD	UWestp	WCAT=3	WCAT=1
Mid West Patron	XGEO=602	X%	XAUD	XSG	XSLRD	MWestp	N/A	XCAT=1
Lower West Patron	YGEO=604	Y%	YAUD	YSG	YSLRD	LWestp	N/A	YCAT=1
Upper East Patron	EGEO=606	E%	EAUD	ESG	ESLRD	UEestp	N/A	ECAT=1
Mid East Patron	FGEO=608	F%	FAUD	FSG	FSLRD	MEastp	N/A	FCAT=1
Lower East Patron	GGEO=610	G%	GAUD	GSG	GSLRD	LEastp	N/A	GCAT=1

All else in the model is mullock

The main Panel in the sub-block routine should look like this

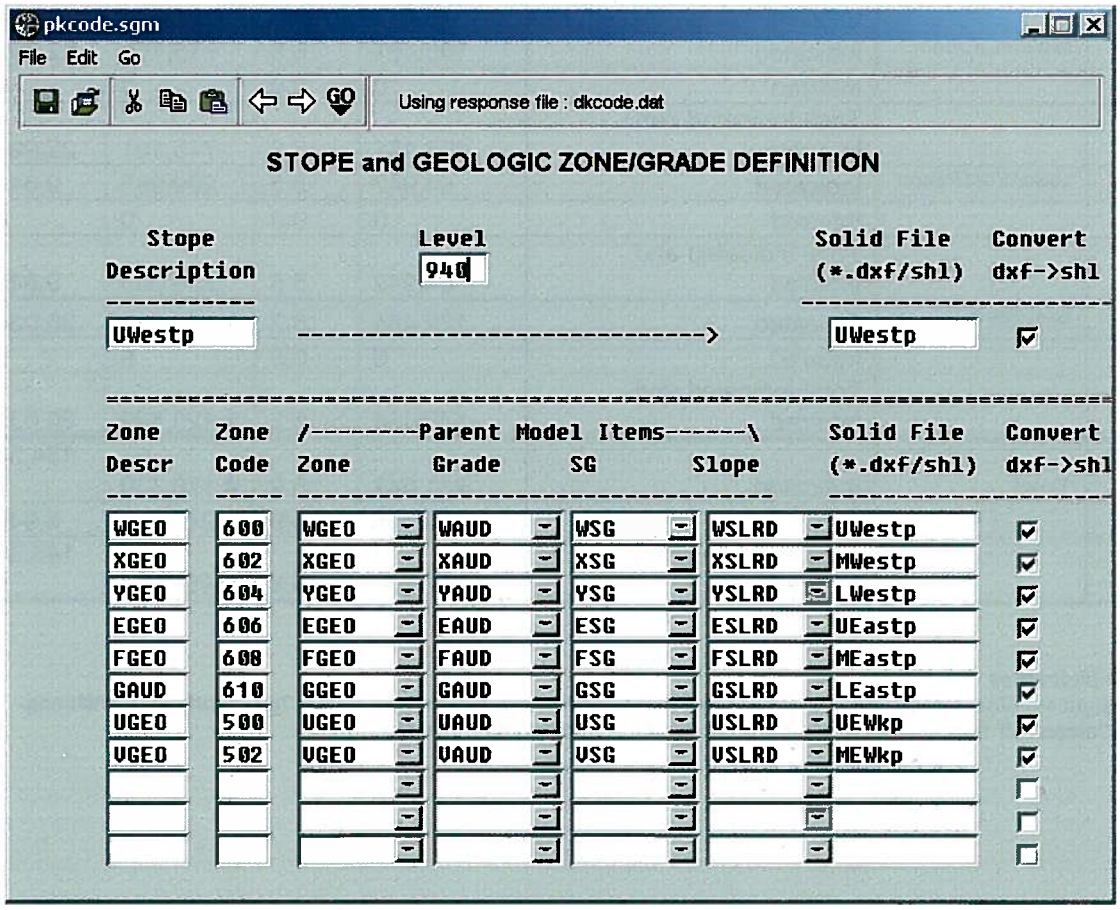

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Figure 13. Normal directional variograms of Au for domain 606. The variogram fan is generated in a plane of strike 125° and dip -55° to the East. The direction N305 is

the average variogram within the plane of the variogram. The variogram N35 is orthogonal to the abovementioned variogram fan. 26

Figure 14. Normal directional variograms of Au for domain 608. The variogram fan is generated in a plane of strike 125° and dip -55° to the East. The directions N125, N143 etc represent the strike of variograms within the abovementioned plane. The variogram N35 is orthogonal to the abovementioned variogram fan. 26

Figure 15. Normal directional variograms of Au for domain 610. The variogram fan is generated in a plane of strike 130° and dip -50° to the East. The directions N130, N140 etc represent the strike of variograms within the abovementioned plane. The variogram N40 is orthogonal to the abovementioned variogram fan. 27

1.0 EXECUTIVE SUMMARY

1.1 Introduction

The GG1 deposit is a part of the Golden Gift resource at Stawell. It is situated approximately 1000m below the surface and is bounded by the Cabernet, Pinot, Shiraz and South faults. The GG1.5 and GG2 resources lie directly above GG1 (Figure 1, Figure 2 and Figure 3). The August 04 resource model was dominated by East and West Patron uninterrupted by faulting. The current model only covers the area south of the 309.5 section. North of the 309.5 section is mined out. With new drilling and development information the modelled area of East and West Patron was divided into 7 fault blocks by sub-horizontal faults. The September 2005 model was generated from ordinary kriging with significant areas of close spaced drilling (15*20m staggered pattern). Data from 61 holes drilled since the June 2005 model run has been incorporated in the new model; the new drill hole data focused above the 960mRL. As presented in Table 1 numerous people have been involved in this project.

Table 1. Identification of the project team.

Resource	Role
Justin Cheyne	Drillhole data input
Carol Durant	Database management and validation of drillhole data
Catherine Smith	Logging
Lachlan Macdonald	Logging
Jane Murray	Mapping, logging and creation of geological domains. Writing the geological section of this report.
Chris De Vitry	Logging, resource model, preparation of this report
Vincent O'Brien	Project review and approval

Table 2. Resource comparison between this model and the same part of the previous model. Resources are above a 4g/t cutoff and include the total Indicated and Inferred resource.

Domain	Tonnes	Grade	Ounces	% change metal
Previous Model West Patron	320,077	6.2	64,225	
Current Model West Patron	587,736	5.7	107,295	167%
Previous Model East Patron	234,211	6.1	45,755	
Current Model East Patron	179,464	6.2	36,035	79%
Total previous model	554,288	6.2	109,980	
Current model	767,200	5.8	143,330	130%

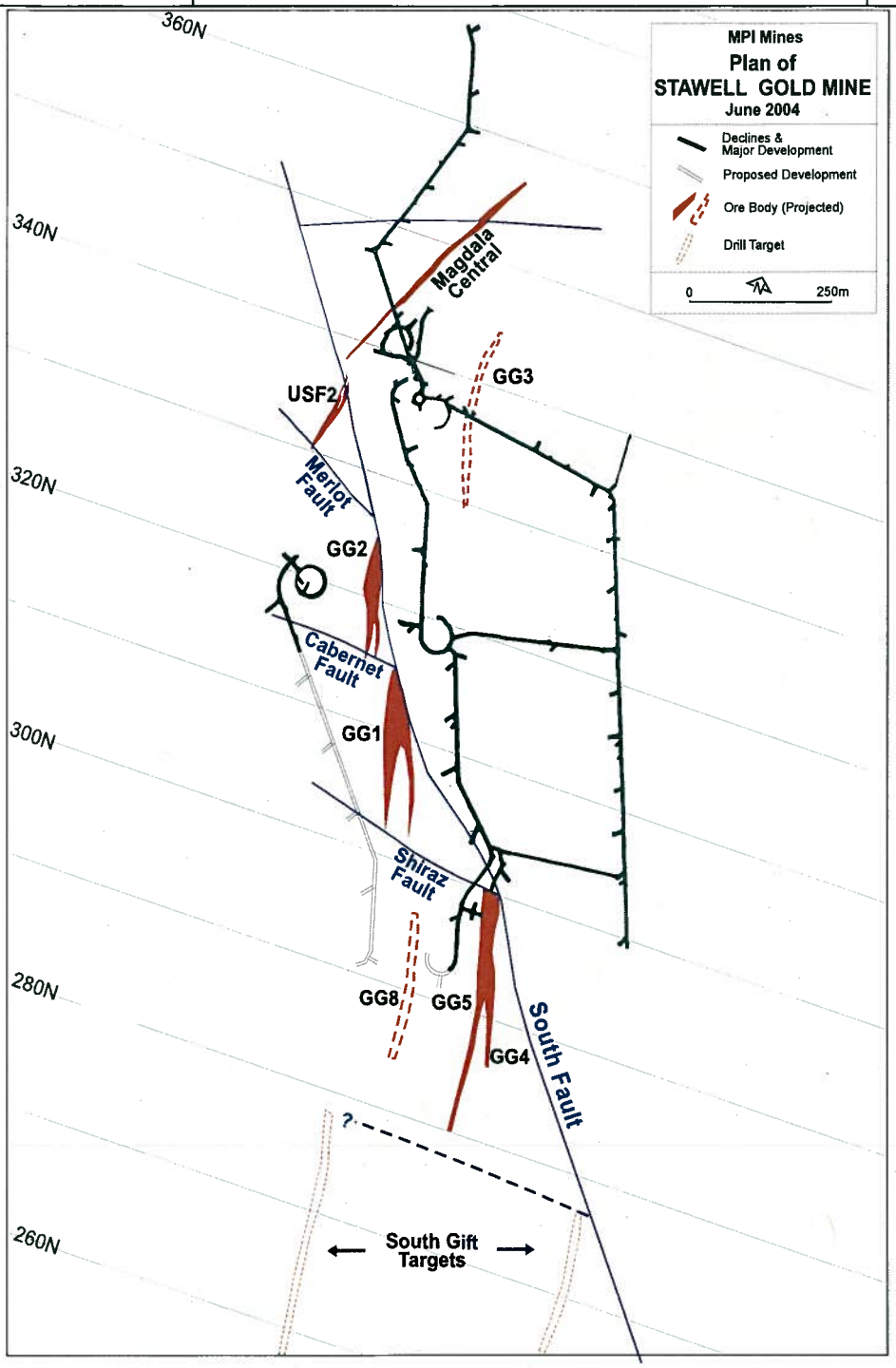


Figure 1. Plan view of the Golden Gift orebodies.

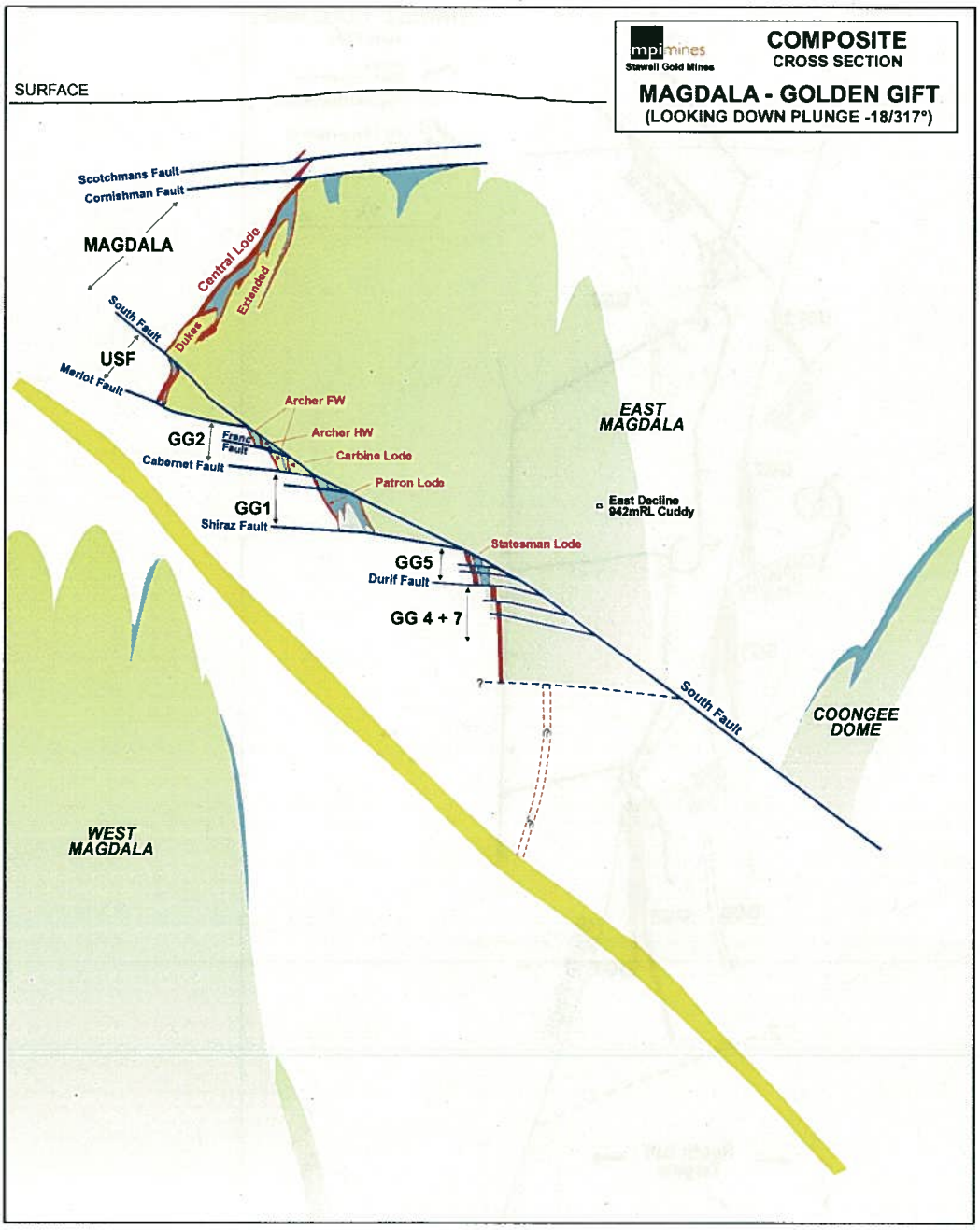


Figure 2. Down plunge cross section of the Magdala and Golden Gift orebodies.

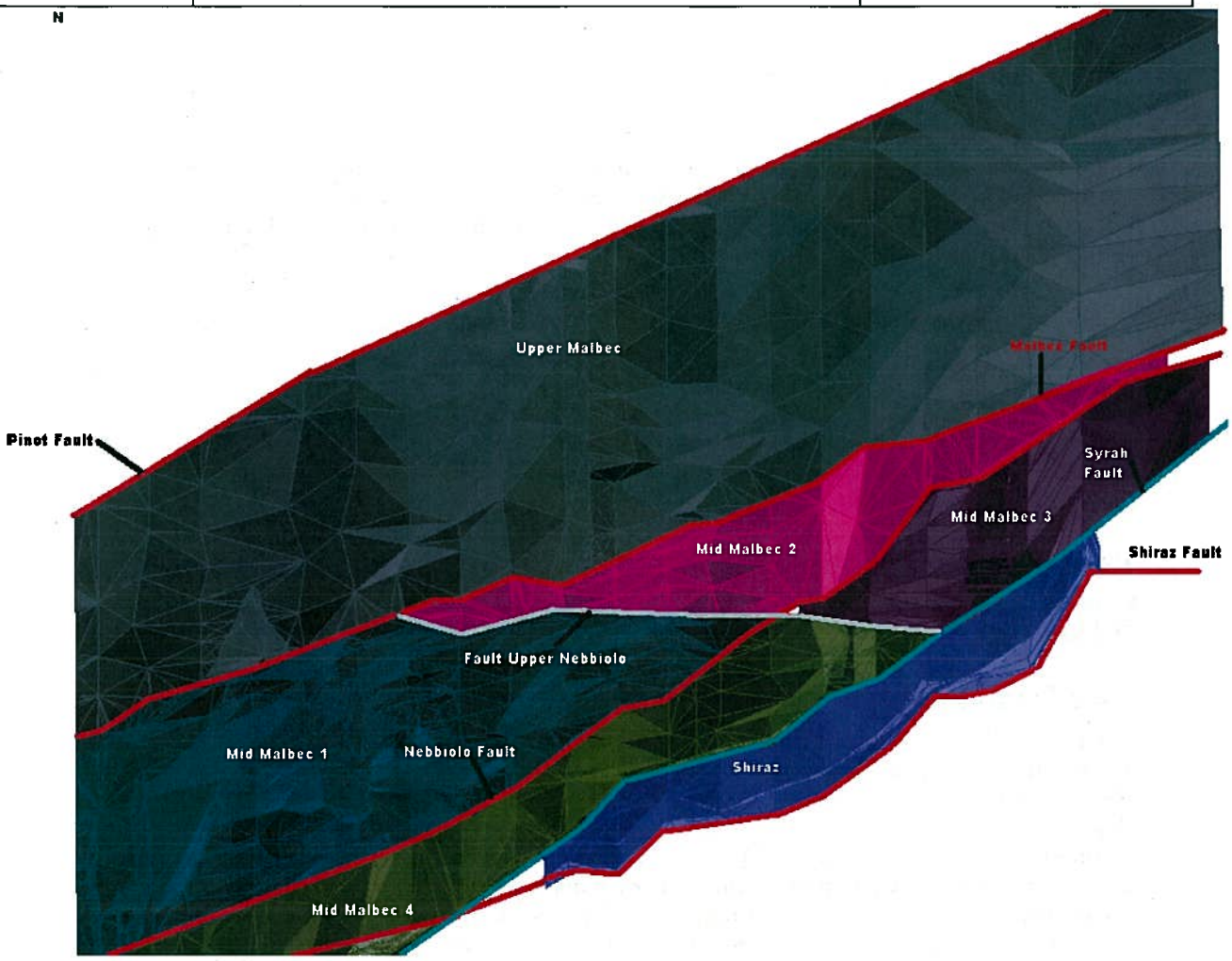


Figure 3. North-South long section showing the position of GG1 fault blocks and faults. Note: New Barbera Fault (sub-parallel to the Pinot and Malbec Faults) subdivides the Upper Malbec. This figure is not yet updated.

1.3 Conclusions

The GG1 resource model is now mostly drilled to Indicated status. The indicated resources are of sufficient quality for preliminary stope design however, the geometry of the faults and the volcanogenic will probably have some significant changes after development and sludge hole drilling.

1.4 Recommendations

1.4.1 Specific gravity

Our current method of measuring SG via length, diameter and weight of the core should be validated against densities determined by water displacement. The benefit of interpolating density should be investigated further.

1.4.2 Recommendations on QA/QC

More detailed analysis of the lab duplicates, blanks and standards should be performed for the GG1 samples.

1.4.3 Recommendations on the geological interpretations

The flat faults that offset the Volcanogenic packages are hard to identify in drill core because the majority of holes are sub parallel to these structures. They are best recognized in the face sketches where the offsets can be seen clearly. Fault geometries will need modification as development progresses.

Domains within the East and West Patron lodes that define barren volcanogenic (< 0.25-0.5 g/t Au) should be investigated further as there may be a requirement for further low grade volcanogenic domains. Further work is required to properly asses wether faults are treated as hard boundaries. The weak volcanogenic domains have too many high gold values the coding of this domain should be more thorough in the next resource estimate.

1.4.4 Improvements to the grade estimation methodology

There are internal basalts and waterloos where grades are assumed be continuous across the basalts. This has resulted in sub-optimal estimation. Fault modelling in Isatis would be the best way to manage this situation as only one variogram would need to be used i.e Isatis can utilise faults in such a way that subdomains are not required. This is done via a fault plane which does not need to pass completely through a domain however, when estimating a block samples in a straight line across the fault are not used in interpolation. Minsight does not model faults in this way and three variograms for three sub-domains would be required. This would require extra variography and time to set up and there would also be less data available for variography per domain.

Because the variograms are so strongly anisotropic minor changes in the orientation of the domains result in suboptimal sample weighting during grade interpolation. Unfolding or variable search orientations would improve local grade estimation. The minimum number of samples should be increased as a small number of blocks may only be estimate by a few samples.

Orientations of maximum continuity are uncertain but appear to vary between flat and a moderate plunge to the North. Variography generated from sludge samples may improve our knowledge of the maximum directions of continuity. Increasing the composite lengths from 1m to 2m should be attempted as this may improve the variography.

2.0 DATA COLLECTION AND VALIDATION

2.1 Overview of the Drilling Data

A total of 181 diamond drill holes were used for interpretation with the diamond drilling being predominantly 44 mm LTK60 and 50.7 mm NQ2 sizes. The is generally minimal core loss in the volcanogenic. The Mine Schist has some core loss but this rock type is not mineralised.

Diamond drill hole collars are surveyed and electronic single shot down hole cameras have been used to survey the hole path. Magnetic intensity is recorded and used to assess the validity of the measurement. Dip change greater than +/- 1.5° and azimuth change greater than +/- 3° is highlighted and assessed.

2.2 QA/QC

The general procedures for QA/QC are outlined below:

- Production diamond drill core is routinely Full Core sampled.
- Routine assaying of diamond drill core samples is undertaken at Aminya laboratories in Ballarat by fire assay (50 gram charges) with an AAS finish. Assays are reported to an LLD of 0.01ppm. For sample reporting below LLD, a value of 0.5xLLD is used in resource estimation.
- Sample QA/QC procedure incorporates routine check assays including repeats (re-assay or repeat assay), duplicate (second sample taken after pulverisation), splits (half coarse sample split, at ratio approx 1:10 samples), and standards.
- SGM Standard reference material has been prepared by Ore Research of Melbourne, from ore samples from Magdala Mine.
- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sampleID created during logging procedure.
- Analysis of received standard values and repeat and duplicate information is used to monitor the performance of the assaying process.

2.3 General Validation Checks

All data is stored within the "acQuire" Database Management System which is structured on a Data Model which applies rules of relational databases to mining and exploration data. The primary key is HoleID.

Diamond drill hole collars are surveyed and single shot down hole cameras have been used to survey the hole path. Electronic single shot instruments are now used in preference to the Eastman mechanical cameras. Magnetic intensity is recorded and used to assess validity of record. Dip +/- 1.5° and azimuth +/- 3° is highlighted and assessed.

All logging of diamond drill core is undertaken using "Datcol" software, to a defined logging scheme. It is a barcode system, eliminating errors involved in keying. Logging is both quantitative and qualitative but in both cases only valid codes or values can be recorded. Additional validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth
- Checks are performed for overlapping samples

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (eg Minesight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted. After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

3.0 GEOLOGICAL OVERVIEW

3.1 Geological Setting

The GG1 orebody contains a large internal basalt separating two 60-70° east-dipping 'volcanogenic' packages: the East Patron and the West Patron lodes. Above the basalt nose the lodes join together in a stockwork zone which generally contains the best gold mineralisation. High grade gold mineralisation is also concentrated as a 'rind' on the basalt contact.

The eastern (hangingwall) margin of the orebody is defined by vesicular basalt and the South Fault. Barren pelitic/psammitic sedimentary 'mine schist' and discontinuous felsic porphyries form the western (footwall) margin of GG1.

Numerous flat faults divide GG1 into 7 blocks, all offset bottom-block-east along these faults. The Shiraz Fault and Cabernet Fault define the base and top of GG1 respectively. Internal faults dipping 0-20° north include (top to bottom) the Syrah, Nebbiolo, Malbec and Pinot Faults, as well as the newly identified Barbera Fault. The Upper Nebbiolo Fault dips south at a similar angle (0-20°). Small scale dip reversals have been mapped in development faces and walls. The Pinot Fault forms the northern boundary of the orebody. GG1 remains open up-plunge to the south.

Recent drilling has defined GG1 up to the 900mRL and south to the 300N crosscuts. Further grade control drilling is required to close the orebody off to the south.

GG1 has only been reinterpreted/ remodelled south of Section 309N (45° Mine Grid) as the orebody is completely mined-out north of this section.

North of Section 309N, East Patron was the weaker of the two lodes with significant mineralisation containing arsenopyrite and pyrite only found on the basalt contacts with a weak volcanogenic package in between. The tenor of East Patron (below the main basalt nose) has increased towards the southern end of GG1 whereas the West Patron limb has thinned and is largely faulted out. The West Patron waterloo persists to the south with significant width and patchy to high grade gold mineralisation.

The mined-out northern end of West Patron had more continuous mineralisation both along strike and down dip but incorporated several sheared internal basalts. Several high grade stockwork zones focused above these internal basalts and a very large high grade zone exists above the main basalt nose. Mineralisation was not consistent across strike with weakly mineralised volcanogenics separating the higher grades. The prominent sulphides of the northern end of the West Patron Lode included arsenopyrite, pyrite and pyrrhotite with visible gold found on occasion.

Several flat north-east dipping thrust faults were identified. These faults offset the orebodies for 1-2m in an east over west direction.

3.2 Methodology for the Creation of Geological Domains

Numerous 3-D shapes were created and then clipped against each other to produce the final geology shapes.

The Shiraz, Syrah, Nebbiolo, Upper Nebbiolo, Malbec, Barbera, Pinot, Cabernet and South Faults were created using information from diamond core and pickups from underground development. These numerous faults divide GG1 into distinct fault blocks. The fault surfaces were created by digitising a line on 45° grid sections where there was enough information to make the interpretation valid. The lines were then joined together to form a surface and later made into a solid to assist the clipping process.

Solids were produced for the footwall Mine Schist, hangingwall Basalt, internal basalt noses, barren volcanogenics zone and porphyry bodies in each fault block. Information from diamond core logging, sludge hole chip logging and development mapping were used in the interpretation of these contacts. Polygons were digitised on 10m sections of the 45° grid and joined together to form solids. These shapes were all clipped off a 'total volcanogenics' solid.

The solids created were then clipped against each other as follows. The Mine Schist, hangingwall Basalt, Porphyry, internal Basalts, barren Volcanogenics zone and total Volcanogenics solids were clipped off the bounding fault planes. Then the bounding Mineschist (FW) and Basalt (HW) shapes were clipped off the total Volcanogenics shape. Finally the footwall Porphyry and Basalt and internal Basalt noses and barren Volcanogenics zone were clipped off the total Volcanogenics. The remaining shape represents the GG1 mineralised 'volcanogenic' lode.

A dividing plane was constructed to separate the East and West Patron Lodes and the total Volcanogenics shape was clipped off this plane to create two separate solids.

3.3 Description of the domains

The hangingwall basalt and the 'mine schist' footwall were used as bounding domains. The 'volcanogenic' package was broken into 8 domains to assist with the modelling process.

The following geological domains, used in previous models, were retained: the western limb (West Patron), the eastern limb (East Patron), and a weak volcanogenic zone in the footwall of the East Patron predominantly occurring above the main basalt nose (Weak Volc).

In addition, the orebody was also domained into Upper, Mid and Lower blocks bounded by flat faults. The Upper West Patron, East Patron and Weak Volc blocks are bounded by the Pinot and Malbec Faults. The Mid West Patron, East Patron and Weak Volc blocks are confined by the Malbec and Nebbiolo Faults. The Lower West Patron and East Patron lie below the Nebbiolo Fault and above the Shiraz Fault.

To create the domains polygons were digitised on 10m sections of the 45° grid and joined together to form solids (see Section 3.2). These solids were then clipped against South Fault, numerous flat faults (Shiraz, Syrah, Nebbiolo, Upper Nebbiolo, Malbec, Barbera, Pinot) and the encompassing Volcanogenics solid to produce the final domains. A total of seven fault blocks were created.

Table 3. Domains interpreted for this resource estimate. The numeric codes are utilised to select domains for modelling and statistical interpretation.

Description (Codes used during grade interpolation)	Numeric Code
Mid weak volcanogenic	VGEO=502
Upper weak volcanogenic	UGEO=500
Upper West Patron	WGEO=600
Mid West Patron	XGEO=602
Lower West Patron	YGEO=604
Upper East Patron	EGEO=606
Mid East Patron	FGEO=608
Lower East Patron	GGEO=610

3.4 Methodology for domain coding

For modelling the 7 fault blocks were re-combined into the upper, middle and lower zones. The Mid Malbec 3, Mid Malbec 4 and the Shiraz domains were combined to form the Lower zone. The

Mid Malbec 1 and Mid Malbec 2 domains were combined the middle zone and the Upper Malbec and Barbara were combined to form the upper zone (Table 4).

The diamond drillholes were coded based on which of the domains they passed through. All drillholes were visually checked and corrected to guarantee that the correct code related to each interval.

Table 4. Domains used for coding.

Description (Codes used during grade interpolation)	Numeric Code	% block coded
Mine Schist	MGEO=1	M%
Basalt	BGEO=4	B%
Mid weak volcanogenic	VGEO=502	V%
Upper weak volcanogenic	UGEO=500	U%
Upper West Patron	WGEO=600	W%
Mid West Patron	XGEO=602	X%
Lower West Patron	YGEO=604	Y%
Upper East Patron	EGEO=606	E%
Mid East Patron	FGEO=608	F%
Lower East Patron	GGEO=610	G%

4.0 DATA ASSESSMENT

4.1 Compositing

There was no length or density weighting during the formation of composites and compositing was within the geological domains. All composites are of equal length except for the last (deepest downhole) composite in each domain which is variable in length. Missing Au values (i.e. unassayed core) were set to 0.01 before compositing. If this is not done errors occur in some composite intervals. In addition, there are intervals of non-sampled internal basalt that must be assigned default grades.

The 1m composite files were validated by comparing them to the uncomposited data to insure that all of the data was composited. The composites were also checked to insure that they are of the correct length and that composites are within geological domains. Removing composites less than 0.5m in length had very little affect on the total length and total Au metal (Table 5).

Table 5. Basic statistics showing the affect of compositing samples to 1m.

UNCOMPOSITED				
Dom	# samp	Mean Au	Max Au	Total Metal
502	157	0.5	19.8	97.7
500	939	0.45	18.4	400
600	1898	4.72	349.0	7990.2
602	694	4.5	57.1	2656.1
604	270	4.41	200.0	1086.2
606	301	3.19	83.2	864.1
608	858	6.59	104.0	5018.5
610	834	3.41	69.8	2774.4

1m COMPOSITES								
Dom	# samp	Mean Au	Max Au	Std Au	Coeff. Var	Total Metal	% Metal kept	% Metal kept
502	196	0.50	13.9	1.59	3.18	96.8	99.1%	100.3%
500	900	0.44	18.4	1.35	3.07	394.0	98.5%	99.0%
600	1695	4.72	192.9	7.92	1.68	7951.1	99.5%	100.1%
602	592	4.52	42.3	5.92	1.31	2631.8	99.1%	100.7%
604	249	4.35	170.4	11.6	2.67	1067.6	98.3%	99.7%
606	276	3.16	83.2	7.16	2.27	855.1	99.0%	100.9%
608	764	6.65	71.5	9	1.35	4961.0	98.9%	101.2%
610	815	3.40	45.1	5.83	1.71	2759.1	99.4%	99.9%

4.2 Cutoff Analysis

A large percentage of metal was cut from the weak volcanogenic domains 500 and 502. This was to compensate for some samples with elevated Au not being representative of the weak volcanogenic domains. Relatively few samples were cut from the remaining domains however in some instances a large amount of metal was cut due to some very high grade Au samples.

Table 6. Statistics for domains after high grade cutting.

Cutting 1m composites							
Dom	Cut Value	# samples cut	Mean Au Cut	Std Au Cut	Coeff. Var Cut	Total Metal	% Metal Kept
502	2.5	7	0.33	0.6	1.82	62.7	64.8%
500	4	16	0.36	0.77	2.14	324	82.2%
600	37	5	4.55	5.8	5.59	7661	96.4%
602	37	1	4.51	5.86	1.30	2628	99.9%
604	37	1	3.82	5.24	1.37	934	87.5%
606	37	2	2.95	5.35	1.81	798	93.3%
608	37	8	6.44	7.93	1.23	4806	96.9%
610	37	5	3.37	5.63	1.67	2730	99.0%

4.3 Statistical Analysis of Domains

The data used in this section has the Au top cuts and minimum sample lengths discussed in the previous sections. Figure 4 to Figure 9 represent histograms for each domain. The volcanogenic domains appear to have significant proportions of unmineralised material. This is most pronounced in domains 606 and 610. There may be some parts of some domains were this unmineralised material can be domained out.

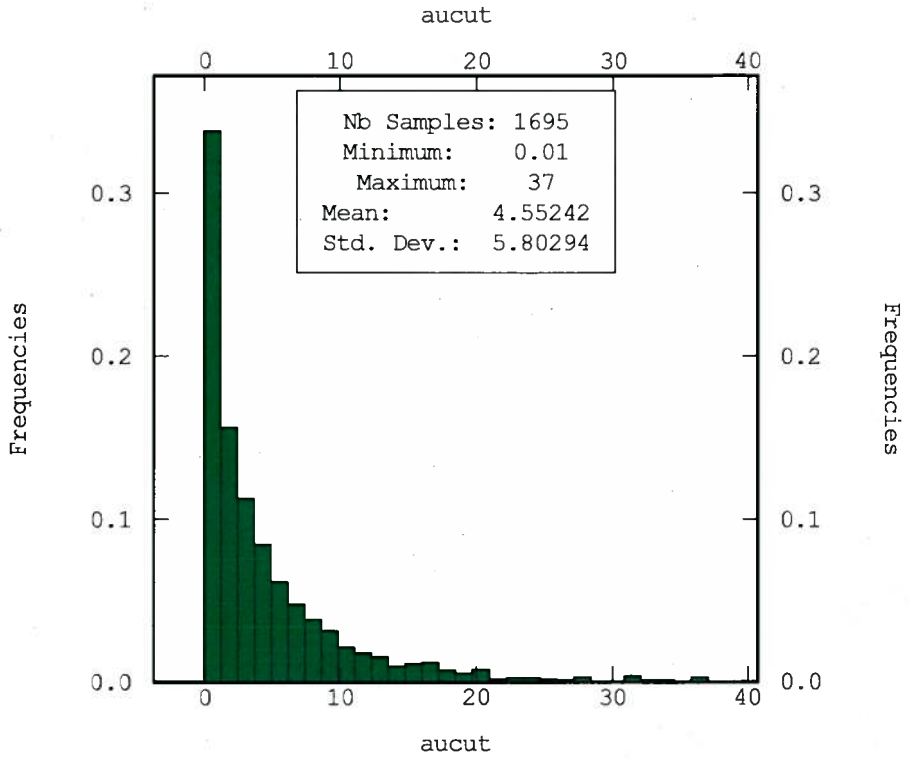


Figure 4. Histogram of gold grades for domain 600.

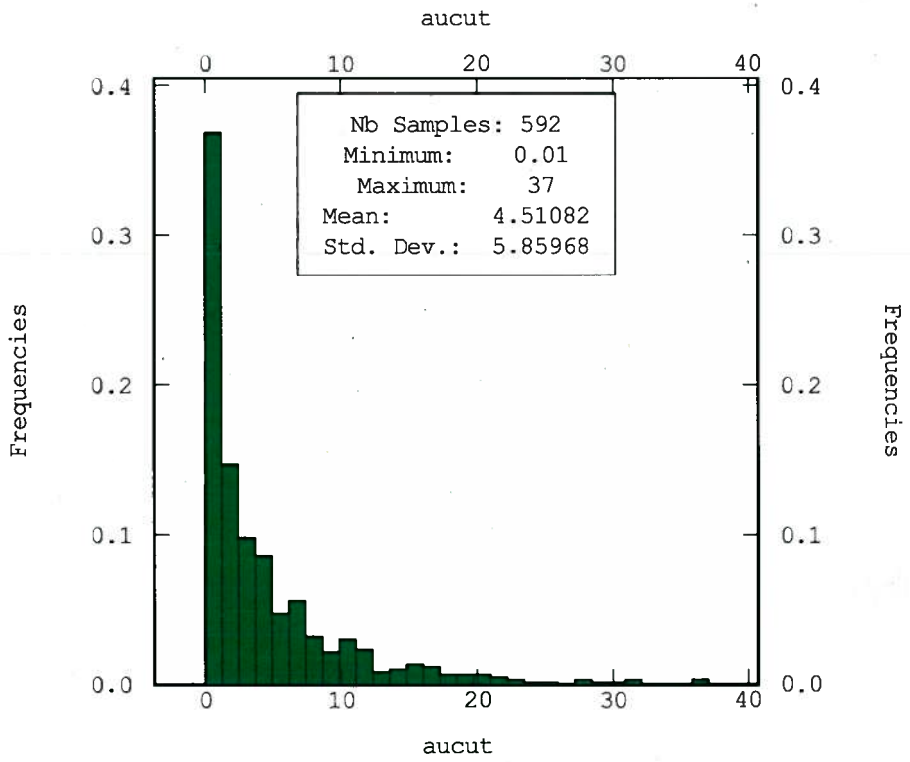


Figure 5. Histogram of gold grades for domain 602.

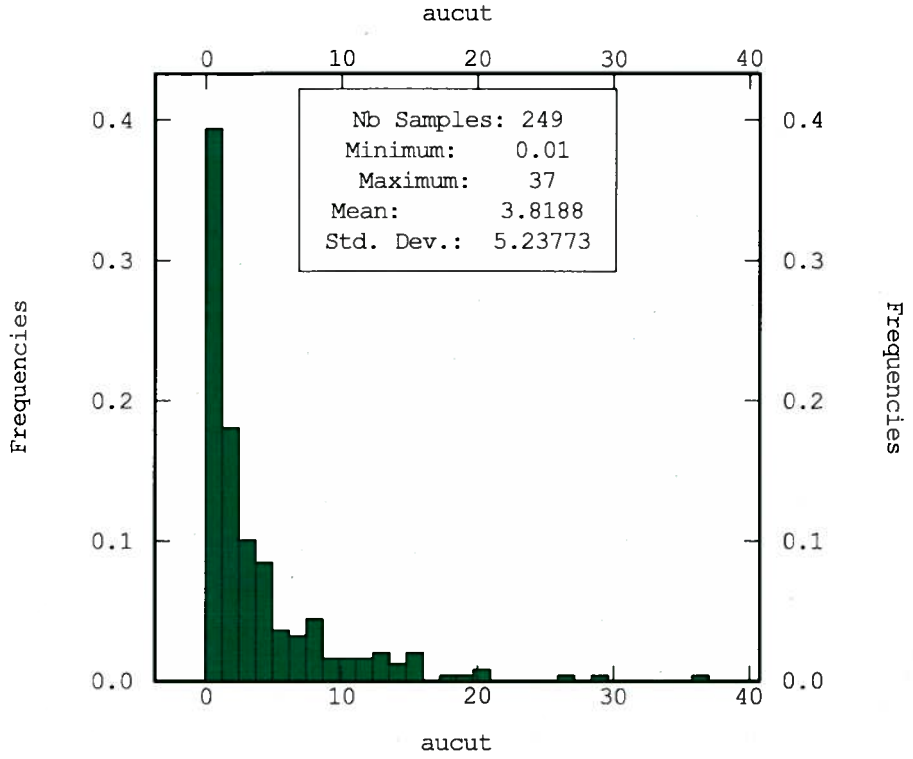


Figure 6. Histogram of gold grades for domain 604.

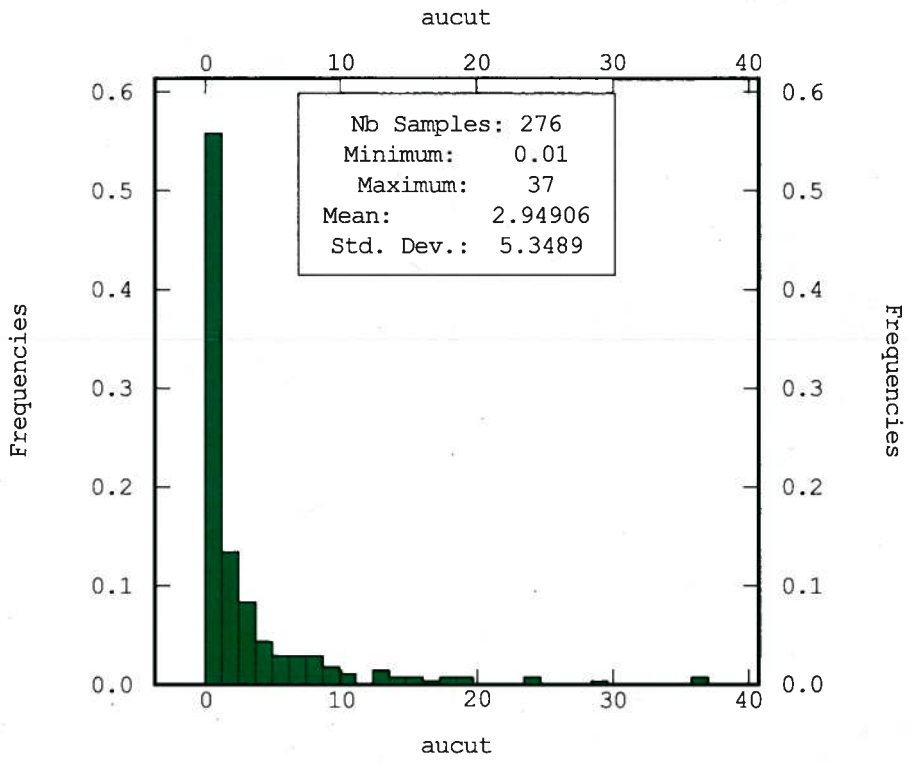


Figure 7. Histogram of gold grades for domain 606.

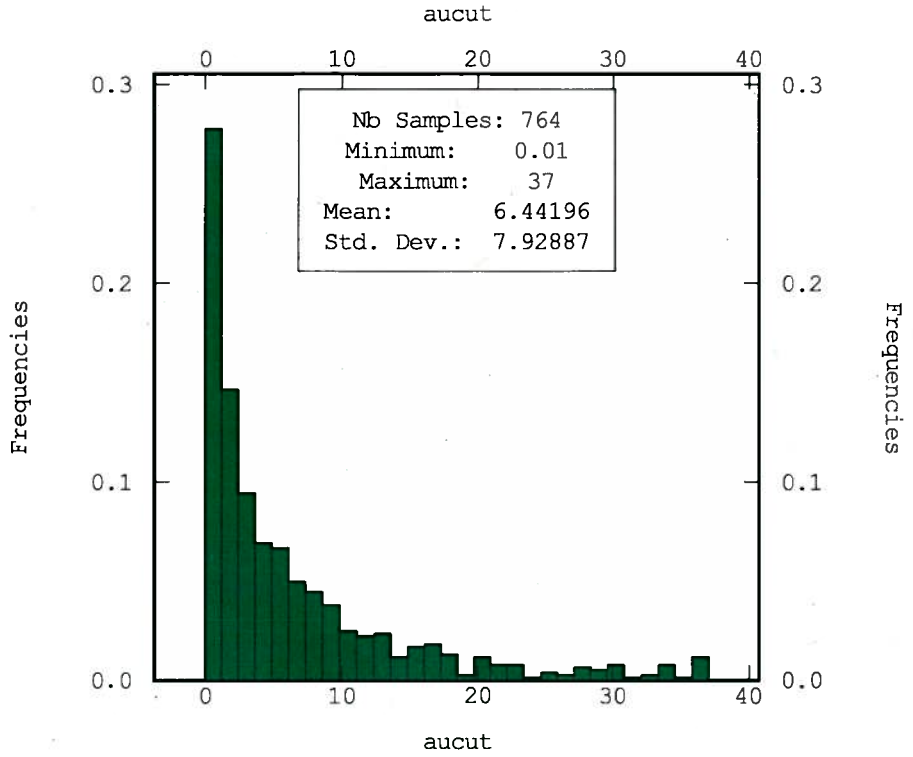


Figure 8. Histogram of gold grades for domain 608.

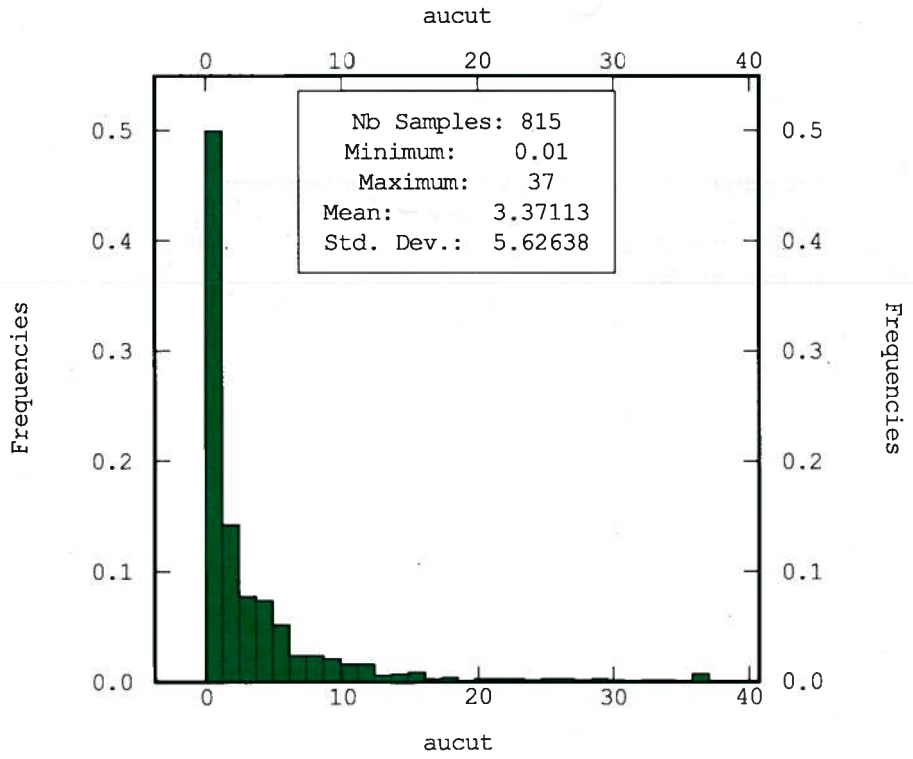


Figure 9. Histogram of gold grades for domain 610.

4.4 Variography

The orientations for variogram fans were generated in the plane of average strike and dip of each domain. Several lag separations, angular tolerances and slicing heights were trialled to produce the best variograms. Gaussian transformation of the Au data improved the variograms slightly to moderately however due to time constraints normal variograms of non-transformed data were utilised.

The variograms and the variogram models are presented in Figure 10 to Figure 15 and Table 7. Maximum directions of continuity were interpreted as subhorizontal however, in some instances a moderate northerly plunge may actually represent the maximum continuity direction. Due to the poor quality of the variograms it was decided to maintain the maximum continuity direction as horizontal that is comparable to the orientation of the basalt noses.

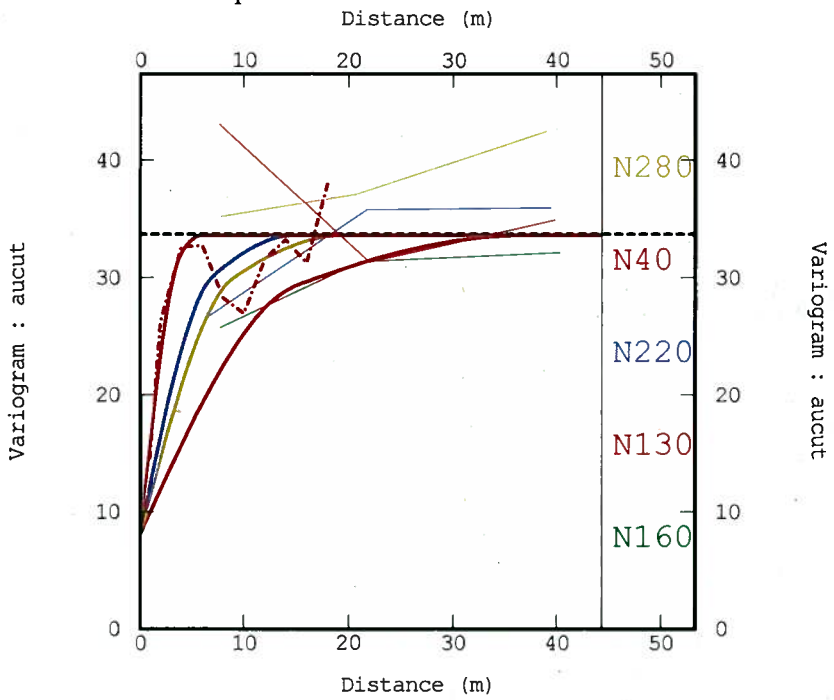


Figure 10. Normal directional variograms of Au for domain 600. The variogram fan is generated in a plane of strike 130° and dip -55° to the East. The directions N130, N160 etc represent the strike of variograms within the abovementioned plane. The variogram N40 is orthogonal to the abovementioned variogram fan.

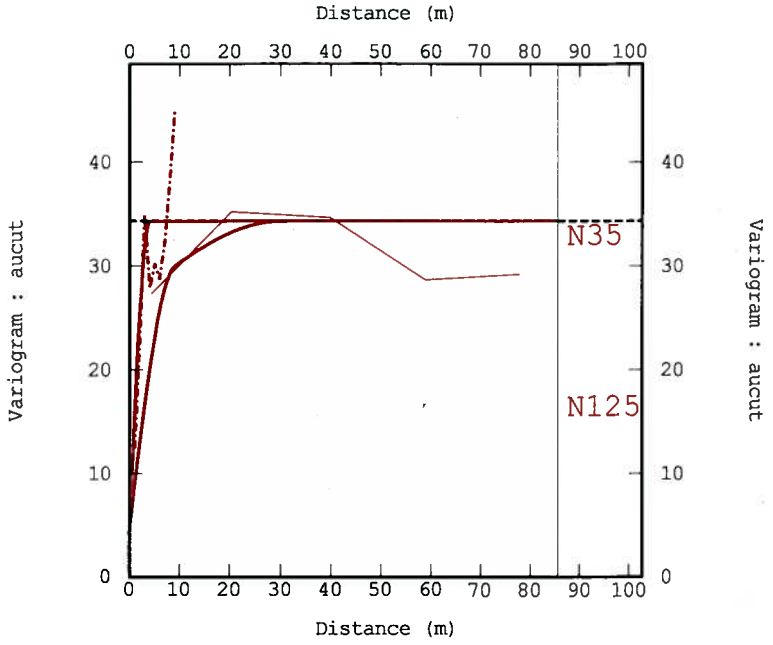


Figure 11. Normal directional variograms of Au for domain 602. The variogram fan is generated in a plane of strike 125° and dip -55° to the East. The direction N305 is the average variogram within the plane of the variogram. The variogram N35 is orthogonal to the abovementioned variogram fan.

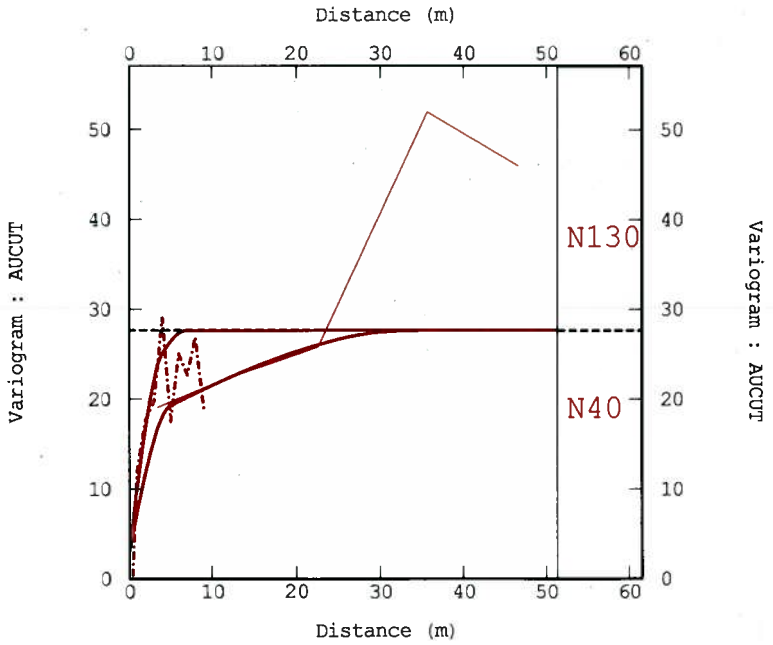


Figure 12. Normal directional variograms of Au for domain 604. The variogram fan is generated in a plane of strike 130° and dip -55° to the East. The direction N305 is the average variogram within the plane of the variogram. The variogram N35 is orthogonal to the abovementioned variogram fan.

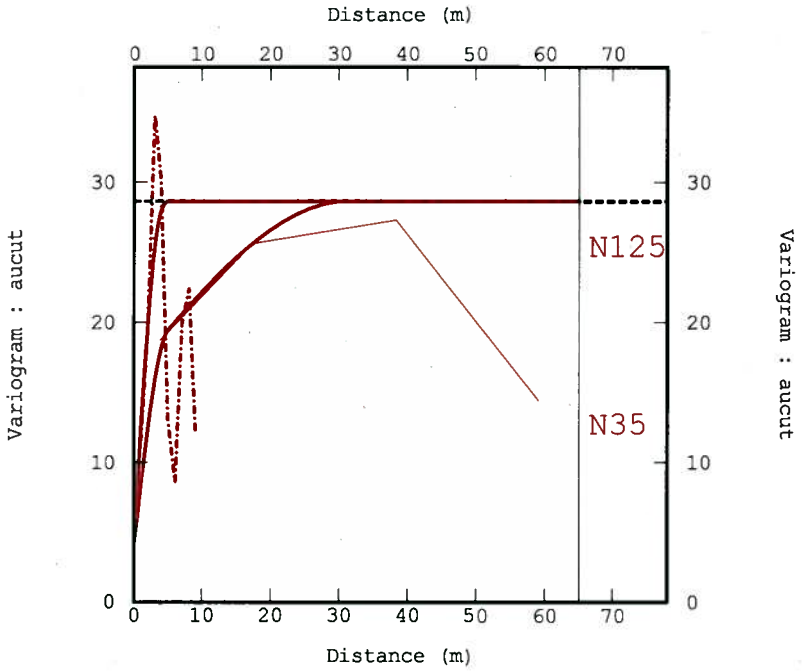


Figure 13. Normal directional variograms of Au for domain 606. The variogram fan is generated in a plane of strike 125° and dip -55° to the East. The direction N305 is the average variogram within the plane of the variogram. The variogram N35 is orthogonal to the abovementioned variogram fan.

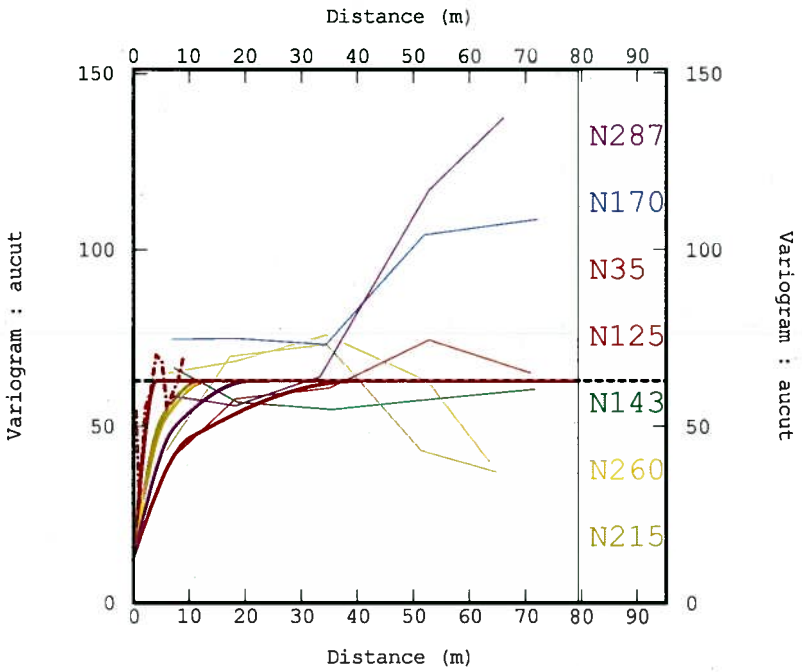


Figure 14. Normal directional variograms of Au for domain 608. The variogram fan is generated in a plane of strike 125° and dip -55° to the East. The directions N125, N143 etc represent the strike of variograms within the abovementioned plane. The variogram N35 is orthogonal to the abovementioned variogram fan.

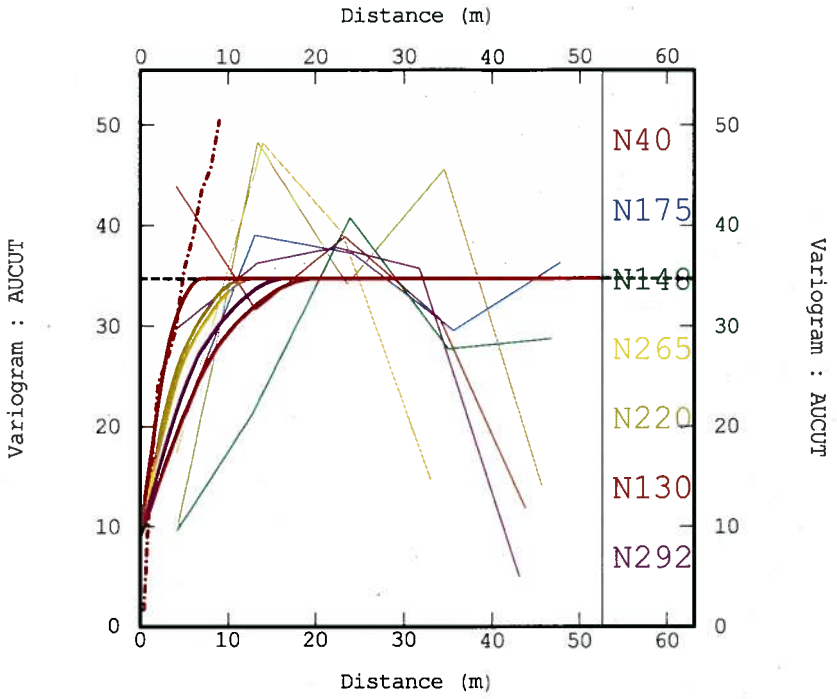


Figure 15. Normal directional variograms of Au for domain 610. The variogram fan is generated in a plane of strike 130° and dip -50° to the East. The directions N130, N140 etc represent the strike of variograms within the abovementioned plane. The variogram N40 is orthogonal to the abovementioned variogram fan.

Table 7. Variogram model parameters utilised for interpolation of Au.

Domain	Rotation (Geologist)	Nugget	Sill 1	Range Horizontal Along Domain	Range Down Dip	Range Across Strike	Sill 2	Range Horizontal Along Domain	Range Down Dip	Range Across Strike
600	-50, 55, -180	8	15.5	15	7	4	10.1	37	15	6
602	-55, 55, -180	5	21.6	9	9	4	7.7	30	30	4
604	-50, 55, -180	3	13.8	5	5	4	10.6	36	36	7
606	-55, 55, -180	4	12.5	5	5	4	12.1	31	31	5
608	-55, 55, -180	13	32.5	10	5	4	14.7	40	12	5
610	-50, 50, -180	9	7	10	5	4	15.5	20	12	7

4.5 Block size selection and search ellipse optimisation

The block size selected for the previous resource estimate was 15m high and 10m wide which is approximately 2/3rds of the drill spacing for the main areas of interest. The model was orientated at the average orientation of the East and West Patron loads, this is a strike of 120 and a dip of -60 East. These parameters have not changed for this resource estimate. A block discretisation of 5*7*2 was utilised. The search parameters have been optimised utilising Isatis (Table 8 and Table 9).

Table 8. Parameters utilised to optimise search ellipse parameters.

Dom	Search ellipse size	Search ellipse orientation	Sample Mean Au	Block Mean Au	Est. error Std	Mean Pos weights	Max Pos Wts	Slope	Mean Dist	# of Blks	# Samp per octant	Total # samp.
600	240*90*48	-50, 55, -180	4.55	4.42	2.57	1.00	1.08	0.50	25	2047	**20**	**80**
602	110*110*20	-55, 55, -180	4.51	4.02	2.74	1.00	1.07	0.47	22.26	711	12	48
604	108*108*14	-50, 55, -180	3.82	3.94	2.77	1.00	1.05	0.54	27.62	446	20	80
606	155*155*25	-55, 55, -180	2.95	2.89	2.80	1.00	1.05	0.43	39.18	882	16	64
608	160*48*20	-55, 55, -180	6.44	5.75	3.20	1.00	1.05	0.36	25.29	617	16	64
610	160*96*56	-50, 50, -180	3.37	3.00	2.79	1.01	1.19	0.33	28.9	697	16	64

Table 9. Search parameters utilised for grade interpolation.

Domain	Rotation (Geologist)	Search	Min no samples	Maximum no. samples	Max samples per quadrant
600	-50, 55, -180	240*90*48	1	80	20
602	-55, 55, -180	110*110*20	1	48	12
604	-50, 55, -180	108*108*20	1	80	20
606	-55, 55, -180	155*155*25	1	64	16
608	-55, 55, -180	240*72*30	1	64	16
610	-50, 50, -180	160*96*56	1	64	16

5.0 Specific Gravity and Default Au

The specific gravity for the previous resource estimate was obtained from the statistical analysis of the specific gravities obtained from composited diamond drillhole samples. The default Au for the weak volcanogenic domains are from the mean grade of the compostes for these domains. The default densities and default Au utilised for this resource estimate are listed in Table 10.

Table 10. Default densities and default Au utilised in the resource estimate.

DOMAIN	Default Au	DRY DENSITY (t/m^3)
Mine Schist	0.01	2.67
Basalt	0.01	2.79
Mineralised Volcanogenics	N/A	2.85
Mid weak volcanogenic	0.33	2.85
Upper weak volcanogenic	0.36	2.85

6.0 Block Model

6.1 Introduction

Table 11 contains the parameters utilised to create the rotated Minsight block model. The block size and model rotation were checked before moving onto domain coding. Domain wireframes were utilised to code the percentage of each block within each domain (Table 12). This was validated on screen to insure correct coding. A single pass ordinary kriging is used to interpolate Au for all the mineralised volcanogenic domains. All domain boundaries of the domains listed in Table 12 are treated as hard.

Table 11. Block model parameters.

	Origin	Rotation	Range (m)	Number of Cells	Cell Dimensions
Easting	658300	1 is 30	560	56	10
Northing	897600	2 is -60	450	30	15
RL	-1270	3 is 0	300	150	2

Table 12. Block model domain coding.

Description (Codes used during grade interpolation)	Numeric Code	% block coded	Au	SG	Slope of Regression	.dxf filename	Inferred Resource	Indicated Resource
Mid weak volcanogenic	VGEO=502	V%	VAUD	VSG		MEWkp		
Upper weak volcanogenic	UGEO=500	U%	UAUD	USG		UEWkp		
Upper West Patron	WGEO=600	W%	WAUD	WSG	WSLRD	UWestp	WCAT=3	WCAT=1
Mid West Patron	XGEO=602	X%	XAUD	XSG	XSLRD	MWestp	N/A	XCAT=1
Lower West Patron	YGEO=604	Y%	YAUD	YSG	YSLRD	LWestp	N/A	YCAT=1
Upper East Patron	EGEO=606	E%	EAUD	ESG	ESLRD	UEestp	N/A	ECAT=1
Mid East Patron	FGEO=608	F%	FAUD	FSG	FSLRD	MEastp	N/A	FCAT=1
Lower East Patron	GGEO=610	G%	GAUD	GSG	GSLRD	LEastp	N/A	GCAT=1

6.2 Model Validation

The model was validated by visually comparing the diamond drilling sample grades for each element to block grades on sections and in 3D. The block size is approximately 2/3rds the size of the drilling grid and parent cell estimation is used. This factor means that individual blocks are not similar to individual holes rather a block grade is most affected by the nearest drillholes. Despite this the comparison between composites and the blocks was considered appropriate for all domains. Table 13 is a comparison of the kriged Au to the declustered Au from 1m composites. Five declustered estimates of Au have been obtained for each domain by changing the declustering block size. The ordinary kriging estimates are acceptably close to the declustered statistics.

Table 13. Comparison of block model Au to declustered Au (1m composites).

	Block Size X, Y, Z							
Domain	Clustered	No Blocks	OK	15*15*500	30*30*500	45*45*500	60*60*500	80*80*500
600	4.55	2047	4.42	4.56	4.40	4.35	4.38	4.38
602	4.51	711	4.02	4.38	3.75	3.81	3.99	4.19
604	3.82	446	3.94	3.76	3.43	3.03	2.89	3.07
606	2.95	882	2.89	3.20	2.73	2.75	3.09	3.21
608	6.44	617	5.75	5.88	5.43	5.16	5.12	5.54
610	3.37	697	3.00	2.85	2.17	2.24	2.76	2.89

7.0 RESOURCE CLASSIFICATION AND REPORTING

It was not within the scope of this report to do significant work on QA/QC for gold assays. The QA/QC is an integral part of resource classification in the JORC code. The resource classification described below assumes that the QA/QC is of a high standard.

Utilising the slope of regression from ordinary kriging and the drill hole geometry, wireframes were created (excludes some material too thin or isolated to be economic) around the areas considered to meet Indicated resource status. This classification was performed in long section view with only complete blocks being classified across the full width of a domain. Not all the material outside the Indicated shapes is considered to be Inferred because some of this Inferred material is thin or isolated or considered to be the result of smearing high grades. Table 14 contains the GG1 tonnage and grade estimates calculated from the Minesight detailed reserves routine. These grades and tonnages do not take into account depletion.

Table 14. Resource at a 4g/t cutoff . These grades and tonnages do not take into account depletion.

Domain	Description	Resource Classification	Tonnes	Grade	Grams	Ounces
600	Upper West Patron	Indicated	427,551	5.7	2,424,214	77,940
		Inferred	55,558	5.0	278,346	8,949
		Total Indicated and Inferred	483,109	5.6	2,702,560	86,889
602	Mid West Patron	Indicated	71,427	5.9	418,562	13,457
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	71,427	5.9	418,562	13,457
604	Lower West Patron	Indicated	33,200	6.5	216,132	6,949
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	33,200	6.5	216,132	6,949
	West Patron	Indicated	532,178	5.7	3,058,908	98,346
		Inferred	55,558	5.0	278,346	8,949
		Total Indicated and Inferred	587,736	5.7	3,337,254	107,295
606	Upper East Patron	Indicated	19,041	5.6	105,678	3,398
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	19,041	5.6	105,678	3,398
608	Mid East Patron	Indicated	105,481	6.8	715,161	22,993
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	105,481	6.8	715,161	22,993
610	Lower East Patron	Indicated	54,942	5.5	299,983	9,645
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	54,942	5.5	299,983	9,645
	East Patron	Indicated	179,464	6.2	1,120,822	36,035
		Inferred	0	0.0	0	0
		Total Indicated and Inferred	179,464	6.2	1,120,822	36,035
	Total	Indicated	711,642	5.9	4,179,730	134,381
		Inferred	55,558	5.0	278,346	8,949
		Total Indicated and Inferred	767,200	5.8	4,458,076	143,330

APPENDIX 1 DRILLHOLES USED IN THIS RESOURCE ESTIMATE

Note these statistics are from accumulations over full width of the domains 600 to 610.

BHID	X	Y	Z	GEOCD	TO	LENGTH	AUCUT
MD2424W1	658589.25	897916.94	-935.14	600	472	5.9	1.787
MD2424W1	658597	897926.62	-948.08	500	502	30	2.362
MD2424W1	658607.81	897940.38	-965.94	600	522	20	2.087
MD2424W1	658616.75	897951.5	-980.69	606	543	21	2.221
MD2424W1	658631.5	897969.06	-1004.25	608	570.8	10.8	0.2
MD2424W1	658639.25	897978.5	-1016.75	608	587.25	8.75	3.469
MD2482	658589.06	897938.5	-979.54	600	526	28	0.765
MD2482	658597.69	897950.75	-993.56	602	537	9	0.39
MD2482	658626.62	897991.12	-1039.68	608	601.8	3.05	0.44
MD3173	658560.88	897981.44	-1016.55	602	403.85	0.85	4.602
MD3180	658579.06	897956.88	-962.57	606	409.25	2.55	0.785
MD3180	658578.31	897955.06	-962.88	500	410.6	1.35	1.137
MD3180	658577.88	897953.94	-963.06	606	411.65	1.05	5.109
MD3180	658574.75	897946.25	-964.37	600	427.45	15.8	2.959
MD3203	658663.94	897959.69	-988.86	608	398.6	2.4	3.76
MD3203	658663.75	897957.88	-989.35	502	400	1.4	0.222
MD3203	658663.06	897952.44	-990.79	608	409.85	9.85	4.104
MD3203	658659.19	897926.25	-997.88	602	432.8	0.9	12
MD3203	658658.88	897924.5	-998.36	602	434.5	0.65	0.03
MD3203	658657.19	897913.44	-1001.35	602	446.05	0.65	1.64
MD3236	658690.25	897969.44	-1026.18	608	358.95	2.1	10.867
MD3236	658683.56	897957.62	-1030.45	604	373.3	2.35	0.077
MD3239	658713	897893.44	-960.93	608	409	6.1	1.619
MD3239	658709.69	897887.25	-961.17	502	417	8	0.221
MD3239	658706.75	897881.88	-961.39	608	421.2	4.2	7.503
MD3239	658702.31	897873.56	-961.71	602	428.9	0.8	8.55
MD3244	658604.12	897931.06	-951.58	606	419.7	2.75	7.211
MD3244	658601.12	897928	-951.67	500	425.55	5.85	0.852
MD3244	658595.25	897922.06	-951.88	600	436.2	10.65	16.058
MD3280	658796.31	897895	-968.31	610	392.7	8.1	1.5
MD3284	658784.94	897877.25	-949.9	610	407.8	4.5	0.062
MD3300A	658659.88	897882.38	-927.8	606	433.1	2.35	1.527
MD3300A	658658.38	897879.94	-927.75	500	436.5	3.4	0.64
MD3300A	658651.94	897870.31	-927.43	600	456.25	19.75	3.241
MD3300B	658583.81	897869.44	-901.04	600	479.55	3.5	6.681
MD3300B	658581.19	897866.19	-900.71	600	482.1	0.3	0.87
MD3406	658689.81	897805.88	-881.19	606	500.2	5.7	2.956
MD3406	658683.94	897799.25	-878.66	500	512.9	12.7	0.937
MD3406	658678.62	897793.31	-876.37	600	516.85	3.95	4.036
MD3868	658568.94	897985.88	-983.36	606	83.6	5.3	0.074
MD3868	658565.88	897975.19	-982.63	600	100.6	17	5.323
MD3868	658561.5	897959.69	-981.59	600	113.5	10.55	3.061
MD3926	658674.75	897941.06	-981.2	608	143.15	23	4.295
MD3926	658653.06	897926.31	-974.64	602	165.4	13.5	6.173
MD3926	658639.44	897917.19	-970.56	600	185.8	20.4	1.994
MD3926	658627.56	897909.25	-967.05	600	193.6	6.65	2.927
MD3927	658705.88	897964.62	-1031.4	610	88.5	0.7	1.61
MD3927	658688.44	897951.62	-1035.19	604	110.55	0.6	1.61
MD3929	658655.56	897963.88	-1003.29	608	136.75	9.35	4.077
MD3929	658608.38	897945.56	-997.25	602	183.6	1.1	1.235
MD3929	658595.31	897940.12	-995.37	602	203.1	11.5	1.609
MD3930	658675.06	897970.94	-1032.65	608	114.95	7.2	9.663
MD3930	658637.56	897954.06	-1038.41	604	154.4	3	2.187
MD3931	658696.56	897913.56	-977.52	608	145.2	17.8	12.182

MD3931	658686	897901	-972.89	602	161.5	16.3	13.224
MD3931	658676.44	897889.69	-968.59	602	172.9	8.3	7.259
MD3932	658692.12	897908.88	-999.47	604	139.5	3.2	0.183
MD3932	658686.75	897902.44	-998.24	604	147.9	3	2.592
MD3941	658628.44	897976.81	-992.93	608	27.8	1.9	12.706
MD3941	658626.38	897975.5	-992.27	502	31	3.2	0.381
MD3941	658619.62	897971.31	-990.09	606	43.4	11.5	0.942
MD3941	658608.94	897964.62	-986.63	500	56.6	11.75	0.373
MD3941	658601.38	897959.81	-984.18	606	63.3	6.7	3.582
MD3941	658586.5	897950.62	-979.48	600	92.85	29.55	7.764
MD3942	658628.44	897978.38	-1000.92	608	26.4	1.8	34.63
MD3942	658626.12	897977.06	-1000.94	502	30	3.6	0.744
MD3942	658623.12	897975.31	-1000.93	608	33.4	3.4	2.791
MD3942	658592.75	897957.38	-1000.94	602	68.3	2.65	0.35
MD3942	658584.19	897952.31	-1000.96	602	79.3	4.8	3.092
MD3943	658623.81	897980.62	-996.52	608	29.4	1	3.9
MD3943	658621.94	897979.88	-996.24	502	32.45	3.05	5.238
MD3943	658614.31	897976.81	-995.09	606	46	13.55	7.236
MD3943	658604.44	897972.81	-993.6	500	54	8	0.437
MD3943	658597.94	897970.25	-992.62	606	60.1	6.1	0.238
MD3943	658579.5	897962.88	-989.81	600	82	9.9	6.708
MD3943	658566.38	897957.5	-987.78	600	95.9	8.9	2.529
MD3944	658621.25	897980.31	-1002.89	608	35.5	8.45	6.487
MD3944	658583.38	897966.12	-1006.41	602	79.4	15	4.513
MD3945	658612.38	897982.25	-1000.31	608	46	14.05	2.958
MD3945	658597.44	897978.5	-1000.15	606	62.7	16.7	4.434
MD3945	658574.56	897972.75	-1000.08	600	80.15	4.45	4.185
MD3945	658560.56	897969.31	-1000.03	600	97.8	10.85	2.612
MD3946	658635.81	897976.88	-1003.64	608	21.3	1.3	3.954
MD3946	658633.69	897974.88	-1004.01	502	25.95	4.65	0.382
MD3946	658630.19	897971.69	-1004.59	608	30.7	4.75	8.917
MD3946	658610.19	897954	-1008.22	602	57	3.4	10.77
MD3946	658602.25	897947	-1009.67	602	66.05	0.25	2.08
MD3947	658633.62	897974.69	-996.77	608	23.95	0.75	1.01
MD3947	658632.12	897973.25	-996.45	502	27.45	3.5	0.371
MD3947	658628.75	897970.12	-995.75	608	33.2	5.75	2.863
MD3947	658602.25	897947.25	-991.03	602	68.3	5.35	1.84
MD3947	658595.94	897941.88	-989.94	602	75.05	2.15	1.808
MD3947	658592	897938.44	-989.25	602	82.05	5.55	3.984
MD3948	658636.75	897970.62	-992.13	502	27.3	4.1	0.369
MD3948	658632.81	897965.38	-990.08	608	37	9.7	4.298
MD3948	658625	897955.38	-986.26	500	49.4	8	3.397
MD3948	658621.31	897950.62	-984.45	606	53.9	4.5	0.981
MD3948	658609.81	897936.06	-978.93	600	88.15	34.25	2.928
MD3949	658644.69	897970	-997.12	608	21.55	1.3	10.896
MD3949	658643.94	897967.81	-996.77	502	24.9	3.35	0.235
MD3949	658641.88	897962.12	-995.85	608	33.7	8.8	3.777
MD3949	658629.31	897929.5	-990.46	602	68.9	8.35	3.791
MD3949	658626.69	897923	-989.41	602	72.85	2.05	8.232
MD3949	658625.94	897921.19	-989.11	602	74.05	0.45	1.99
MD3950	658645.06	897970.19	-1003.08	502	21.25	1.6	0.129
MD3950	658643.88	897966.88	-1003.42	608	26.8	5.55	3.184
MD3950	658633.88	897941.12	-1006.11	602	56	8.65	2.522
MD3950	658629.06	897929	-1007.39	602	65.8	2	0.01
MD3958	658608.56	897972.12	-985.77	606	52.4	9	1.618
MD3958	658595.94	897966.56	-981.71	500	72.2	19.8	0.399
MD3958	658587	897962.75	-978.96	606	72.6	0.4	5.43
MD3958	658572.25	897956.31	-974.38	600	105.7	33.1	4.712
MD3959	658613.88	897986.38	-1007.61	608	42.3	9.9	5.701

MD3959	658567.69	897979.88	-1015.43	602	90.25	11.3	6.367
MD3960	658622	897964.75	-984.84	606	43.9	5.9	0.787
MD3960	658611.38	897955.25	-979.39	500	68.5	24.6	0.282
MD3960	658591.56	897938	-969.17	600	100.2	31.7	6.106
MD3961	658641.44	897967.44	-990.29	608	26.7	1.9	3.448
MD3961	658640.56	897965.5	-989.48	502	29.35	2.65	0.067
MD3961	658638.12	897960.31	-987.3	608	39	9.65	5.046
MD3961	658632.06	897947.38	-981.83	602	54.95	10.85	2.578
MD3961	658628.5	897939.88	-978.7	500	61.75	6.8	0.352
MD3961	658622.81	897927.94	-973.5	600	83.4	21.65	3.123
MD3961	658616.75	897915.25	-967.99	600	91.3	7.2	2.5
MD3990	658555.81	897957.88	-962.04	600	104	19.9	1.773
MD3994	658532.94	897936.56	-938.88	600	138.1	17.5	3.718
MD4091	658647.38	897971.75	-1008.27	608	45.55	3.3	1.283
MD4091	658626	897951.38	-1014.3	602	74.7	1.5	0.01
MD4092	658657.06	897970	-1013.3	608	41.15	2.6	5.679
MD4092	658636.38	897940.19	-1026.26	602	78.8	0.8	4.65
MD4093	658664.75	897963.62	-1000.02	608	40.1	1.85	0.993
MD4093	658664.19	897962.25	-1000.06	502	41.2	1.1	2.87
MD4093	658662.81	897958.81	-1000.16	608	47.55	6.35	9.106
MD4093	658651.44	897931.19	-1001.1	602	75.8	3.15	3.545
MD4093	658645.25	897916	-1001.71	602	90.8	0.4	1.18
MD4094	658678.25	897942.56	-983.93	608	56.15	23.5	9.32
MD4094	658670.44	897909	-974.46	602	89.2	18.1	10.475
MD4094	658666.75	897893.88	-970.16	602	101.1	9.6	3.719
MD4095	658678.5	897936.69	-977.11	608	62.3	21.1	6.044
MD4095	658674.81	897920.25	-970.99	500	77.1	14.8	1.241
MD4095	658672.31	897909.31	-966.92	600	86.15	9.05	3.775
MD4095	658669.38	897896.75	-962.26	600	99.5	8.35	5.409
MD4095	658665.94	897883	-957.15	600	119	17.15	3.631
MD4096	658678.06	897955.12	-990.73	608	33.45	4.8	23.464
MD4096	658677.12	897952.06	-990.16	502	35.15	1.7	0.462
MD4096	658674.56	897943.69	-988.59	608	51.3	16.15	9.194
MD4096	658665.06	897913.69	-983.09	602	76	1.7	22.918
MD4096	658663.94	897910.19	-982.44	602	79.45	1.05	0.05
MD4096	658660.75	897900.12	-980.63	602	91.3	3.3	1.443
MD4098	658640.5	897956.56	-977.15	606	64.1	2.75	0.491
MD4098	658628.5	897943.06	-970.83	500	99.7	35.6	0.277
MD4098	658613.19	897926.5	-963.32	600	111.6	11.9	2.871
MD4098	658604.75	897917.31	-959.15	600	125	12.35	7.493
MD4099	658645.88	897953.12	-975.74	606	63.6	1.85	0.197
MD4099	658635.06	897937.69	-968.72	500	102	38.4	0.051
MD4099	658621.56	897919.31	-960.41	600	112.15	10.15	3.935
MD4099	658613.88	897908.75	-955.66	600	129.1	16.25	12.376
MD4100	658667.56	897980.5	-990.08	502	50	50	0.01
MD4100	658654.88	897958.25	-980.78	502	54.5	4.5	0.025
MD4100	658650.62	897950.88	-977.67	608	65.1	7.55	0.068
MD4100	658640.94	897934.31	-970.54	500	98.4	33.3	0.709
MD4100	658630.06	897916.5	-962.81	600	109.5	11.1	3.605
MD4100	658623.31	897905.44	-957.99	600	123.7	11.8	1.285
MD4100	658618.75	897898.06	-954.79	600	127.7	1.3	0.102
MD4101	658661.81	897961.12	-992.87	608	43.8	1	0.65
MD4101	658661.19	897959.81	-992.67	502	45.75	1.95	0.093
MD4101	658658.94	897954.81	-991.93	608	54.9	9.15	3.001
MD4101	658643.56	897920.25	-987.41	602	92.2	7.5	1.414
MD4101	658640.19	897912.62	-986.52	602	98.05	2.45	0.548
MD4102	658654.06	897947.88	-973.71	606	64.8	3.05	0.052
MD4102	658645.62	897930.31	-965.99	500	103.5	38.7	0.128
MD4102	658635.56	897910	-957.24	600	113.4	9.9	1.342
MD4102	658630.69	897900.31	-953.16	600	123.7	7.35	5.049

MD4103	658673	897945.12	-976.1	608	49.75	7.35	0.236
MD4103	658670.12	897937.38	-972.31	606	56.3	2.2	49.142
MD4103	658665.75	897925.75	-966.74	500	81.15	24.85	0.346
MD4103	658655.25	897897.56	-953.26	600	122.3	41.15	2.993
MD4104	658679.75	897947.81	-992.51	608	46.1	17.45	11.184
MD4104	658671.56	897911.94	-988.67	602	74.85	1	0.65
MD4104	658671	897909.69	-988.43	602	77.2	1	0.13
MD4104	658668.94	897901	-987.52	602	86	0.65	0.32
MD4105	658680.12	897934.62	-974.1	608	55.15	1	0.02
MD4105	658679.25	897929.12	-971.72	606	65.3	9.15	4.904
MD4105	658676.25	897911.75	-964.35	500	89	18.3	0.287
MD4105	658671.25	897884.94	-953.2	600	129.7	40.7	3.825
MD4106	658679.38	897948.81	-992.74	608	45.05	16.95	10.454
MD4106	658670.94	897911.75	-989.12	602	75.25	1	0.03
MD4106	658670.5	897909.81	-988.94	602	77.1	0.75	0.01
MD4106	658668.88	897902.69	-988.28	602	84.55	0.9	3.3
MD4107	658676.94	897939.31	-981.3	608	58.95	21.15	5.388
MD4107	658670.12	897911.44	-972.48	602	90.45	24.05	3.935
MD4107	658665.44	897892.75	-966.63	602	104.5	11.9	6.557
MD4108	658692.06	897935.75	-988.13	608	59.4	20.7	6.987
MD4108	658694	897906.38	-983.24	602	83.2	8.7	0.346
MD4108	658694.75	897892.81	-980.98	602	96	6.6	4.604
MD4108	658695.12	897881.88	-979.2	604	104.2	0.8	0.01
MD4116	658639.62	897980.19	-1018.6	608	49.3	2	5.055
MD4116	658608.25	897962	-1033.74	602	91.45	7.65	0.6
MD4117	658629	897984	-1014.24	608	54.8	0.4	0.47
MD4117	658625.94	897982.88	-1015.13	608	60.6	5.2	5.999
MD4117	658604.12	897974.44	-1021.51	602	82.8	1	9.99
MD4117	658587.44	897968	-1026.35	602	101.4	1.3	1.711
MD4117	658584.94	897967.06	-1027.08	602	103.85	0.65	4.1
MD4118	658613.19	897989.88	-1022.22	608	73.5	7.1	2.065
MD4118	658588.5	897985.62	-1030.32	602	96.75	0.95	5.88
MD4132	658651.06	897963.38	-989.02	608	49.05	2.4	0.018
MD4132	658648.88	897960.56	-988.26	502	53.9	4.85	0.075
MD4132	658645.69	897956.38	-987.13	608	59.9	6	3.27
MD4132	658627.06	897931.88	-980.43	602	100.25	23.75	3.841
MD4132	658617.06	897918.94	-976.75	602	106.5	2.8	7.194
MD4132	658614.31	897915.38	-975.72	600	110.25	1	0.01
MD4133	658670	897959.44	-993.86	608	41.1	1	3.51
MD4133	658669.62	897958	-993.7	502	43.1	2	0.26
MD4133	658667.81	897950.69	-992.85	608	56.3	13.2	4.866
MD4133	658659.19	897917.06	-989.22	602	85.05	0.9	7.96
MD4133	658658.62	897914.94	-989	602	87.6	1.65	0.01
MD4134	658670.69	897943.06	-976.61	608	55.2	14	0.956
MD4134	658664.38	897927.69	-969.47	500	77.45	22.25	0.128
MD4134	658653.19	897900.62	-957.1	600	118.8	41.35	2.396
MD4135	658680.62	897942.44	-988	608	55.15	24.3	9.95
MD4135	658674.31	897908.38	-981.2	602	81.8	7.05	2.983
MD4135	658672	897895.5	-978.61	602	94.4	5.65	1.274
MD4136	658684.62	897947.94	-984.91	608	39	2.1	0.01
MD4136	658683.12	897931.75	-980	608	63.3	16.7	1.672
MD4136	658681.81	897918.56	-976.09	608	70.45	3.3	2.078
MD4136	658680.56	897906.69	-972.54	602	91.9	21.45	5.782
MD4136	658678.75	897889.06	-967.42	602	103.3	7.35	2.452
MD4137	658691.44	897910.88	-964.9	608	84.4	9	3.133
MD4137	658691.12	897901.81	-961.2	502	95	10.6	1.979
MD4137	658690.81	897892.25	-957.31	500	105	10	0.556
MD4137	658690.56	897883.31	-953.66	500	111	2.7	1.785
MD4137	658690.38	897877.19	-951.15	600	121.6	10.6	38.211

MD4137	658689.62	897863.38	-945.54	600	138.55	14.6	3.589
MD4138	658705.25	897913.56	-977.6	608	79.25	8.6	8.192
MD4138	658706.88	897906.06	-975.71	502	86.5	7.25	2.017
MD4138	658709.31	897894.31	-972.76	608	104	17.5	6.67
MD4138	658712.94	897875.75	-968.11	602	115.5	1.5	1.16
MD4138	658713.25	897874.12	-967.71	604	117.4	1.9	2.126
MD4139	658700.81	897932.88	-992.27	608	59.65	13.9	7.651
MD4139	658705.94	897910.94	-990.49	602	75.7	0.85	13.6
MD4139	658706.81	897907.31	-990.2	604	82.3	6.6	10.623
MD4139	658710	897893.06	-989.08	604	97.5	7.6	3.966
MD4140	658686.19	897915.44	-972.05	608	84.6	23.6	7.153
MD4140	658685.75	897902.5	-967.69	502	88.3	3.7	3.932
MD4140	658685.5	897898.31	-966.27	602	93.5	5.2	1.627
MD4140	658685.12	897891.44	-964.03	602	99.25	2.25	14.933
MD4140	658684.25	897873.38	-958.05	602	125.6	16.7	0.579
MD4147	658688.81	897942.88	-993.15	608	50	17.6	6.257
MD4147	658689	897912	-990.66	602	73.5	2.5	11.192
MD4147	658689	897908.31	-990.37	602	77	2.1	0.597
MD4147	658688.94	897899.88	-989.71	602	88	7.2	5.33
MD4155	658698	897949.5	-1008.51	608	74.6	6.65	8.108
MD4156	658679.19	897962.06	-1014.4	608	59.65	3.35	27.45
MD4157	658669.44	897963.25	-1010.37	608	49.8	1.05	3.53
MD4157	658672.12	897964.81	-1011.04	608	54.05	3.15	1.212
MD4158	658664.12	897970.38	-1022.83	608	56.15	5.8	13.842
MD4159	658665.75	897976.38	-1033.59	608	65.6	5.05	4.541
MD4160	658652.88	897975.88	-1025.51	608	52.65	2.5	3.329
MD4161	658648.81	897979.62	-1031.31	608	57.05	3.2	8.723
MD4289	658743.31	897812.19	-894.97	500	95.45	6.85	0.052
MD4289	658740.69	897810.81	-899.52	600	99.5	4.05	11.538
MD4300	658757.19	897903.69	-983.21	610	33.35	33.35	6.983
MD4300	658736.56	897884.5	-985.59	604	46.75	3.6	0.301
MD4301	658750	897896.88	-971.43	610	51.75	47.6	2.129
MD4301	658719.56	897868.5	-957.76	604	75.35	7.2	2.571
MD4301	658711.19	897861.06	-954.05	602	86.65	6.25	0.021
MD4302	658740.12	897898.5	-981.91	610	23.9	20.45	1.82
MD4303	658763.25	897887.62	-968.08	610	32.9	31.8	2.902
MD4304	658775.62	897899.69	-992.39	604	46.45	27.3	2.816
MD4304	658796.06	897923.94	-1004.25	610	87.05	40.6	1.702
MD4305	658774.88	897899.5	-979.93	610	46.45	33.45	0.753
MD4307	658747.5	897899.31	-986.46	610	28.35	28.35	2.712
MD4307	658748.62	897936.38	-1002.81	610	55.5	1.65	4.817
MD4308	658753.62	897890.44	-984.46	610	14.1	14.1	1.78
MD4308	658768.12	897904.81	-995.42	604	46	31.5	1.704
MD4308	658779.5	897915.62	-1003.83	610	50	4	0.108
MD4309	658761.06	897905.75	-989.96	610	50	50	5.725
MD4309	658778.62	897930.81	-1002	610	65.8	15.8	2.576
MD4310	658752.62	897892.12	-985.67	610	17.2	17.2	2.818
MD4310	658760.69	897904.56	-995.01	604	35.05	17.85	1.074
MD4310	658770.38	897918.06	-1005.51	604	47.2	2.75	2.02
MD4310	658771.75	897920	-1006.99	610	50	2.8	3.159
MD4328	658766.75	897910.88	-988.98	610	15.35	15.35	7.826
MD4328	658761.94	897904.12	-998.69	604	24.45	8	0.848
MD4329	658753.88	897902	-989.66	610	38.2	38.2	1.845
MD4329	658771.19	897945.69	-1015.73	610	86.1	26.5	3.788
MD4330	658749.75	897895.5	-986.6	610	21.85	21.85	1.148
MD4330	658752.12	897906.25	-993.21	604	25.65	3.8	6.907
MD4330	658762	897946.88	-1017.85	610	78.45	12.35	6.281
MD4331	658744.69	897897.31	-986.41	610	24.35	23.45	1.642
MD4331	658742.56	897908.56	-992.14	604	26.4	2.05	5.244
MD4332	658804.81	897912.19	-985.07	610	40.5	8.3	0.222

MD4337	658792.62	897870.88	-943.72	610	79.55	3.7	0.128
MD4364	658754.62	897891.75	-969.45	610	20.7	18.05	0.887
MD4365	658752.44	897905.44	-981.17	610	37.9	36.6	3.838
MD4365	658716.62	897883.12	-981.09	604	68.9	14.15	2.656
MD4366	658749.12	897906.88	-981.26	610	43.25	42.25	5.562
MD4366	658707.44	897889	-981.37	604	73.2	11.55	5.665
MD4367	658743.75	897909.25	-981.52	610	51.6	50	3.25
MD4367	658718.75	897902.44	-981.45	610	53.5	1.9	8.667
MD4367	658714.12	897901.38	-981.44	608	61	7.5	1.795
MD4367	658694.06	897894.88	-981.31	602	80.4	4.1	8.675
MD4367	658684.19	897891.69	-981.26	602	95.85	14.2	3.841
MD4368	658752	897913.5	-981.3	610	33	30.5	6.332
MD4368	658720.06	897908.44	-980.56	608	51.1	2.05	0.02
MD4368	658715.12	897907.62	-980.45	502	59	7.9	0.038
MD4368	658708.88	897906.69	-980.31	608	63.8	4.8	5.971
MD4368	658691.25	897904.06	-979.93	602	86.1	13.75	9.327
MD4368	658660.75	897899.75	-979.05	602	113.35	6.75	0.229
MD4369	658765.38	897903.5	-987.17	610	50	50	3.329
MD4369	658785.44	897923.44	-995.07	610	58.65	8.65	0.523
MD4394	658678.56	897943.75	-993.11	608	22.45	22.45	11.385
MD4394	658650.25	897915.62	-975.42	602	64.6	19.4	1.771
MD4394	658641.62	897907.19	-970.13	600	71.55	6.95	0.871
MD4394	658637.31	897903	-967.5	600	76.8	4.35	6.766
MD4395	658685.31	897950.62	-1001.27	608	3.8	3.8	7.433
MD4395	658673.94	897939.12	-1015.1	602	23.7	1.1	0.011
MD4395	658667.62	897932.81	-1022.75	604	35.8	1.8	0.369
MD4396	658671.31	897950.81	-995.05	608	17.7	17.7	14.185
MD4396	658645.88	897925.81	-981.58	602	48.45	3	20.238
MD4396	658638.5	897918.75	-977.74	602	66.6	17.4	2.765
MD4396	658629.75	897910.56	-973.33	602	72.35	3.35	4.068
MD4398	658660.31	897955	-995.21	608	11.4	11.4	5.739
MD4398	658638.69	897934.31	-978.35	602	46.2	12.35	6.57
MD4398	658632.5	897928.38	-973.57	500	53.5	7.3	0.408
MD4398	658625.19	897921.38	-967.92	600	69.4	15.9	3.961
MD4398	658613.81	897910.5	-959.18	600	88.6	18.35	8.674
MD4399	658660.94	897955.44	-996.93	608	8.95	8.95	2.313
MD4399	658631.81	897925.94	-981.52	602	59.3	21.35	3.724
MD4399	658621.62	897915.62	-976.19	602	65.75	3.25	3.187
MD4400	658663.06	897957.44	-1000.89	608	2.6	2.6	2.492
MD4400	658648	897941.5	-1011.8	602	27.7	3.75	12.952
MD4400	658637.69	897930.56	-1019.33	602	43.65	2	0.015
MD4415	658733.38	897935.81	-997.19	610	10.2	3.1	0.021
MD4416	658770.5	897927.44	-996.37	610	60.55	36.75	4.313
MD4417	658764.06	897917.94	-996.87	610	43.35	12.2	3.863
MD4417	658780.75	897913.5	-996.67	610	64.7	20.45	6.533
MD4418	658772.62	897907.25	-997.94	604	55.1	12.55	15.633
MD4419	658748.06	897905.81	-997.14	604	38.35	19.35	5.692
MD4420	658731.69	897906.19	-997.08	604	28.55	17.55	6.431
MD4421	658719.44	897910.56	-997.52	604	17.5	4.95	7.263
MD4422	658721.94	897917.19	-1005.77	604	12.4	2.9	9.43
MD4422	658720	897911.25	-1011.11	604	19.65	1	0.01
MD4423	658711.5	897911.75	-997.87	604	19.25	3.9	16.456
MD4423	658703.06	897894.69	-998.34	604	40.45	8.25	2.657
MD4424	658705.5	897914.81	-1000.59	604	18.7	0.6	19.43
MD4424	658698.81	897908.19	-1002.07	604	28.7	1.6	0.239
MD4424	658691.38	897900.94	-1003.76	604	41.7	6.6	3.449
MD4425	658710.25	897919.31	-1007.12	604	15.3	1	8.25
MD4425	658704.69	897913.62	-1013.02	604	25.4	1.4	0.274
MD4426	658703.81	897927.62	-1009.61	604	11.7	0.3	35.7
MD4427	658698.25	897935.94	-998.1	608	1.1	1.1	10.174

MD4427	658678.75	897915.19	-998.3	602	29.6	1.2	2.05
MD4427	658675.44	897911.81	-998.34	602	34.95	2.45	0.699
MD4427	658671.31	897907.44	-998.38	602	40.1	0.65	0.65
MD4428	658698.56	897936.25	-999.15	608	0.2	0.2	1.56
MD4428	658688.62	897925.56	-1009.37	604	18.45	1.05	16.663
MD4428	658682.75	897919.25	-1015.41	604	29	1.1	0.336
MD4429	658687.25	897936.12	-992.1	608	18.75	18.75	17.823
MD4429	658665.25	897914.19	-974.35	602	51.5	12.7	10.548
MD4429	658655.06	897904.56	-966.4	600	71.05	19.55	7.231
MD4429	658644.88	897895	-958.55	600	79.85	5.15	4.496
MD4430	658692.06	897941.06	-998.91	608	3.05	3.05	12.162
MD4430	658677.12	897925.31	-1002.46	602	24.35	1.7	20.059
MD4430	658672.44	897920.5	-1003.57	602	31.5	2.3	0.098
MD4430	658668.12	897916	-1004.59	602	38.75	4.25	0.123
MD4431	658693.12	897942.12	-1000.61	608	1.5	1.5	15.587
MD4431	658687.31	897936.44	-1017.96	604	20.4	0.95	0.89
MD4432	658651.44	897959.38	-997.61	608	7.3	7.3	1.897
MD4432	658625	897931.62	-985.69	602	52.6	17.7	4.237
MD4432	658616.06	897921.88	-981.8	602	58.75	2.4	6.649
MD4432	658613.69	897919.31	-980.77	602	62.4	2.4	0.016
MD4433	658653.38	897961.31	-1000.89	608	2.1	2.1	3.101
MD4433	658638.25	897946.25	-1010.96	602	26.2	3.05	1.491
MD4433	658629.25	897936.69	-1017.4	602	40	1.4	3.05
MD4435	658641.06	897963.31	-998.39	608	9	9	5.242
MD4435	658606.5	897929.25	-988.5	602	55.55	2.9	8.608
MD4436	658620.44	897977.44	-999.05	502	1	1	0.34
MD4436	658619.38	897975.19	-998.2	608	5.25	4.25	2.087
MD4436	658605.31	897946.19	-987.96	602	39	4.05	7.384
MD4436	658601.5	897938.19	-985.39	600	53.4	14.4	3.353
MD4438	658618.62	897977.56	-1000.94	608	2.7	2.7	12.413
MD4438	658598.44	897962.12	-1008.9	602	29.1	2.2	1.601
MD4438	658589.56	897955.06	-1012.57	602	40.05	0.35	0.01
MD4439	658618.12	897976.62	-998.18	608	5.5	5.5	0.807
MD4439	658613	897973.12	-995.89	606	13.3	7.8	7.194
MD4439	658606.69	897968.38	-993.18	500	22.2	8.9	0.245
MD4439A	658617.81	897976.06	-997.89	608	6.8	6.8	3.703
MD4439A	658614.5	897972.12	-995.95	606	11	4.2	0.028
MD4439A	658609.88	897966.69	-993.27	500	22	11	1.37
MD4439A	658606	897961.94	-991.19	606	24	2	4.025
MD4439A	658595.62	897948.81	-985.51	600	47.7	14	5.079
MD4440	658742.12	897880	-950.65	608	32.45	2.35	3.352
MD4440	658733.62	897872.69	-946.67	502	53.9	21.45	0.28
MD4440	658725.56	897865.81	-942.89	602	55	1.1	0.04
MD4440	658719.38	897860.56	-940.04	500	71	16	0.13
MD4440	658707.88	897851.06	-934.8	600	86.65	15.65	4.599
MD4440	658692.94	897838.69	-928.04	600	107	15.35	4.917
MD4441	658754.25	897880.31	-953.3	608	25.5	5.1	2.754
MD4441	658747.81	897869.38	-948.67	502	47.5	22	0.062
MD4441	658742.38	897860.06	-944.79	602	48.4	0.9	0.07
MD4476	658626.81	897904.94	-934.42	606	90.45	1.05	0.02
MD4476	658624.62	897902.62	-933.87	500	95.75	5.3	0.087
MD4476	658620.38	897898.56	-932.87	600	102.4	6.65	2.151
MD4476	658612.38	897890.81	-930.97	600	113.3	5.95	4.067
MD4477	658636.62	897915.88	-945.93	606	74.05	0.85	0.09
MD4477	658631.25	897910.5	-945.33	500	88.45	14.4	0.44
MD4477	658625.69	897905	-944.7	600	89.8	1.35	4.02
MD4477	658617.69	897897.5	-943.78	600	103.55	6.75	7.286
MD4477	658611.19	897891.44	-943.04	600	109.3	0.45	0.25
MD4478	658651.5	897929.12	-961.86	606	56.25	3.3	1.772
MD4478	658645.25	897922.56	-963.47	500	71.3	15.05	0.093

MD4478	658636.81	897913.69	-965.63	600	81.2	9.9	2.971
MD4478	658631	897907.5	-967.11	600	87.5	5.35	2.094
MD4479	658606.19	897877.88	-913.75	606	130	3.6	8.829
MD4479	658603.19	897874.88	-912.55	500	135.2	5.2	0.21
MD4479	658601.25	897872.94	-911.77	600	135.7	0.5	2.61
MD4479	658593.5	897865.25	-908.67	600	152.4	11.3	8.539
MD4480	658640.94	897905.31	-939.44	606	95.4	1.95	0.781
MD4480	658636.25	897901.69	-938.97	500	105.35	9.95	1.378
MD4480	658632.12	897898.56	-938.57	600	105.75	0.4	4.13
MD4480	658622.31	897891.06	-937.6	600	122.7	9.5	5.404
MD4482	658595.69	897925.81	-945.02	606	81.2	0.5	2.55
MD4482	658593	897924.38	-944.81	500	86.9	5.7	0.144
MD4482	658584.44	897919.81	-944.16	600	100.75	13.85	4.715
MD4483	658616.25	897910.88	-937.32	606	75	0.95	1.18
MD4483	658614.62	897909.25	-936.93	500	78.7	3.7	0.164
MD4483	658608.19	897902.75	-935.36	600	93.6	14.9	5.161
MD4483	658600.19	897894.81	-933.46	600	97.7	0.25	0.07
MD4484	658627.56	897930.19	-957.14	606	54.8	4.7	0.193
MD4484	658622.75	897926	-957.89	500	62.85	8.05	0.072
MD4484	658611.69	897916.44	-959.61	600	84.35	21.5	3.972
MD4485	658612.25	897918.25	-942.17	606	73.8	3.95	4.51
MD4485	658609.12	897915.75	-941.75	500	77.9	4.1	0.24
MD4485	658596.69	897905.56	-940.1	600	96.2	8.4	9.132
MD4485	658591.62	897901.44	-939.42	600	99.15	1.2	0.02
MD4486	658596.75	897906.75	-929.21	606	93.8	1.9	0.757
MD4486	658595.62	897905.88	-928.91	600	94.8	1	0.07
MD4486	658592.81	897903.62	-928.15	500	101.2	6.4	0.264
MD4486	658585.88	897898.19	-926.29	600	112.85	11.65	1.422
MD4486	658573.5	897888.5	-923.02	600	124.5	2.9	2.109
MD4487	658629.19	897939.31	-963.34	606	47.2	0.9	2.59
MD4487	658623.62	897935.44	-965.16	500	60.2	13	0.355
MD4487	658613.44	897928.69	-968.54	600	72.6	12.4	1.024
MD4487	658604.25	897922.56	-971.58	600	82	8.25	5.333
MD4488	658594.44	897917.69	-935.66	606	88.05	1.75	0.646
MD4488	658593	897916.75	-935.38	500	89.85	1.8	0.02
MD4488	658583.75	897911.12	-933.93	600	110	20.15	10.597
MD4488	658570.31	897903	-931.89	600	119.65	7.75	1.964
MD4489	658624.75	897917.5	-947.02	606	63.65	1.5	0.19
MD4489	658623.44	897916.12	-946.93	500	66	2.35	0.054
MD4489	658615.38	897907.62	-946.37	600	87	21	4.28
MD4489	658605.25	897896.94	-945.69	600	91.8	1.1	0.01
MD4490	658611.75	897939.31	-959.71	606	62	1.95	0.382
MD4490	658608.19	897937.62	-960.3	500	67.95	5.95	0.034
MD4490	658594.62	897931.25	-962.66	600	92.45	24.5	4.155
MD4491	658558.88	897917.31	-933.48	600	132.15	26.45	3.533
MD4492	658587.06	897931.69	-949	606	86.6	1	12.4
MD4492	658585.75	897931.06	-949	600	88.55	1.95	0.125
MD4492	658582.44	897929.69	-949	500	93.75	5.2	0.225
MD4492	658575.38	897926.62	-949.01	600	103.95	10.2	4.198
MD4493	658606.69	897951.56	-968.64	606	68.65	10.25	0.862
MD4493	658598.44	897949.56	-970.92	500	75.9	7.25	1.897
MD4493	658593.62	897948.38	-972.28	600	79	3.1	29.897
MD4494	658563.31	897932.12	-942.57	600	117.7	18.75	1.763
MD4494	658550.69	897928.06	-941.57	500	125.5	7.8	0.039
MD4494	658544.81	897926.19	-941.09	600	130.1	4.6	11.725
MD4497	658701.62	897894.94	-959.82	608	71.05	4.95	2.823
MD4497	658698.81	897891.06	-960.76	502	75.9	4.85	0.044
MD4497	658696.94	897888.56	-961.37	608	77.4	1.5	1.323
MD4497	658694.44	897885.06	-962.21	608	81.75	1.5	17.49
MD4497	658688.5	897876.94	-964.19	602	95.1	7.8	2.039

MD4498	658687.94	897886.81	-944.19	606	83.45	2.6	5.707
MD4498	658682.19	897880.06	-943.99	600	95.7	9.55	10.276
MD4498	658673.31	897869.69	-943.76	600	112.5	15.8	3.771
MD4499	658673	897871.06	-925.83	606	106.4	1.25	4.97
MD4499	658671.44	897869.38	-925.42	500	109.9	3.5	0.624
MD4499	658662	897859.25	-922.98	600	134.5	24.6	4.584
MD4500	658640.06	897835.44	-893.69	500	166.6	11.6	0.424
MD4500	658634.31	897829.31	-890.76	600	172.8	6.2	2.178
MD4502	658686.75	897895.12	-952.3	606	77.75	1.9	10.342
MD4502	658683.69	897892.06	-952.6	500	83.6	4.9	1.57
MD4502	658673.69	897882.25	-953.56	600	106.75	23.15	7.12
MD4503	658672.56	897883.38	-933.4	606	96.9	2.25	2.712
MD4503	658668.81	897879.88	-932.59	500	105	8.1	0.108
MD4503	658657.75	897869.5	-930.22	600	127.6	22.6	5.24
MD4504	658647.62	897858.12	-913.28	500	141.7	14.45	0.612
MD4504	658635.06	897846.12	-909.12	600	163	21.3	2.039
MD4505	658623.75	897844.31	-895.59	500	170.5	10.9	1.024
MD4505	658617.19	897838.62	-893.05	600	177.8	7.3	8.601
MD4506	658671.5	897892.94	-941.08	606	90	1.2	2.99
MD4506	658657.31	897881.06	-939.86	600	121.85	27.75	5.89
MD4507	658634.06	897866.44	-912.2	606	139.55	0.85	0.16
MD4507	658632.31	897865.19	-911.7	500	143.1	3.55	0.063
MD4507	658629.5	897863.06	-910.88	606	146.85	3.75	3.339
MD4507	658617.25	897854.06	-907.34	600	174.4	27.55	4.218
MD4509	658668	897823.19	-892.09	606	116.4	2.7	0.038
MD4509	658663.75	897819.19	-891.22	500	125.5	9.1	0.089
MD4509	658656.62	897812.5	-889.78	600	136.2	10.7	1.143
MD4510	658692.75	897846.5	-911.96	606	82.1	6.1	0.103
MD4510	658682	897836.5	-911.62	500	105.3	23.2	0.396
MD4510	658668.19	897823.81	-911.22	600	119.7	14.4	6.143
MD4511	658716.31	897867.31	-929.29	606	51.25	2.8	0.172
MD4511	658713.5	897864.44	-930.53	500	56.9	5.65	0.36
MD4511	658704.38	897855.31	-934.46	600	78.2	21.3	2.557
MD4513	658669.81	897836.88	-903.71	500	108.5	11.4	0.143
MD4513	658664.81	897833	-903.26	606	109.8	1.3	4.12
MD4513	658656.56	897826.56	-902.5	600	129.5	19.7	5.457
MD4514	658696.12	897858.5	-922.94	606	69.45	1.25	0.05
MD4514	658692.19	897855.31	-923.55	500	78	8.1	0.214
MD4514	658682.44	897847.44	-925.04	600	95	17	5.825
MD4515	658633.94	897828.62	-885.9	500	140.6	3.7	0.495
MD4515	658630.94	897826.81	-885.2	600	144	3.4	0.502
MD4516	658672.31	897849.81	-913.36	606	94.15	3.95	1.664
MD4516	658665.5	897845.38	-913.25	500	106.35	12.2	0.131
MD4516	658655	897838.69	-913.05	600	119	12.65	2.867
MD4517	658694.75	897869.88	-929.95	606	64.9	1.5	0.443
MD4517	658692.81	897868.81	-930.46	500	68	3.1	0.087
MD4517	658679.56	897861.12	-933.96	600	96.3	28.3	4.237
MD4518	658709.38	897879.12	-948.98	606	58.9	5.6	3.544
MD4518	658705.19	897876.81	-952.32	500	64.95	6.05	0.31
MD4518	658701.56	897874.81	-955.21	608	69	4.05	1.78
MD4518	658694.56	897870.94	-960.81	602	80.7	7.85	0.781
MD4519	658636.44	897892.19	-928.85	500	108.8	1.7	0.26
MD4519	658628.25	897884.81	-926.84	600	129.45	20.65	6.662
MD4520	658659.81	897911.69	-946.4	606	76.5	2	12.217
MD4520	658653.75	897905.81	-946	500	91.45	14.95	0.192
MD4520	658644.88	897897.31	-945.48	600	101.1	9.65	4.925
MD4520	658636.94	897889.62	-945	600	111.75	8.8	4.164
MD4521	658631.88	897868.88	-914.71	500	133.95	6.25	0.226
MD4521	658629.5	897866.31	-913.74	606	134.95	1	3.01
MD4521	658619.25	897855.75	-909.75	600	164.35	29.4	3.896

MD4522	658652.31	897890.88	-933.73	606	99.35	3.9	0.058
MD4522	658650.31	897888.62	-933.21	500	101.6	2.25	0.231
MD4522	658642.31	897879.62	-931.13	600	123.7	22.1	5.775
MD4523	658674.06	897914.94	-956.78	606	65	2.55	2.522
MD4523	658668.81	897908.69	-957.48	500	78.8	13.8	1.298
MD4523	658657.62	897895.38	-958.95	600	99.9	21.1	6.478
MD4524	658720.94	897826.38	-905.93	500	130.8	24.5	0.031
MD4524	658706.69	897811.31	-905.33	600	147.9	17.1	4.167
MD4525	658737.38	897848.62	-921.61	500	102	20.75	0.053
MD4525	658724.56	897836.44	-923.49	600	116.85	14.85	4.889
MD4526	658716.44	897822.38	-896.09	606	126.85	2.85	0.072
MD4526	658709.44	897815.56	-895.15	500	143.65	16.8	0.036
MD4526	658696.69	897803.69	-893.5	600	161.9	18.25	4.854
MD4528	658609.38	897870.69	-909.38	606	148	3.5	0.647
MD4528	658606.25	897868.06	-908.22	500	153	5	0.068
MD4528	658603.44	897865.75	-907.19	600	155.55	2.55	0.401
MD4528	658595.44	897859	-904.21	600	170.4	10.3	9.512
MD4529	658726.38	897879.19	-944.67	606	45	4	12.917
MD4529	658724.75	897877.69	-946.64	500	46.95	1.95	1.755
MD4529	658721.38	897874.56	-950.66	502	57.15	10.2	0.59
MD4529	658717.69	897871.12	-955.15	608	60.5	3.35	2.033
MD4529	658715.44	897869.06	-957.8	602	64.2	2.75	2.474
MD4529	658714.56	897868.25	-958.85	604	64.65	0.45	4.53
MD4530	658719.19	897857.06	-922.97	606	66.85	5.8	1.116
MD4530	658710.56	897848.94	-924.31	500	84.95	18.1	0.167
MD4530	658703	897841.69	-925.5	600	87.9	2.95	10.192
MD4530	658696.25	897835.38	-926.52	600	98.95	6.35	3.947
MD4531	658694.81	897834.19	-904.07	606	98.2	1.45	2.1
MD4531	658686.44	897826.06	-902.79	500	120.15	21.95	0.198
MD4531	658672.81	897813.62	-900.93	600	135.3	15.15	2.002
MD4532	658663.38	897812.19	-884.72	606	141.75	7.75	4.13
MD4532	658658.94	897808.38	-883.56	500	146	4.25	0.719
MD4532	658654.12	897804.19	-882.31	600	154.7	8.7	2.558
MD4533	658738.44	897865.31	-928.6	606	49.4	5.4	0.13
MD4533	658732.94	897857.81	-931.27	500	63.45	14.05	0.044
MD4533	658727.75	897850.75	-933.7	600	67.5	4.05	5.914
MD4533	658725.5	897847.75	-934.76	602	71.3	3.8	3.516
MD4534	658721.94	897847.88	-915.92	606	70.4	3.4	0.388
MD4534	658717	897841.88	-916.03	500	82.5	12.1	0.015
MD4534	658706.5	897829	-916.27	600	103.7	21.2	5.112
MD4535	658689	897816.69	-893.59	500	122.4	14	0.082
MD4535	658678.5	897805.38	-890.87	600	139.8	17.4	4.284
MD4537	658762.19	897831.19	-912.01	500	113.25	9.55	0.046
MD4537	658753.81	897823.44	-912.69	600	126.55	13.3	3.967
MD4538	658745	897800.19	-888.37	500	145.3	2.7	0.027
MD4538	658738.75	897792.75	-887.64	600	161.9	16.6	1.519
MD4552	658751.06	897791.69	-885.83	500	173.95	11.95	0.057
MD4552	658744.38	897786.38	-885.48	600	179.1	5.15	4.395

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX F

Golden Gift 2 Resource Model

November 2005

Golden Gift 2
Resource Model

November 2005

1 Introduction

The updated Golden Gift 2 resource model covers the area from 320N (6400N) to 341N (6820N) (see Figure 4, Long section). Bounded by the Cabernet Fault at the base and the South Fault above (both of which have be extruded north and south several sections and 'tied in' with the latest information from the Golden Gift 1 model) the GG2 model consists of a new interpretation for the area between 312.5N (6250N) and 322.75N (6455N). All the recent information including diamond drill holes, sludge sampling (not used for estimation but to define geological boundaries) and face and backs mapping has occurred since the Dec03 estimation. Beyond 322.75N (6455N) to the north no new information is available thus the original interpretation from August 2003 has be 'linked' together with the southern area to form the complete solid which spans 370m of strike.

Project directory S:\Mining-Geology\Golden Gift 2\Minesight\Geology\Current Model

Model is a rotated model. Model limits and rotation parameters are listed below.

Origin Point (AMG co-ordinates)
658400 E
898000 N
-1350 RL

Rotation Parameters
Rotation 1 – 310 (about Z axis)
Rotation 2 – 0 (about rotated X axis)
Rotation 3 – 15 (about rotated Y axis looking from end of axis towards origin)

Table 1: Model Limits

	Min	*Max*	*Cell Size*	*No. Cells*
East	0	240	2.5	96
North	0	525	7.5	70
RL	0	487.5	7.5	65

This documentation covers aspects of resource modelling specific to the Golden Gift 2 Model area. For general information such as sample techniques, assay quality, database storage, and resource estimation methodology the reader is referred to SGM Resource and Reserves Documentation, June 2002.

2 Data Sources

2.1 Data added since December 2004

2.1.1 Drilling

A total of 43 diamond holes have been added to the GG2 area since the last resource estimate in August 2004. This includes:

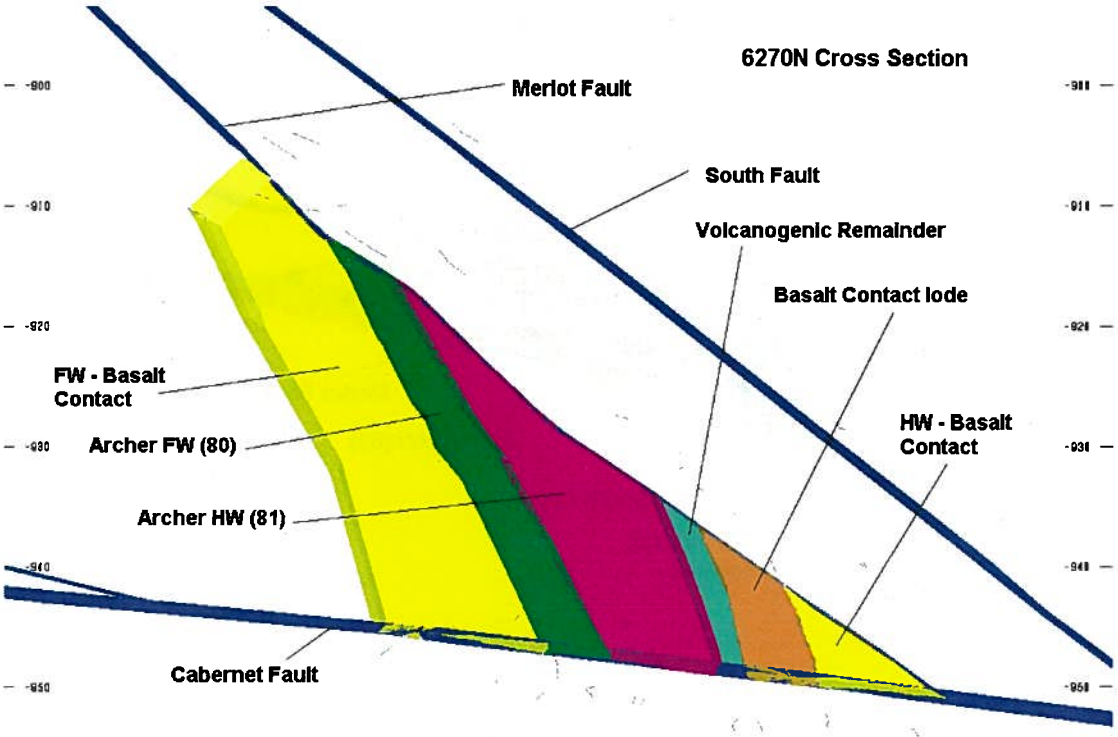


Figure 1: Cross section view at 6270N August 2004 Interpretation

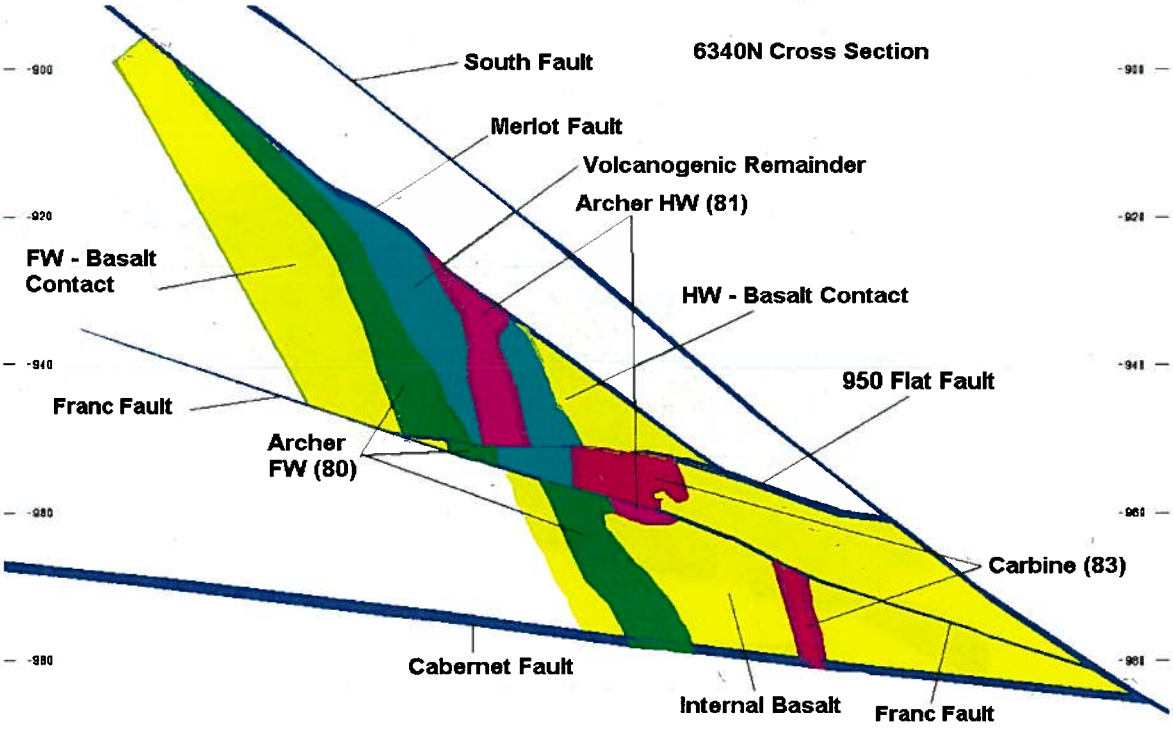


Figure 2: Cross section view at 6340N August 2004 Interpretation

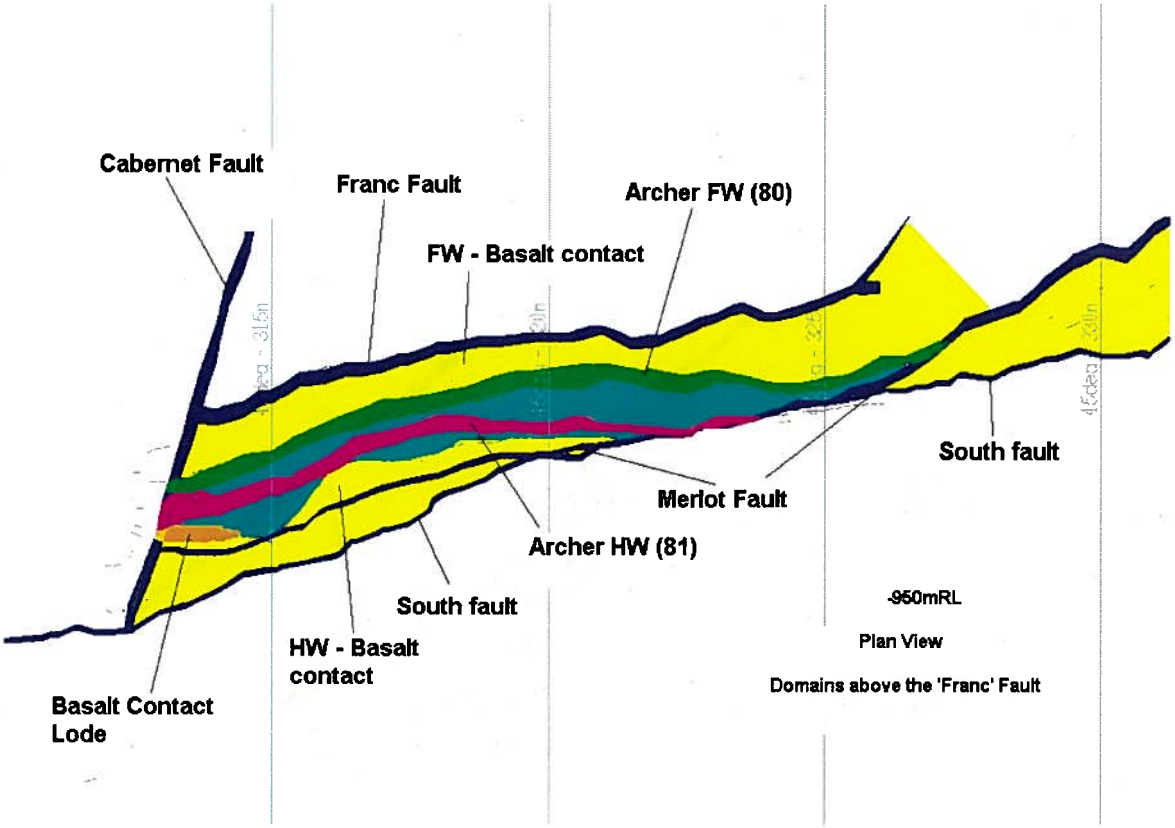


Figure 3: Plan View of August 2004 Interpretation at -945mRL.

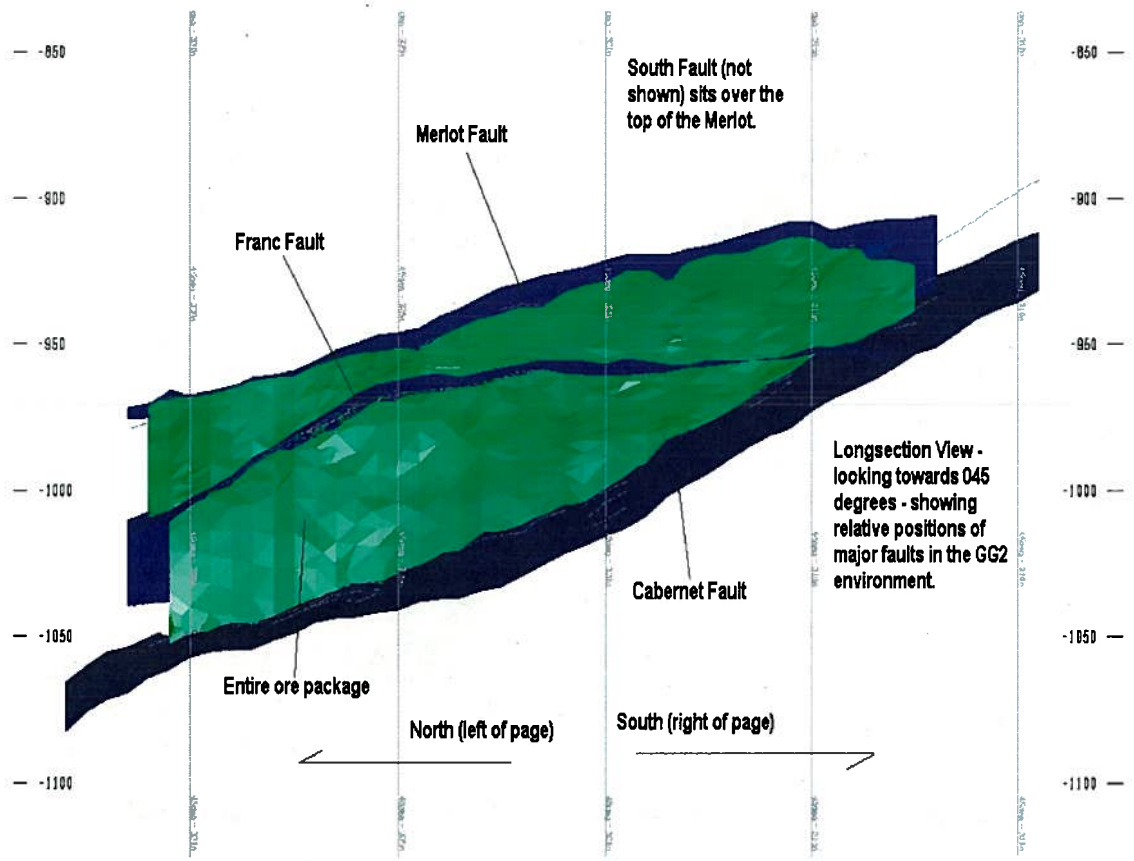


Figure 4: Long Section View (looking towards 045°)

Geological interpretation was initially undertaken on 10m paper sections on the 45 degree grid based on diamond drilling and limited development in the 950L during late 2003 by Dean Fredericksen and myself. The December 03 interpretation formed the base for the current August 04 update and given the strong continuity of geological features between drill holes the interpretation was carried out on screen in Minesight. Solids were constructed by then linking strings to form wireframe solids. Solids were overlapped, and then intersected against each other to form common surfaces. Considerable time was spent matching the South and Cabernet fault structures into the new GG1 interpretation. While the South fault extrudes through to GG1 with high confidence the Cabernet fault remains 25 to 30 metres in RL apart (see Figure 5). A drill program has been designed to test a 'GG1.5' lode (offset between the Archer and Patron (GG1) lodes).

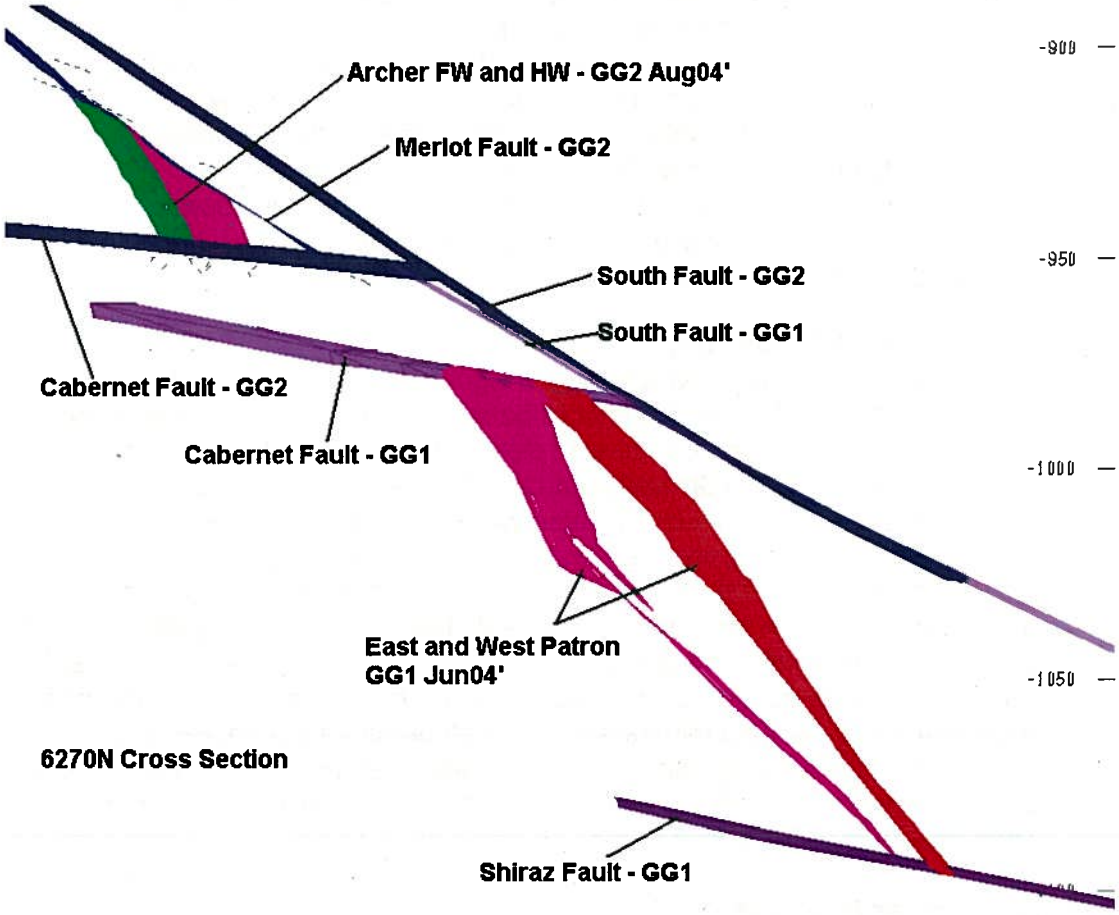


Figure 5: Cross section view at 6270N - GG2 and GG1

The following primary geological domains have been interpreted. Domain boundaries are based primarily on lithological or structural contacts.

Domain	Description
Archer FW Lode	Stawell database number 80 – most westerly of GG2 ore bodies found on the east dipping footwall
Archer HW Lode	Stawell database number 81 – the sometime massive sulphide lode east of the Archer FW
Volcanogenics Remaining	Volcanogenic remainder domain, which as the name suggests is the remaining Volcanogenic package after intersecting off the other domains.
Basalt Contact	Small domain designed to 'stem' the flow of strong grade found in MD4007 into the Archer HW lode or Volcanogenic remainder
Carbine Lode	Stawell database number 83 - Waterloo style ore package east of an internal basalt nose – only below '950 Flat' fault

Table 2: Domain Summaries

3.1.1.1 Archer FW Lode (80)

The Archer FW (80) lode has a historical significance at Stawell Gold Mines in that it was the first Golden Gift lode to be mined. The most westerly of the GG2 domains, the ore is found on the east dipping footwall which mostly consists of a thin lens of basalt 0.5 to 3m wide but can pinch out to leave a volcanogenic to mine schist contact. Given the position which can the be compared to a 'Central' lode in Magdala, the ore zone displays none of the strong quartz shear zone properties typical of Central lode, instead displaying an often 'dirty' graphitic sheared look with the only sulphide mineral visible being disseminated pyrrhotite, similar to the mineralisation seen higher up in the USF#1 block. Quartz tension veins with associated arsenopyrite occur intermittently but only account for a small percentage of gold mineralisation. Small basalt lenses appear and disappear in the space of 20m of strike and are dealt with on a stope by stope basis in consultation with the mine engineers due to their difficulties to model. *Lode indicated in figure 1, 2 and 3.*

3.1.1.2 Archer HW Lode (81)

The Archer HW (81) lode lies to the east of the of the Archer FW lode separated by a distance that can measure between 0 and 12 metres (weak volcanogenics fill the space between the lodes and are included in the 'volcanogenic remainder' domain). The boundaries of this lode are indicated by a sharp rise in grade associated with increases in sulphide and quartz percentages. The lode is usually located on the HW basalt contact, but occasionally situated several metres off the contact within the volcanogenic package. Towards the south end the ore body turns into an extremely strong massive pyrite zone which produced 60m of strike development on the 950-322-81Sth with consistent 10g/t plus face grades. Note that no Archer HW lode domain for the area between the 950 Flat fault and the Franc fault as the grade is concentrated on the FW lode in this area. *Lode indicated in figure 1, 2 and 3.*

3.1.1.3 Carbine Lode (83)

The Carbine (83) lode is a narrow basalt contact waterloo below the Franc fault on the east side of an internal basalt nose (similar to an 'oriental' type nose in Magdala). Currently there is 90 metres of strike development on the 990 Level which has exposed the Carbine lode. Face, sludge and several diamond holes have indicated that while the volcanogenics package is up to 12 metres wide, the ore zone is very narrow (although quite rich) averaging between 0.5 and 2 metres. I have domained out a small (but wider) zone between the Franc and 950 Flat fault which contains strong grade from one diamond hole to prevent the grade smearing into the 'Volcanogenic remainder' domain. *Lode is indicated in Figure 2.*

3.1.1.4 Basalt Contact Lode

The 'Basalt contact' volume contains just over 4600t and was created to prevent the smearing of very strong grade in MD4007. Unlike other holes in the 'stockwork' area at the southern end of GG2 which placed the Archer HW a considerable distance off the HW basalt contact (which pre MD4007 was thought to be barren), MD4007 contained strong mineralisation it's entire length with several plus 20g/t assay results located east of the massive sulphide zone which belongs to the Archer HW. *Lode is indicated in Figure 1.*

3.1.1.5 Volcanogenic Remainder

Which as the name suggests is the remaining volcanogenic package after intersecting off the other domains. Towards the very northern end of the model between 6570N and 6620N (-965mRL and -990mRL) this domain does contain some stronger (plus 4g/t) grades on what will eventually become the Archer HW lode, but at this stage requires follow-up drilling as currently the domain cannot be 'linked' through. *Lode is indicated in Figure 2 and visible in Figure 1, 3 and 4.*

4 Drillhole Data

Type of data	Assay	Survey	Composite	Description
Diamond drillholes, coded by geological domain.	gg0211.ddh	gg0212.ddh	gg0209.dat	1m ddh composites, all geologic domains.

Table 3: Drill hole Data Files

Diamond drill data was dumped from Acquire and loaded to Medsystem files gg0211.ddh and gg0212.ddh. These were the survey and assay files created as part of the 'gg02' project used in the Dec03' estimation and were updated using the <updatedh.sgm> procedure in 'compass'. 38 new holes were updated of which 33 holes intersected the GG2 model.

Due to the number of new holes and the extension of several domains since the Dec03' model a decision was made after careful consideration to 're-spear' the model. I understand the necessity of re-checking all coding following this procedure and spent several hours rechecking all drill holes section by section. 8 drill holes were manually changed due mostly to several 'self intersecting triangles' around the 'link' between the new interpretation and the north end model of Aug03'.

Note that MD2060W1 was edited back to a 'null' value as its position is suspect due to the length and direction after wedging, plus unlike the parent hole does not match

the geology. This hole was one of the original drilled from the 688 cuddy on the south decline steeply down through the GG2 domains.

5 Geological Coding

Drillholes, composites and block model utilise the same geological coding for domains. Codes values for the interim interpolation model (gg0215.dat) are:

Table 4: Geological Domains for Interim GG2 model

Domain	Model Code	DXF Filename
Archer Hangingwall between Merlot Fault and Franc Fault	601	601.dxf
Archer Hangingwall between Franc Fault and Tannat Fault	602	602.dxf
Archer Hangingwall between Tannat Fault and Cabernet Fault	603	603.dxf
Archer Stockwork between Merlot Fault and Franc Fault	621	621.dxf
Archer Stockwork between Franc Fault and Tannat Fault	622	622.dxf
Archer Stockwork between Tannat Fault and Cabernet Fault	623	623.dxf
Archer Volcanogenic between Merlot Fault and Franc Fault	501	501.dxf
Archer Volcanogenic between Franc Fault and Tannat Fault	502	502.dxf
Archer Volcanogenic between Tannat Fault and Cabernet Fault	503	503.dxf
Basalt Carbine between Merlot Fault and Franc Fault	521	521.dxf
Basalt Carbine between Franc Fault and Tannat Fault	522	522.dxf
Basalt Carbine between Tannat Fault and Cabernet Fault	523	523.dxf
Basalt Hangingwall	4	4.dxf
Mine Schist	1	1.dxf

Table 5: Geological Domains for Final GG2 Model

Codes values for the final interpolation model (gg0215.gg2) are:

Domain	Model Code	DXF Filename
Archer Hangingwall	600	600.dxf
Archer Stockwork	620	620.dxf
Archer Volcanogenic	500	500.dxf
Basalt Carbine	520	520.dxf
Basalt Hangingwall	4	4.dxf
Mine Schist	1	1.dxf

6 Composites

Experimentation at Stawell has shown that for the narrow, tabular lode structures like the Archer lodes, a 1m composite length is optimal. Figure 6 shows a histogram of interval length with a mean value of 1.02 metres

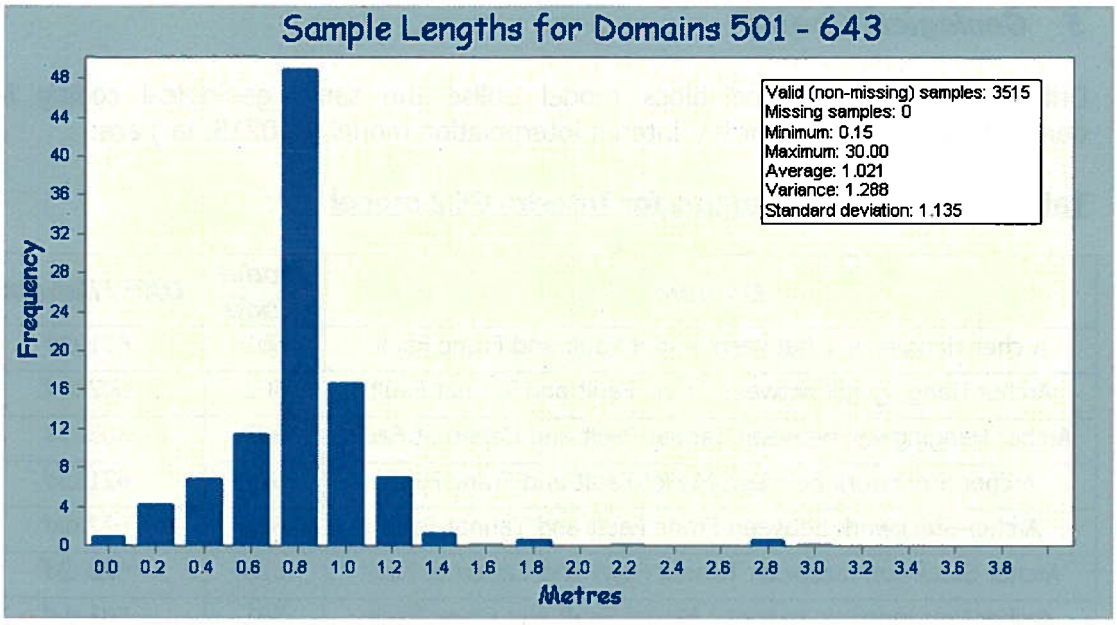


Figure 6: Histogram of interval length.

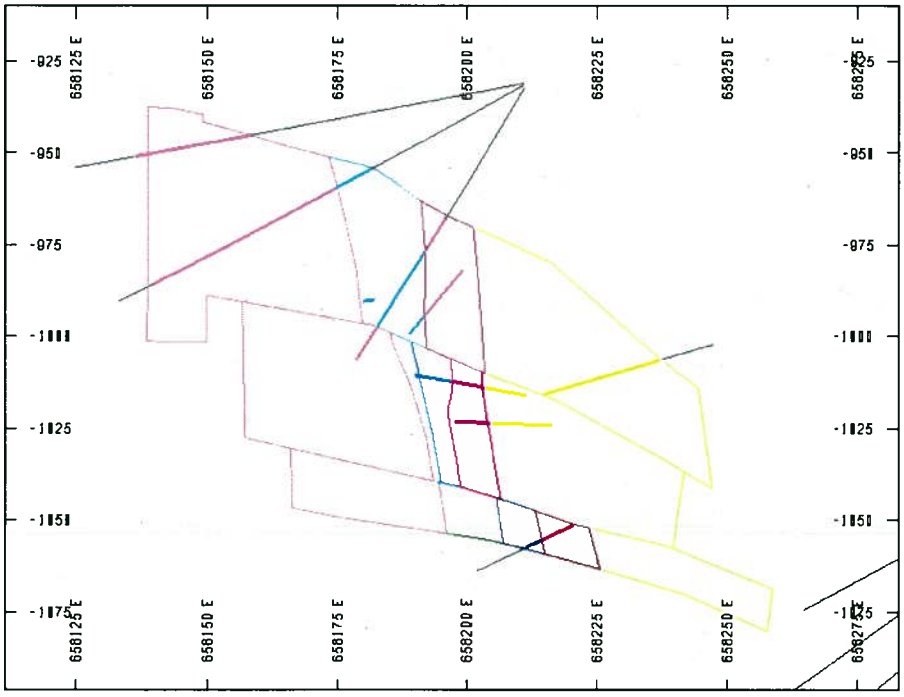


Figure 7: Cross Section 6610N showing coded sample intervals.

Table 5 Domain Composite Gold Statistics

Domain	#	min	max	mean	variance	c.v.
Mine Schist	485	0.005	16.189	0.21	1.154	5.116
Basalt Hangingwall	397	0.005	25.1	0.39	4.735	5.574
Archer Volcanogenic						
501	98	0.005	24.29	2.11	17.656	1.991
502	287	0.005	14.5	0.49	2.238	3.053
503	199	0.005	13.8	0.63	2.540	2.530
Basalt Carbine						
521	23	0.005	0.07	0.01	0.0003	1.830
522	168	0.005	17.44	1.88	18.577	2.293
523	49	0.005	12.603	1.36	8.775	2.178
Archer Hangingwall						
601	368	0.005	37.1	2.24	25.722	2.264
602	457	0.005	36.1	3.21	31.951	1.761
603	228	0.005	67.44	5.44	141.119	2.184
Archer Stockwork						
621	739	0.005	17.657	1.24	6.024	1.980
622	902	0.005	41.483	2.07	17.480	2.020
623	255	0.005	78.300	5.32	120.964	2.067

Table 6. Domain Composite SG Statistics

Domain	#	min	max	mean	variance	Cut SG
Mine Schist	359	1.03	4.61	2.52	0.115	2.54
Basalt Hangingwall	255	2.05	3.48	2.75	0.033	2.75
Archer Volcanogenic 501-503	309	1.75	3.49	2.66	0.055	2.66
Basalt Carbine 521-523	118	1.54	4.16	2.71	0.070	2.71
Archer Hangingwall 601-603	706	1.70	3.91	2.74	0.064	2.74
Archer Stockwork 621-623	1146	1.43	5.71	2.70	0.074	2.70

Table 6 Composite Statistics with Topcut Applied

Domain	#	Topcut Au g/t
Mine Schist	485	15
Basalt Hangingwall	397	20
Archer Volcanogenic		
501	98	20
502	287	10
503	199	10
Basalt Carbine		
521	23	-
522	168	15
523	49	-
Archer Hangingwall		
601	368	25
602	457	25
603	228	25
Archer Stockwork		
621	739	12
622	902	25
623	255	35

Print outs of the Excel 'Assay cutting' spreadsheet for each domain are supplied within the Appendix E.

7 Variography

Downhole variograms (correlograms) were calculated for all geological zones\units. Each geological zone has been offset\dislocated by the Franc and Tannat faults resulting in three separate fault blocks as described in Section 5 Geological Modelling. The three fault blocks for each geological domain were combined for the modeling of the downhole variograms.

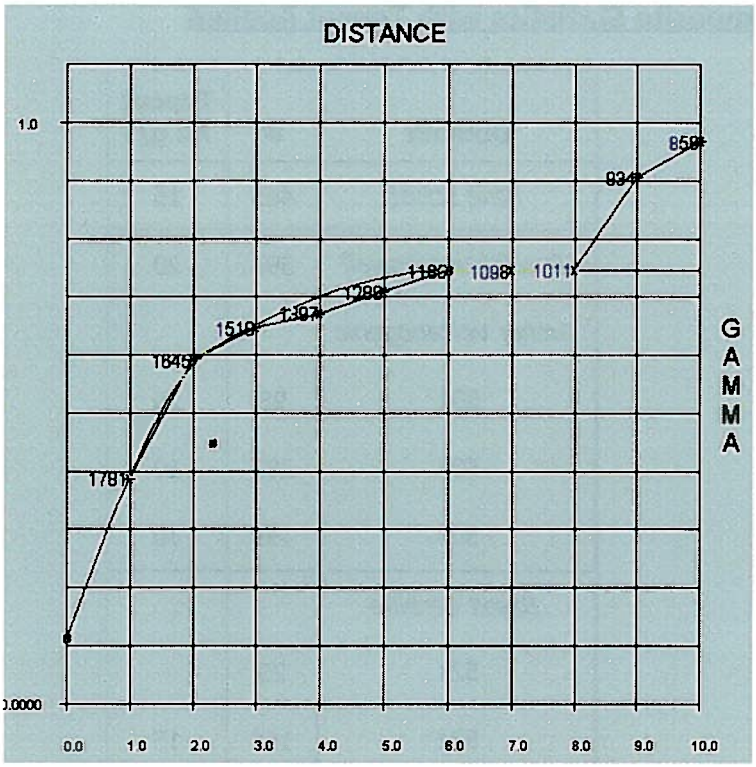


Figure 7: Archer Stockwork Domains 621-622-623 downhole variogram

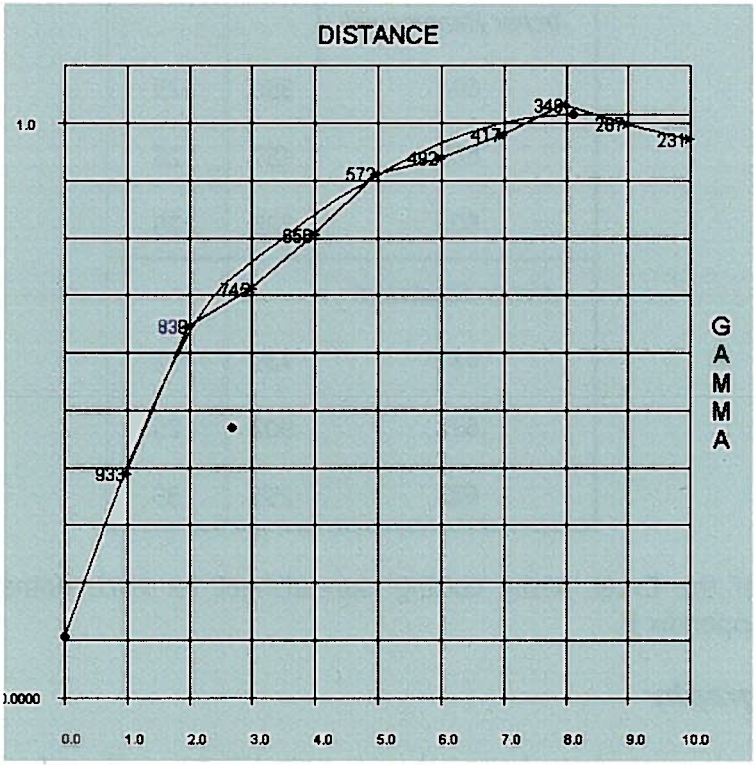


Figure 7: Archer Hangingwall Domains 601-602-603 downhole variogram

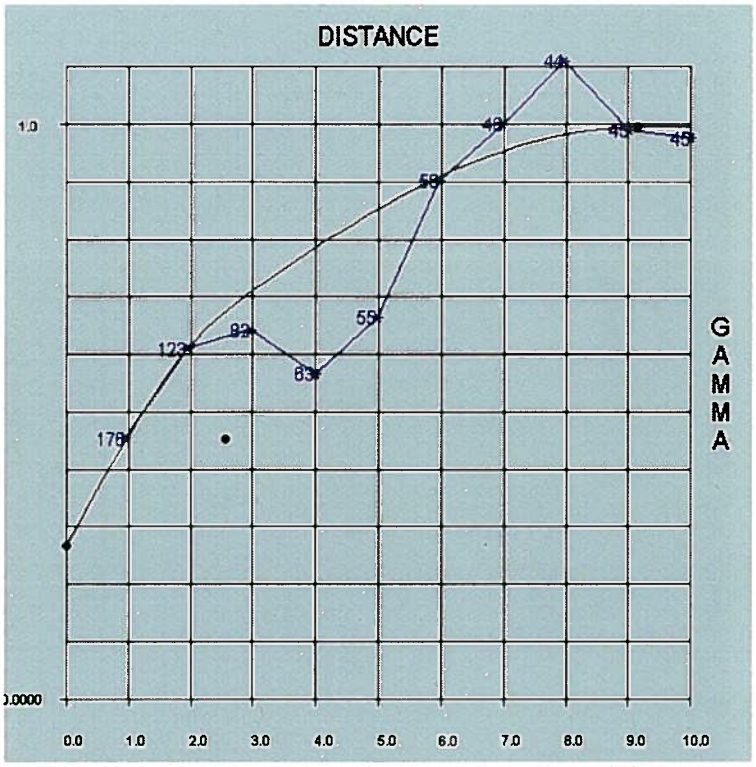


Figure 7: Basalt Carbine Domains 521-522-523 downhole variogram

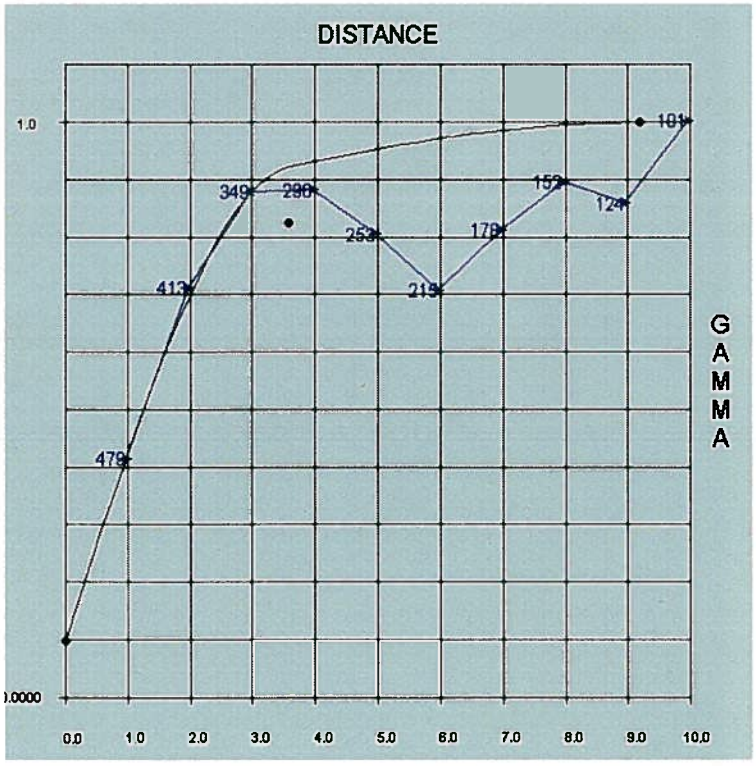


Figure 7: Archer Volcanogenics Domains 501-502-503 downhole variogram

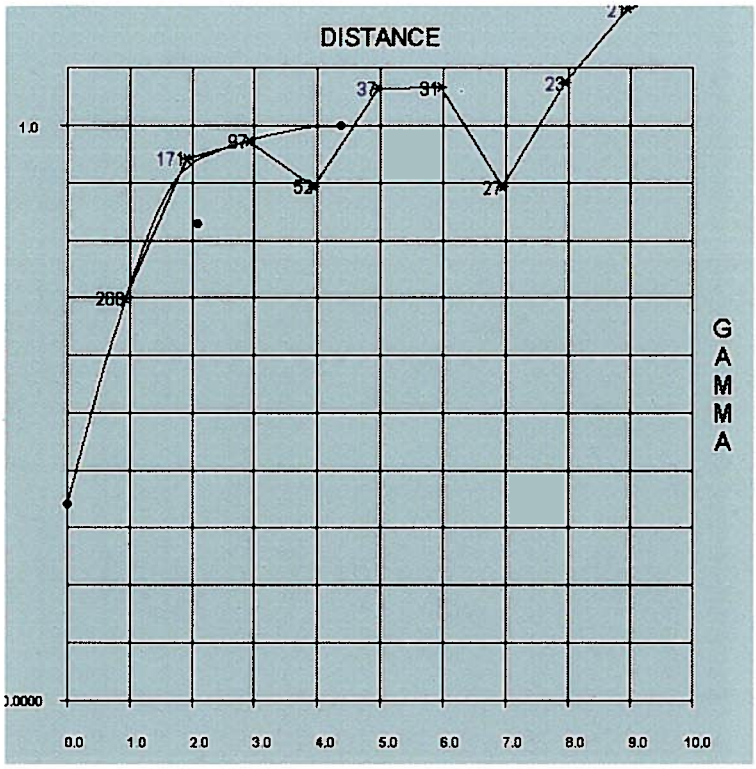


Figure 7: Basalt Hangingwall Domain 4 downhole variogram

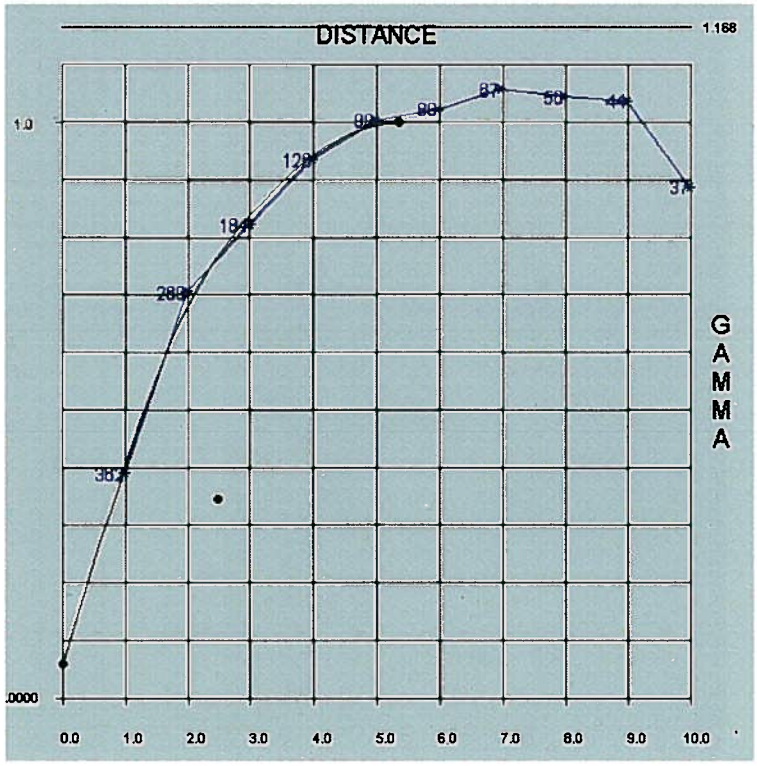


Figure 7: Mine Schist Domain 1 downhole variogram

Directional variograms tended to be poor. A combination of normalised normal variograms and correllograms were calculated, with correllograms producing the preferred results. For many of the fault block domains, the small datasets produced poorly structured experimental variograms.

The experimental variogram and models used are shown below:

Archer Volcanogenics

This geological zone, but more specifically each fault block within the domain, has very few composites with Domain 502 (fault block between the Franc and Tannat faults) has the most composites with a total of 287. These domains (501,502 and 503) were combined for the calculation and modelling of the variogram.

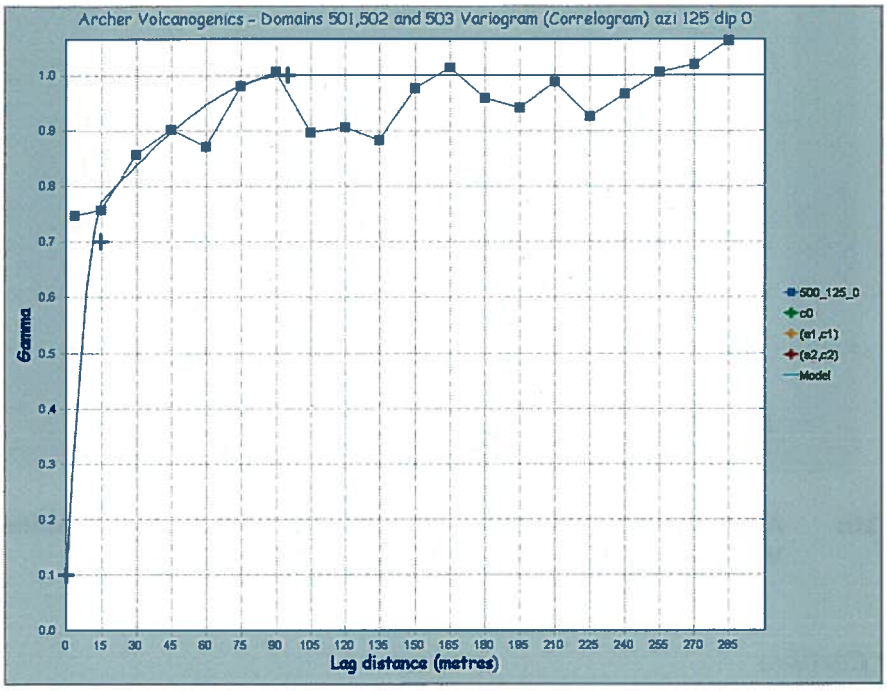


Figure 9: **Archer Volcanogenics – Domain 501,502 and 503 Combined Au Variogram – Major axis - Dip Dirn 125 Dip 0**

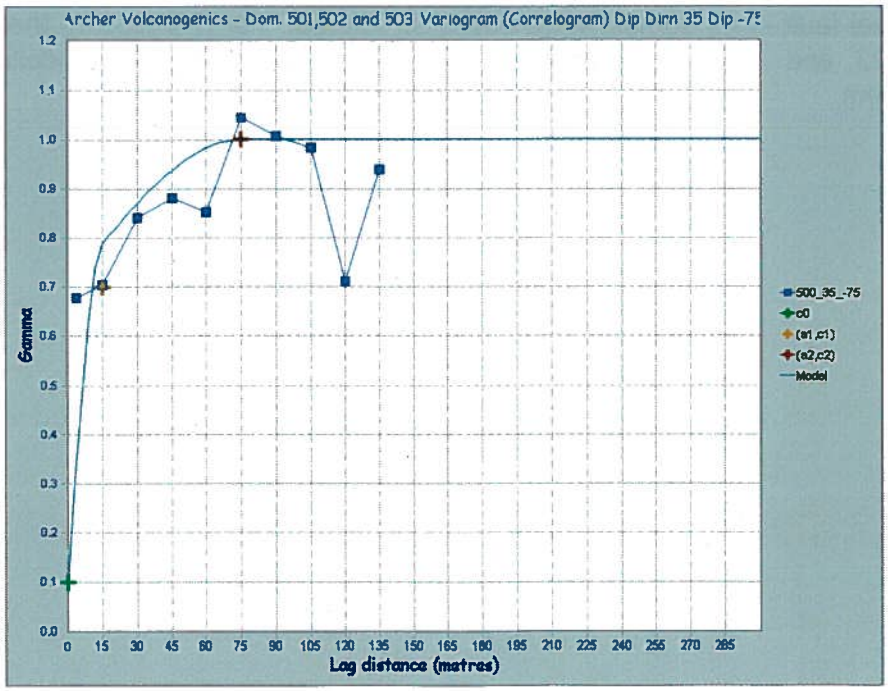


Figure 10: Archer Volcanogenics – Domain 501,502 and 503 Combined Au Variogram – Semi Major axis – Dip Dirn 35 Dip -75

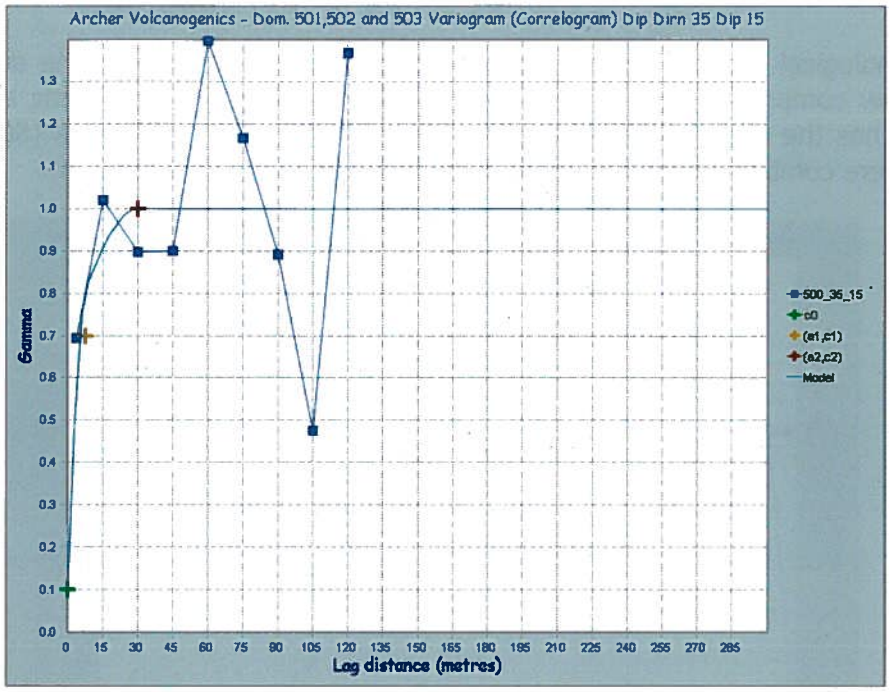


Figure 10: Archer Volcanogenics – Domain 501,502 and 503 Combined Au Variogram – Minor axis – Dip Dirn 35 Dip 15

Basalt Carbine

This geological zone, but more specifically each fault block within the domain, has very few composites with Domain 521, 522 and 523 having 23, 168 and 49 composites respectively. Due to the very small datasets, variograms for each individual fault block domain were poorly structured. For this reason, these domains (521,522 and 523) were combined for the calculation and modelling of the variogram.

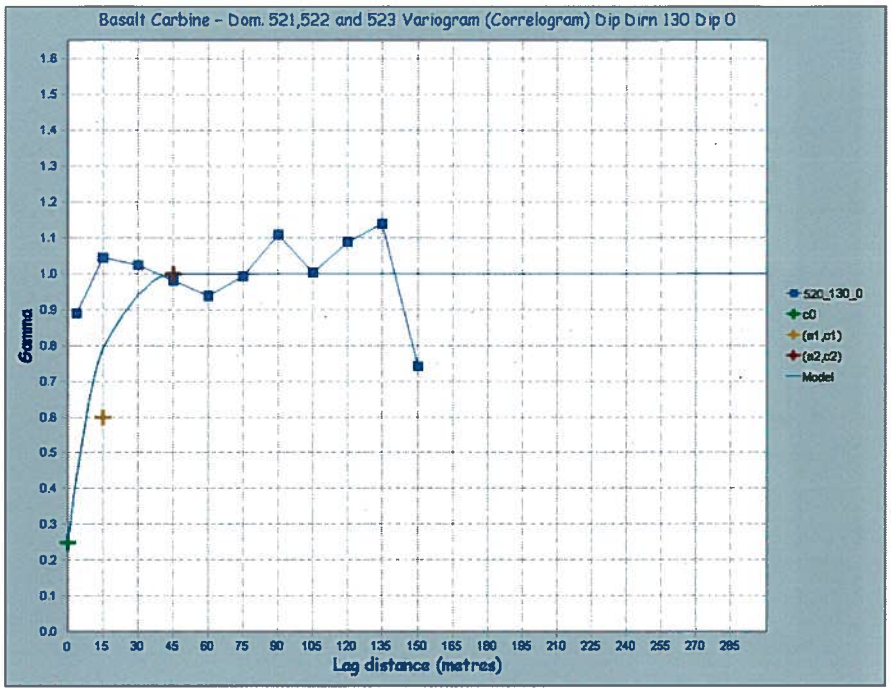


Figure 10: Basalt Carbine – Domain 521,522 and 523 Combined Au Variogram – Major axis – Dip Dirn 130 Dip 0

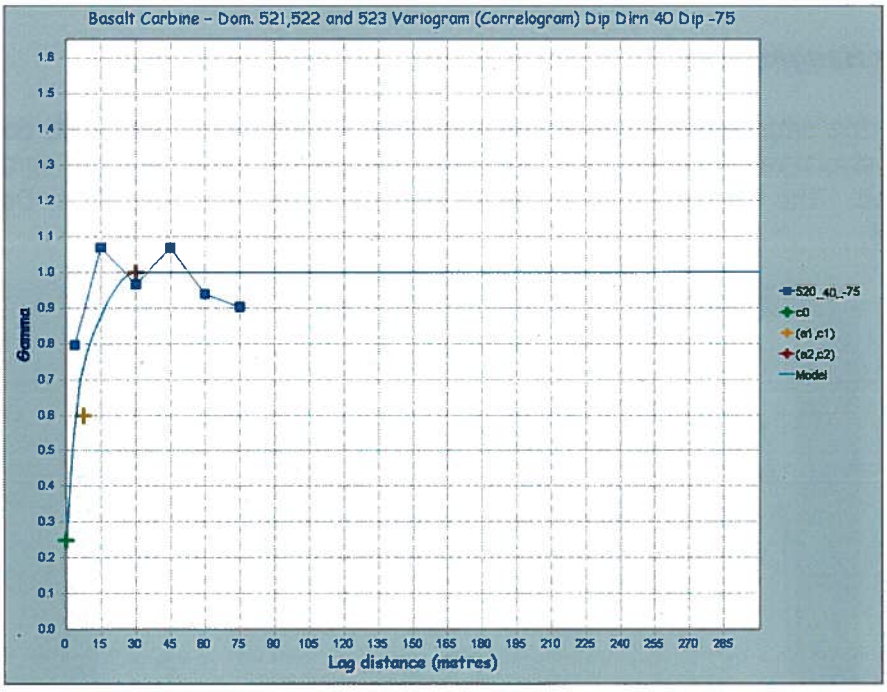


Figure 10: Basalt Carbine – Domain 521,522 and 523 Combined Au Variogram – Semi Major axis – Dip Dirn 40 Dip -75

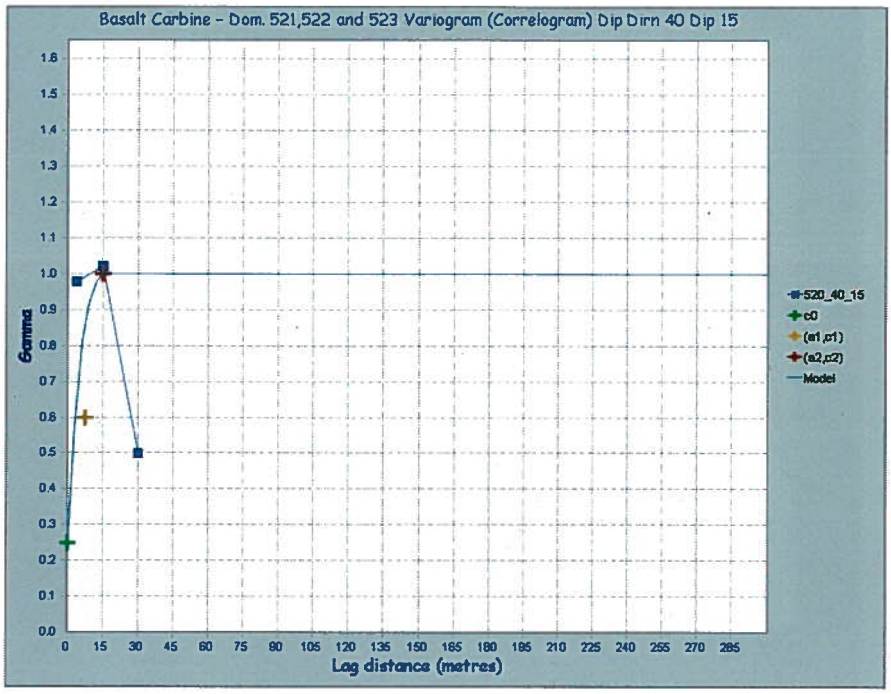


Figure 10: Basalt Carbine – Domain 521,522 and 523 Combined Au Variogram – Minor axis – Dip Dirn 40 Dip 15

Archer Hangingwall

Due to the small datasets, variograms for each individual fault block domain were poorly structured but Domain 602 had sufficiently structure variograms to be modelled. The variogram model for Domain 602 has been applied to Domains 601 and 603.

Figure 10: Archer Hangingwall – Domain 601,602 and 603 Combined Au Variogram – Major axis – Dip Dirn 140 Dip 10

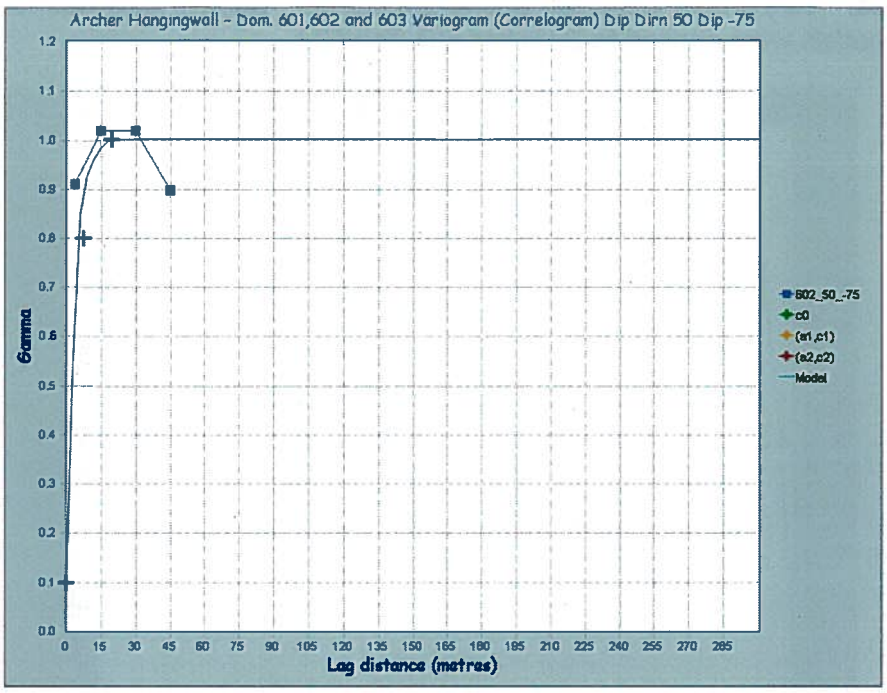


Figure 10: Archer Hangingwall – Domain 601,602 and 603 Combined Au Variogram – Semi Major axis – Dip Dirn 50 Dip -75

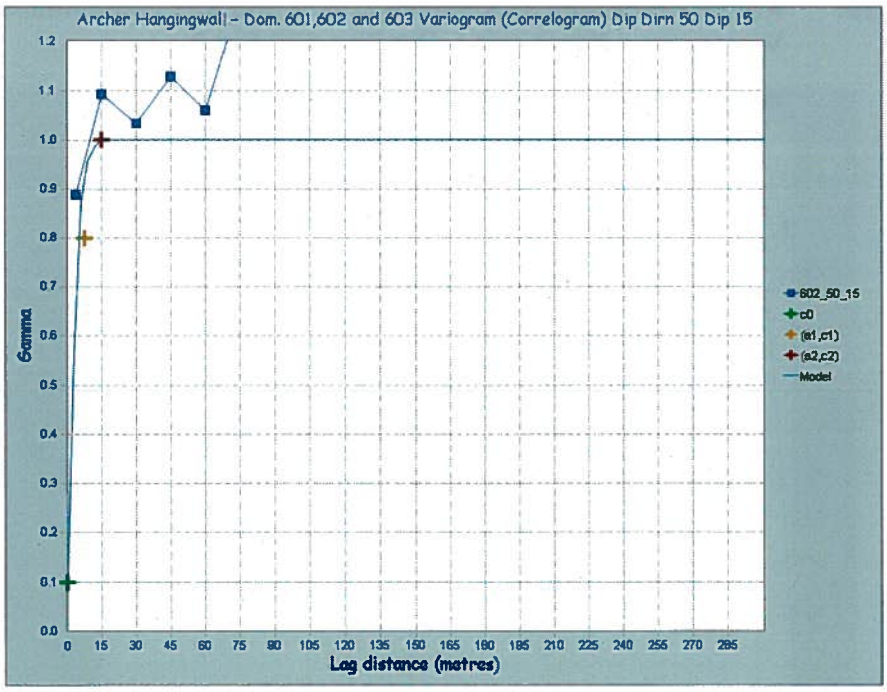


Figure 10: Archer Hangingwall – Domain 601,602 and 603 Combined Au Variogram – Minor axis – Dip Dirn 50 Dip 15

Archer Stockwork

These fault block domains (621,622 and 623) had the most composites compared to any other domains. Variograms for each individual fault block domain were modelled. Variograms were still poorly structured. Domain 622 was modeled and these models were applied to Domains 601 and 603.

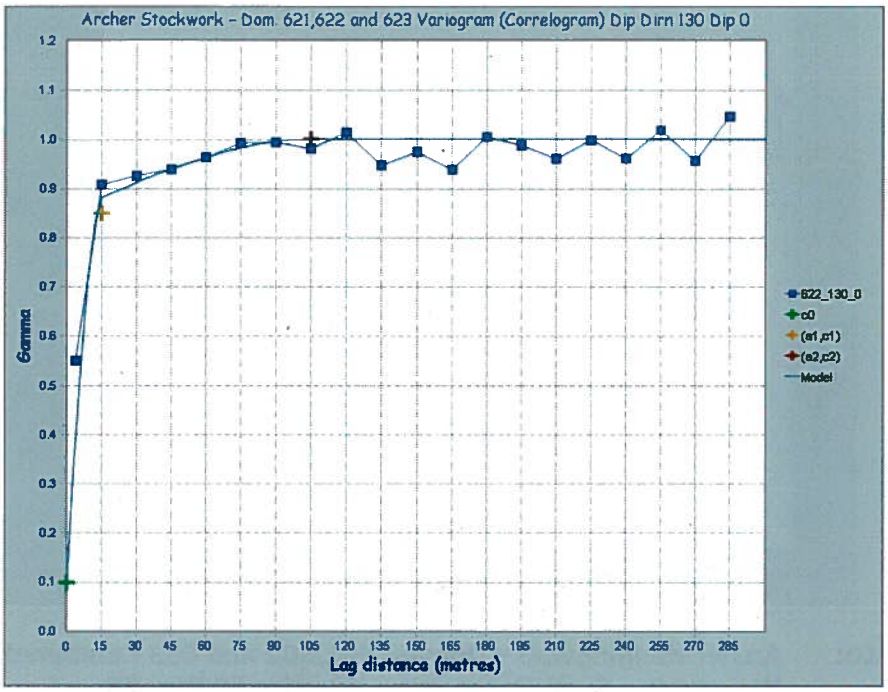


Figure 10: Archer Stockwork – Domain 621,622 and 623 Combined Au Variogram – Major axis – Dip Dirn 130 Dip 0

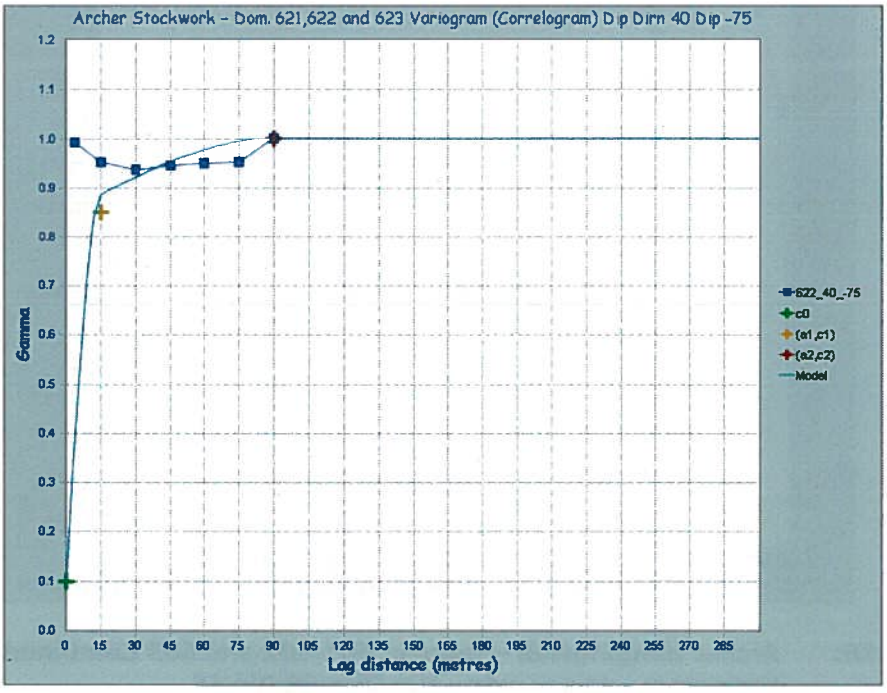


Figure 10: Archer Stockwork – Domain 621,622 and 623 Combined Au Variogram – Semi Major axis – Dip Dirn 40 Dip -75

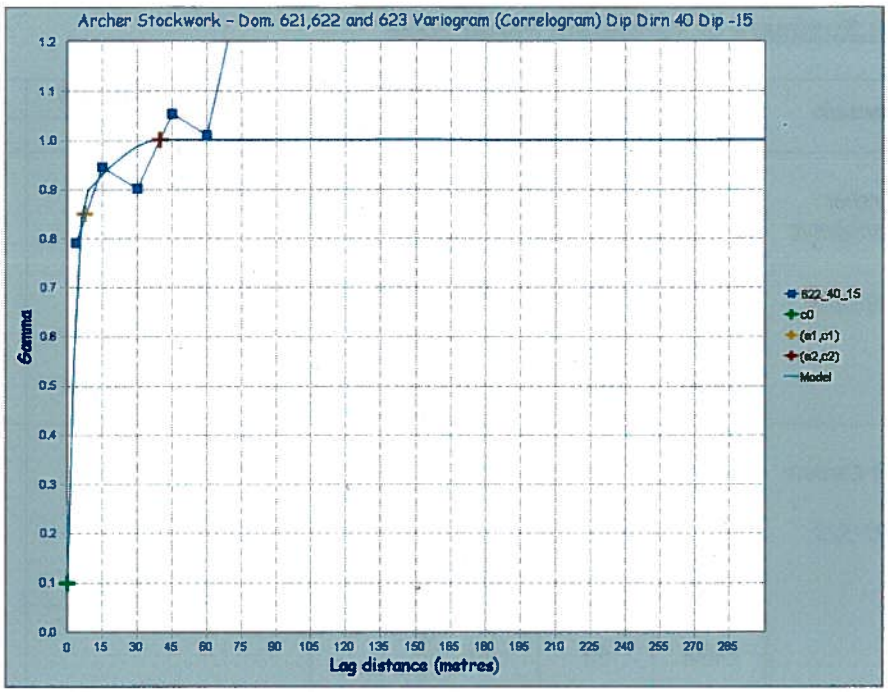


Figure 10: Archer Stockwork – Domain 621,622 and 623 Combined Au Variogram – Minor axis – Dip Dirn 40 Dip 15

Table 6: Summary of Variogram Parameters

Domain				Nugget	1st sill	2nd sill	Principal axis	
	Axis	Dip Dirn	Dip	C0	C1	C2	R1	R2
Archer Volcanogenic *501-503*	Major	125	0	0.1	0.6	0.3	15	95
	Semi Major	35	-75				15	75
	Minor	35	15				7.5	30
Basalt Carbine *521-523*	Major	130	0	0.25	0.35	0.4	15	45
	Semi Major	40	-75				7.5	30
	Minor	40	15				7.5	15
Archer Hangingwall *601-603*	Major	140	0	0.1	0.7	0.2	7.5	50
	Semi Major	50	-75				7.5	20
	Minor	50	15				7.5	15
Archer Stockwork *621-623*	Major	130	0	0.1	0.75	0.15	15	105
	Semi Major	40	-75				15	90
	Minor	40	15				7.5	40

8 Block Model

Block model was coded directly from 3D geological solids. Geology code (?GEO)and percentage (?%) of block inside solid were stored. Each geological domain "family" has its own set of block model items.

Table 7: Domain codes for Interim model (gg0215.dat)

Structure	*Fault Block*	*Model Item*	*Code*
Mine Schist		NGEO	1
Basalt Hangingwall		MGEO	4
Archer Volcanogenics	Merlot and Franc	GGEO	501
	Franc and Tannat	HGEO	502
	Tannat and Cabernet	IGEO	503
Basalt Carbine	Merlot and Franc	JGEO	521
	Franc and Tannat	KGEO	522
	Tannat and Cabernet	LGEO	523
Archer Hangingwall	Merlot and Franc	AGEO	601
	Franc and Tannat	BGEO	602
	Tannat and Cabernet	CGEO	603
Archer Stockwork	Merlot and Franc	DGEO	621
	Franc and Tannat	EGEO	622
	Tannat and Cabernet	FEGO	623

Table 7: Domain codes for Final model (gg0215.gg2)

Structure	*Fault Block*	*Model Item*	*Code*
Mine Schist		AGEO	1
Basalt Hangingwall		BGEO	4
Archer Volcanogenics	Merlot and Franc	CGEO	500
Basalt Carbine	Merlot and Franc	DGEO	520
Archer Hangingwall	Merlot and Franc	EGEO	600
Archer Stockwork	Merlot and Franc	FGEO	620

Block model is rotated into the plane of the structure, so that the rotated Y axis lies along strike, the rotated x axis lies up dip and the z direction is perpendicular to the orebody. Block dimensions are:

X direction	7.5m
Y direction	7.5m
Z direction	2.5m

The block dimensions have not been altered compared to the August 2003 model but the extents have been modified for this estimation.

The block model has been assigned the standard SGM default SG of 2.85.

Table 8: Block Model file – gg0215.dat Items and descriptions

Item	Min	Max	Prec.	Description
TOPO	0	100	0.01	Block % below TOPOG
A%	0	100	0.1	Archer Hangingwall % of block
AAUK	0	100	0.01	Kriged gold grade
ADIST	0	50	0.01	Distance to closest composite
AGEO	1	999	1	Geology code item
ASG	0	5	0.01	Bulk density
ASLOP	0	2	0.01	Slope of regression
B%	0	100	0.1	Archer Hangingwall % of block
BAUK	0	100	0.01	Kriged gold grade
BDIST	0	50	0.01	Distance to closest composite
BGEO	1	999	1	Geology code item
BSG	0	5	0.01	Bulk density
BSLOP	0	2	0.01	Slope of regression
C%	0	100	0.1	Archer Hangingwall % of block
CAUK	0	100	0.01	Kriged gold grade
CDIST	0	50	0.01	Distance to closest composite
CGEO	1	999	1	Geology code item
CSG	0	5	0.01	Bulk density
CSLOP	0	2	0.01	Slope of regression
D%	0	100	0.1	Archer Stockwork % of block
DAUK	0	100	0.01	Kriged gold grade
DDIST	0	50	0.01	Distance to closest composite
DGEO	1	999	1	Geology code item
DSG	0	5	0.01	Bulk density
DSLOP	0	2	0.01	Slope of regression
E%	0	100	0.1	Archer Stockwork % of block
EAUK	0	100	0.01	Kriged gold grade
EDIST	0	50	0.01	Distance to closest composite
EGEO	1	999	1	Geology code item
ESG	0	5	0.01	Bulk density
ESLOP	0	2	0.01	Slope of regression
F%	0	100	0.1	Archer Stockwork % of block
FAUK	0	100	0.01	Kriged gold grade
FDIST	0	50	0.01	Distance to closest composite
FGEO	1	999	1	Geology code item
FSG	0	5	0.01	Bulk density
FSLOP	0	2	0.01	Slope of regression
G%	0	100	0.1	Archer Volcanogenics % of block
GAUK	0	100	0.01	Kriged gold grade
GDIST	0	50	0.01	Distance to closest composite
GGEO	1	999	1	Geology code item
GSG	0	5	0.01	Bulk density
GSLOP	0	2	0.01	Slope of regression
H%	0	100	0.1	Archer Hangingwall % of block
HAUK	0	100	0.01	Kriged gold grade
HDIST	0	50	0.01	Distance to closest composite
HGEO	1	999	1	Geology code item
HSG	0	5	0.01	Bulk density

Table 8: Block Model file – gg0215.dat Items and descriptions (cont.)

Item	Min	Max	Prec.	Description
HSLOP	0	2	0.01	Slope of regression
I%	0	100	0.1	Archer Hangingwall % of block
IAUK	0	100	0.01	Kriged gold grade
IDIST	0	50	0.01	Distance to closest composite
IGEO	1	999	1	Geology code item
ISG	0	5	0.01	Bulk density
ISLOP	0	2	0.01	Slope of regression
J%	0	100	0.1	Basalt Carbine % of block
JAUK	0	100	0.01	Kriged gold grade
JDIST	0	50	0.01	Distance to closest composite
JGEO	1	999	1	Geology code item
JSG	0	5	0.01	Bulk density
JSLOP	0	2	0.01	Slope of regression
K%	0	100	0.1	Basalt Carbine % of block
KAUK	0	100	0.01	Kriged gold grade
KDIST	0	50	0.01	Distance to closest composite
KGEO	1	999	1	Geology code item
KSG	0	5	0.01	Bulk density
KSLOP	0	2	0.01	Slope of regression
L%	0	100	0.1	Basalt Carbine % of block
LAUK	0	100	0.01	Kriged gold grade
LDIST	0	50	0.01	Distance to closest composite
LGEO	1	999	1	Geology code item
LSG	0	5	0.01	Bulk density
LSLOP	0	2	0.01	Slope of regression
M%	0	100	0.1	Basalt Hangingwall % of block
MAUK	0	100	0.01	Kriged gold grade
MDIST	0	50	0.01	Distance to closest composite
MGEO	1	999	1	Geology code item
MSG	0	5	0.01	Bulk density
MSLOP	0	2	0.01	Slope of regression
N%	0	100	0.1	Mine Schist % of block
NAUK	0	100	0.01	Kriged gold grade
NDIST	0	50	0.01	Distance to closest composite
NGEO	1	999	1	Geology code item
NSG	0	5	0.01	Bulk density
NSLOP	0	2	0.01	Slope of regression

Coding was visually checked within MineSight.

Table 8: Block Model file – gg0215.gg2 Items and descriptions

Item	Min	Max	Prec.	Description
TOPO	0	100	0.01	Block % below TOPOG
A%	0	100	0.1	Mine Schist % of block
AAUK	0	100	0.01	Kriged gold grade
ADIST	0	50	0.01	Distance to closest composite
AGEO	1	999	1	Geology code item
ASG	0	5	0.01	Bulk density
ASLOP	0	2	0.01	Slope of regression
ACAT	0	5	1	Resource Category
B%	0	100	0.1	Basalt Hangingwall % of block
BAUK	0	100	0.01	Kriged gold grade
BDIST	0	50	0.01	Distance to closest composite
BGEO	1	999	1	Geology code item
BSG	0	5	0.01	Bulk density
BSLOP	0	2	0.01	Slope of regression
BCAT	0	5	1	Resource Category
C%	0	100	0.1	Archer Volcanogenic % of block
CAUK	0	100	0.01	Kriged gold grade
CDIST	0	50	0.01	Distance to closest composite
CGEO	1	999	1	Geology code item
CSG	0	5	0.01	Bulk density
CSLOP	0	2	0.01	Slope of regression
CCAT	0	5	1	Resource Category
D%	0	100	0.1	Basalt Carbine % of block
DAUK	0	100	0.01	Kriged gold grade
DDIST	0	50	0.01	Distance to closest composite
DGEO	1	999	1	Geology code item
DSG	0	5	0.01	Bulk density
DSLOP	0	2	0.01	Slope of regression
DCAT	0	5	1	Resource Category
E%	0	100	0.1	Archer Hangingwall % of block
EAUK	0	100	0.01	Kriged gold grade
EDIST	0	50	0.01	Distance to closest composite
EGEO	1	999	1	Geology code item
ESG	0	5	0.01	Bulk density
ESLOP	0	2	0.01	Slope of regression
ECAT	0	5	1	Resource Category

Table 8: Block Model file – gg0215.gg2 Items and descriptions (cont.)

F%	0	100	0.1	Archer Stockwork % of block
FAUK	0	100	0.01	Kriged gold grade
FDIST	0	50	0.01	Distance to closest composite
FGEO	1	999	1	Geology code item
FSG	0	5	0.01	Bulk density
FSLOP	0	2	0.01	Slope of regression
FCAT	0	5	1	Resource Category

9 Resource Estimation

Resource estimation method used is Ordinary Kriging. For explanation of the appropriateness and implementation of this estimation method the reader is referred to the following QG reports:

Vann, J., 2001. Review of Geological Interpretation, Domain Decision-Making and Kriging Plan at the Stawell Gold Mine, Victoria, Australia. *Confidential consultant report by Quantitative Geoscience Pty Ltd. for Stawell Gold Mine.*

Vann, J., 2001. Reviews of June 2001 Resource Estimate (North Mine, Central Lode) & Laboratory QA-QC at the Stawell Gold Mine, Victoria, Australia. *Confidential consultant report by Quantitative Geoscience Pty Ltd. for Stawell Gold Mine. August, 2001.* QG Project SGM 00403.

Bertolli, O., 2001. Proposed Improvements for In-situ Resource Estimates and Recoverable Resource Estimation at the Stawell Gold Mine, Victoria, Australia. *Confidential consultant report by Quantitative Geoscience Pty Ltd. for Stawell Gold Mine. September 2001*

9.1.1.1 Quantified Kriging Neighborhood Analysis and Block Estimation

Selection of appropriate kriging plans builds on review of similar resource models and work carried out on Kriging Neighborhood Analysis.

The MineSight kriging m624 run-files used in interpolation are attached in Appendix D.

9.1.1.2 Archer FW Lode

Data Selection

Data selection is controlled using an isotropic ellipsoidal search of 100x100x30. A minimum of 10 composites, a maximum of 24 composites, a maximum of 10 composites per drillhole, and maximum distance to closest composite of 40m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 6. (file run624.510)

9.1.1.3 Archer HW Lode

Data Selection

Data selection is controlled using an isotropic ellipsoidal search of 100x100x30. A minimum of 10 composites, a maximum of 40 composites, a maximum of 20 composites per drillhole, and maximum distance to closest

composite of 40m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 10. (file run624.511)

9.1.1.4 Carbine Lode

Data Selection

Data selection is controlled using an isotropic ellipsoidal search of 100x100x30. A minimum of 10 composites, a maximum of 32 composites, a maximum of 6 composites per drillhole, and maximum distance to closest composite of 40m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 8. (file run624.512)

9.1.1.5 Basalt contact

Data Selection

Data selection is controlled using an isotropic ellipsoidal search of 100x100x30. A minimum of 10 composites, a maximum of 40 composites, a maximum of 20 composites per drillhole, and maximum distance to closest composite of 40m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 10. (file run624.513)

9.1.1.6 Volcanogenics

Data Selection

Data selection is controlled using an isotropic ellipsoidal search of 100x100x30. A minimum of 10 composites, a maximum of 40 composites, a maximum of 10 composites per drillhole, and maximum distance to closest composite of 40m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 8. (file run624.501)

10 Validation

10.1 Test Kriged Blocks

Approximately 5 to 7 blocks were test kriged for each of the domains for each set of search parameters trialled. Test kriging generates the search ellipse, and reports the number of composites used, kriging weights, minimum and maximum distances to samples and the slope of regression. The search ellipse and weighting strings were then visualized in Minesight to ensure all suitable composites were used.

Slope of regression values appear closely related to the distance of the closest sample. Blocks at the periphery of the data set generally showed much lower slope of regression, and longer search distances to data. Comparisons between areas with broad spaced Resource Definition drilling and those with infill Grade Control drilling indicated that a single pass grade estimation strategy would suffice.

Several sets of estimation parameters were trialled for Archer HW, as there were significant negative kriging weights in well informed areas. These negative weights were reduced to an acceptable level with the parameters selected.

11 Reporting

Reporting from the model should be carried out in S:\Mining-Geology\Golden Gift\MineSight\Geology\Current Model using gg0215.gg2.

0.0g/t cutoff			
	Tonnes	**Grade**	**Ounces**
Archer Volcanogenic			
Indicated	494,054	0.54	8,538.7
Inferred	5,612	0.50	89.4
Total	**499,666**	**0.54**	**8,628.1**
Basalt Carbine			
Indicated	219,768	0.79	5,601.6
Inferred	20,159	0.90	584.5
Total	**239,927**	**0.80**	**6,186.1**
Archer Hangingwall			
Indicated	395,879	2.48	31,597.2
Inferred	326,874	2.29	23,420.0
Total	**722,753**	**2.37**	**55,017.2**
Archer Stockwork			
Indicated	1,034,850	1.62	53,971.1
Inferred	26,286	1.06	895.9
Total	**1,061,136**	**1.61**	**54,867.0**
Indicated	2,144,551	1.44	99,708.6
Inferred	378,931	2.10	24,989.8
Total	**2,523,482**	**1.54**	**124,698.4**

4.0g/t cutoff			
	Tonnes	**Grade**	**Ounces**
Archer Volcanogenic			
Indicated	3,915	6.57	827.1
Inferred	-	-	-
Total	3,915	6.57	827.1
Basalt Carbine			
Indicated	2,003	5.39	347.1
Inferred	-	-	-
Total	2,003	5.39	347.1
Archer Hangingwall			
Indicated	54,111	6.00	10,437.0
Inferred	34,667	5.53	6,158.3
Total	**88,778**	**5.81**	**16,595.3**
Archer Stockwork			
Indicated	64,741	7.93	16,509.0
Inferred	132	4.42	18.8
Total	**64,873**	**7.92**	**16,527.9**
Indicated	124,770	7.01	28,120.2
Inferred	34,799	5.53	6,177.1
Total	**159,569**	**6.69**	**34,297.3**

Table 9: Resource Summary for GG2

Resource classification is based on a visual assessment of drillhole spacing, interpolation slope of regressions and number of samples.

Indicated blocks interpolated with drillhole with approximated spacing within 25-30m and have a slope of regression of greater than 0.3.

Measured = 1
Indicated = 2
Inferred = 3

Table 7: Classification Items for Final model (gg0215.gg2)

Structure	*Model Item*	*Code*
Mine Schist	ACAT	1
Basalt Hangingwall	BCAT	4
Archer Volcanogenics	CCAT	500
Basalt Carbine	DCAT	520
Archer Hangingwall	ECAT	600
Archer Stockwork	FCAT	620

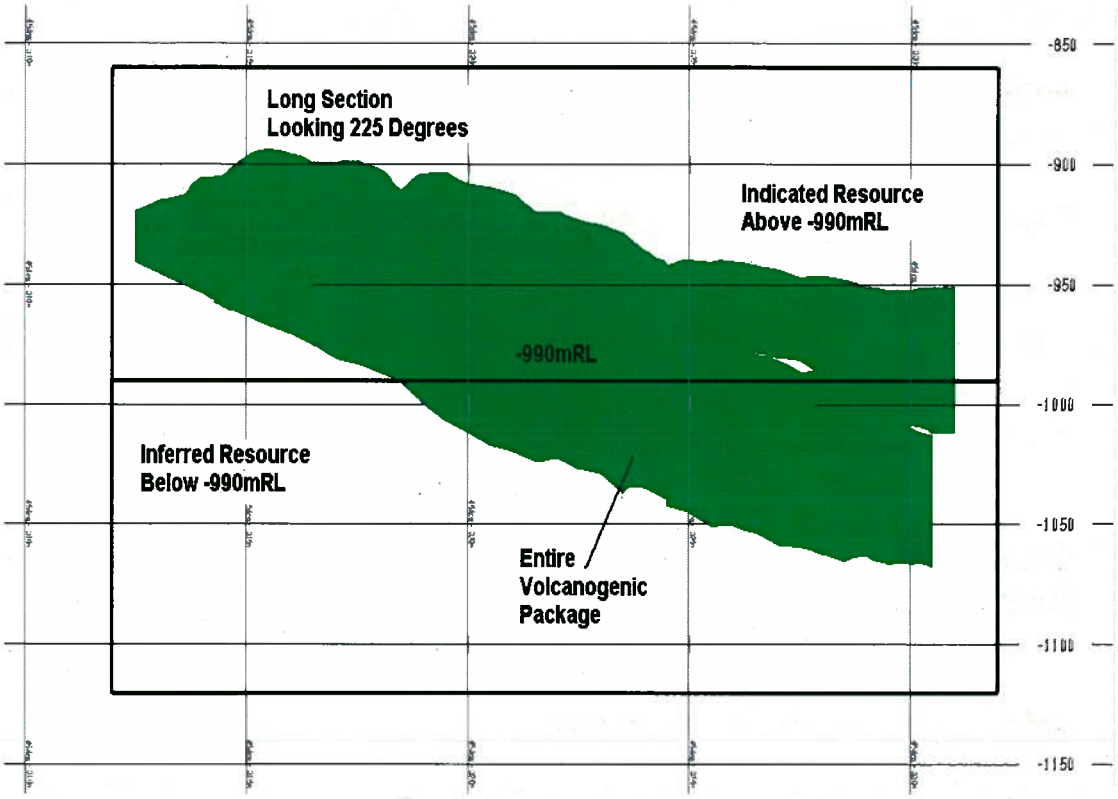


Figure 18: Long Section view looking 225 Degrees showing areas above and below the -990mRL level.

12 Model Comparisons

This August 2004 model replaces the corresponding Indicated Resource from the previous December 2003 model. A comparison of the Indicated Resources from these models is presented in Table 10, with the respective areas shown in Figure 19. The significant increase in grade is due in part to the discovery of a stockwork lode on the south end which was previously unknown.

	Tonnes	Grade	Oz's
0.0g/t cutoff			

Dec-03	954,007	2.35	72,079.9
Aug-04	874,068	2.77	77,842.4
Difference	-79,939		5762.5
4.0g/t cutoff			
Dec-03	157,068	6.87	34,692.5
Aug-04	207,818	7.58	50,645.8
Difference	50,750	9.78	15,953.5

Table 10: Comparison of Indicated Resources from Dec 03' and Aug 04'.

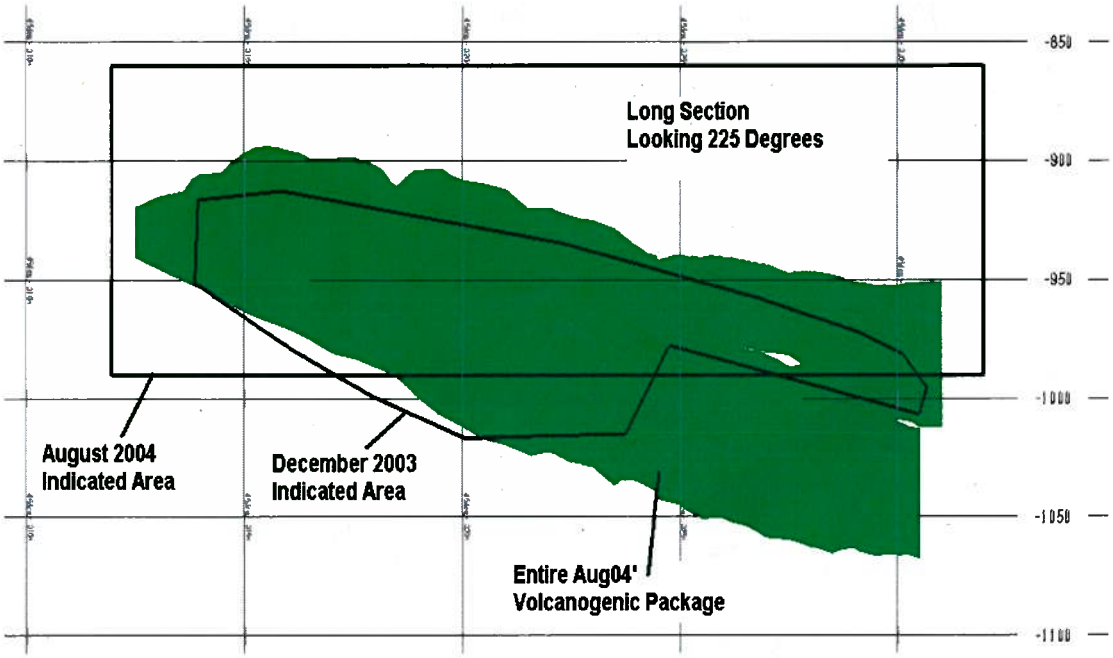


Figure 19: Increase in 'area' of 'confidence' between Dec03 and Aug04 models.

As a further comparison the December 2003 'Indicated resource area' (shown in Figure 19) was run through the August 2004 block model to ascertain the impact of the new interpretation. Table 11 shows the comparison between the models. The first obvious change is in total tonnes at zero cutoff with a 143,112t difference between the two models. The majority of this change is attributed to the better understanding of the Merlot fault (especially after its exposure in the 930-322-81Sth) and the new position of the Cabernet fault after intersection with several drill holes. This is illustrated in Figure 20 which shows the two interpretations.

	Tonnes	Grade	Oz's
0.0g/t cutoff			
Dec-03	954,007	2.35	72,079.9
Aug-04	810,895	2.84	74,041.3
Difference	-143,112		1961.4
4.0g/t cutoff			
Dec-03	157,068	6.87	34,692.5
Aug-04	195,830	7.46	46,968.8
Difference	38,762	9.85	12,276.3

Table 11: Comparison Indicated Resources within the area modeled in the Dec03 model.

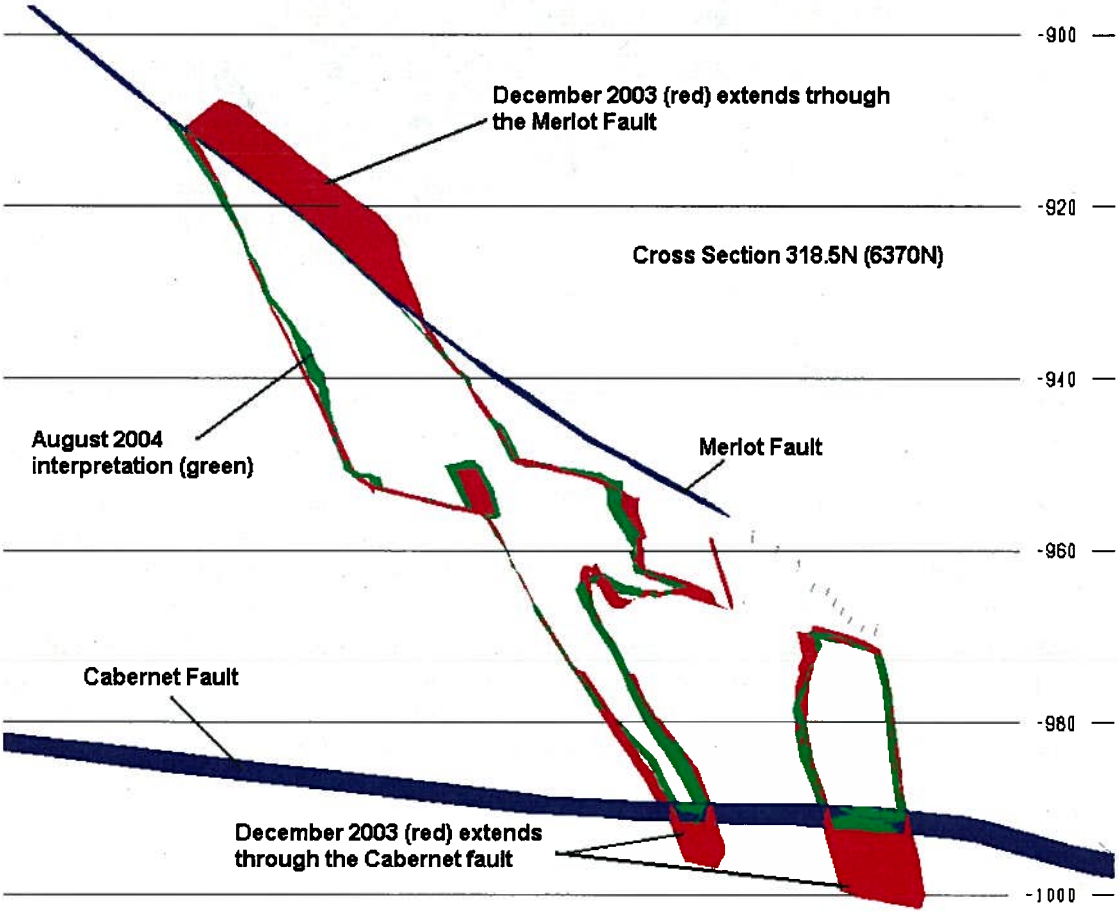


Figure 20: Change in tonnes between December 2003 and August 2004 interpretations.

Note that the 15,953.5 ounce increase in the August 2004 model, (as reported in Table 10) 77% of 'new' ounces lie within the December 2003 'Indicated resource area' (shown in Figure 19).

13 Comparison of new 'Kriged' model with October 2002 Median Case Conditional Simulation Resource

To compare the results of the August 2004 model with the conditional simulation work completed in October 2002, the block modeling statistics calculation <p60801.dat> was run in the 'Simulation' project and filtered by 'Bench' (level). It was found (and validated on screen) that by limiting the bench to between '0 and 9' (-900mRL and -990mRL) the model could represent the 'Aug04 Indicated Resource area' as shown in Figure 18. Likewise, by limiting the bench to between '10 and 18' (-990mRL and -1080mRL) the model could represent the 'Aug04 Inferred Resource area as shown in Figure 18. Note that the October 2002 Median Case geology model extends >100m north of the August 2004 current model (see Figure 21).

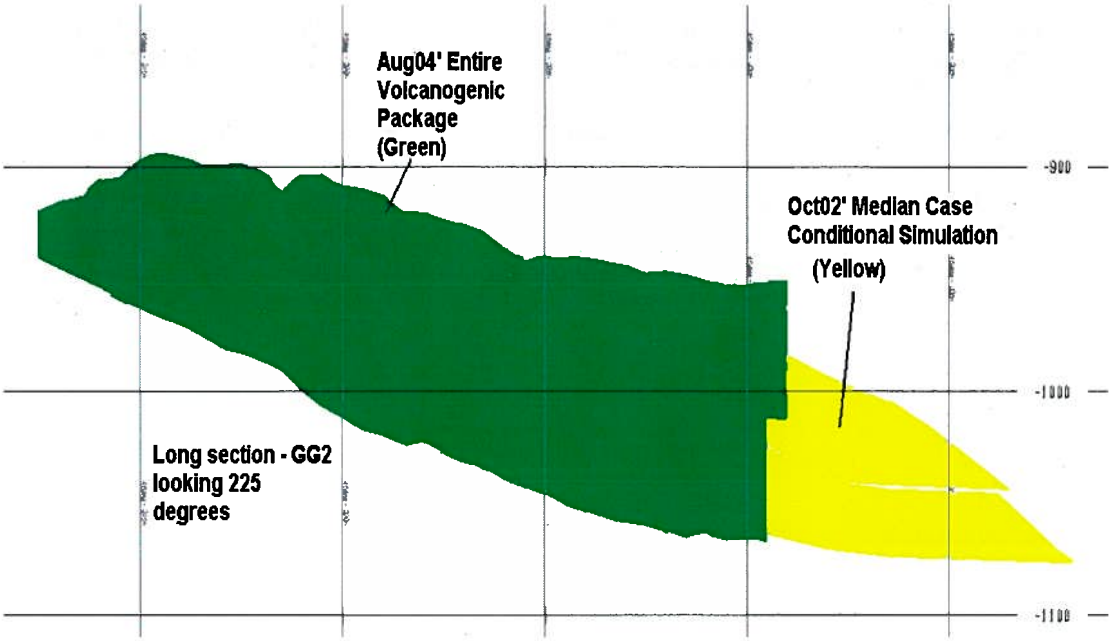


Figure 21: Long section showing the two geology models. Note that the Oct02' model extends beyond the northern end of the new Aug04' model

Comparison with Conditional Simulation Model

Median Case Con Sim for benches 0 to 9 (-900mRL and -990mRL) on 10m x 10m x5m at 4.0g/t cutoff.
101,175 tonnes at 6.57g/t for 21,371.2 ounces
This is 31% of the total 'Median Case' ounces

As a comparison, August 2004 Resource model for Indicated Resource area (see Fig 18.) based on 7.5m x 7.5m x 2.5m model at 4.0g/t cutoff
207,818 tonnes at 7.58g/t for 50,645.8 ounces
This represents over 73% of the total Median Case scenario ounces for GG2.
Grade of 7.58g/t is remarkably similar to that of Median Case total of 7.37g/t.

According to the Median Case Con Sim model of October 2002 the remaining area of GG2 (inferred) not tested by recent infill drilling can be represented by benches 10 to 18 (-990mRL and -1080mRL) see Figure 18.

Medium Con Sim for the benches 10 to 18 (-990mRL and -1080mRL) on 10m x 10m x5m at 4.0g/t cutoff.
Suggests 188,100 tonnes at 7.80g/t for 47,170 ounces

The conditional simulation is based on the broad spaced drilling available at the time and represents a plausible global estimate of ounces for the GG2 block.

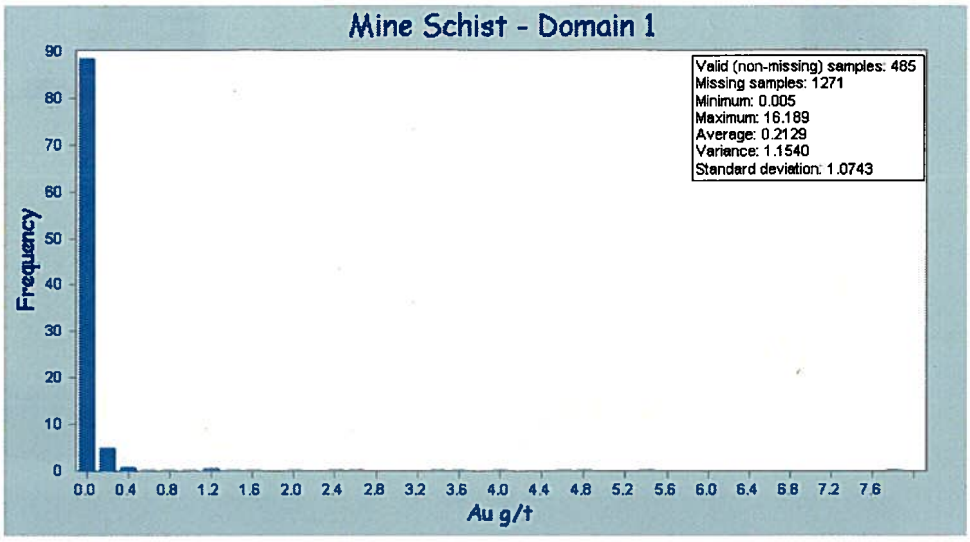
With >73% of ounces now accounted for by local estimation above the 990 level it would appear that further infill diamond drilling below this point should discover more ounces than the Median Case Conditional Simulation total for GG2 of 68,541 ounces.

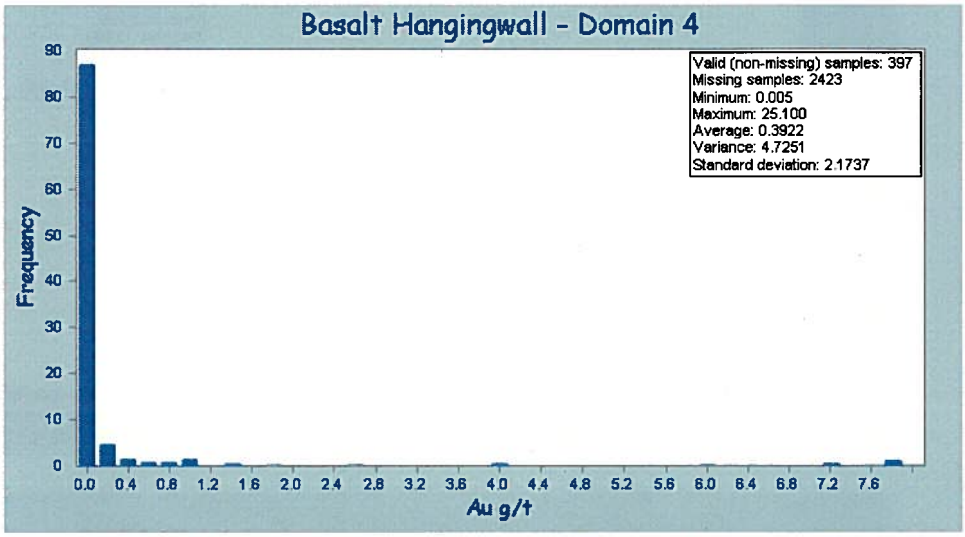
14 Grade Shells

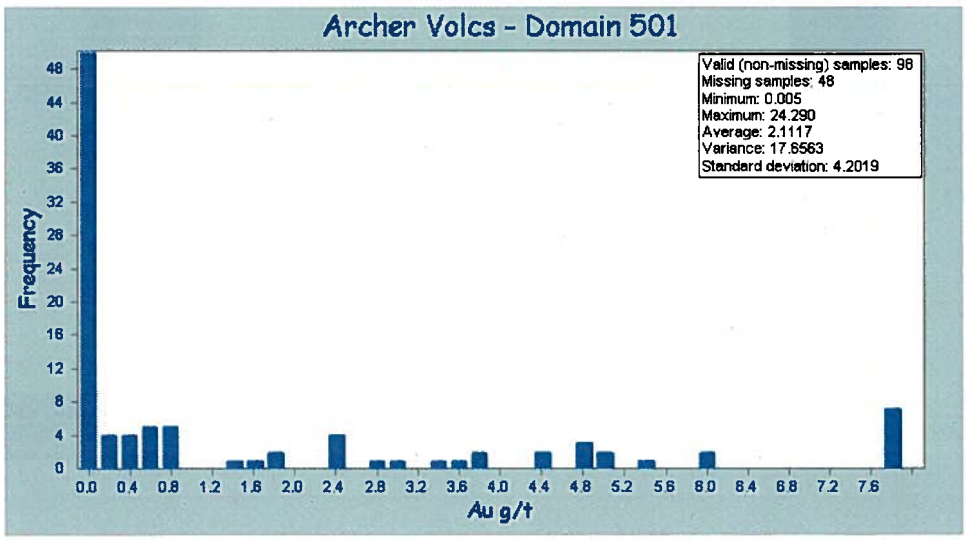
Grade shells, clipped against the geology solids, were generated for each of the surfaces. The following cutoff grades were used for grade shell generation:

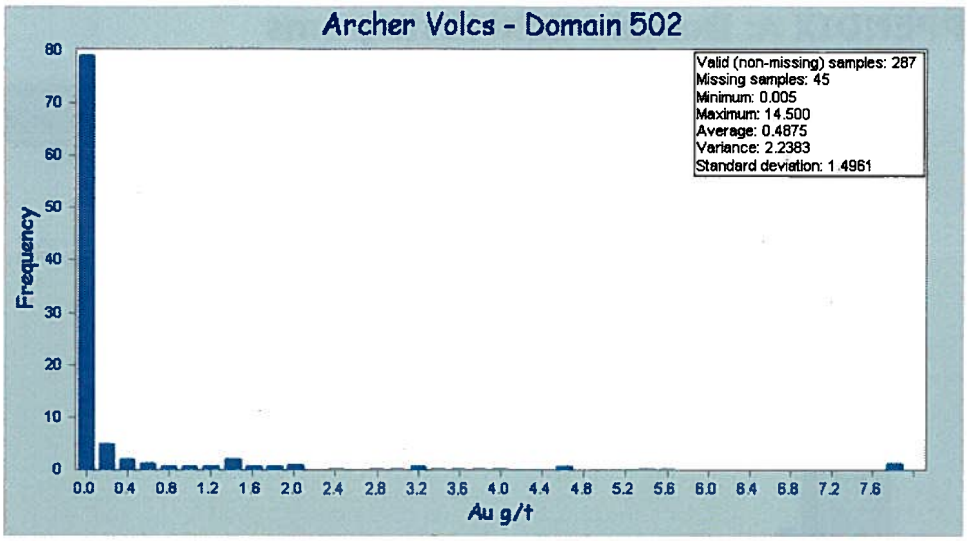
Archer FW	Archer HW	Carbine	Basalt Contact	Volcanogenics Remainder
>4.0 g/t au	>4.0 g/t au	>4.0 g/t au	>4.0 g/t au	>4.0 g/t au
>7.0 g/t au	>7.0 g/t au	>7.0 g/t au	>7.0 g/t au	>7.0 g/t au
> 9.0 g/t au	> 9.0 g/t au	> 9.0 g/t au	> 9.0 g/t au	> 9.0 g/t au

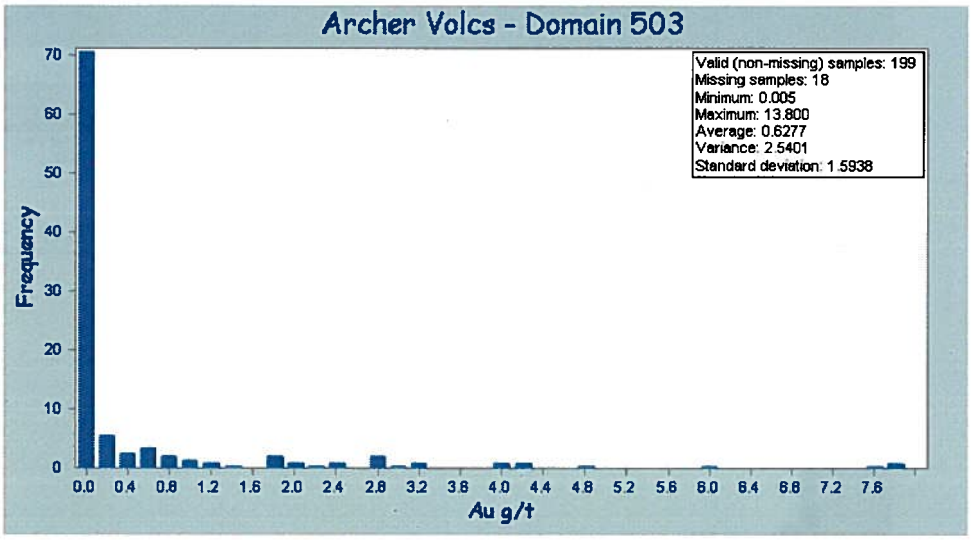
APPENDIX A: Domain Gold Histograms

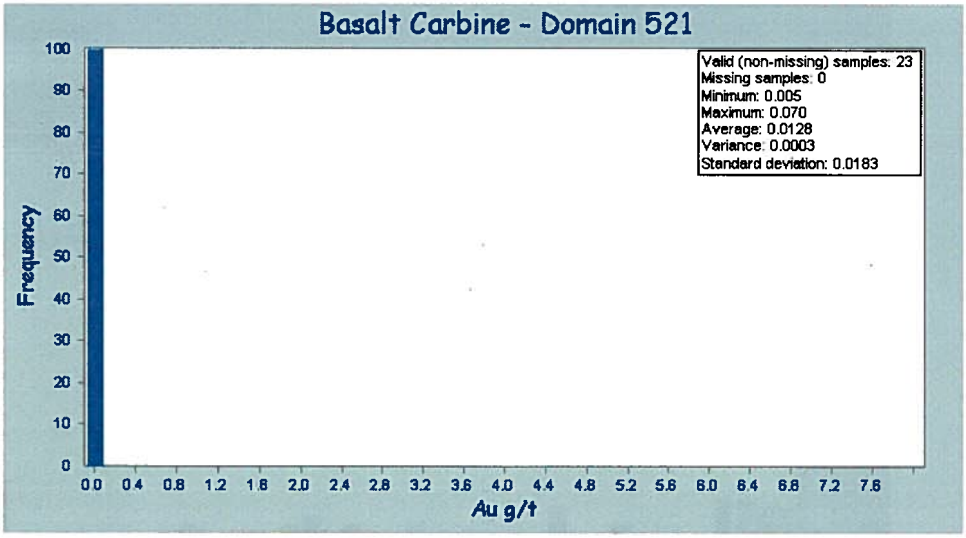


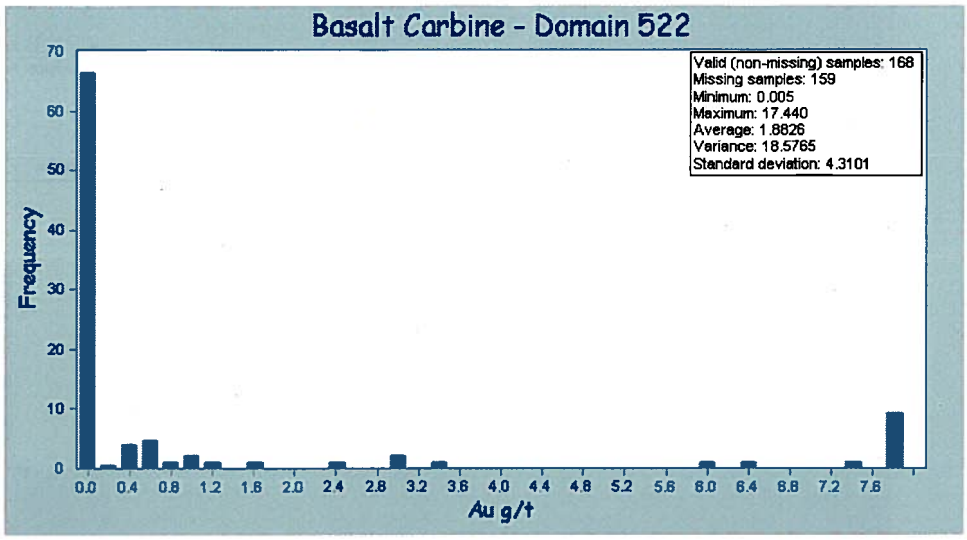


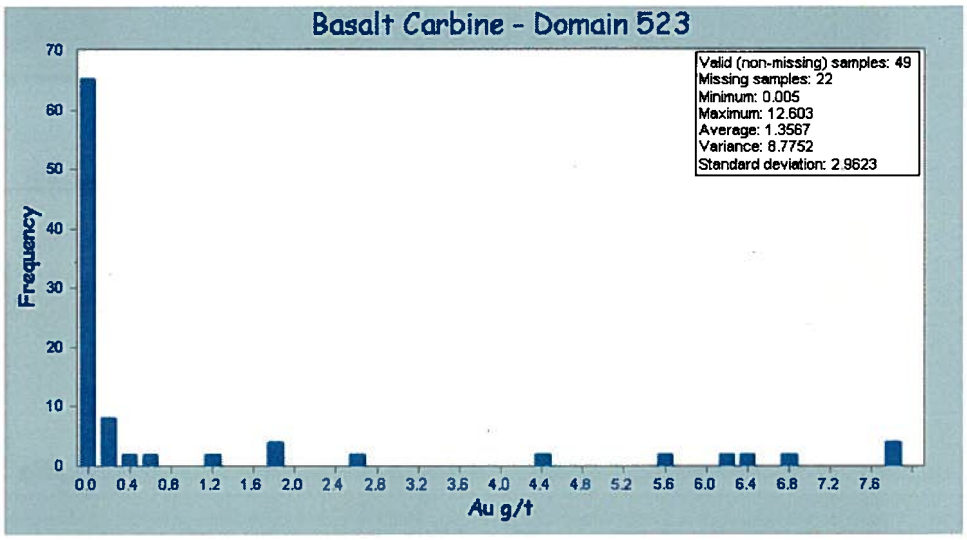


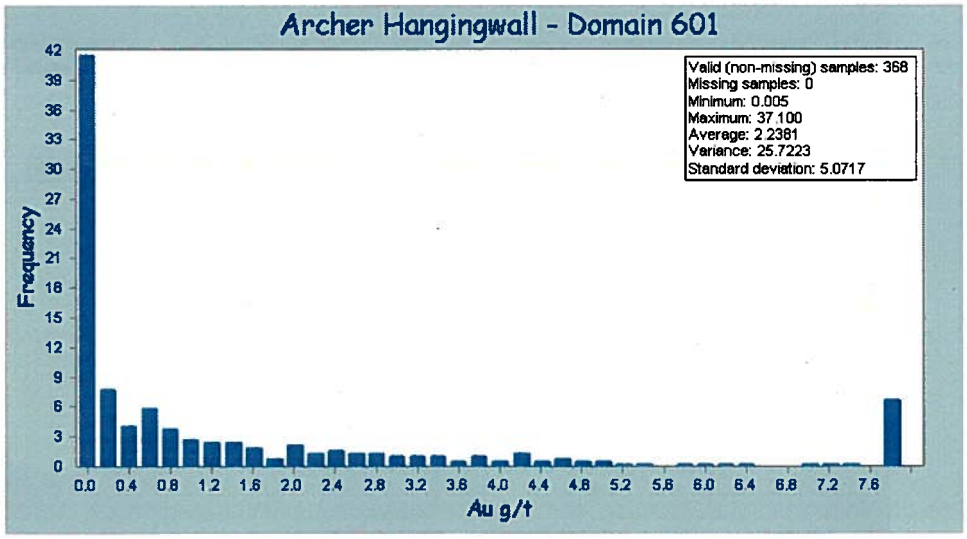

Archer Volcs - Domain 502
Valid (non-missing) samples: 287
Missing samples: 45
Minimum: 0.005
Maximum: 14.500
Average: 0.4875
Variance: 2.2383
Standard deviation: 1.4961
Frequency
Au g/t

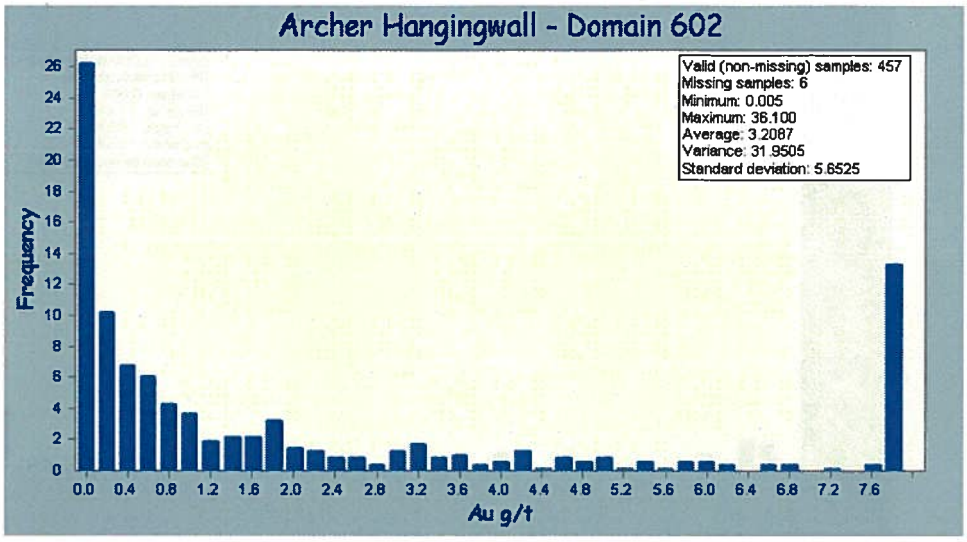

Archer Volcs - Domain 503
Valid (non-missing) samples: 199
Missing samples: 18
Minimum: 0.005
Maximum: 13.800
Average: 0.6277
Variance: 2.5401
Standard deviation: 1.5938
Frequency
Au g/t

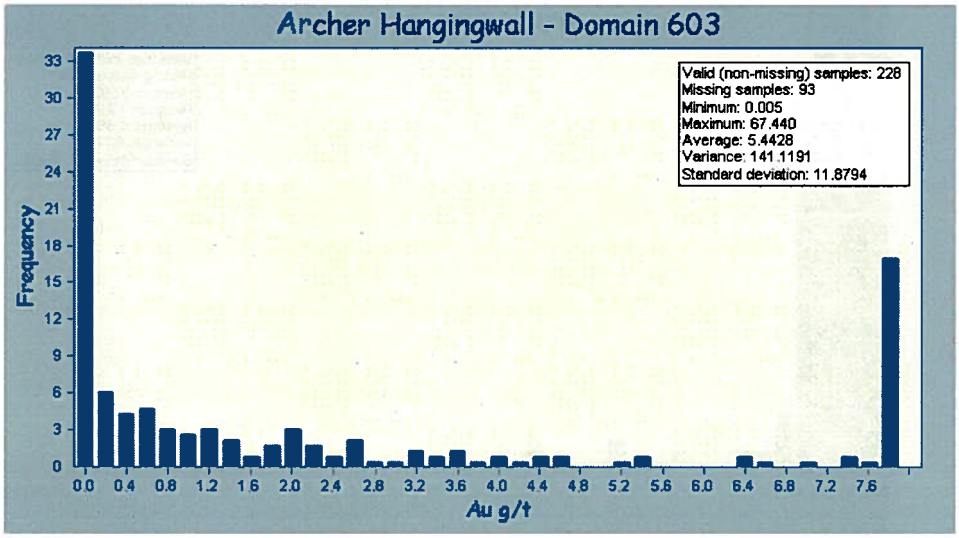

Basalt Carbine - Domain 521
Valid (non-missing) samples: 23
Missing samples: 0
Minimum: 0.005
Maximum: 0.070
Average: 0.0128
Variance: 0.0003
Standard deviation: 0.0183
Frequency
Au g/t

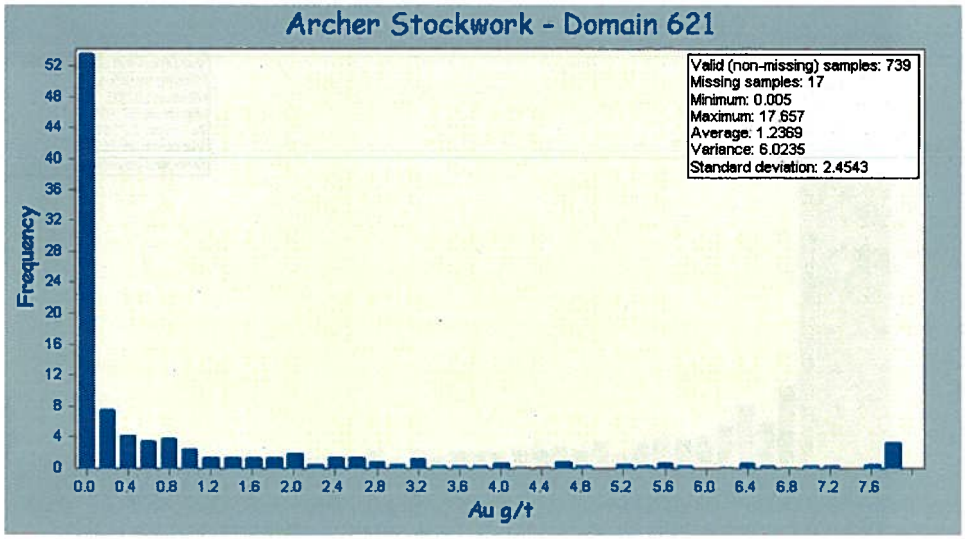

Basalt Carbine - Domain 522
Valid (non-missing) samples: 168
Missing samples: 159
Minimum: 0.005
Maximum: 17.440
Average: 1.8826
Variance: 18.5765
Standard deviation: 4.3101
Frequency
Au g/t

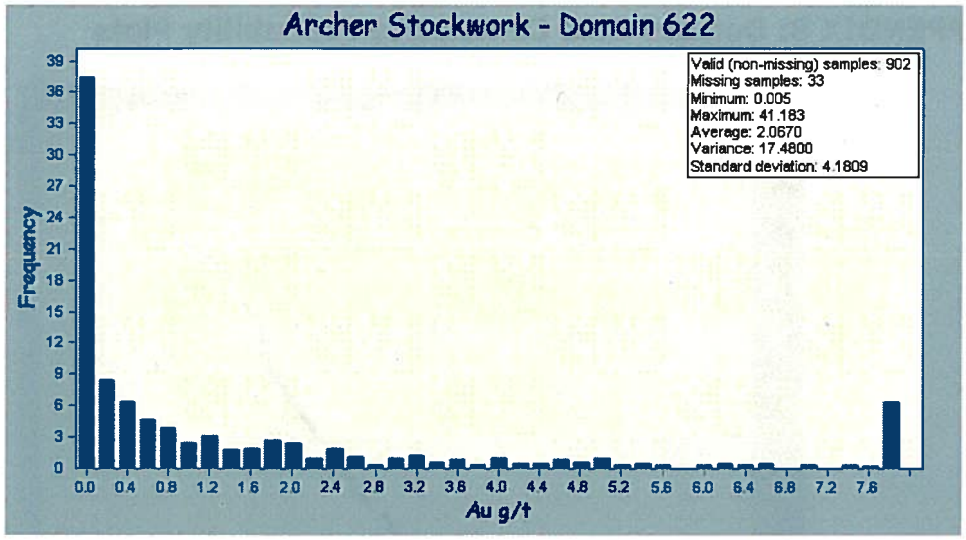

Basalt Carbine - Domain 523
Valid (non-missing) samples: 49
Missing samples: 22
Minimum: 0.005
Maximum: 12.603
Average: 1.3567
Variance: 8.7752
Standard deviation: 2.9623
Frequency
Au g/t

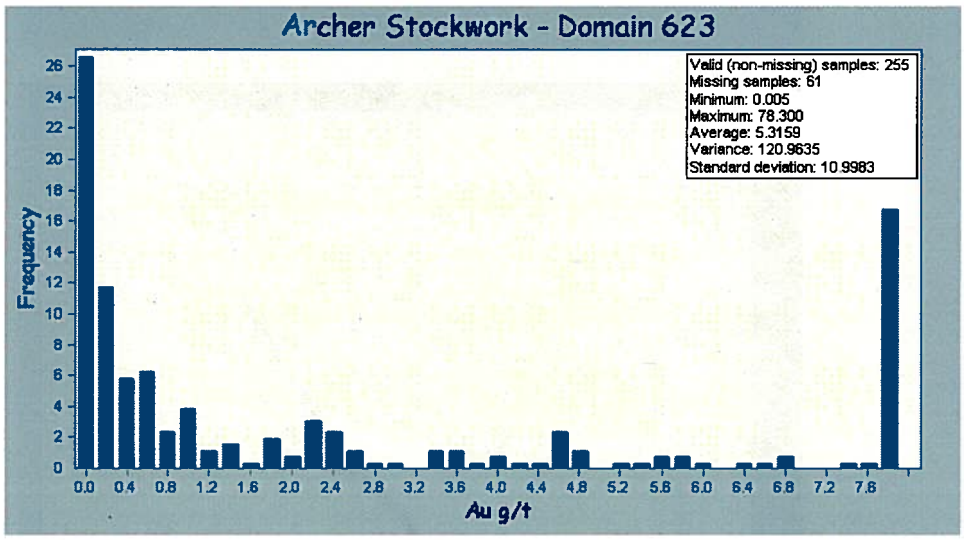

Archer Hangingwall - Domain 601
Valid (non-missing) samples: 368
Missing samples: 0
Minimum: 0.005
Maximum: 37.100
Average: 2.2381
Variance: 25.7223
Standard deviation: 5.0717
Frequency
Au g/t


Archer Hangingwall - Domain 602
Valid (non-missing) samples: 457
Missing samples: 6
Minimum: 0.005
Maximum: 36.100
Average: 3.2087
Variance: 31.9505
Standard deviation: 5.6525
Frequency
Au g/t

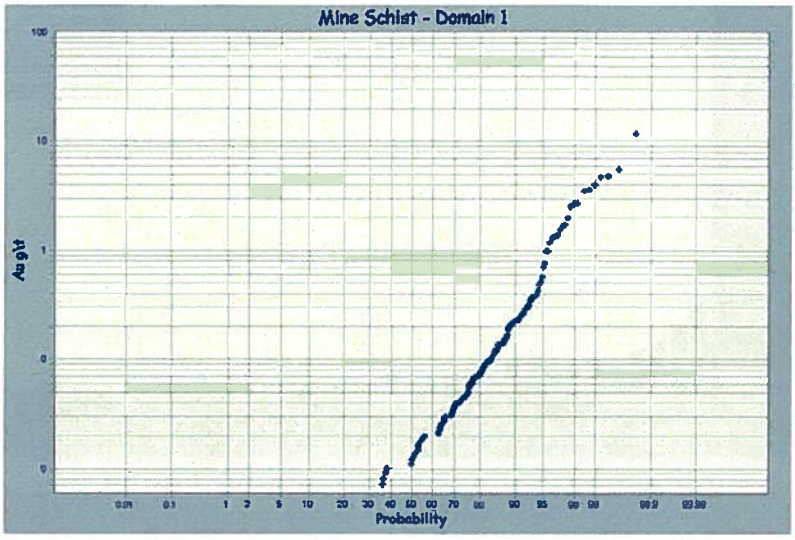

Archer Hangingwall - Domain 603
Valid (non-missing) samples: 228
Missing samples: 93
Minimum: 0.005
Maximum: 67.440
Average: 5.4428
Variance: 141.1191
Standard deviation: 11.8794
Frequency
Au g/t

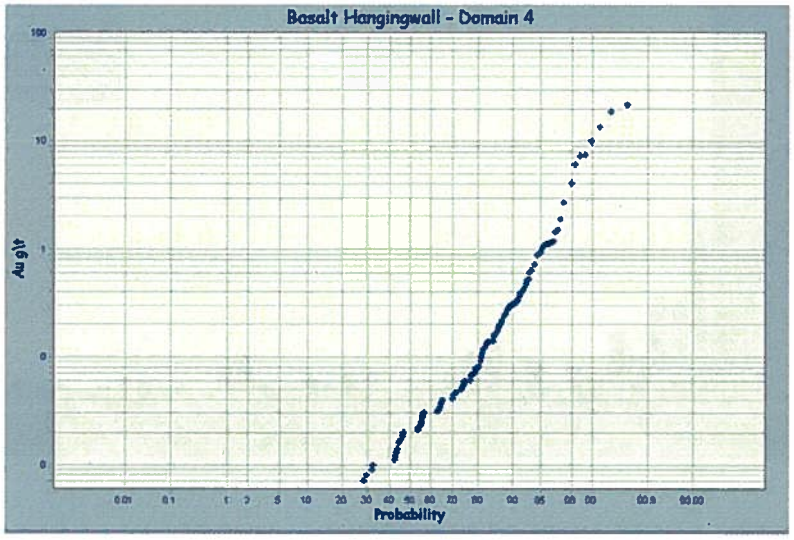

Archer Stockwork - Domain 621
Valid (non-missing) samples: 739
Missing samples: 17
Minimum: 0.005
Maximum: 17.657
Average: 1.2369
Variance: 6.0235
Standard deviation: 2.4543
Frequency
Au g/t

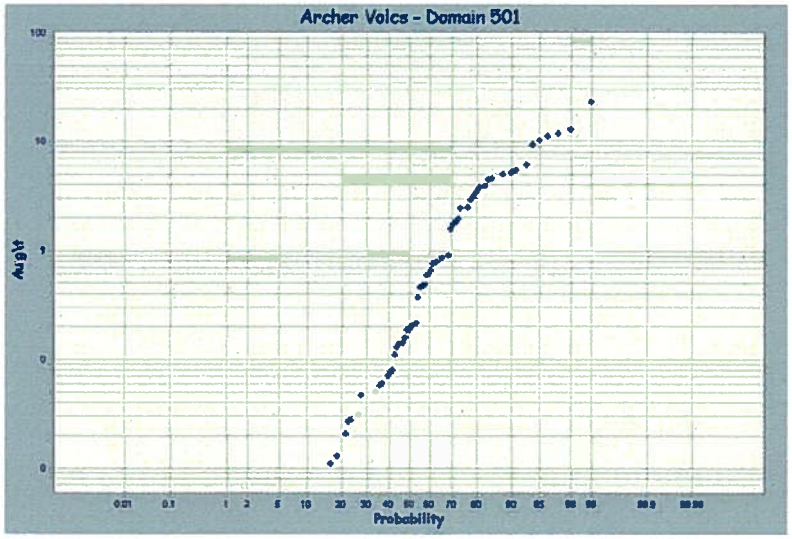

Archer Stockwork - Domain 622
Valid (non-missing) samples: 902
Missing samples: 33
Minimum: 0.005
Maximum: 41.183
Average: 2.0670
Variance: 17.4800
Standard deviation: 4.1809
Frequency
Au g/t

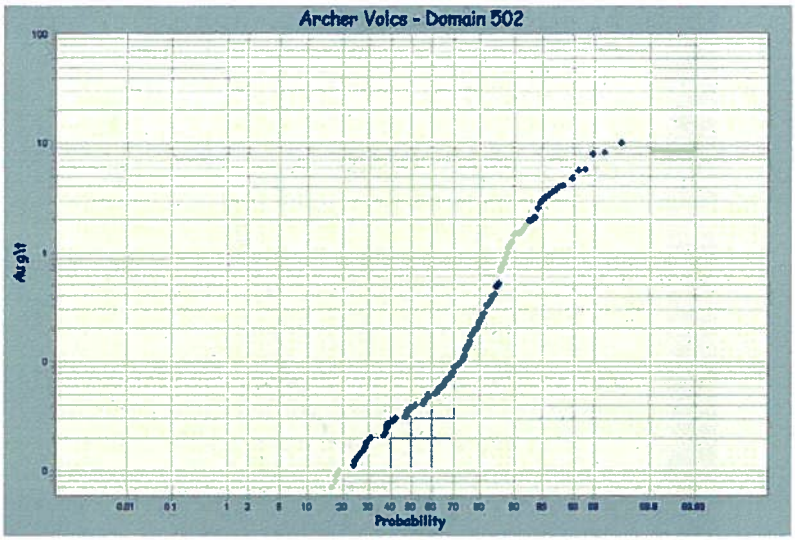

Archer Stockwork - Domain 623
Valid (non-missing) samples: 255
Missing samples: 61
Minimum: 0.005
Maximum: 78.300
Average: 5.3159
Variance: 120.9635
Standard deviation: 10.9983
Frequency
Au g/t

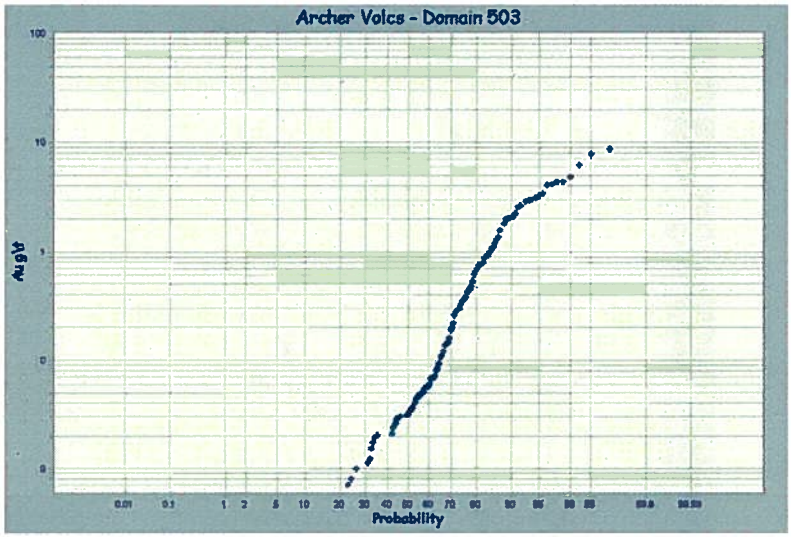
APPENDIX B: Domain Gold Cumulative Probability Plots

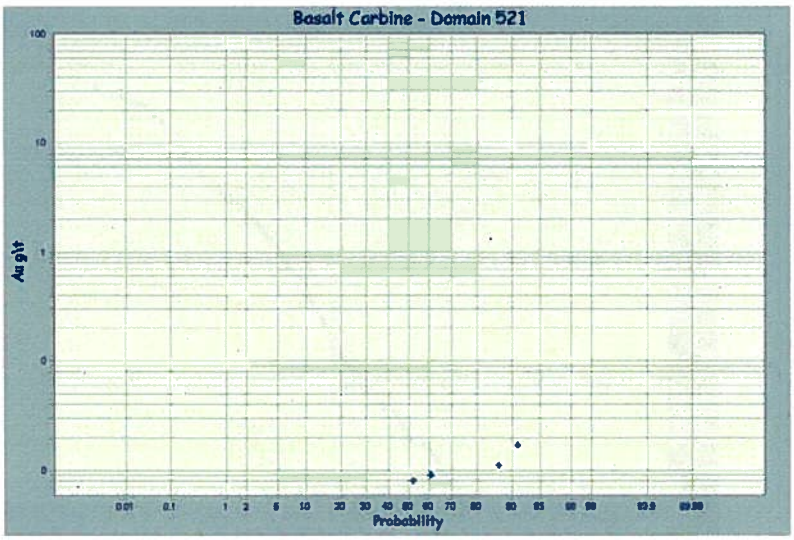


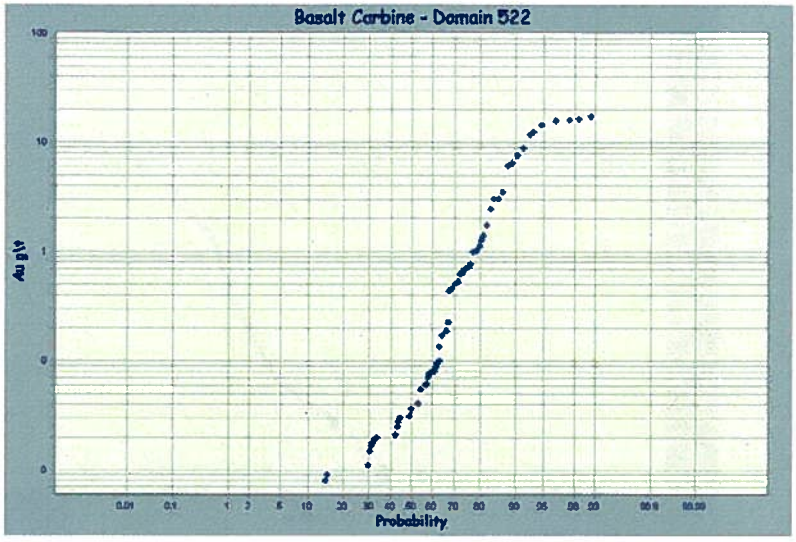


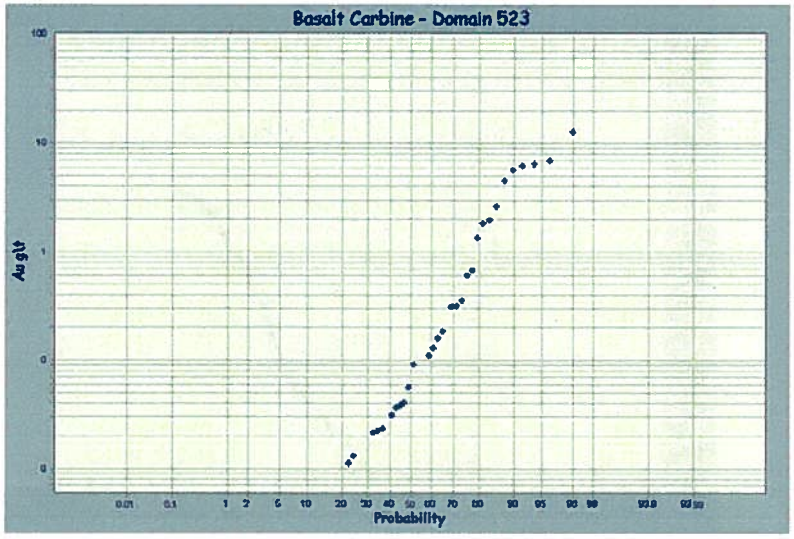


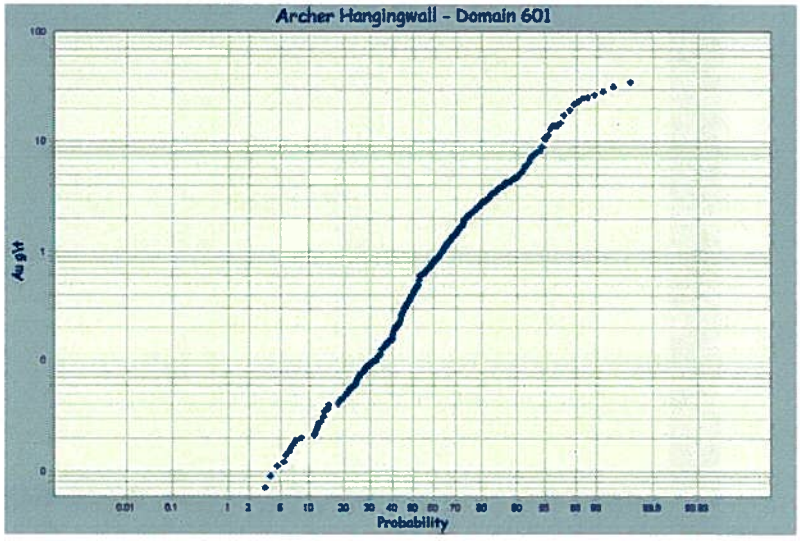

Archer Volcs - Domain 502
Augit
Probability

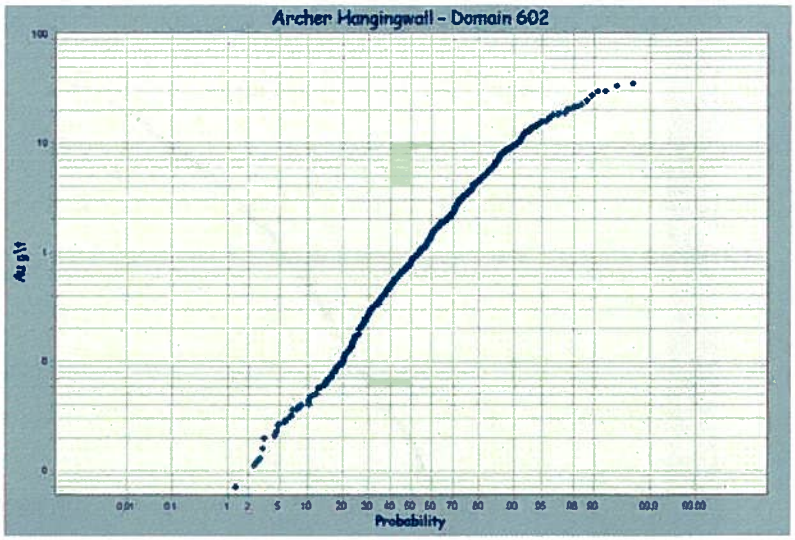

Archer Volcs - Domain 503
Augit
Probability

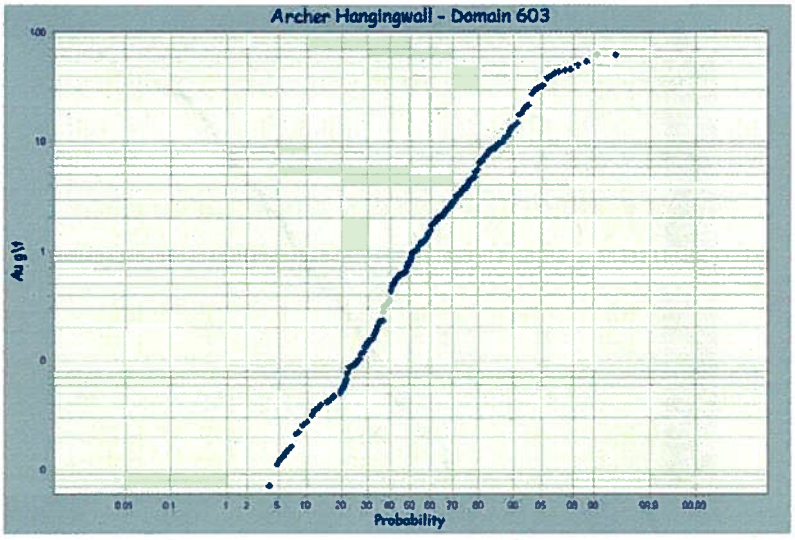

Basalt Carbine - Domain 521
Augit
Probability

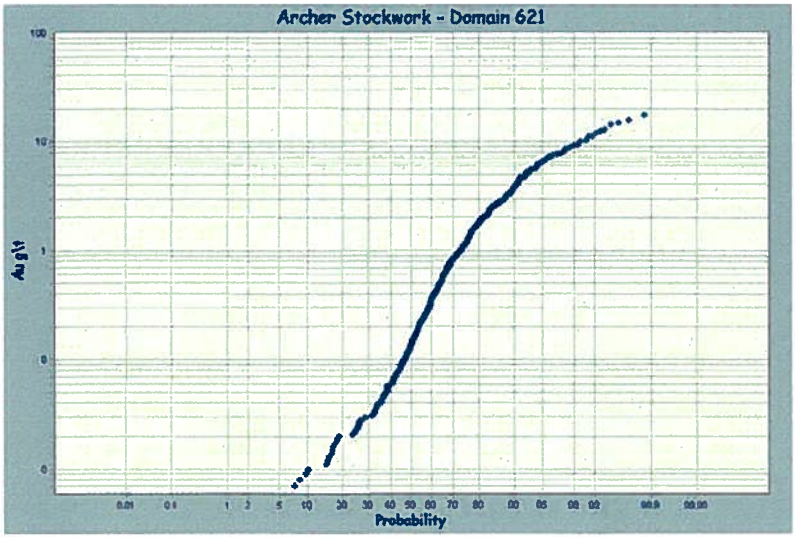

Basalt Carbine - Domain 522
Au g/t
Probability

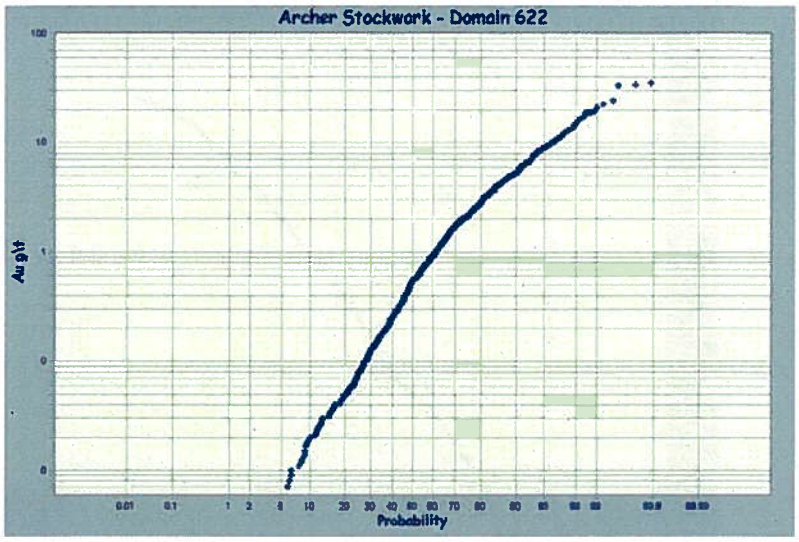

Basalt Carbine - Domain 523
Au g/t
Probability

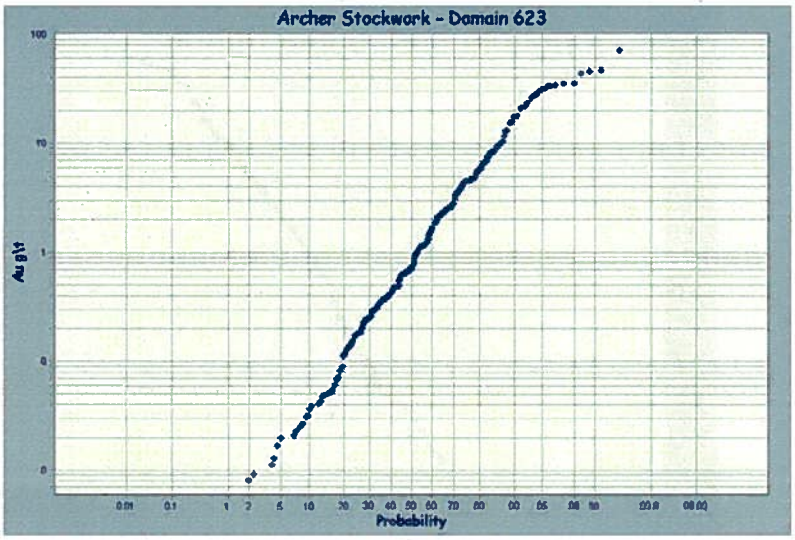

Archer Hangingwall - Domain 601
Au g/t
Probability


Archer Hangingwall - Domain 602
Au g/t
Probability

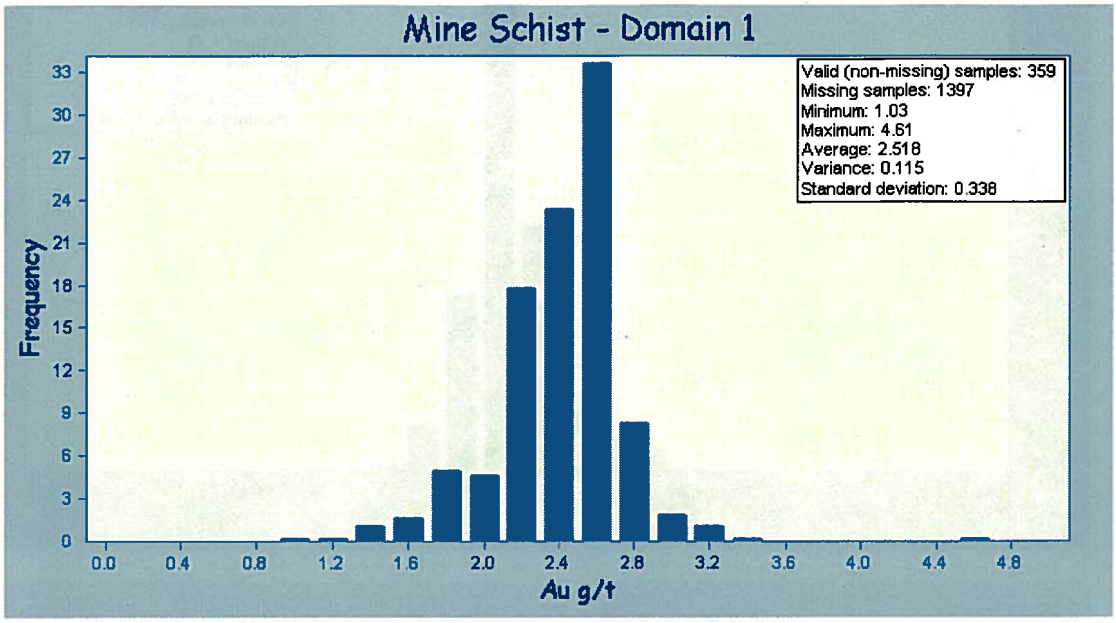

Archer Hangingwall - Domain 603
Au g/t
Probability

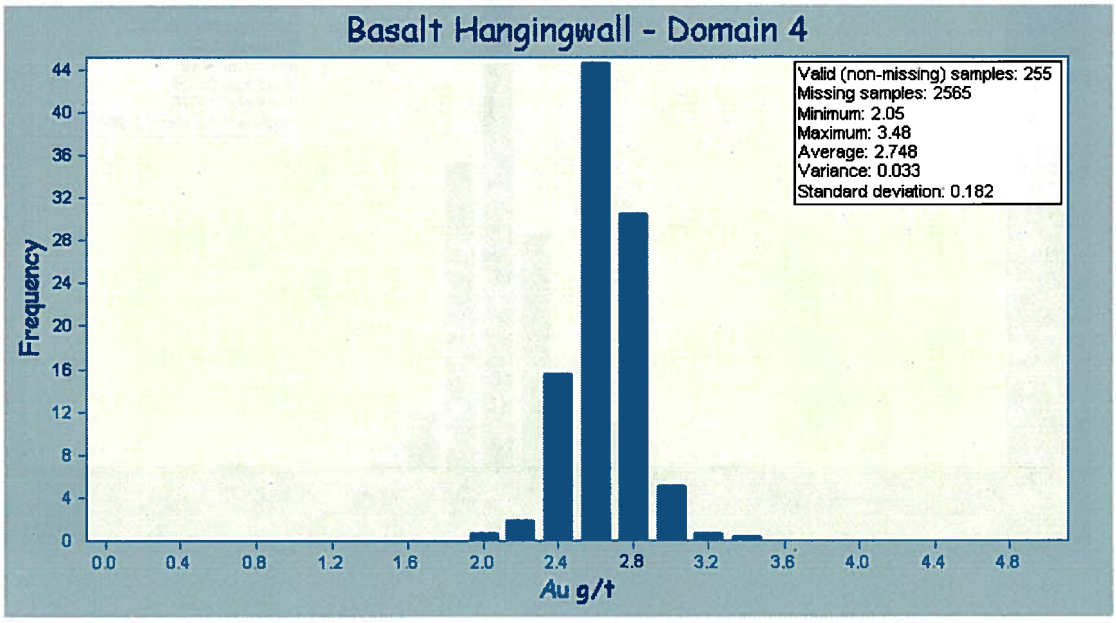

Archer Stockwork - Domain 621
Au g/t
Probability

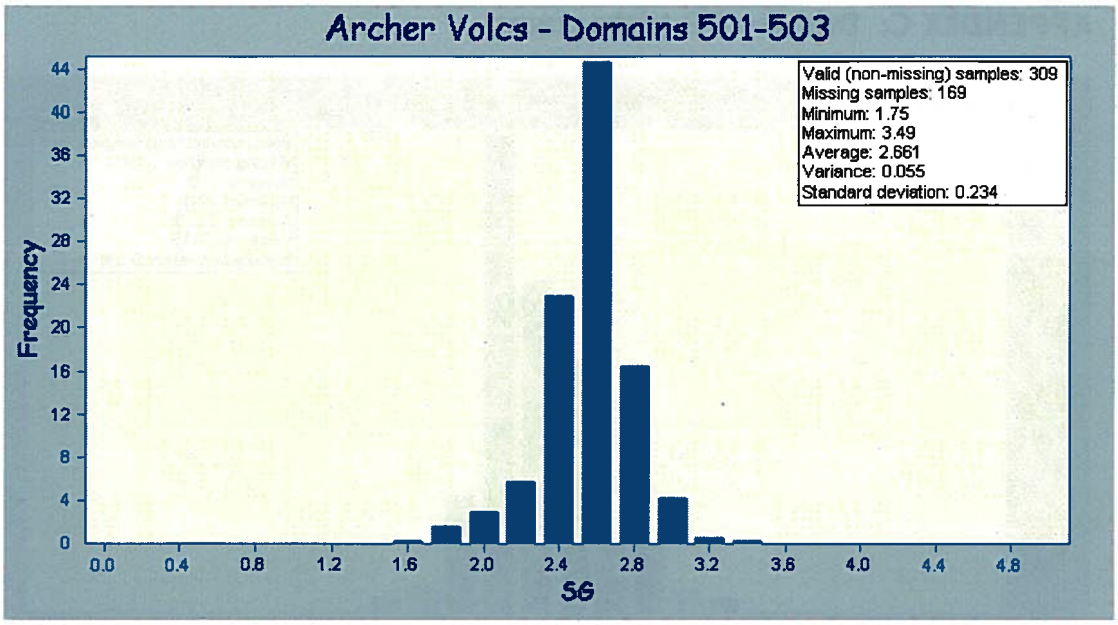

Archer Stockwork - Domain 622
Au g/t
Probability

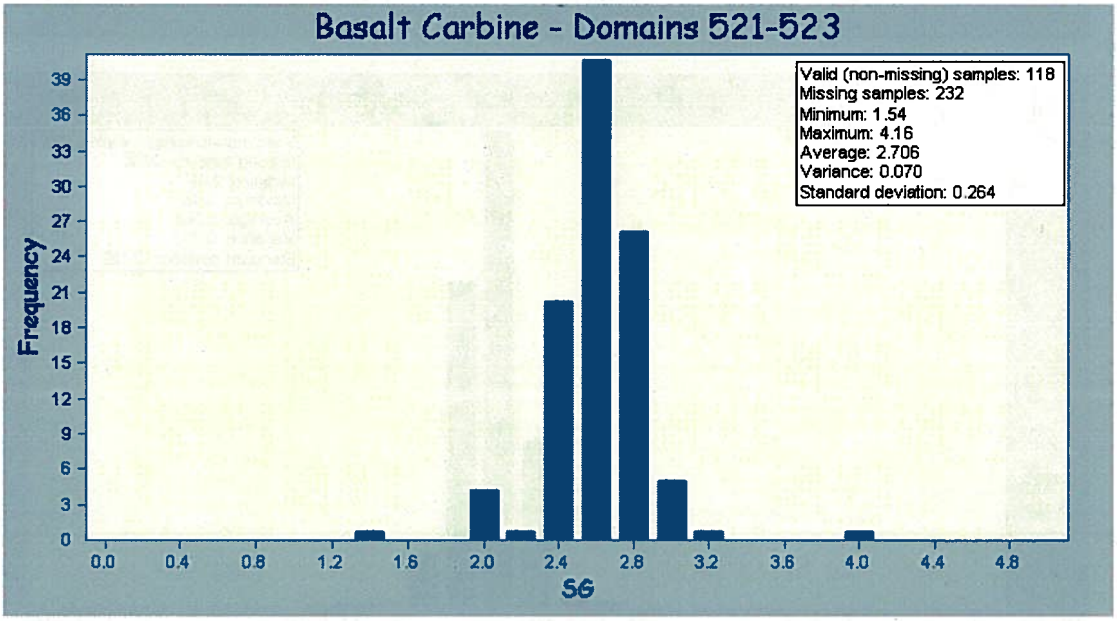

Archer Stockwork - Domain 623
Au g/t
Probability

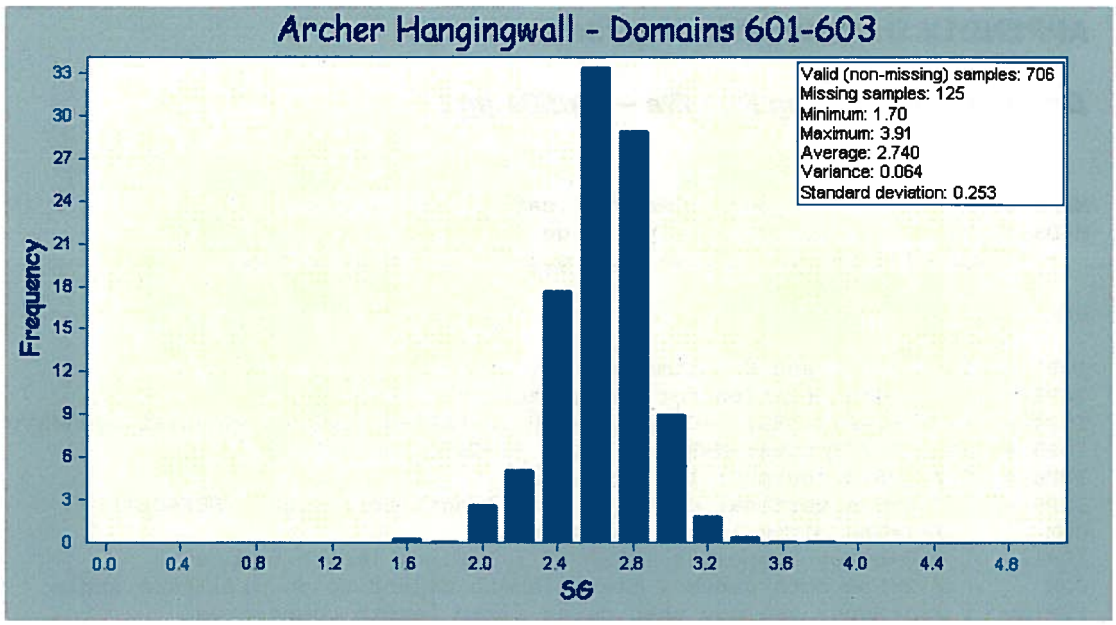
APPENDIX C: Domain SG Histograms

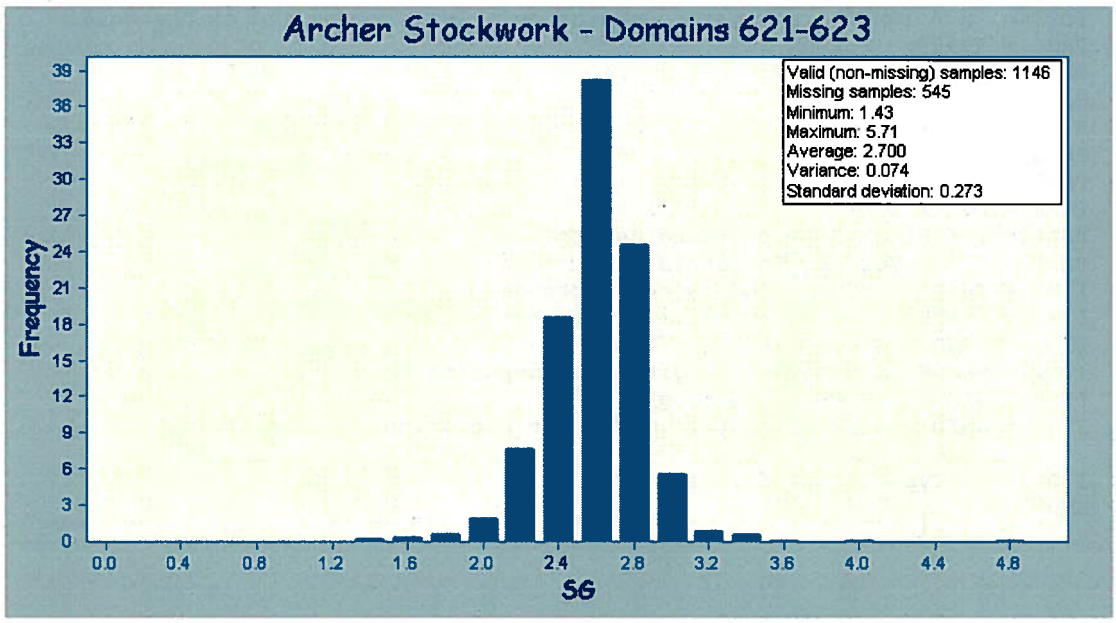





Archer Volcs - Domains 501-503
Valid (non-missing) samples: 309
Missing samples: 169
Minimum: 1.75
Maximum: 3.49
Average: 2.661
Variance: 0.055
Standard deviation: 0.234
Frequency
SG


Basalt Carbine - Domains 521-523
Valid (non-missing) samples: 118
Missing samples: 232
Minimum: 1.54
Maximum: 4.16
Average: 2.706
Variance: 0.070
Standard deviation: 0.264
Frequency
SG


Archer Hangingwall - Domains 601-603
Valid (non-missing) samples: 706
Missing samples: 125
Minimum: 1.70
Maximum: 3.91
Average: 2.740
Variance: 0.064
Standard deviation: 0.253
Frequency
SG


Archer Stockwork - Domains 621-623
Valid (non-missing) samples: 1146
Missing samples: 545
Minimum: 1.43
Maximum: 5.71
Average: 2.700
Variance: 0.074
Standard deviation: 0.273
Frequency
SG

APPENDIX D: Estimation run files

Drillhole Compositing Run file – run501.gg2

```
MEDS-501V1  10=gg0210.dat  11=gg0211.dat  12=gg0212.dat;
MEDS-501V1   9=gg0209.dat  3=rpt501.gg2
  **  COMPOSITE DRILLHOLE DATA FROM gg0211.dat TO gg0209.dat  **

USR =   /

IOP1 =  1  0 / M12 and N12 limits
IOP3 =  1 / Start location for composites in file 9
IOP4 =  -1 / -1=All DHs; 0=Composite DH in PAR5-8 limit; 1=Special boundary
IOP5 =  -1 / 0=PCF toes; N=Read table; -1=Calc. a table
IOP6 =  0 / 0=Std. output; 1=Print details
IOP9 =  1 / 0=Use vertical or horizontal length for weight (DEFAULT)
COM       / 1=Use slope length for weight
IOP11=  1 / 0=bench composite (DEFAULT); 1=Fixed length composite
COM       / 2=Use both bench & fixed length depending on drillhole angle
IOP12=  1 / 1=Honor geology when doing fixed length composites
IOP13=  0 / 1=Store DHID in the composite file
IOP14=  0 / 1=Merge all ints, 2=Merge only ints at the end of the hole.
PAR1 = 99999.  / ZMAX
PAR2 = -99999.  / ZMIN
PAR3 = 0.  / DZ
PAR5 = 0.  / XMIN
PAR6 = 0.  / XMAX
PAR7 = 0.  / YMIN
PAR8 = 0.  / YMAX
PAR10= 1 / Block Size or Fixed length
PAR12= 0.5 / Merging Threshold
ITM1 = THICK  LNGTH / Composite thickness
ITM2 = DEPTH  -TO- / Composite DH depth
ITM3 = GRADE  GEOCD  GEOCD   /GEOL
ITM4 = GRADE  AU  AU  / 1st grade to composite
ITM5 = GEOL   GEOCD   / Geology
ITM6 = ZMID   ELEV.   / Store mid-point elevation

I-O = 1 / Type a line for each DH
END
```

Ordinary Kriging Run file – run624.501

Archer Volcanogenic – Domain 501

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.501  30=
** KRIGING of 3-D block values for GAUK **

RUN = OMIT

USR =   /  Thu Nov 10, 2005 9:28:40 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 95. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 95. 75. 30. 125. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.6 15. 15. 7.5 125. 0. 75.
CMD = SPH GSLIB 0.3 95. 75. 30. 125. 0. 75.

ITM1 = GAUK AUCUT CALC KRIGE
ITM2 = GSLOP Block error
ITM3 = GGEO Block limit
ITM4 = GEOCD RANGE 501. 501.
ITM5 = GDIST Block calc rings
ITM6 = GSAMP Block calc #comp
CMD = Block limit GGEO 501
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.502

Archer Volcanogenic – Domain 502

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.502  30=
** KRIGING of 3-D block values for HAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:33:19 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 95. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 95. 75. 30. 125. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.6 15. 15. 7.5 125. 0. 75.
CMD = SPH GSLIB 0.3 95. 75. 30. 125. 0. 75.

ITM1 = HAUK AUCUT CALC KRIGE
ITM2 = HSLOP Block error
ITM3 = HGEO Block limit
ITM4 = GEOCD RANGE 502. 502.
ITM5 = HDIST Block calc rings
ITM6 = HSAMP Block calc #comp
CMD = Block limit HGEO 502
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.503

Archer Volcanogenic – Domain 503

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.503  30=
** KRIGING of 3-D block values for IAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:28:46 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 95. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 95. 75. 30. 125. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.6 15. 15. 7.5 125. 0. 75.
CMD = SPH GSLIB 0.3 95. 75. 30. 125. 0. 75.

ITM1 = IAUK AUCUT CALC KRIGE
ITM2 = ISLOP Block error
ITM3 = IGEO Block limit
ITM4 = GEOCD RANGE 503. 503.
ITM5 = IDIST Block calc rings
ITM6 = ISAMP Block calc #comp
CMD = Block limit IGEO 503
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.521

Basalt Carbine – Domain 521

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.521  30=
** KRIGING of 3-D block values for JAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:23:02 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 45. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 45. 30. 15. 130. 0. 75.
CMD = NUG 0.25
CMD = SPH GSLIB 0.35 15. 7.5 7.5 130. 0. 75.
CMD = SPH GSLIB 0.4 45. 30. 15. 130. 0. 75.

ITM1 = JAUK AUCUT CALC KRIGE
ITM2 = JSLOP Block error
ITM3 = JGEO Block limit
ITM4 = GEOCD RANGE 521. 521.
ITM5 = JDIST Block calc rings
ITM6 = JSAMP Block calc #comp
CMD = Block limit JGEO 521
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.522

Basalt Carbine – Domain 522

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.522  31=comps.xyz 30=
** KRIGING of 3-D block values for KAUK

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:26:40 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= 2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octants;
COM         4=Split Quadrants
COM        Note: Negative #=use rotated search. By Default,
COM         octants & quadrants are defined using the
COM         project's orthogonal axes).
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 45. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 45. 30. 15. 130. 0. 75.
CMD = NUG 0.25
CMD = SPH GSLIB 0.35 15. 7.5 7.5 130. 0. 75.
CMD = SPH GSLIB 0.4 45. 30. 15. 130. 0. 75.

ITM1 = KAUK AUCUT CALC KRIGE
ITM2 = KSLOP Block error
ITM3 = KGEO Block limit
ITM4 = GEOCD RANGE 522. 522.
ITM5 = KDIST Block calc rings
ITM6 = KSAMP Block calc #comp
CMD = Block limit codes 522
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.523

Basalt Carbine – Domain 523

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.523  30=
** KRIGING of 3-D block values for LAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:18:00 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 45. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 45. 30. 15. 130. 0. 75.
CMD = NUG 0.25
CMD = SPH GSLIB 0.35 15. 7.5 7.5 130. 0. 75.
CMD = SPH GSLIB 0.4 45. 30. 15. 130. 0. 75.

ITM1 = LAUK AUCUT CALC KRIGE
ITM2 = LSLOP Block error
ITM3 = LGEO Block limit
ITM4 = GEOCD RANGE 523. 523.
ITM5 = LDIST Block calc rings
ITM6 = LSAMP Block calc #comp
CMD = Block limit LGEO 523
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.601

Archer Hangingwall – Domain 601

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.601  30=
** KRIGING of 3-D block values for AAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:16:10 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 105. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 50. 20. 15. 140. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.7 7.5 7.5 7.5 140. 0. 75.
CMD = SPH GSLIB 0.2 50. 20. 15. 140. 0. 75.

ITM1 = AAUK AUCUT CALC KRIGE
ITM2 = ASLOP Block error
ITM3 = AGEO Block limit
ITM4 = GEOCD RANGE 601. 601.
ITM5 = ADIST Block calc rings
ITM6 = ASAMP Block calc #comp
CMD = Block limit AGEO 601
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.602

Archer Hangingwall – Domain 602

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.602  30=
** KRIGING of 3-D block values for BAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:13:33 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 105. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 50. 20. 15. 140. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.7 7.5 7.5 7.5 140. 0. 75.
CMD = SPH GSLIB 0.2 50. 20. 15. 140. 0. 75.

ITM1 = BAUK AUCUT CALC KRIGE
ITM2 = BSLOP Block error
ITM3 = BGEO Block limit
ITM4 = GEOCD RANGE 602. 602.
ITM5 = BDIST Block calc rings
ITM6 = BSAMP Block calc #comp
CMD = Block limit BGEO 602
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.603

Archer Hangingwall – Domain 603

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.603  30=
** KRIGING of 3-D block values for CAUK

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:11:21 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octants;
COM         4=Split Quadrants
COM        Note: Negative #=use rotated search. By Default,
COM         octants & quadrants are defined using the
COM         project's orthogonal axes).
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 105. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 50. 20. 15. 140. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.7 7.5 7.5 7.5 140. 0. 75.
CMD = SPH GSLIB 0.2 50. 20. 15. 140. 0. 75.

ITM1 = CAUK AUCUT CALC KRIGE
ITM2 = CSLOP Block error
ITM3 = CGEO Block limit
ITM4 = GEOCD RANGE 603. 603.
ITM5 = CDIST Block calc rings
ITM6 = CSAMP Block calc #comp
CMD = Block limit codes 603
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.621

Archer Stockwork – Domain 621

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.621  30=
** KRIGING of 3-D block values for DAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 10:00:59 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 105. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 105. 90. 40. 130. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.75 15. 15. 7.5 130. 0. 75.
CMD = SPH GSLIB 0.15 105. 90. 40. 130. 0. 75.

ITM1 = DAUK AUCUT CALC KRIGE
ITM2 = DSLOP Block error
ITM3 = DGEO Block limit
ITM4 = GEOCD RANGE 621. 621.
ITM5 = DDIST Block calc rings
ITM6 = DSAMP Block calc #comp
CMD = Block limit DGEO 621
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.622

Archer Stockwork – Domain 622

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.622  30=
** KRIGING of 3-D block values for EAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 9:44:55 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 105. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 105. 90. 40. 130. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.75 15. 15. 7.5 130. 0. 75.
CMD = SPH GSLIB 0.15 105. 90. 40. 130. 0. 75.

ITM1 = EAUK AUCUT CALC KRIGE
ITM2 = ESLOP Block error
ITM3 = EGEO Block limit
ITM4 = GEOCD RANGE 622. 622.
ITM5 = EDIST Block calc rings
ITM6 = ESAMP Block calc #comp
CMD = Block limit EGEO 622
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

Ordinary Kriging Run file – run624.623

Archer Stockwork – Domain 623

```
MEDS-624V1  10=gg0210.dat  9=gg0209.dat  15=gg0215.dat;
MEDS-624V1   3=rpt624.623  30=
** KRIGING of 3-D block values for FAUK **

RUN = OMIT

USR =   /  Wed Nov 9, 2005 9:57:37 PM W. Australia Standard Time

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM         4=Split Quadrant.
COM         Note: Negative#=use rotated search.  By Default,
COM         octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 20 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 105. / Limiting search distance
PAR7 = 40. / Max distance to closest point
PAR8 = 40. / Max distance to project single composite
PAR27= 5 6 6 / Block discretization in X,Y,Z
PAR31= 15. / Outlier cutoff to change the search
PAR32= 15. / Max. 3D search distance at PAR31

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH GSLIB 105. 90. 40. 130. 0. 75.
CMD = NUG 0.1
CMD = SPH GSLIB 0.75 15. 15. 7.5 130. 0. 75.
CMD = SPH GSLIB 0.15 105. 90. 40. 130. 0. 75.

ITM1 = FAUK AUCUT CALC KRIGE
ITM2 = FSLOP Block error
ITM3 = FGEO Block limit
ITM4 = GEOCD RANGE 623. 623.
ITM5 = FDIST Block calc rings
ITM6 = FSAMP Block calc #comp
CMD = Block limit FGEO 623
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

APPENDIX E: 'Assay cutting' Excel Spreadsheets

Mine Schist - Domain 1

Raw assay stats

# assays	485
metres	474.9
Max value	16.189
Mean	0.22
CV	5.062

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
20	0.22	5.062	0.00%	0.00%	0
15	0.21	4.951	1.16%	0.21%	1
10	0.20	4.413	7.72%	0.41%	2
5	0.18	3.780	17.99%	0.62%	3
4	0.17	3.601	22.47%	1.24%	6
3	0.15	3.339	29.92%	1.86%	9
2	0.13	2.979	40.54%	2.68%	13
1	0.09	2.419	56.67%	4.54%	22

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
485	0.22	5.062		
1	0.22	5.057	0.00%	0.00%
2	0.22	5.052	0.00%	0.01%
3	0.22	5.047	0.00%	0.01%
4	0.22	5.041	0.00%	0.02%
5	0.22	5.036	0.00%	0.02%
6	0.22	5.031	0.00%	0.03%
7	0.22	5.026	0.00%	0.03%
8	0.22	5.021	0.00%	0.04%
9	0.22	5.015	0.00%	0.04%
10	0.22	5.010	0.00%	0.05%
11	0.22	5.005	0.00%	0.05%
12	0.22	5.000	0.00%	0.06%
13	0.22	4.995	0.00%	0.06%
14	0.22	4.989	0.00%	0.07%
15	0.22	4.984	0.00%	0.07%
16	0.22	4.979	0.00%	0.08%
17	0.22	4.974	0.01%	0.09%
18	0.23	4.969	0.00%	0.09%
19	0.23	4.963	0.00%	0.10%
20	0.23	4.958	0.00%	0.10%

Basalt hangingwall - Domain 4

Raw assay stats

# assays	397
metres	357.6
Max value	25.1
Mean	0.37
CV	5.827

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
30	0.37	5.827	0.00%	0.00%	0
25	0.37	5.822	0.07%	0.25%	1
20	0.35	5.543	4.78%	0.50%	2
15	0.32	5.133	12.39%	0.76%	3
10	0.28	4.664	22.59%	1.01%	4
5	0.21	3.831	42.23%	2.02%	8
4	0.19	3.573	47.62%	2.52%	10
3	0.17	3.230	54.36%	2.52%	10

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
397	0.37	5.827		
1	0.37	5.822	0.04%	0.04%
2	0.37	5.817	0.04%	0.08%
3	0.37	5.811	0.04%	0.11%
4	0.37	5.806	0.03%	0.15%
5	0.37	5.800	0.03%	0.18%
6	0.37	5.793	0.02%	0.19%
7	0.37	5.787	0.03%	0.22%
8	0.37	5.781	0.03%	0.25%
9	0.37	5.775	0.02%	0.28%
10	0.37	5.775	0.13%	0.41%
11	0.37	5.768	0.01%	0.42%
12	0.38	5.760	0.00%	0.42%
13	0.38	5.753	0.00%	0.42%
14	0.38	5.746	0.00%	0.43%
15	0.38	5.741	0.05%	0.48%
16	0.38	5.733	0.01%	0.48%
17	0.38	5.727	0.02%	0.50%
18	0.38	5.723	0.07%	0.57%
19	0.38	5.716	0.02%	0.58%
20	0.38	5.709	0.00%	0.59%

Archer Volcanogenics - Domain 501

Raw assay stats

# assays	98
metres	92.0
Max value	24.29
Mean	2.11
CV	2.112

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
25	2.11	2.112	0.00%	0.00%	0
22	2.07	2.058	1.75%	2.04%	2
20	2.03	1.996	3.81%	2.04%	2
15	1.92	1.858	8.97%	2.04%	2
10	1.74	1.702	17.43%	6.12%	6
9	1.67	1.650	20.74%	7.14%	7
8	1.59	1.593	24.35%	7.14%	7
7	1.52	1.539	27.96%	7.14%	7

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
98	2.11	2.112		
1	2.13	2.100	0.03%	0.03%
2	2.15	2.089	0.11%	0.14%
3	2.13	2.119	2.02%	2.16%
4	2.10	2.161	2.67%	4.84%
5	2.09	2.179	1.29%	6.13%
6	2.11	2.177	0.47%	6.60%
7	2.13	2.164	0.04%	6.64%
8	2.15	2.150	0.02%	6.65%
9	2.18	2.136	0.00%	6.65%
10	2.21	2.122	0.03%	6.68%
11	2.23	2.107	0.01%	6.69%
12	2.26	2.093	0.03%	6.71%
13	2.29	2.079	0.03%	6.74%
14	2.32	2.064	0.01%	6.74%
15	2.35	2.049	0.00%	6.75%
16	2.38	2.034	0.00%	6.75%
17	2.41	2.019	0.00%	6.75%
18	2.44	2.004	0.00%	6.75%
19	2.47	1.988	0.01%	6.77%
20	2.42	2.036	3.16%	9.92%

Archer Volcanogenics - Domain 502

Raw assay stats

# assays	287
metres	268.7
Max value	14.5
Mean	0.47
CV	2.944

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
15	0.47	2.944	0.00%	0.00%	0
14	0.47	2.936	0.24%	0.35%	1
13	0.46	2.921	0.72%	0.35%	1
12	0.46	2.908	1.19%	0.35%	1
11	0.46	2.897	1.67%	0.35%	1
10	0.46	2.887	2.17%	0.70%	2
9	0.45	2.842	3.44%	0.70%	2
8	0.44	2.795	4.92%	1.05%	3

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
287	0.47	2.944		
1	0.47	2.939	0.00%	0.00%
2	0.47	2.933	0.00%	0.01%
3	0.47	2.928	0.02%	0.03%
4	0.47	2.923	0.02%	0.05%
5	0.48	2.917	0.00%	0.05%
6	0.48	2.912	0.00%	0.06%
7	0.48	2.907	0.02%	0.08%
8	0.48	2.901	0.01%	0.09%
9	0.48	2.895	0.00%	0.09%
10	0.48	2.890	0.00%	0.10%
11	0.49	2.884	0.00%	0.10%
12	0.49	2.878	0.01%	0.11%
13	0.49	2.873	0.01%	0.12%
14	0.49	2.867	0.01%	0.12%
15	0.49	2.861	0.00%	0.13%
16	0.49	2.856	0.00%	0.13%
17	0.50	2.850	0.00%	0.14%
18	0.50	2.844	0.01%	0.15%
19	0.50	2.839	0.02%	0.17%
20	0.50	2.835	0.06%	0.23%

Archer Volcanogenics - Domain 503

Raw assay stats

# assays	199
metres	187.2
Max value	13.8
Mean	0.61
CV	2.470

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
15	0.61	2.470	0.00%	0.00%	0
13	0.61	2.452	0.49%	0.50%	1
12	0.61	2.431	1.10%	0.50%	1
11	0.60	2.412	1.71%	0.50%	1
10	0.60	2.396	2.32%	0.50%	1
9	0.60	2.382	2.92%	0.50%	1
8	0.59	2.351	4.09%	1.01%	2
7	0.58	2.292	6.20%	1.51%	3

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
199	0.61	2.470		
1	0.61	2.484	0.80%	0.80%
2	0.61	2.486	0.37%	1.17%
3	0.61	2.487	0.26%	1.43%
4	0.62	2.486	0.24%	1.67%
5	0.62	2.481	0.10%	1.77%
6	0.62	2.475	0.03%	1.80%
7	0.62	2.468	0.01%	1.80%
8	0.63	2.460	0.00%	1.81%
9	0.63	2.454	0.03%	1.84%
10	0.63	2.451	0.17%	2.00%
11	0.63	2.448	0.16%	2.16%
12	0.64	2.440	0.01%	2.17%
13	0.64	2.434	0.03%	2.20%
14	0.62	2.480	3.51%	5.72%
15	0.62	2.488	0.54%	6.26%
16	0.62	2.481	0.03%	6.29%
17	0.63	2.473	0.02%	6.30%
18	0.63	2.466	0.00%	6.31%
19	0.63	2.458	0.00%	6.31%
20	0.64	2.450	0.01%	6.32%

Basalt Carbine - Domain 521

Raw assay stats

# assays	23
metres	20.0
Max value	0.07
Mean	0.01
CV	1.417

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
45	0.01	1.417	0.00%	0.00%	0
40	0.01	1.417	0.00%	0.00%	0
35	0.01	1.417	0.00%	0.00%	0
30	0.01	1.417	0.00%	0.00%	0
25	0.01	1.417	0.00%	0.00%	0
30	0.01	1.417	0.00%	0.00%	0
35	0.01	1.417	0.00%	0.00%	0
20	0.01	1.417	0.00%	0.00%	0

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
23	0.01	1.417		
1	0.01	1.412	2.07%	2.07%
2	0.01	1.406	2.07%	4.14%
3	0.01	1.399	2.07%	6.20%
4	0.01	1.390	2.07%	8.27%
5	0.01	1.379	2.07%	10.34%
6	0.02	1.367	2.07%	12.41%
7	0.02	1.334	1.24%	13.65%
8	0.02	1.400	6.62%	20.27%
9	0.01	1.017	28.96%	49.22%
10	0.01	0.402	15.93%	65.15%
11	0.01	0.402	4.14%	69.29%
12	0.01	0.421	3.31%	72.60%
13	0.01	0.447	2.90%	75.49%
14	0.01	0.475	2.48%	77.97%
15	0.01	0.495	2.07%	80.04%
16	0.01	0.540	3.31%	83.35%
17	0.01	0.545	1.65%	85.01%
18	0.01	0.567	4.14%	89.14%
19	0.01	0.384	4.14%	93.28%
20	0.01	0.479	1.86%	95.14%

Basalt Carbine - Domain 522

Raw assay stats

# assays	168
metres	159.2
Max value	17.44
Mean	1.89
CV	2.323

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
20	1.89	2.323	0.00%	0.00%	0
17	1.88	2.319	0.29%	1.19%	2
16	1.87	2.311	1.00%	1.79%	3
15	1.83	2.291	2.92%	5.36%	9
14	1.78	2.262	5.74%	6.55%	11
13	1.72	2.229	9.11%	6.55%	11
12	1.65	2.196	12.60%	7.14%	12
11	1.57	2.157	16.73%	8.33%	14

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
168	1.89	2.323		
1	1.89	2.321	0.25%	0.25%
2	1.90	2.323	0.37%	0.63%
3	1.90	2.335	0.81%	1.43%
4	1.82	2.378	4.71%	6.15%
5	1.73	2.418	5.25%	11.40%
6	1.70	2.459	2.15%	13.55%
7	1.67	2.501	2.03%	15.57%
8	1.56	2.537	5.80%	21.38%
9	1.47	2.584	5.11%	26.49%
10	1.47	2.597	0.57%	27.07%
11	1.40	2.662	3.87%	30.93%
12	1.41	2.659	0.17%	31.10%
13	1.40	2.688	1.01%	32.11%
14	1.38	2.718	1.02%	33.12%
15	1.39	2.715	0.15%	33.28%
16	1.40	2.715	0.24%	33.51%
17	1.40	2.706	0.01%	33.52%
18	1.41	2.696	0.00%	33.53%
19	1.42	2.687	0.03%	33.55%
20	1.43	2.691	0.33%	33.88%

Basalt Carbine - Domain 523

Raw assay stats

# assays	49
metres	44.3
Max value	12.603
Mean	1.09
CV	2.050

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
15	1.09	2.050	0.00%	0.00%	0
10	1.03	2.003	5.25%	4.08%	2
9	1.01	1.991	7.33%	4.08%	2
8	0.99	1.984	9.40%	4.08%	2
7	0.96	1.983	11.47%	4.08%	2
6	0.92	1.977	15.30%	10.20%	5
5	0.83	1.918	23.46%	12.24%	6
4	0.73	1.839	33.25%	14.29%	7

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
49	1.09	2.050		
1	1.07	2.106	3.78%	3.78%
2	1.09	2.085	0.23%	4.00%
3	1.10	2.058	0.02%	4.03%
4	1.12	2.031	0.01%	4.04%
5	1.13	2.025	0.86%	4.89%
6	1.16	1.998	0.04%	4.94%
7	1.19	1.970	0.02%	4.96%
8	1.10	2.080	7.84%	12.80%
9	1.13	2.058	0.33%	13.13%
10	1.16	2.029	0.06%	13.19%
11	1.19	1.997	0.02%	13.21%
12	1.21	1.982	0.64%	13.86%
13	1.05	2.119	13.29%	27.14%
14	0.81	2.127	18.15%	45.30%
15	0.83	2.091	0.01%	45.31%
16	0.86	2.054	0.01%	45.32%
17	0.89	2.017	0.01%	45.33%
18	0.92	1.978	0.01%	45.34%
19	0.95	1.939	0.02%	45.36%
20	0.99	1.909	0.23%	45.59%

Archer Hangingwall - Domain 601

Raw assay stats

# assays	368
metres	350.4
Max value	37.1
Mean	2.23
CV	2.235

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
40	2.23	2.235	0.00%	0.00%	0
35	2.23	2.236	0.03%	0.27%	1
30	2.21	2.201	0.95%	0.82%	3
25	2.16	2.134	2.96%	1.63%	6
20	2.06	2.006	7.40%	2.45%	9
19	2.04	1.975	8.53%	2.99%	11
18	2.01	1.941	9.83%	2.99%	11
17	1.98	1.907	11.16%	3.26%	12

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
368	2.23	2.235		
1	2.23	2.232	0.00%	0.00%
2	2.23	2.228	0.01%	0.01%
3	2.23	2.225	0.01%	0.03%
4	2.23	2.221	0.00%	0.03%
5	2.22	2.229	0.48%	0.50%
6	2.22	2.225	0.00%	0.50%
7	2.22	2.221	0.00%	0.50%
8	2.22	2.218	0.00%	0.51%
9	2.22	2.214	0.02%	0.52%
10	2.22	2.211	0.00%	0.52%
11	2.22	2.207	0.00%	0.52%
12	2.22	2.206	0.13%	0.65%
13	2.22	2.203	0.00%	0.65%
14	2.23	2.199	0.00%	0.66%
15	2.23	2.195	0.00%	0.66%
16	2.23	2.192	0.00%	0.66%
17	2.23	2.188	0.00%	0.66%
18	2.23	2.187	0.11%	0.77%
19	2.24	2.187	0.13%	0.90%
20	2.24	2.183	0.00%	0.90%

Archer Hangingwall - Domain 602

Raw assay stats

# assays	457
metres	437.4
Max value	36.1
Mean	3.20
CV	1.773

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
40	3.20	1.773	0.00%	0.00%	0
35	3.20	1.770	0.10%	0.44%	2
30	3.17	1.742	0.96%	0.88%	4
25	3.11	1.689	2.81%	1.31%	6
20	3.02	1.623	5.67%	2.63%	12
19	2.99	1.607	6.50%	2.63%	12
18	2.96	1.588	7.52%	3.94%	18
17	2.92	1.565	8.78%	4.16%	19

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
457	3.20	1.773		
1	3.21	1.770	0.00%	0.00%
2	3.21	1.768	0.04%	0.04%
3	3.21	1.767	0.03%	0.07%
4	3.19	1.775	0.99%	1.06%
5	3.16	1.782	1.13%	2.19%
6	3.17	1.780	0.00%	2.20%
7	3.18	1.777	0.00%	2.20%
8	3.18	1.774	0.00%	2.20%
9	3.18	1.779	0.36%	2.56%
10	3.17	1.784	0.40%	2.96%
11	3.18	1.783	0.08%	3.04%
12	3.18	1.781	0.01%	3.05%
13	3.19	1.780	0.09%	3.14%
14	3.19	1.782	0.22%	3.36%
15	3.20	1.780	0.01%	3.37%
16	3.20	1.777	0.00%	3.37%
17	3.21	1.774	0.00%	3.38%
18	3.22	1.772	0.01%	3.38%
19	3.23	1.770	0.03%	3.41%
20	3.23	1.769	0.05%	3.46%

Archer Hangingwall - Domain 603

Raw assay stats

# assays	228
metres	219.9
Max value	67.44
Mean	5.44
CV	2.246

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
70	5.44	2.246	0.00%	0.00%	0
60	5.38	2.215	1.12%	1.32%	3
50	5.22	2.148	3.99%	2.19%	5
45	5.11	2.105	6.18%	3.07%	7
40	4.91	2.039	9.72%	4.39%	10
35	4.67	1.960	14.19%	4.82%	11
30	4.39	1.872	19.31%	6.14%	14
25	4.05	1.767	25.62%	7.02%	16

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
228	5.44	2.246		
1	5.47	2.240	0.00%	0.00%
2	5.49	2.236	0.07%	0.07%
3	5.49	2.242	0.47%	0.54%
4	5.50	2.242	0.25%	0.78%
5	5.52	2.237	0.05%	0.83%
6	5.55	2.232	0.02%	0.85%
7	5.57	2.227	0.03%	0.88%
8	5.59	2.222	0.05%	0.93%
9	5.58	2.232	0.70%	1.63%
10	5.57	2.243	0.73%	2.36%
11	5.56	2.252	0.59%	2.95%
12	5.51	2.273	1.25%	4.20%
13	5.49	2.289	0.95%	5.15%
14	5.49	2.292	0.35%	5.50%
15	5.49	2.300	0.54%	6.05%
16	5.48	2.310	0.64%	6.69%
17	5.47	2.318	0.54%	7.22%
18	5.47	2.316	0.16%	7.38%
19	5.47	2.320	0.39%	7.77%
20	5.48	2.324	0.39%	8.16%

Archer Stockwork - Domain 621

Raw assay stats

# assays	739
metres	714.9
Max value	17.657
Mean	1.21
CV	1.998

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
20	1.21	1.998	0.00%	0.00%	0
15	1.20	1.976	0.60%	0.41%	3
14	1.20	1.959	1.08%	0.68%	5
13	1.19	1.941	1.63%	0.81%	6
12	1.18	1.919	2.34%	1.08%	8
11	1.17	1.896	3.22%	1.35%	10
10	1.16	1.869	4.34%	1.62%	12
9	1.14	1.836	5.87%	2.30%	17

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
739	1.21	1.998		
1	1.21	1.997	0.01%	0.01%
2	1.22	1.995	0.01%	0.02%
3	1.22	1.994	0.00%	0.02%
4	1.22	1.992	0.01%	0.03%
5	1.22	1.991	0.01%	0.04%
6	1.22	1.989	0.01%	0.05%
7	1.22	1.988	0.01%	0.06%
8	1.22	1.986	0.01%	0.07%
9	1.23	1.986	0.06%	0.12%
10	1.23	1.984	0.00%	0.13%
11	1.23	1.983	0.00%	0.13%
12	1.23	1.981	0.01%	0.13%
13	1.23	1.980	0.03%	0.16%
14	1.23	1.979	0.03%	0.19%
15	1.23	1.978	0.00%	0.20%
16	1.24	1.976	0.00%	0.20%
17	1.24	1.976	0.08%	0.28%
18	1.24	1.979	0.21%	0.49%
19	1.23	1.983	0.28%	0.77%
20	1.23	1.987	0.31%	1.08%

Archer Stockwork - Domain 622

Raw assay stats

# assays	902
metres	871.9
Max value	41.183
Mean	2.08
CV	2.060

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
45	2.08	2.060	0.00%	0.00%	0
40	2.08	2.055	0.07%	0.11%	1
35	2.07	2.036	0.35%	0.22%	2
30	2.04	1.966	1.58%	0.55%	5
25	2.02	1.898	2.96%	0.55%	5
20	1.97	1.813	5.11%	1.11%	10
15	1.89	1.694	8.99%	2.11%	19
10	1.74	1.530	16.07%	4.10%	37

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
902	2.08	2.060		
1	2.07	2.063	0.24%	0.24%
2	2.07	2.067	0.32%	0.55%
3	2.06	2.073	0.49%	1.04%
4	2.06	2.079	0.41%	1.46%
5	2.05	2.083	0.25%	1.70%
6	2.05	2.084	0.12%	1.82%
7	2.05	2.087	0.22%	2.04%
8	2.05	2.090	0.20%	2.24%
9	2.05	2.089	0.03%	2.27%
10	2.05	2.088	0.01%	2.28%
11	2.06	2.087	0.00%	2.28%
12	2.06	2.085	0.00%	2.28%
13	2.06	2.084	0.00%	2.28%
14	2.06	2.082	0.00%	2.28%
15	2.06	2.085	0.18%	2.47%
16	2.06	2.088	0.20%	2.67%
17	2.06	2.092	0.28%	2.95%
18	2.04	2.098	0.75%	3.70%
19	2.03	2.104	0.54%	4.24%
20	2.04	2.104	0.04%	4.27%

Archer Stockwork - Domain 623

Raw assay stats

# assays	255
metres	242.6
Max value	78.3
Mean	5.09
CV	2.146

Cut stats

Cap Value	Cut Mean	CV	%metal	%data	# cut
80	5.09	2.146	0.00%	0.00%	0
70	5.05	2.112	0.74%	0.78%	2
60	4.97	2.046	2.36%	0.78%	2
50	4.89	1.990	3.98%	0.78%	2
40	4.76	1.929	6.38%	1.96%	5
35	4.67	1.893	8.16%	3.53%	9
30	4.46	1.821	12.37%	5.49%	14
25	4.15	1.725	18.44%	7.45%	19

Exclusion stats

# excluded	Mean	CV	% metal	% metal cum
255	5.09	2.146		
1	5.05	2.165	1.26%	1.26%
2	5.05	2.169	0.37%	1.63%
3	4.98	2.192	1.70%	3.33%
4	4.85	2.215	2.88%	6.21%
5	4.80	2.234	1.17%	7.38%
6	4.82	2.229	0.03%	7.41%
7	4.84	2.225	0.04%	7.46%
8	4.86	2.221	0.04%	7.50%
9	4.88	2.216	0.01%	7.51%
10	4.90	2.211	0.01%	7.52%
11	4.92	2.206	0.00%	7.52%
12	4.94	2.200	0.00%	7.53%
13	4.96	2.196	0.03%	7.56%
14	4.98	2.193	0.09%	7.64%
15	5.00	2.189	0.06%	7.70%
16	5.02	2.184	0.03%	7.73%
17	5.04	2.180	0.04%	7.77%
18	5.06	2.175	0.04%	7.81%
19	5.08	2.171	0.04%	7.85%
20	5.09	2.168	0.09%	7.94%

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX G

Golden Gift 3 Geology and Resource Model Report

November 2007

PERSEVERANCE CORPORATION LIMITED

STAWELL GOLD MINES

Golden Gift 3 Geology and Resource Model Report

November 2007

Technical Note TN 005

SUMMARY

The Golden Gift 3 resource model covers the area from 325N to 350N (mine grid) and between -1050mRL to -1450mRL. The ore body is bound by the Cabernet Fault above, Durif Fault to the north and the Wildcat porphyry at depth. The main lodes are the Dalray Footwall, Dalray Hangingwall and the Fury lode. The Dalray lodes are located adjacent to the hangingwall basalt whilst the Fury lode is located in the waterloo position within the basalt. Similar to other areas in the Golden Gift, GG3 contains internal basalt noses which appear to facilitate the mineralisation of broad quartz stockwork zones. The lodes are offset by several 'early south fault' orientated structures dipping predominantly to grid north.

Significant faulting occurs within the geological model with. The model is broken up into fourteen different fault blocks with differing offset between each fault. The ore body is bound by the major fault structures of the Cabernet to the top, Durif to the north and the Wildcat Porphyry at Depth. Three major structures, Pinot, New Fault, and Kamado Fault have been interpreted. These fault dip shallowly to the north and act as natural boundaries within the orebody. Between the Pinot and Cabernet Faults in the southern end of the GG3 ore body are a series of smaller faults which still cause significant offset of ore blocks.

The total Modelled Inventory at a cut off grade of 4.0g/t Au is 77,400oz. A listing of the global inventory at a series of cut-off gold grades is provided in **Error! Reference source not found.** (not depleted with current mining). Note the change in interpretation and re-estimation included a volume addition.

PERSEVERANCE CORPORTATION LIMITED	GG3 RESOURCE MODEL
Resource Classification	**Measured, Indicated, Inferred**
Mined Status	**Mined and Unmined**
Other Restrictions	**None**
Filter Applied to Evaluation	**No filter applied**
Previous Model Name	**ggnt15.dat**
New Model Name	**ggnt15.dat**

RESCAT	Cutoff (g/t)	PREVIOUS MODEL			NEW MODEL			Change			Change %		
		quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)
MEASURED INDICATED INFERRED	**0.0**	**1,960,000**	**2.3**	**147,400**	**2,852,800**	**2.1**	**191,200**	892,800	-0.3	43,800	**46%**	**-11%**	**30%**
	2.0	**892,000**	**4.0**	**115,700**	**1,008,200**	**4.2**	**134,900**	116,200	0.1	19,200	**13%**	**3%**	**17%**
	4.0	**352,700**	**5.8**	**65,400**	**369,900**	**6.5**	**77,400**	17,200	0.7	12,000	**5%**	**13%**	**18%**
	5.0	**190,100**	**6.9**	**42,100**	**211,900**	**7.9**	**53,700**	21,800	1.0	11,600	**11%**	**14%**	**28%**
	7.0	**65,000**	**8.9**	**18,600**	**88,300**	**10.7**	**30,300**	23,300	1.7	11,700	**36%**	**20%**	**63%**
	9.0	**27,800**	**10.5**	**9,400**	**47,000**	**13.1**	**19,800**	19,200	2.6	10,400	**69%**	**25%**	**111%**
MEASURED	**0.0**	-	-	-	-	-	-	0	0.0	0			
	2.0	-	-	-	-	-	-	0	0.0	0			
	4.0	-	-	-	-	-	-	0	0.0	0			
	5.0	-	-	-	-	-	-	0	0.0	0			
	7.0	-	-	-	-	-	-	0	0.0	0			
	9.0	-	-	-	-	-	-	0	0.0	0			
INDICATED	**0.0**	1,960,000	2.3	147,400	2,563,400	1.9	155,400	603,400	-0.5	8,000	**31%**	**-19%**	**5%**
	2.0	892,000	4.0	115,700	850,200	3.8	103,500	-41,800	-0.2	-12,200	**-5%**	**-6%**	**-11%**
	4.0	352,700	5.8	65,400	294,200	5.6	52,700	-58,500	-0.2	-12,700	**-17%**	**-3%**	**-19%**
	5.0	190,100	6.9	42,100	152,200	6.6	32,300	-37,900	-0.3	-9,800	**-20%**	**-4%**	**-23%**
	7.0	65,000	8.9	18,600	41,500	8.8	11,800	-23,500	-0.1	-6,800	**-36%**	**-1%**	**-37%**
	9.0	27,800	10.5	9,400	15,000	10.7	5,200	-12,800	0.2	-4,200	**-46%**	**2%**	**-45%**
INFERRED	**0.0**	-	-	-	289,400	3.9	35,900	289,400	3.9	35,900			
	2.0	-	-	-	158,000	6.2	31,400	158,000	6.2	31,400			
	4.0	-	-	-	75,800	10.1	24,700	75,800	10.1	24,700			
	5.0	-	-	-	59,700	11.1	21,400	59,700	11.1	21,400			
	7.0	-	-	-	46,800	12.3	18,500	46,800	12.3	18,500			
	9.0	-	-	-	32,000	14.2	14,600	32,000	14.2	14,600			
POTENTIAL	**0.0**	-	-	-	-	-	-	-	0.0	-			
	2.0	-	-	-	-	-	-	-	0.0	-			
	4.0	-	-	-	-	-	-	-	0.0	-			
	5.0	-	-	-	-	-	-	-	0.0	-			
	7.0	-	-	-	-	-	-	-	0.0	-			
	9.0	-	-	-	-	-	-	-	0.0	-			

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Figure 6 Histograms of Au (left) and length (right) in 2.0m composites of Domain 501. 23

Figure 7. Histograms of Au (left) and length (right) in 2.0m composites of Domain 503. 23

Figure 8. Histograms of Au (left) and length (right) in 2.0m composites of Domain 504. 24

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1 INTRODUCTION

Substantial drilling activity occurred during the May – November 2007 period to improve the geological model and also to complete a resource model by November. The objective was to complete a reserve update for production.

1.1 Geology

The Golden Gift 3 resource model covers the area from 325N to 350N (mine grid) and between -1050mRL to -1450mRL. The ore body is bound by the Cabernet Fault above, Durif Fault to the north and the Wildcat porphyry at depth. The main lodes are the Dalray Footwall, Dalray Hangingwall and the Fury lode. The Dalray lodes are located adjacent to the hangingwall basalt whilst the Fury lode is located in the waterloo position within the basalt. Similar to other areas in the Golden Gift, GG3 contains internal basalt noses which appear to facilitate the mineralisation of broad quartz stockwork zones. The lodes are offset by several 'early south fault' orientated structures dipping predominantly to grid north. Drilling has located a high grade stockwork zone south of 331N above the Pinot Fault and below the Cabernet Fault. It appears that this zone is the offset along the Cabernet Fault from the high grade stockwork previously mined in the southern end of the GG2 orebody on the 950 and 1052 Levels.

At the time of modelling diamond drillholes intersecting the ore body mainly on a 20m x 30m intercept grid but in places the intercept grid was closed down to 15m x 15m. This close spaced drilling was considered necessary due to the large number of fault blocks encountered within the GG3 ore body. The current interpretation differs from that of the May 2007 model with the addition of 5 faults in the lower portion of the ore body below the New Fault which were identified due to the deeper drilling done during this period. Also further grade control drilling allowed for minor tweaking of many existing faults. This model concentrates on the entire GG3 ore body from -1050mRL to -1450mRL.

1.2 Alteration Styles

The alteration observed at Golden Gift 3 is dominated by classic volcanogenic alteration (chlorite/carbonate/silicate/sulphide) as seen elsewhere in the golden gift ore body.

1.3 Structural Geology

Significant faulting occurs within the geological model with. The model is broken up into fourteen different fault blocks with differing offset between each fault. The ore body is bound by the major fault structures of the Cabernet to the top, Durif to the north and the Wildcat Porphyry at Depth. Three major structures, Pinot, New Fault, and Kamado Fault have been interpreted. These fault dip shallowly to the north and act as natural boundaries within the orebody. Between the Pinot and Cabernet Faults in the southern end of the GG3 ore body are a series of smaller faults which still cause significant offset of ore blocks. These faults are the flat fault, fault A, steep fault and eastern steep fault. Apart from the Flat Fault, this dips shallowly to the north, the other faults in this block dip steeply to the NW. Between the Pinot and New Fault is the Saignee fault which produces another offset and mimics the Pinot fault in strike and dip. Between New Fault and the Wildcat Porphyry are the Tandoori Fault, Vindaloo Fault, Korma Fault, Kamado Fault and the Naan Fault. All of these faults dip shallowly to the North except for the Naan Fault which dips steeply to the NW.

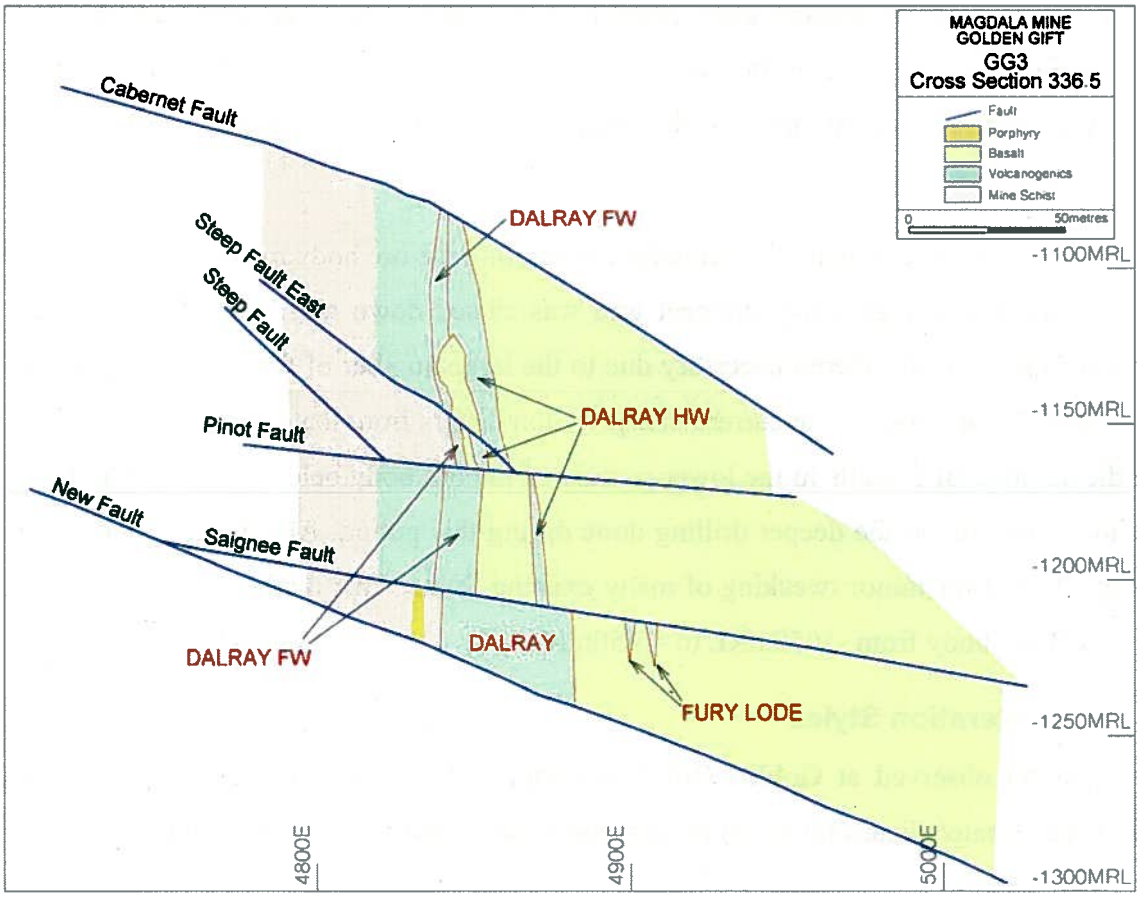

Figure 1 Cross section 336.5 illustrating the geology and faulting of GG3.

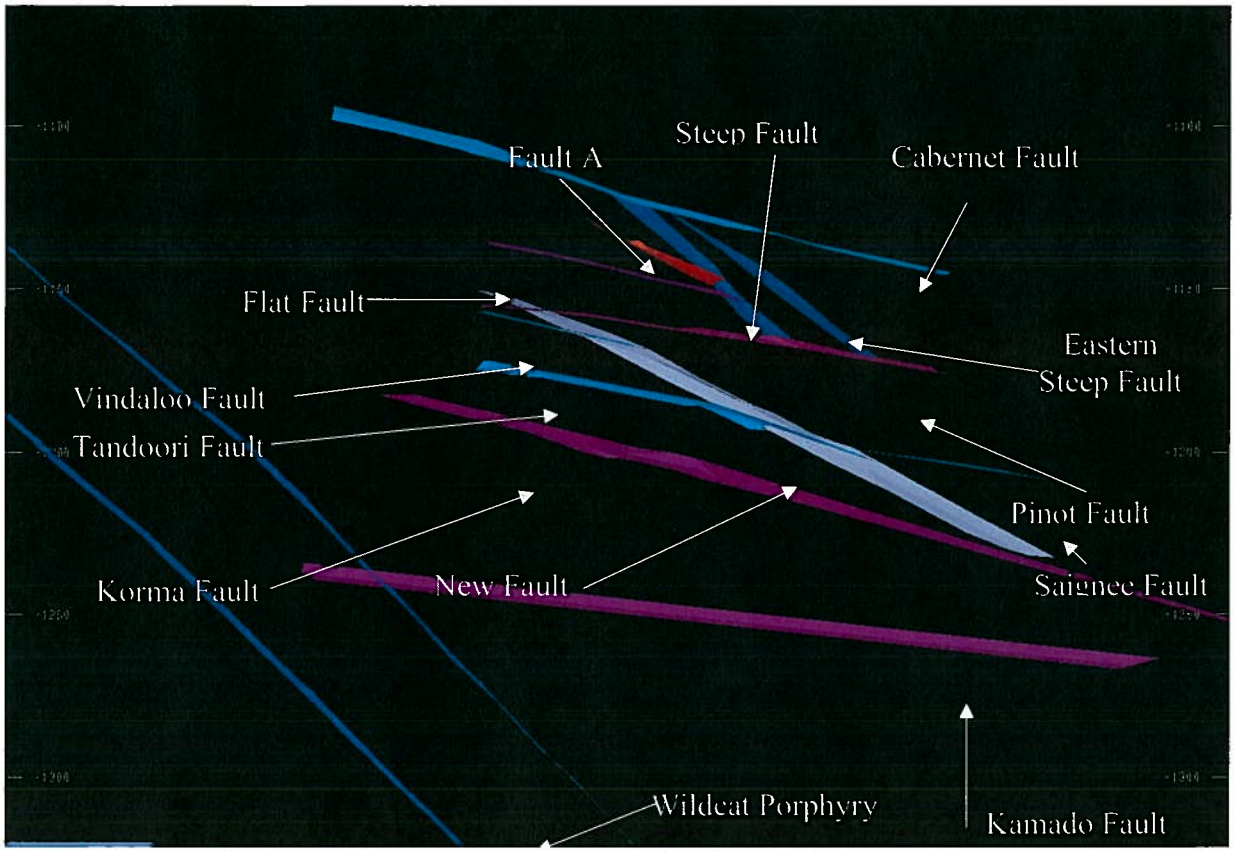

Figure 2 Cross section showing faulting within the GG3 ore body (Section 6870mN).

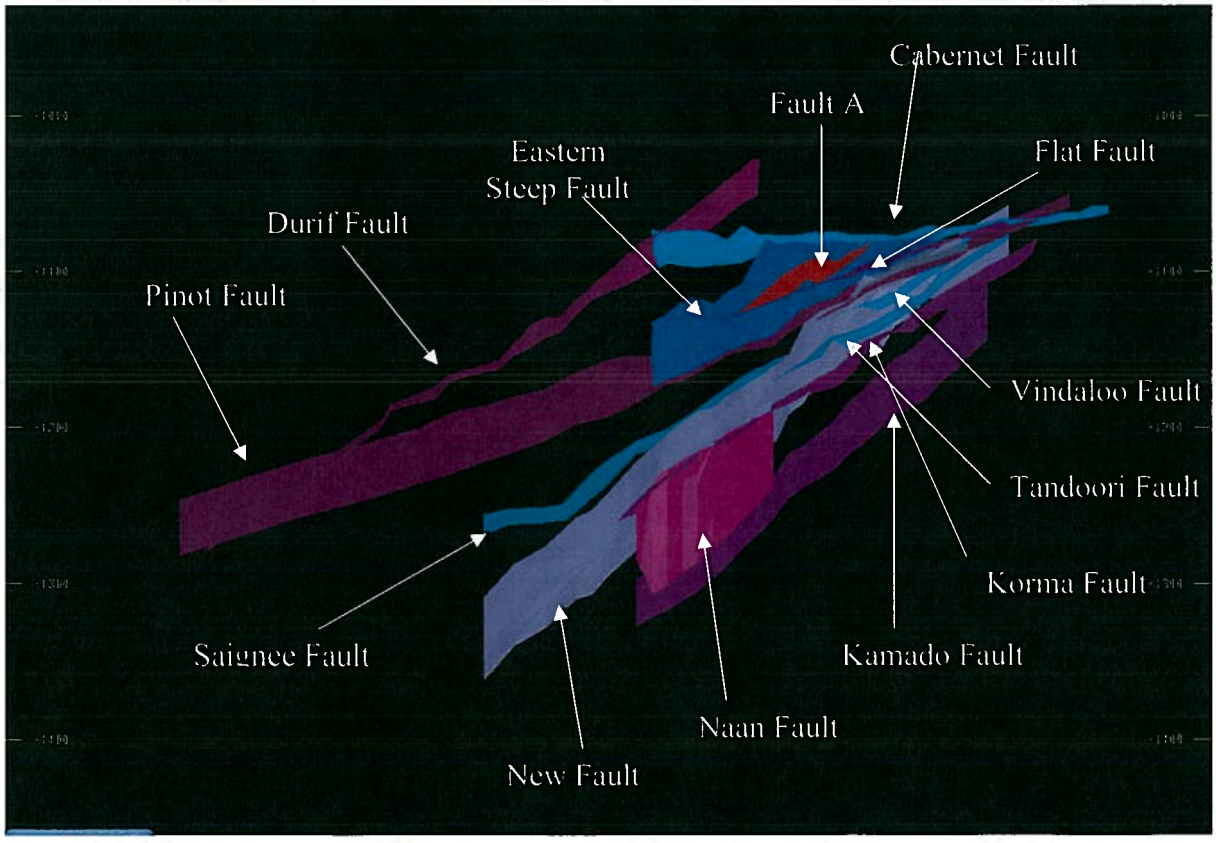

Figure 3 Long section toward 045° showing faulting within the GG3 ore body.

2 Drilling and Sampling Quality

All drill intercepts used in grade estimation were drilled from the 985 loading bay, 1004 crosscut, GG3 decline, 1032-331 crosscut, 950 stockpile, 1033 drill cuddy, 966 crosscut, 1032-331-80 Sth drive, 1103 drill cuddy, 1103 crosscut, 1116 crosscut, 1127, 1116-333-88 Sth drive, 1137, 1173 crosscut and the 1153. Holes used were drilled with both NQ2 and LTK60 equipment, with core diameters of 50.7mm and 44mm respectively.

The average drillhole spacing in the Indicated areas of the Model Inventory is 15mN x 15mRL and 20mN x 20mRL. The average drillhole spacing in the Inferred areas is 15mN x 15mRL and 20mN x 20mRL.

Additional diamond drilling data was used to update the Model Inventory.

Geological logging of diamond drill core is completed to the detail required for resource modelling using the standard SGM geological legend. Lithology, alteration, and structural characteristics known to affect mineralisation are collected. In addition, diamond holes are photographed.

Diamond drilling is assumed to provide 100% sample recovery unless recorded otherwise. The ore zones are generally competent and sample recovery is high. Core loss is recorded by the logging geologist and can be queried in the geological database. Regular rig audits and inspections are also used to assist with maintaining the high level of sample quality.

Drill core is prepared by cutting to intervals specified by the geologist. Sampling is within geological boundaries and intervals of one metre are desired. Predominantly the diamond core was submitted as full core for analysis. The metallurgical test work was performed on pulp samples.

2.1 QAQC Assessment

2.1.1 Quality Management Measures

Sampling of the diamond drill core is performed by samplers at the core farm once the core has been marked for sampling by the geologists.

Prior to February 2005 all drillcore was sent to Aminya laboratories in Ballarat for assaying. Ongoing QA/QC issues with Aminya led to a change in February whereby SGM Resource

Definition and Grade Control drillcore has since been sent to Genalysis in Kalgoorlie and Amdel in South Australia for assay. From August 2007 samples have been sent to ALS laboratories in Orange for assaying.

The general procedures for QA/QC are outlined below:

- Production diamond drill core is routinely Full Core sampled with one in five sent as half core.
- Routine assaying of diamond drill core samples is undertaken by fire assay (50 gram charges) with an AAS finish. Assays are reported to an LLD of 0.01ppm. For sample reporting below LLD, a value of 0.5xLLD is used in resource estimation.
- Sample QA/QC procedure incorporates routine check assays including repeats (re-assay or repeat assay), duplicate (second sample taken after pulverisation), splits (half coarse sample split, at ratio approx 1:10 samples), and standards.
- An allocation of recovery is applied to sample intervals during logging of the diamond core.
- SGM Standard reference material has been prepared by Ore Research of Melbourne, from ore samples from Magdala Mine.
- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sample ID created during logging procedure.
- Analysis of received standard values and repeat and duplicate information is used to monitor the performance of the assaying process.

2.1.2 Location of Data Points

Diamond drillhole collars are surveyed by the Survey Department and single shot down hole cameras have been used to survey the drillhole path. Electronic single shot instruments are now used in preference to the Eastman mechanical cameras. Magnetic intensity is recorded and used to assess validity of record. Deviation of Dip +/- 1.5° and azimuth +/- 3° is highlighted and assessed.

All data is stored within the "acQuire" Database Management System which is structured on a Data Model which applies rules of relational databases to mining and exploration data. The primary key is HoleID.

The drill holes with inaccurate hole collar locations have been excluded from the resource model.

2.1.3 Database Integrity

All logging of diamond drill core is undertaken using "Datcol" software, to a defined logging scheme, S:\Geology\QA_QC_Protocol, Geol Codes.xls. It is a barcode system, eliminating errors involved in keying. Logging is both quantitative and qualitative but in both cases, only valid codes or values can be recorded.

Additional validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth
- Checks are performed for overlapping samples

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (eg Minesight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted.

After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

3 Bulk Density

Bulk dry density of the volcanogenic zone averaged 2.85g/cm^3 and 2.97 g/cm^3 for the Dalray HW and FW lodes. A detailed analysis is outlined in Golden Gift 3 Resource Model, December 2005.

4 Metallurgical Test Work

5 Geological Modelling

The re-interpretation of GG3 in November 2007 was initially undertaken on paper 10m spaced cross sections and 15m spaced level plans. These were digitized and imported into Minesight. Solids were constructed by then linking strings to form wireframe solids. Solids were overlapped, and then clipped against each other to form common surfaces.

The following general geological domains have been interpreted. These domains have been split further into individual fault blocks for estimation purposes, due to offsets across fault boundaries, but the general geological characteristics remain the same throughout.

Domain	**Description**
Dalray HW	Sulphide dominated Lode is located with the waterloo position, typically bounded by a hangingwall basalt and basalt nose in the footwall.
Dalray FW	Sulphide domainated Lode is located on the open flank of the basalt nose/footwall basalt.
Fury	Sulphide dominated Lode located in the Waterloo position.
Volcanogenic	Variably mineralised volcanogenics
Porphyry	Porphyry cuts the lode position at depth
Hangingwall Basalt	Basalt
Basalt Nose	Narrow basalt flow, widening with depth
Footwall Basalt	Thin basalt flows in FW of Volcanogenics
Mineschist	Mineschist in footwall to Volcanogenics

The GG3 ore body has been offset by a series of faults which have been described earlier. Due to the magnitude of the apparent offset across these faults each geological domain has been split further into individual fault blocks. Within the GG3 ore body fourteen fault blocks have been classified, they are outline below. Refer to Figure 4.

Fault Block	**Bounding Faults**

Cabernet	Bound by the Cabernet Fault above, Durif Fault to the north, Pinot below and Eastern Steep Fault to the south.
Eastern Steep Fault	Bound by the Cabernet Fault above, Eastern Steep Fault to the north, Pinot Fault below and the Steep Fault to the south.
Fault A	Bound by the Cabernet Fault above, Steep Fault to the north and east, Flat Fault below and Fault A to the south.
Steep Fault	Bound by the Cabernet Fault above, Fault A to the north, Steep Fault to the east, and Flat Fault below and to the south.
Flat Fault	Bound by the Flat fault above, Steep Fault to the north and east, Pinot Fault and Pinot Splay below and the Cabernet Fault to the south.
Pinot Splay	Bound by the Pinot Splay Fault above and the Pinot Fault below.
Pinot	Bound by the Pinot Fault above, Wildcat Porphyry to the north, Saignee Fault beneath, Cabernet Fault to the southeast and the New Fault to the south and west.
Saignee	Bound by the Saignee Fault above, Wildcat Porphyry to the north and the New Fault to the south and west.
New Fault	Bound by the Cabernet and Pinot Faults above, New Fault to the north, Wildcat Porphyry below and is currently open to the south.
Vindaloo	Bound by the New Fault above and the Vindaloo fault below.
Tandoori	Bound by the Vindaloo Fault above, the New Fault to the north and the Tandoori fault below.
Korma	Bound by the Tandoori fault above, the New Fault to the north and the Korma fault below.
Kamado	Bound by the Korma fault above, the Wildcat Porphyry at depth and the Kamado fault below.

Naan — Bound by the Korma fault above, the New fault to the north, the Kamado fault below and the Wildcat Porphyry at depth.

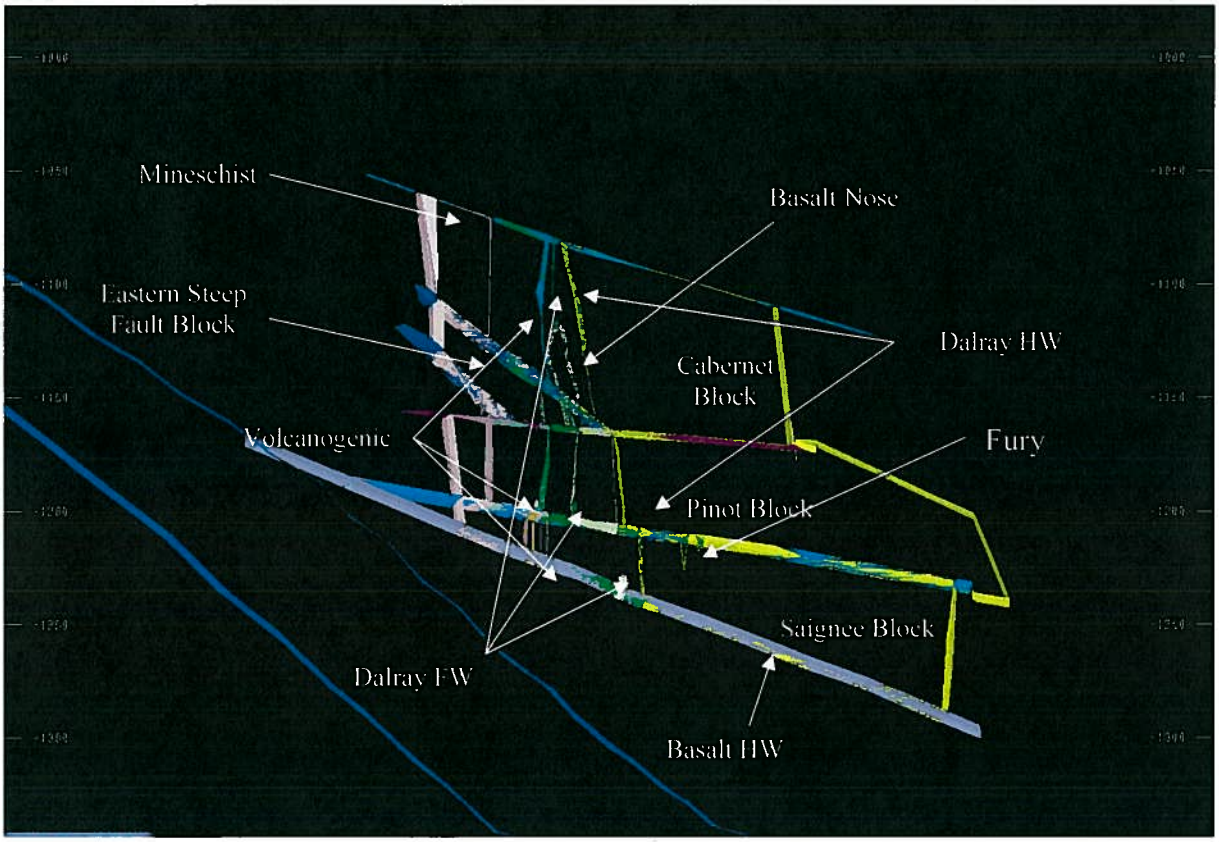

Figure 4 Cros ssetion at 6730mN highlighting the different fault block and geological domains.

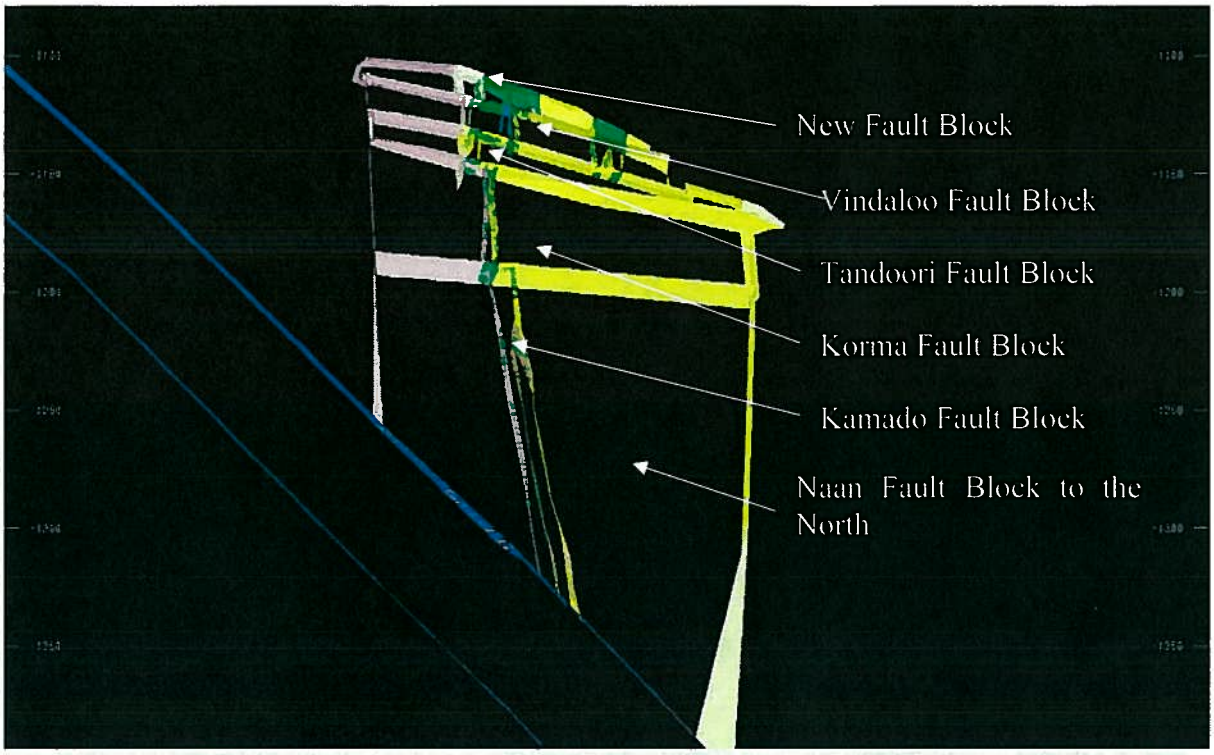

Figure 5 Cross section at 6600mN highlighting the additional blocks below the new fault in the November 2007 model.

5.1 Coding drillholes in mineralised zones

Drillholes, composites and block model utilize the same geological coding for domain. Code values are:

Fault Block	Solid	Code	Model Item	Shell Filename
Cabernet	Dalray HW	601	BAUD	601DalrayHW
Cabernet	Dalray FW	701	CAUD	701Dalray
Cabernet	Volc	501	AAUD	501volc
Eastern Steep Fault	Dalray HW	602	DAUD	602DalrayHW
Eastern Steep Fault	Dalray FW	702	EAUD	702DalrayFW
Fault A	Dalray FW	603	DAUD	603Dalray
Fault A	Volc	503	BAUD	503volc
Steep Fault	Dalray HW	604	BAUD	604DalrayHW
Steep Fault	Dalray FW	704	CAUD	704DalrayFW
Steep Fault	Volc	504	AAUD	504volc
Flat Fault	Dalray HW	605	DAUD	605DalrayHW
Flat Fault	Dalray FW	705	GAUD	705DalrayFW.
Flat Fault	Volc	505	FAUD	505volc
Pinot Splay	Dalray HW	609	AAUD	609DalrayHW
Pinot Splay	Dalray FW	709	EAUD	709DalrayFW
Pinot	Dalray HW	606	CAUD	606DalrayHW
Pinot	Dalray FW	706	HAUD	706DalrayFW
Pinot	Volc	506	BAUD	506volc
Saignee	Dalray HW/Fury	607	AAUD	607Fury
Saignee	Dalray	707	AAUD	707Dalray
Saignee	Volc	507	CAUD	507volc
New Fault	Dalray HW/Fury	608	BAUD	608Fury
New Fault	Dalray FW	708		708DalrayFW
New Fault	Dalray HW	508	DAUD	508DalrayHW
Vindaloo	Fury	610	HAUD	610Fury
Vindaloo	Dalray FW	710	AAUD	710DalrayFW
Vindaloo	Dalray HW	510	CAUD	510DalrayHW.
Tandoori	Waterloo	611	FAUD	611Waterloo
Tandoori	Volc	711	DAUD	711Volc
Korma	Dalray FW	712	BAUD	712Volc
Naan	Volc	713	HAUD	713Volc
Kamado	Volc	714	CAUD	714volc
General	Basalt	400	WAUD	basalt
General	Mineschist/Porphyry	1	WAUD	msporh

During coding of the data set a few intervals were manually adjusted. Below is a summary of the adjusted intervals:

Fault Block	Drillhole ID	Adjustment
All	MD4951	Assays not received back from lab thus coded blank
Fault A	MD4899	Wireframe not snapped to drillhole so coded to drillhole geology
Steep Fault	MD4909	Basalt in fault pinch out of Dalray HW, intervals coded blank

Steep Fault	MD4648	Basalt in fault pinch out of Dalray HW, intervals coded for basalt (400)
Flat Fault	MD4649, MD4647, MD4596	Survey errors, geology down hole does match geology observed in 1116-333-88Sth drive. Holes coded to interp or blank where down hole geology conflicted with interp
Pinot	MD4820	Down hole geology conflicts with the down hole geology of MD4774. Deemed only a small conflict. Both hole coded for down hole geology.
Saignee	MD4089	Dalray HW/Fury lode interpreted position doesn't match drillhole. Hole coded blank for this estimate. Currently drilling Dalray HW/Fury lode to update estimate at a later date.
Saignee	MD4131	Down hole geology doesn't match interpreted Dalray lode in a fault pinch out position. Intervals coded blank.
Saignee	MD2880AW1	Wireframes not snapped to drillhole. Intervals coded to hole geology rather than interpretation.
New Fault	MD4299	Down hole geology doesn't match interpretation. Intervals coded to down hole geology. (Survey concerns)
New Fault	MD4783	Down hole geology doesn't match interpretation. Intervals where there is a difference between drillhole and interpretation geology coded blank. (Survey concerns)
New Fault	MD4921	Wireframes not snapped to drillhole. Intervals coded to drillhole geology.
New Fault	MD4457	Wireframes not snapped to drillhole. Intervals coded to drillhole geology.
New Fault	MD2880AW1	Drill hole on edge of northern extent of the interpretation. Has been coded to be included in the estimate.
New Fault	MD4981, MD4982, MD4983, MD4984, MD4985, MD4986, MD4987	These drills holes have only recently been drilled targeting Dalray HW/Fury from the 1116-333-88Sth. Their collar position has yet to be picked up by survey and no assays have been received. They have all been coded blank for this estimate.

5.2 Sub-Domaining

No further sub-domaining of the major mineralised surfaces has been carried out. But areas which may require further work and sub-domaining have been identified. These have been grouped according to fault blocks.

Eastern Steep Fault Block:

- The Dalray FW domain has potential to be split out from a lower grade area which could be added to the volcanogenic domain, but at this stage there is not enough data to make a definitive split

Fault A Block:

- The Dalray FW domain has potential to be split out from a lower grade volcanogenic domain which is not present at this time. Once again, at this stage there is not enough data to make a definitive split

Saignee Fault Block:

- The Dalray FW domain has potential to be split out from a lower grade area which could be added to the volcanogenic domain, but at this stage there is not enough data to make a definitive split
- Need to resolve the geometry of the Fury solid. Drilling is currently underway to resolve this issue.

New Fault Block:

- Need to resolve the geometry of the Fury solid. Drilling is currently underway to resolve this issue.
- South of section 6560mN data becomes limited and a tweak of the geometry of the domains may be required.

5.3 Domaining and Descriptive Statistics

As described previously based on the current drillhole information, the geological model is divided into fourteen fault blocks. As a general practice, the raw sample results have been composited to 2.0m intervals and a top-cut applied, if deemed necessary, after compositing.

A listing of descriptive statistics is provided in **Error! Reference source not found.**. The composite sets selected for the resource interpolation are highlighted in yellow. Histograms of the 2.0m composite lengths and the gold assay values are shown in Figure 6 to Figure 37. A

general description of the domains and the top-cut applied is outlined in **Error! Reference source not found.**.

PERSEVERANCE CORPORTATION LIMITED								
GG3 RESOURCE MODEL								
Date:	Nov-2007							
DOMAIN	Data Type(s)	Number of Samples	Minimum g/t	Maximum g/t	Mean g/t	Std Dev g/t	Variance g/t2	Coeff of Var
Domain 601								
Au Raw	DD	326	0.005	111	3.498	11.3871	129.67	3.26
Au 2.0m Composites	DD	151	0.005	88.83	3.498	8.9092	79.37	2.55
Au 2.0m Composites TC 40	DD	151	0.005	40	3.1673	6.2326	38.85	1.97
Length Raw	DD	312	0.15	1.7	0.91	0.354		
Length 2.0m Composites	DD	151	0.5	2.95	1.96	0.32		
Domain 602								
Au Raw	DD	6	0.03	8.72	2.886	4.1313	17.07	1.43
Au 2.0m Composites	DD	3	0.17	6.68	2.8857	3.2645	10.66	1.13
Length Raw	DD	6	0.6	1.5	0.9	0.3114		
Length 2.0m Composites	DD	3	1.4	2	1.8	0.28		
Domain 603								
Au Raw	DD	38	0.01	7.68	1.522	2.3178	5.37	1.52
Au 2.0m Composites	DD	19	0.03	7.49	1.5221	2.0359	4.14	1.34
Length Raw	DD	31	0.3	1.5	0.96	0.2688		
Length 2.0m Composites	DD	19	0.4	2.45	1.91	0.44		
Domain 604								
Au Raw	DD	103	0.005	112	7.921	14.5051	210.40	1.83
Au 2.0m Composites	DD	44	0.02	46.22	7.9209	10.7193	114.90	1.35
Au 2.0m Composites TC 35	DD	44	0.02	35	7.4757	9.2668	85.87	1.24
Au 2.0m Composites TC 25	DD	44	0.02	25	6.8638	7.6085	57.89	1.11
Length Raw	DD	84	0.15	3	0.901	0.388		
Length 2.0m Composites	DD	44	1.15	2.55	1.95	0.23		
Domain 605								
Au Raw	DD	61	0.005	55.5	9.029	13.9357	194.20	1.54
Au 2.0m Composites	DD	27	0.13	47.85	9.0292	10.5403	111.10	1.17
Au 2.0m Composites TC 25 - 1 sample	DD	27	0.133	25	8.1438	7.8021	60.87	0.96
Length Raw	DD	55	0.15	4.5	0.982	0.5916		
Length 2.0m Composites	DD	27	1	2.95	2.01	0.39		
Domain 606								
Au Raw	DD	272	0.005	87.8	4.461	8.3636	69.95	1.87
Au 2.0m Composites	DD	120	0	32.03	4.461	5.9002	34.81	1.32
Length Raw	DD	275	0.15	7	0.915	0.5373		
Length 2.0m Composites	DD	120	0.3	2.9	2	0.34		
Domain 607								
Au Raw	DD	30	0.005	28.1	2.349	5.3368	28.48	2.27
Au 2.0m Composites	DD	15	0	16.71	2.3493	4.1648	17.35	1.77
Length Raw	DD	36	0.15	2.05	0.939	0.3864		
Length 2.0m Composites	DD	15	0.95	2.9	1.85	0.5		
Domain 608								
Au Raw	DD	38	0.1	158	21.185	32.8274	1,077.64	1.55
Au 2.0m Composites	DD	20	1.11	117	21.1848	30.1539	909.26	1.42
Au 2.0m Composites 60 - 2 samples	DD	20	1.11	60	17.1715	19.4785	379.41	1.13
Au 2.0m Composites 30 - 3 samples	DD	20	1.11	30	12.8786	10.7545	115.66	0.84
Length Raw	DD	38	0.35	1.6	0.988	0.291		
Length 2.0m Composites	DD	20	1	2.4	1.88	0.31		

Domain 609								
Au Raw	DD	14	0.02	52.8	18.673	20.304	412.25	1.09
Au 2.0m Composites	DD	7	0.82	46.19	18.626	17.54	307.65	0.94
Au 2.0m Composites 20	DD	7	0.82	20	11.6783	9.4389	89.09	0.81
Length Raw	DD	14	0.6	1.4	0.986	0.2332		
Length 2.0m Composites	DD	7	1.5	2.3	1.97	0.22		
Domain 610								
Au Raw	DD	76	0.005	121	20.263	22.603	510.90	1.12
Au 2.0m Composites	DD	33	0	81.47	20.2632	18.2699	333.79	0.90
Au 2.0m Composites TC 50 - 1 samples	DD	33	0	50	19.3341	15.721	247.15	0.81
Length Raw	DD	77	0.15	1.5	0.882	0.399		
Length 2.0m Composites	DD	33	1.05	2.9	2.05	0.4		
Domain 611								
Au Raw	DD	36	0.01	33.4	8.329	7.3534	54.07	0.88
Au 2.0m Composites	DD	18	1.33	23.52	8.3295	6.0024	36.03	0.72
Length Raw	DD	38	0.15	3.3	0.988	0.5532		
Length 2.0m Composites	DD	18	1.1	2.95	1.88	0.42		
Domain 701								
Au Raw	DD	286	0.005	51.7	4.828	6.5306	42.65	1.35
Au 2.0m Composites	DD	136	0.01	26.59	4.8279	4.5075	20.32	0.93
Length Raw	DD	259	0.25	3	0.961	0.3298		
Length 2 0m Composites	DD	136	0.9	2.95	1.97	0.3		
Domain 702								
Au Raw	DD	109	0.005	19.5	1.201	2.913	8.49	2.43
Au 2.0m Composites	DD	50	0.005	8.79	1.201	1.9821	3.93	1.65
Au 2.0m Composites TC 6 - 2 samples	DD	50	0.01	6	1.1211	1.7211	2.96	1.54
Length Raw	DD	103	0.15	4.9	0.97	0.5089		
Length 2.0m Composites	DD	50	0.3	2.9	2	0.36		
Domain 704								
Au Raw	DD	126	0.005	75	9.046	14.0994	198.79	1.56
Au 2.0m Composites	DD	55	0	52.98	9.0456	11.8737	140.98	1.31
Au 2.0m Composites	DD	55	0	35	8.3249	9.5321	90.86	1.15
Au 2.0m Composites	DD	55	0	25	7.6894	7.9519	63.23	1.03
Length Raw	DD	97	0.15	1.5	0.866	0.3088		
Length 2.0m Composites	DD	55	0.35	2.85	1.94	0.39		
Domain 705								
Au Raw	DD	194	0.005	49.2	4.312	6.7118	45.05	1.56
Au 2.0m Composites	DD	81	0.005	31.04	4.3119	5.1638	26.66	1.20
Au 2.0m Composites	DD	81	0.005	20	4.1405	4.4261	19.59	1.07
Length Raw	DD	158	0.15	0.195	0.846	0.3431		
Length 2.0m Composites	DD	81	0.6	2.9	1.97	0.33		
Domain 706								
Au Raw	DD	695	0.005	63.7	2.64	5.0343	25.34	1.91
Au 2.0m Composites	DD	327	0.008	35.61	2.6396	3.7971	14.42	1.44
Length Raw	DD	639	0.1	3	0.93	0.3365		
Length 2.0m Composites	DD	327	1	2.9	1.97	0.22		
Domain 707								
Au Raw	DD	797	0.005	59.1	2.917	4.9574	24.58	1.70
Au 2.0m Composites	DD	375	0.005	30	2.917	3.79	14.36	1.30
Length Raw	DD	750	0.15	0.994	7.7	0.5358		
Length 2.0m Composites	DD	375	0.4	2.95	1.97	0.23		

Domain 708								
Au Raw	DD	28	0.02	17.5	2.487	3.7296	13.91	1.50
Au 2.0m Composites	DD	13	0.06	6.24	2.487	2.2289	4.97	0.90
Length Raw	DD	25	0.35	1.45	0.938	0.2924		
Length 2.0m Composites	DD	13	1.35	2	1.91	0.21		
Domain 709								
Au Raw	DD	31	0.02	18.1	5.208	4.9545	24.55	0.95
Au 2.0m Composites	DD	13	0.02	12.19	5.208	3.9322	15.46	0.76
Length Raw	DD	23	0.25	0.85	1.45	0.3763		
Length 2.0m Composites	DD	13	1	2.75	1.91	0.4		
Domain 710								
Au Raw	DD	91	0.005	20.4	2.967	4.3291	18.74	1.46
Au 2.0m Composites	DD	45	0.01	12.58	2.967	3.5989	12.95	1.21
Length Raw	DD	72	0.25	2	0.94	0.3103		
Length 2.0m Composites	DD	45	0.25	2	1.86	0.39		
Domain 711								
Au Raw	DD	62	0.005	65.5	6.867	13.248	175.51	1.93
Au 2.0m Composites	DD	31	0.005	43.106	6.8666	9.8601	97.22	1.44
Length Raw	DD	61	0.2	6.2	1.055	0.7469		
Length 2.0m Composites	DD	31	0.05	2.9	1.91	0.55		
Domain 712								
Au Raw	DD	259	0.005	55.93	4.214	6.5876	43.40	1.56
Au 2.0m Composites	DD	119	0	23.72	4.214	4.5777	20.96	1.09
Length Raw	DD	246	0.1	3.85	0.903	0.4481		
Length 2.0m Composites	DD	119	0.15	2.9	1.89	0.4		
Domain 713								
Au Raw	DD	72	0.01	22.6	2.69	4.0306	16.25	1.50
Au 2.0m Composites	DD	41	0.01	11.44	2.69	2.7346	7.48	1.02
Length Raw	DD	76	0.15	2.25	1.105	0.4593		
Length 2.0m Composites	DD	41	0.25	2.8	1.87	0.39		
Domain 714								
Au Raw	DD	48	0.005	6.23	0.424	1.174	1.38	2.77
Au 2.0m Composites	DD	22	0	3.09	0.4244	0.8167	0.67	1.92
Length Raw	DD	25	0.4	1.65	1.104	0.2685		
Length 2.0m Composites	DD	22	1.05	2.75	1.94	0.29		

Table 1. Descriptive statistics of the 2.0m composites per domain.

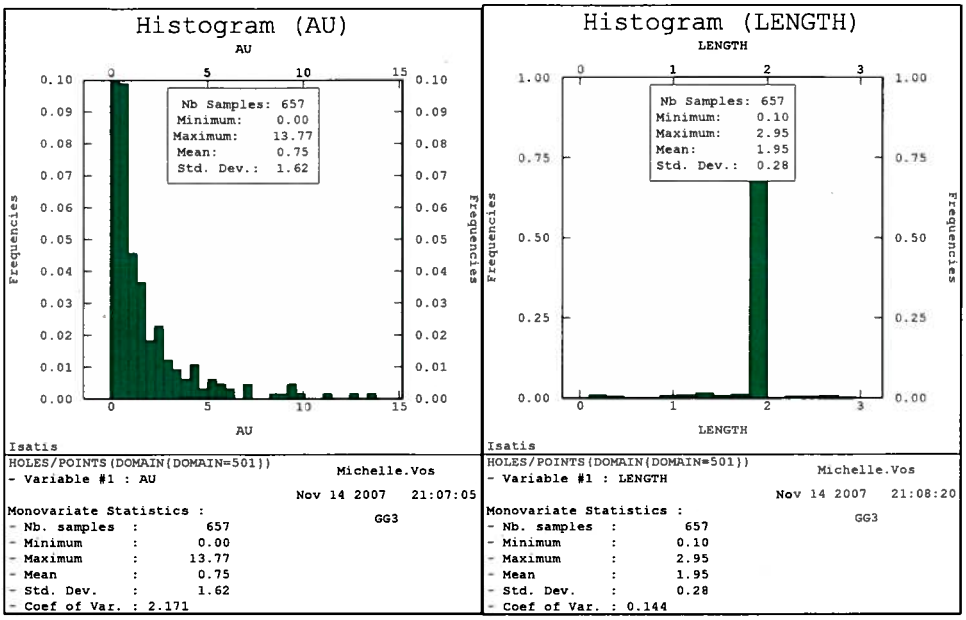

Figure 6 Histograms of Au (left) and length (right) in 2.0m composites of Domain 501.

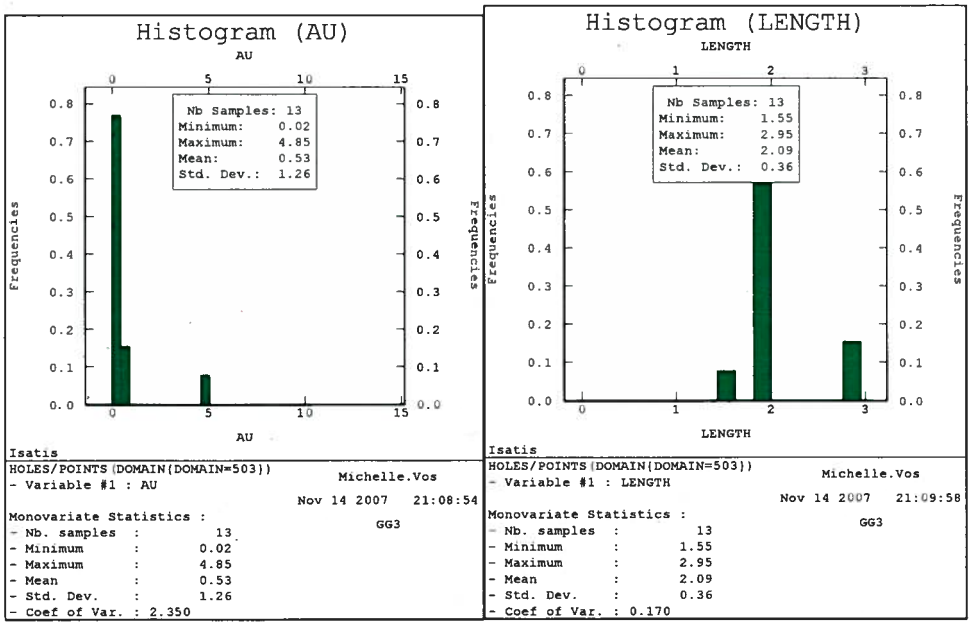

Figure 7. Histograms of Au (left) and length (right) in 2.0m composites of Domain 503.

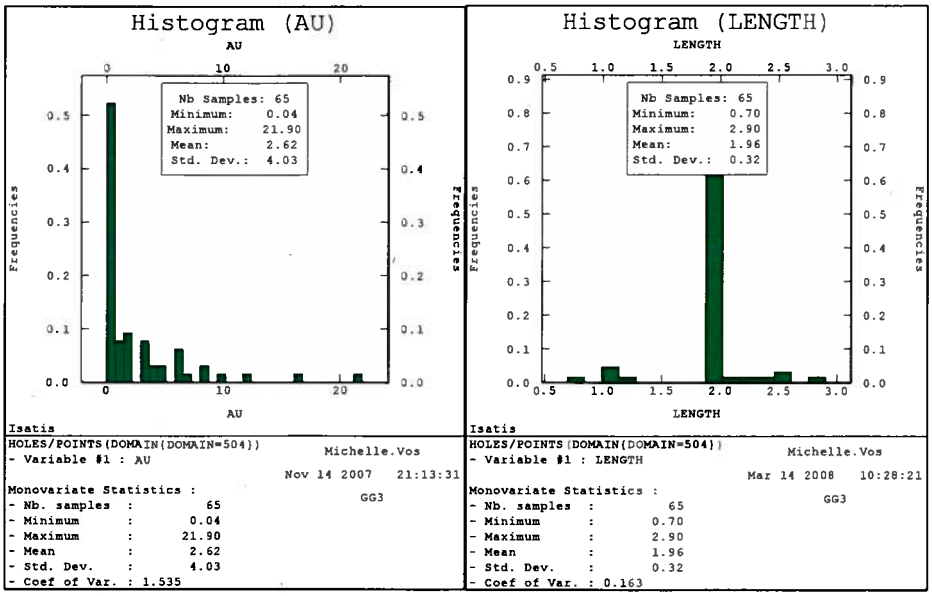

Figure 8. Histograms of Au (left) and length (right) in 2.0m composites of Domain 504.

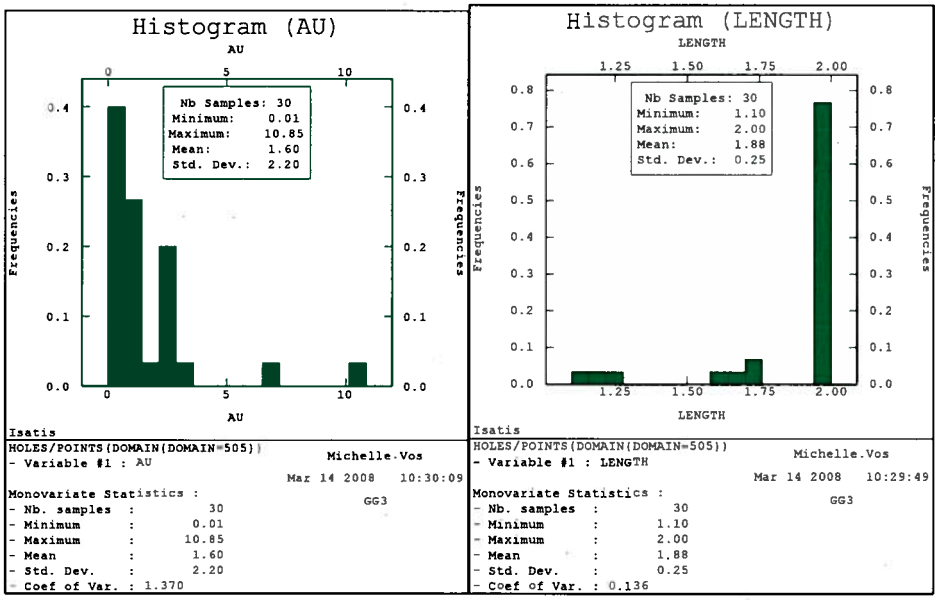

Figure 9 Histograms of Au (left) and length (right) in 2.0m composites of Domain 505.

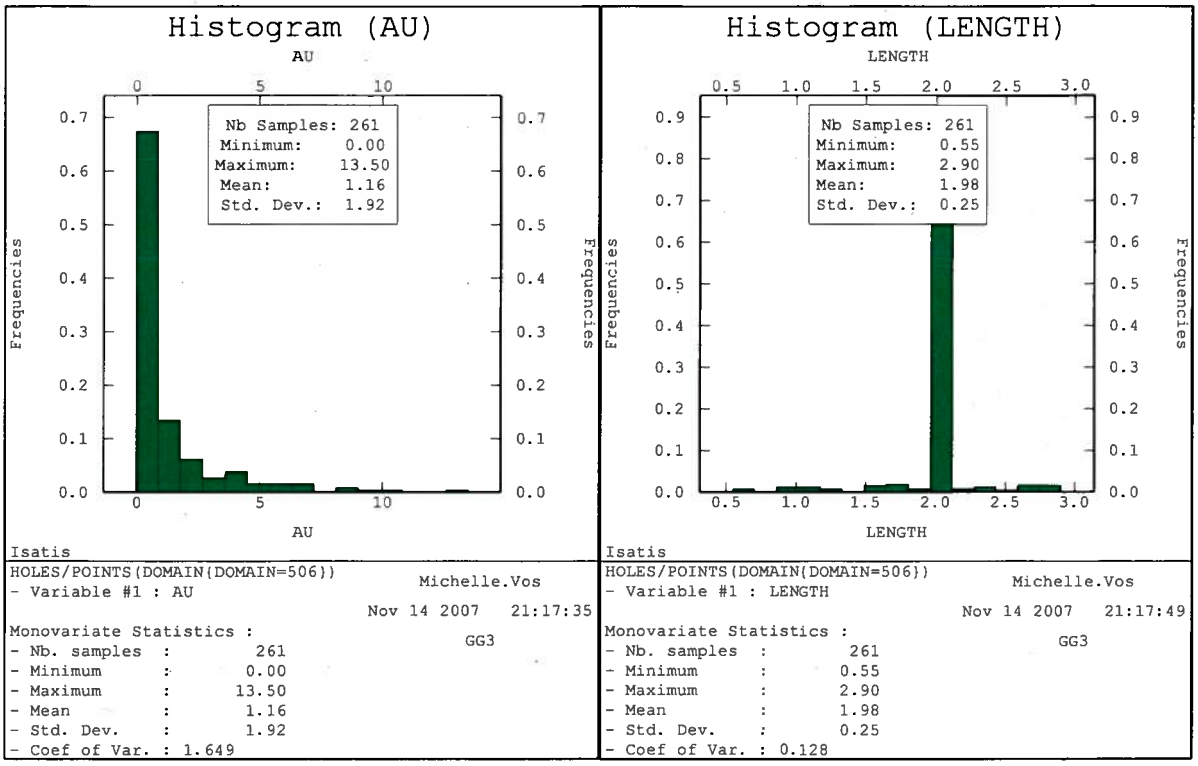

Figure 10 Histograms of Au (left) and length (right) in 2.0m composites of Domain 506.

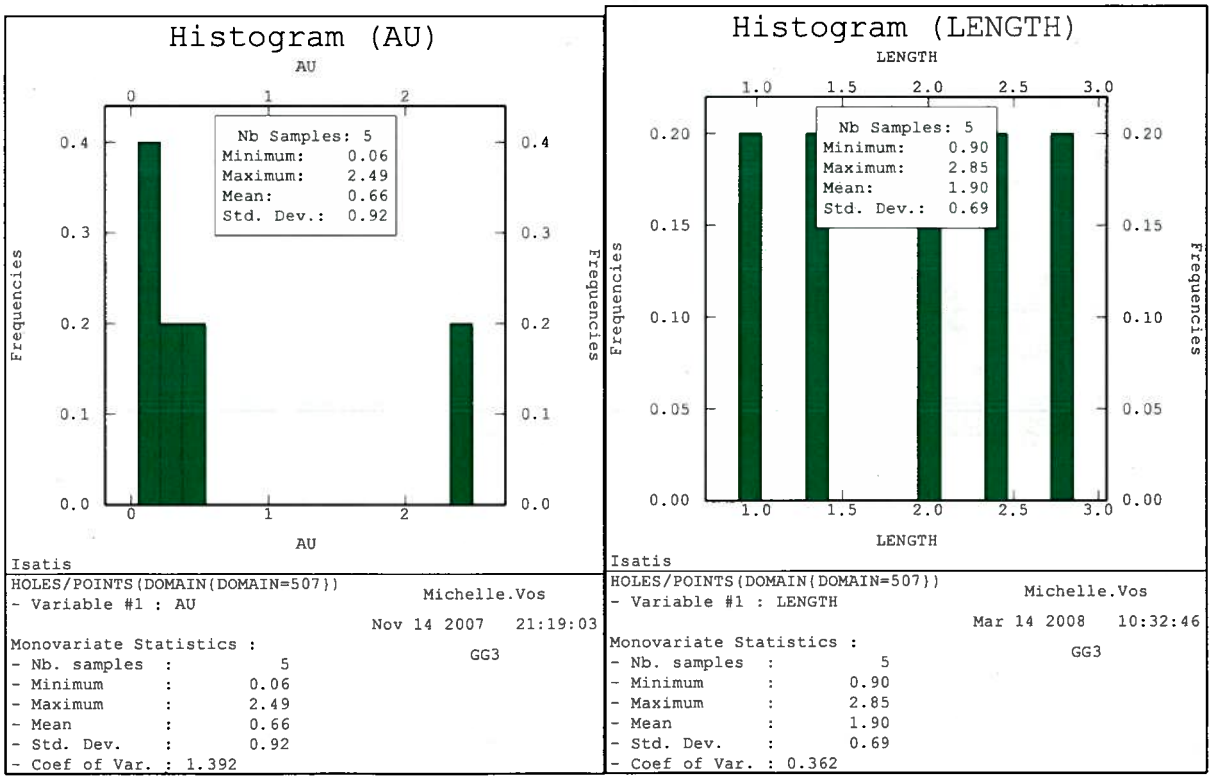

Figure 11 Histograms of Au (left) and length (right) in 2.0m composites of Domain 507.

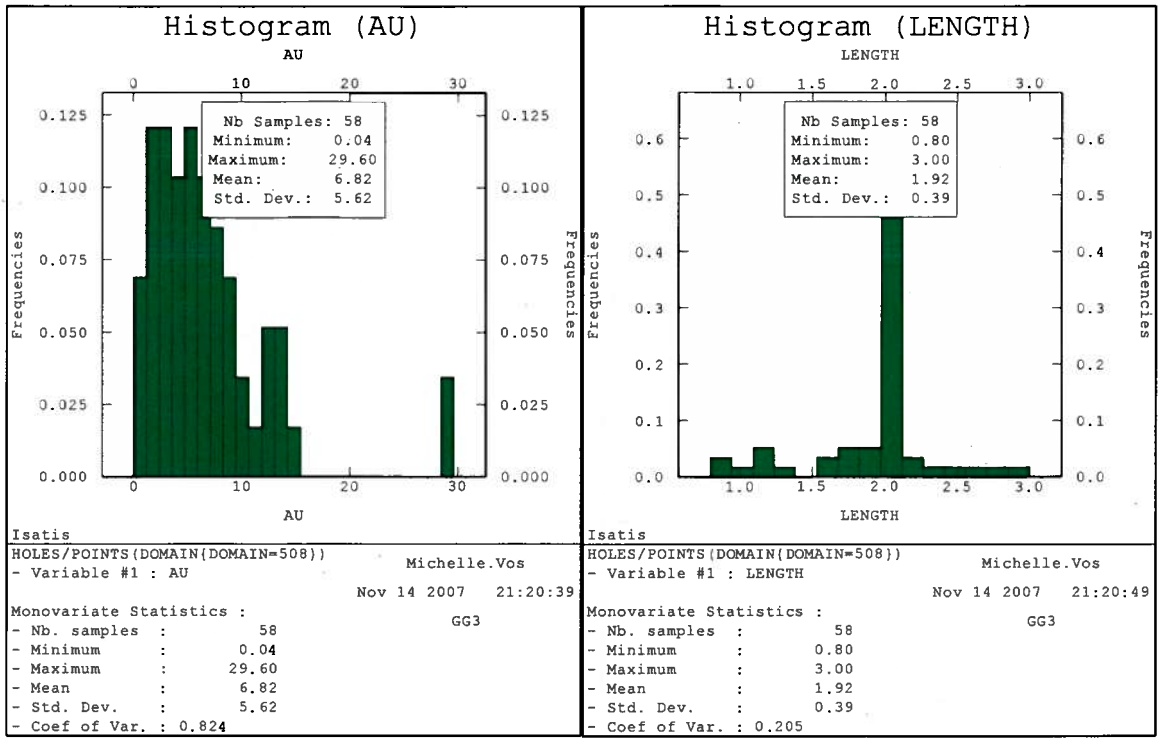

Figure 12 Histograms of Au (left) and length (right) in 2.0m composites of Domain 508.

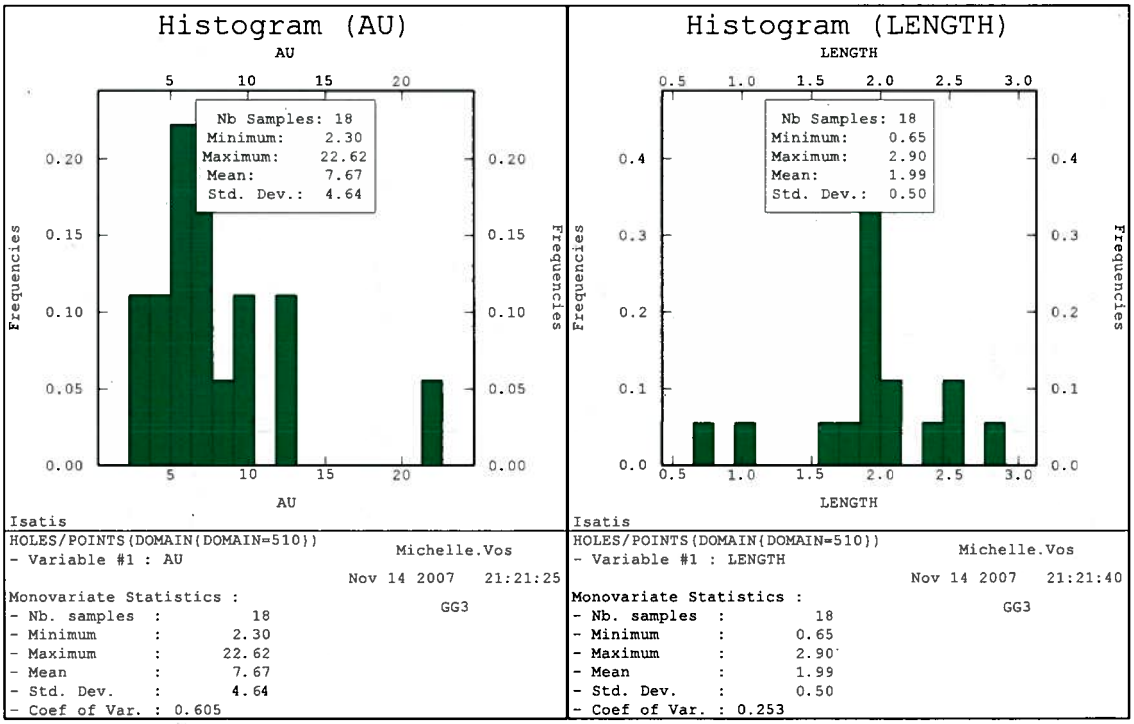

Figure 13 Histograms of Au (left) and length (right) in 2.0m composites of Domain 510.

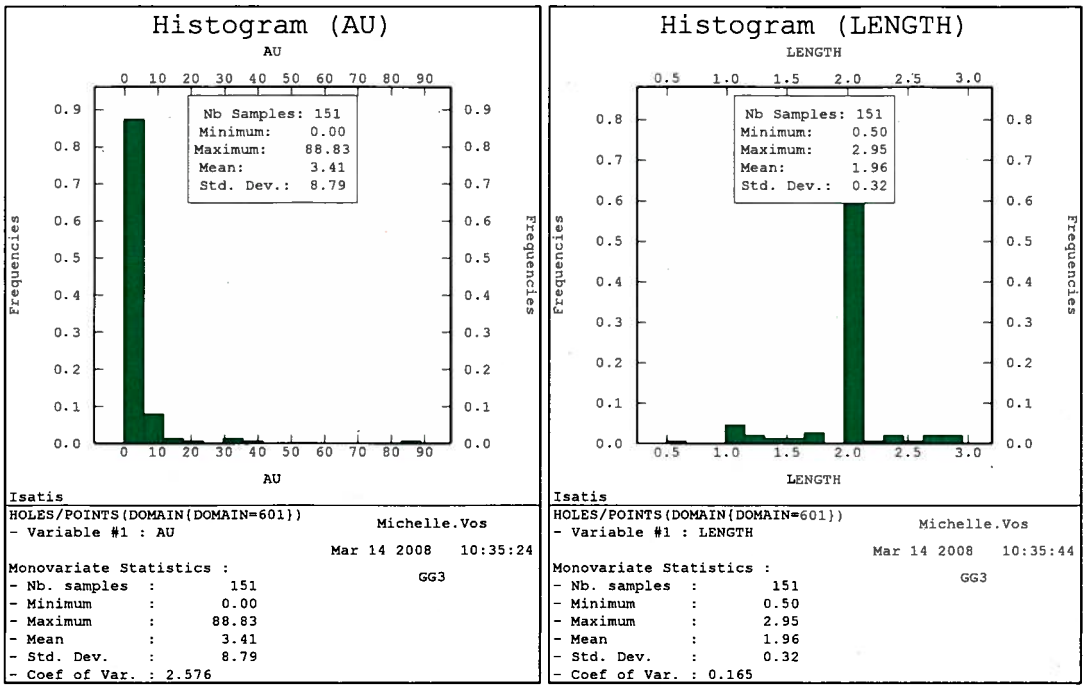

Figure 14 Figure Histograms of Au (left) and length (right) in 2.0m composites of Domain 601.

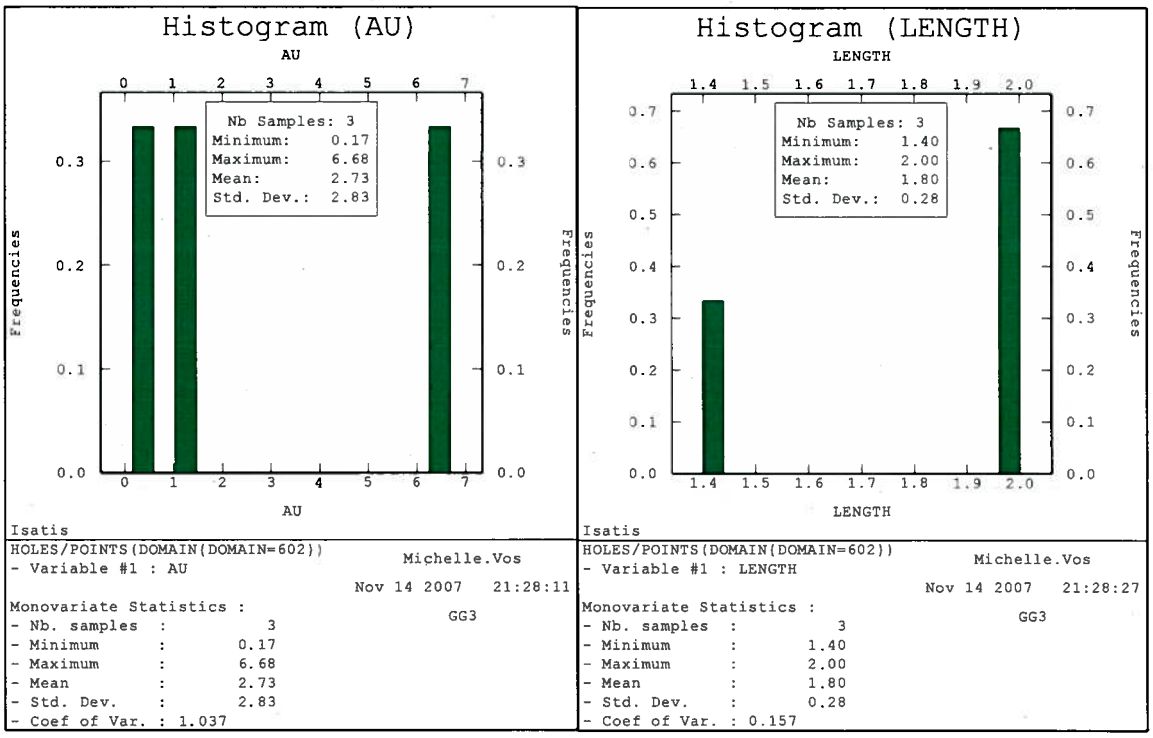

Figure 15 Histograms of Au (left) and length (right) in 2.0m composites of Domain 602.

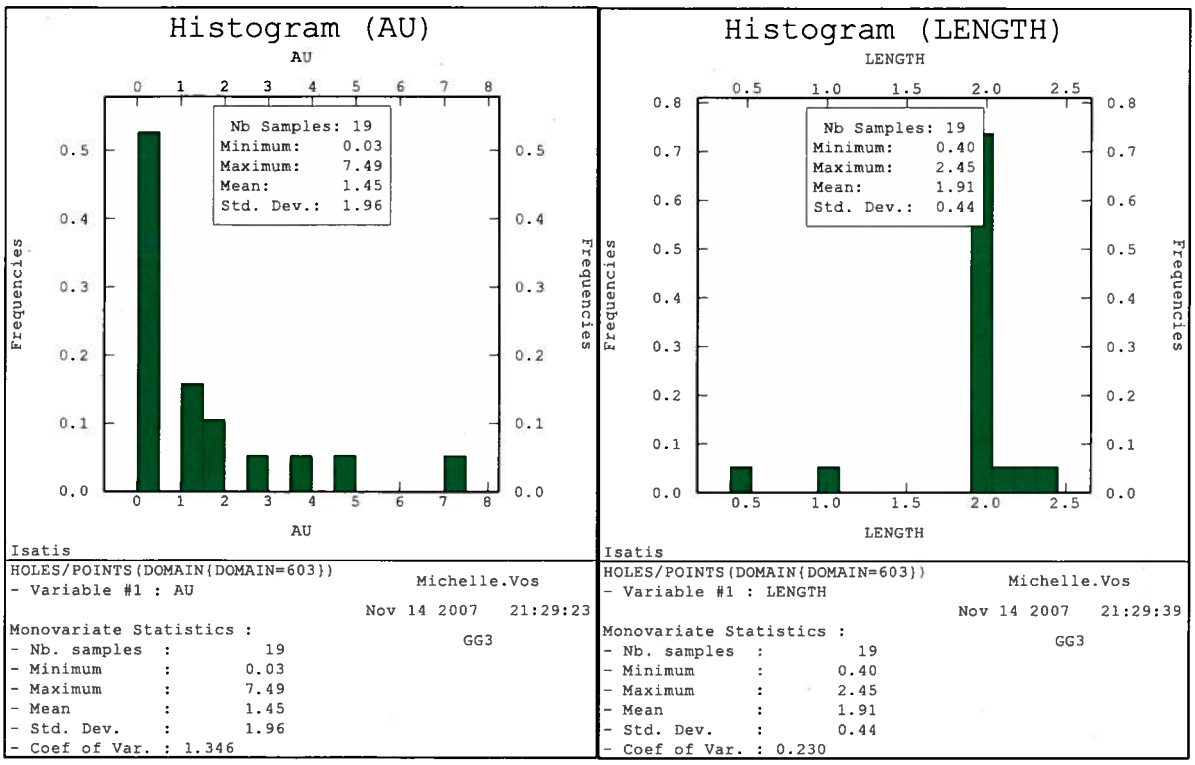

Figure 16 Histograms of Au (left) and length (right) in 2.0m composites of Domain 603.

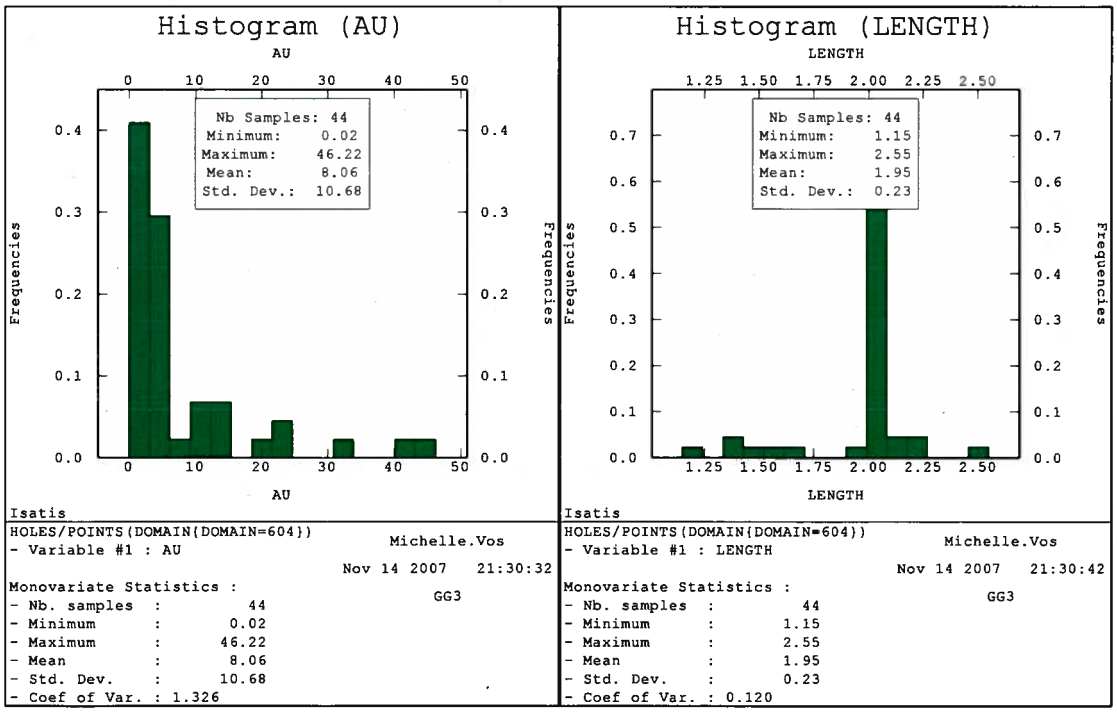

Figure 17 Histograms of Au (left) and length (right) in 2.0m composites of Domain 604.

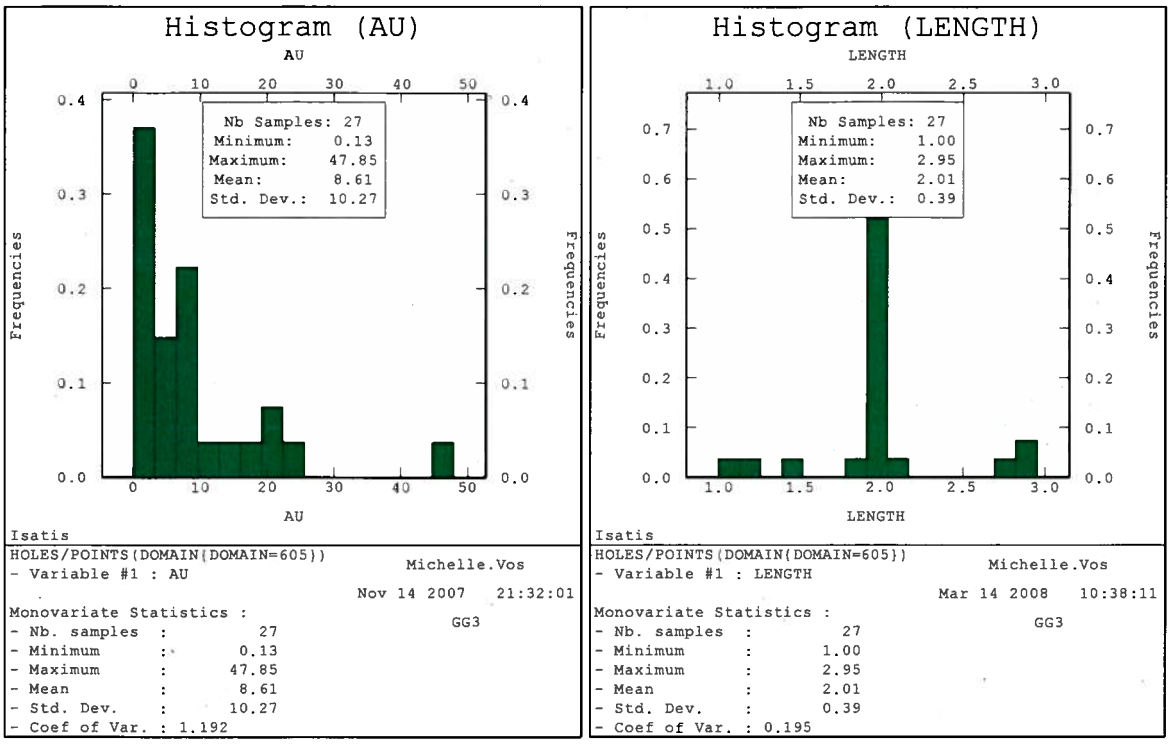

Figure 18 Histograms of Au (left) and length (right) in 2.0m composites of Domain 605.

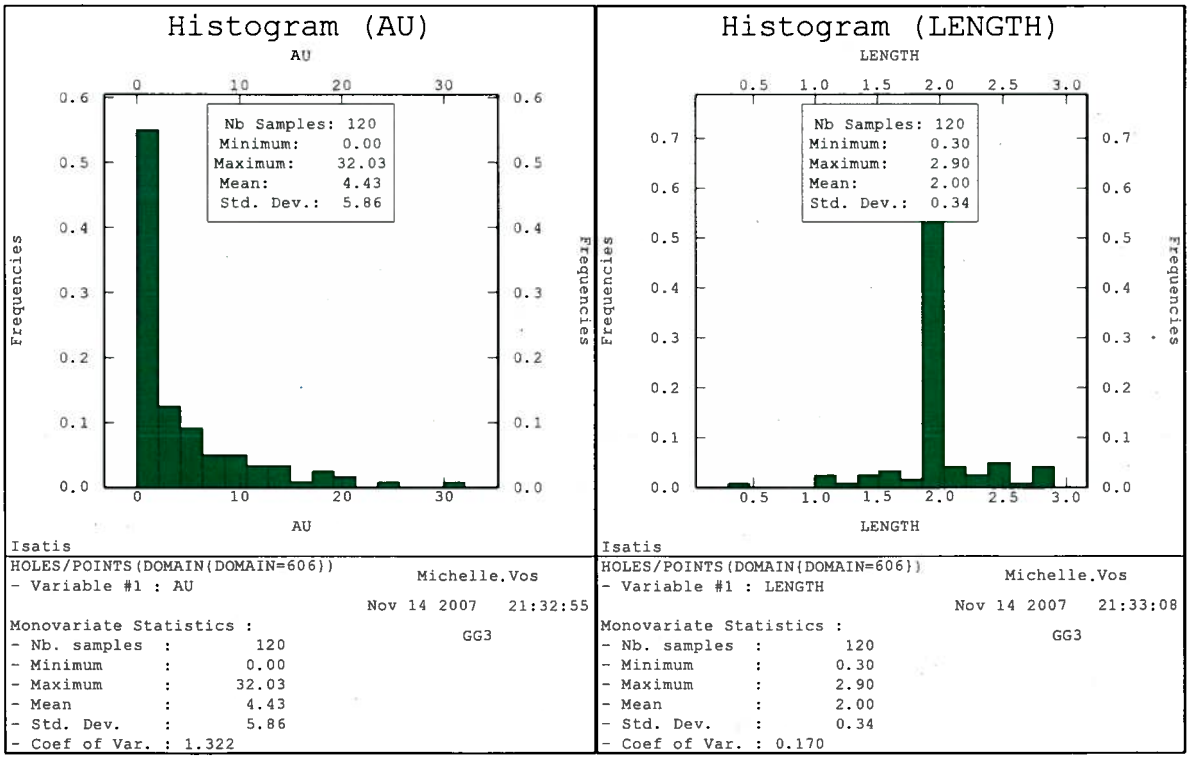

Figure 19 Histograms of Au (left) and length (right) in 2.0m composites of Domain 606.

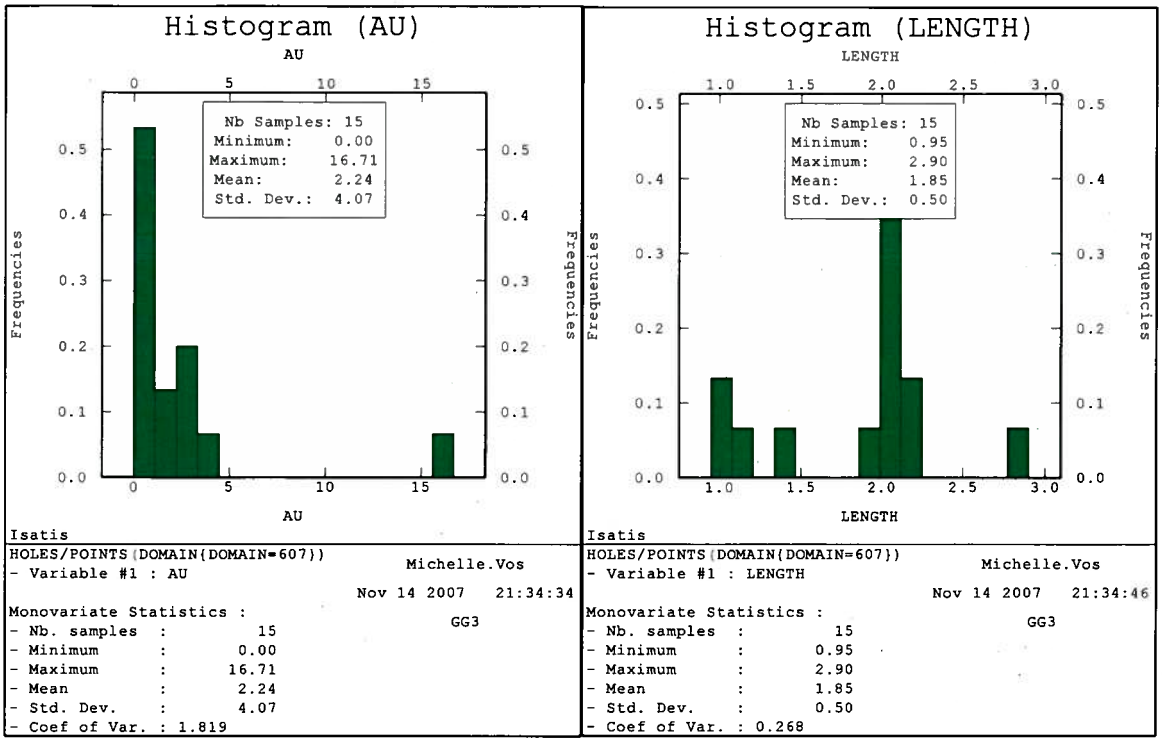

Figure 20 Histograms of Au (left) and length (right) in 2.0m composites of Domain 607.

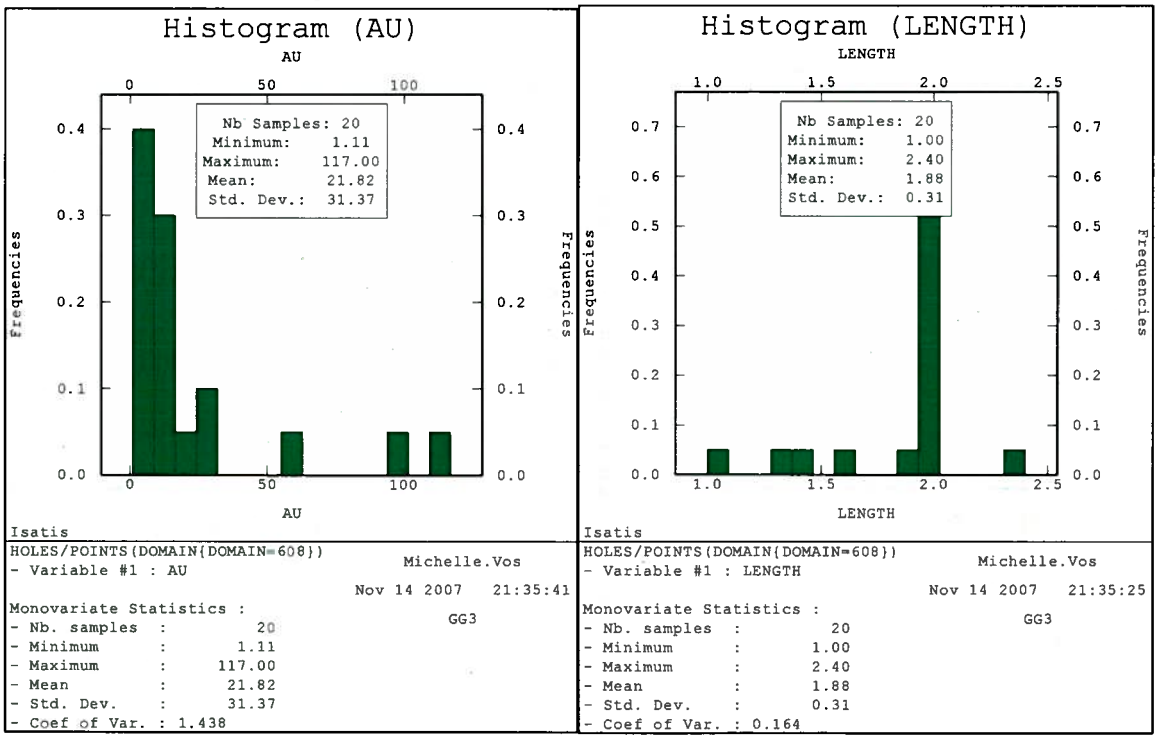

Figure 21 Histograms of Au (left) and length (right) in 2.0m composites of Domain 608.

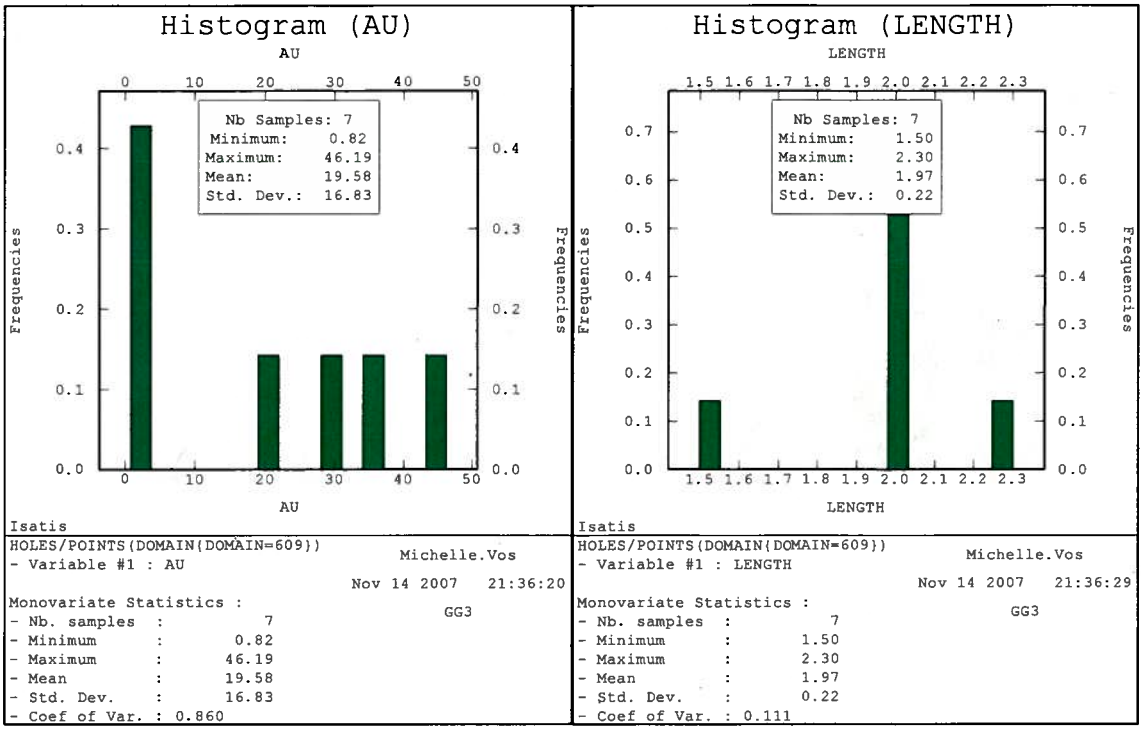

Figure 22 Histograms of Au (left) and length (right) in 2.0m composites of Domain 609.

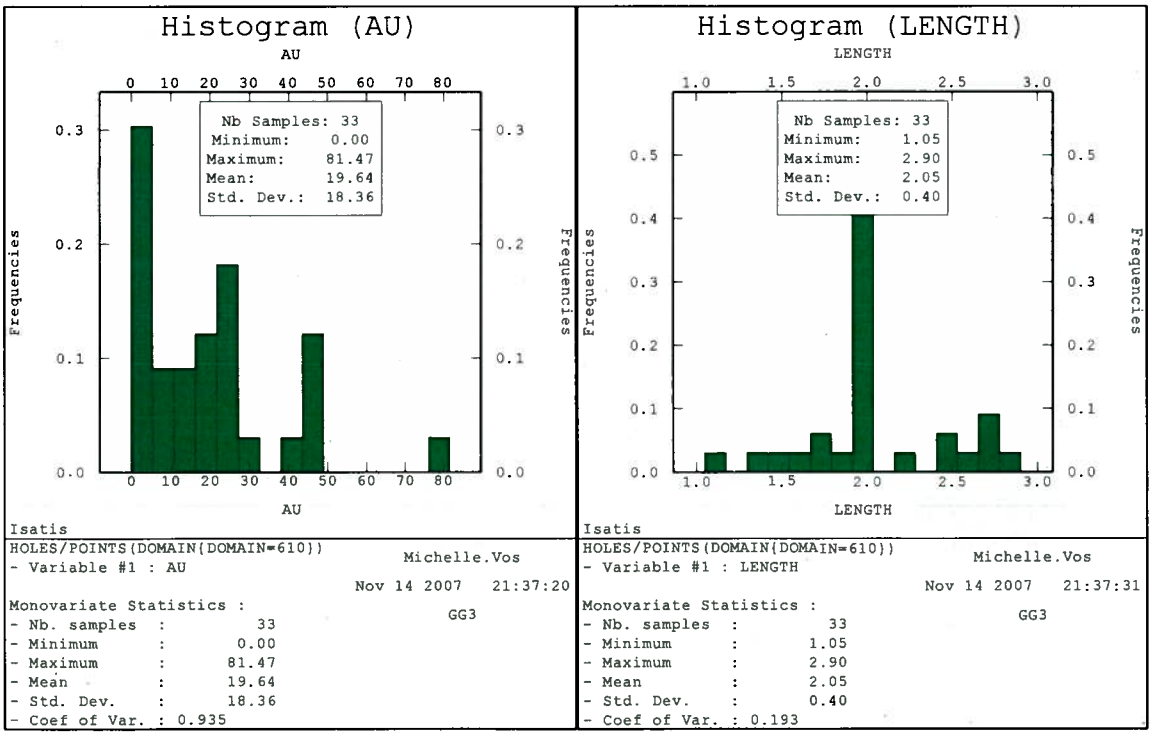

Figure 23 Histograms of Au (left) and length (right) in 2.0m composites of Domain 610.

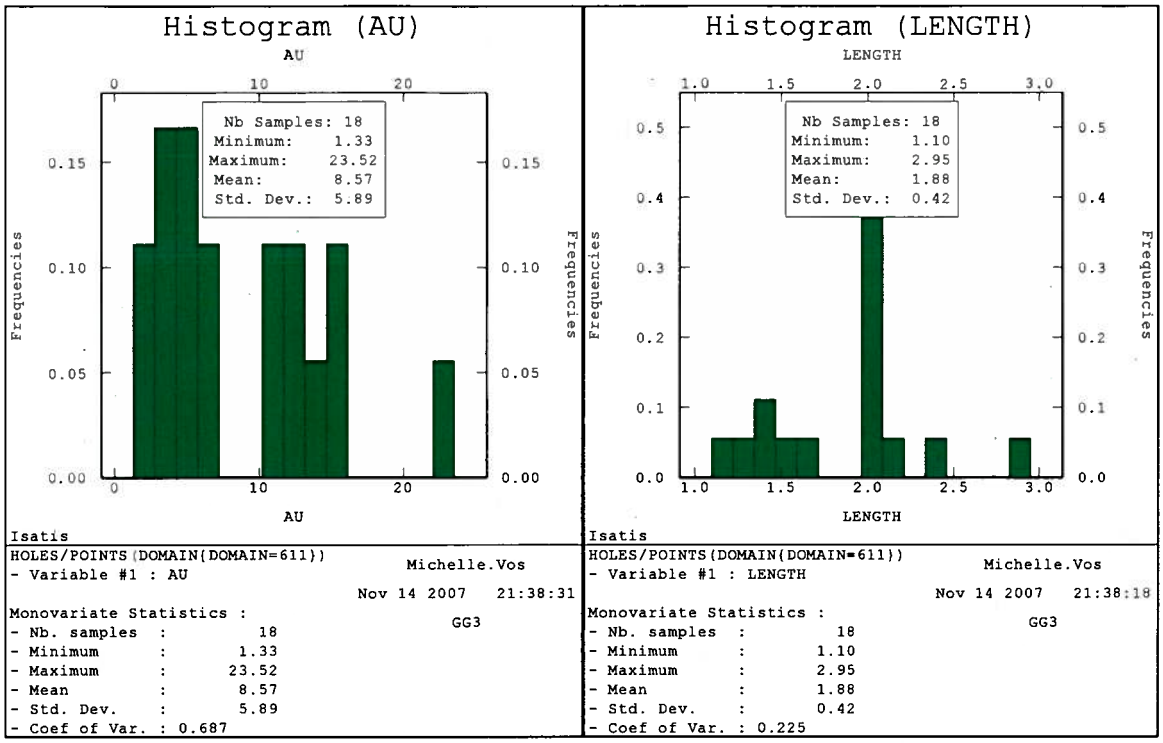

Figure 24 Histograms of Au (left) and length (right) in 2.0m composites of Domain 611.

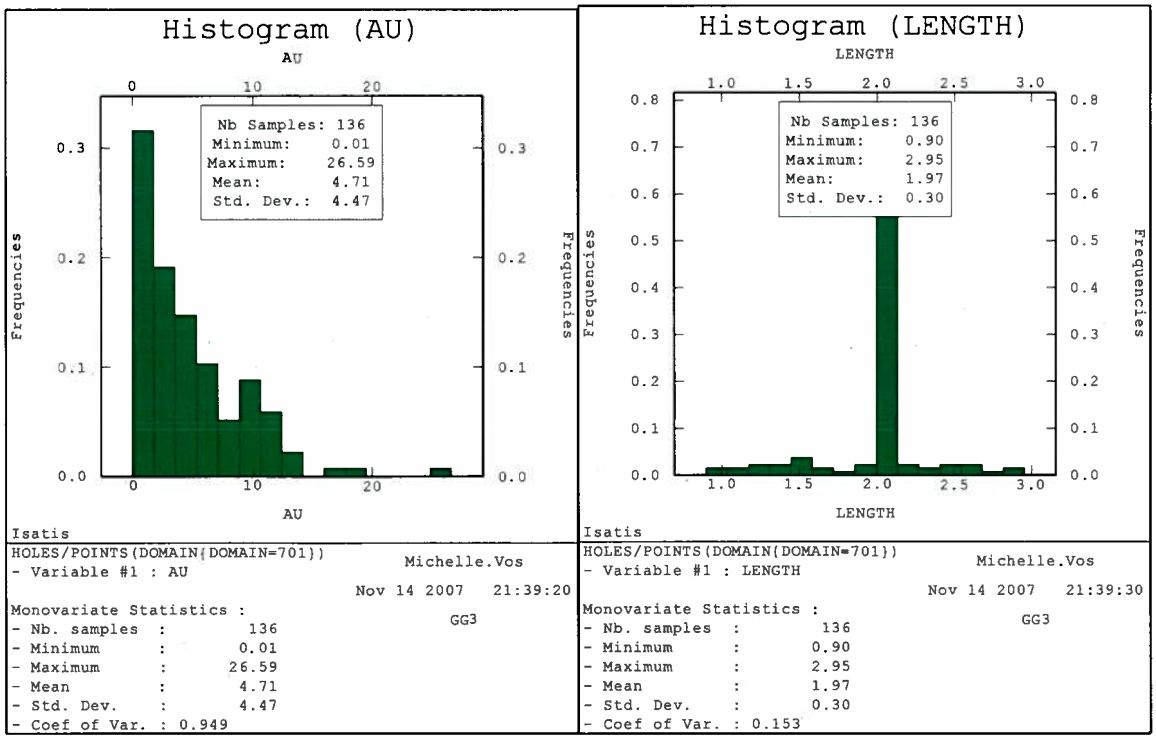

Figure 25 Histograms of Au (left) and length (right) in 2.0m composites of Domain 701.

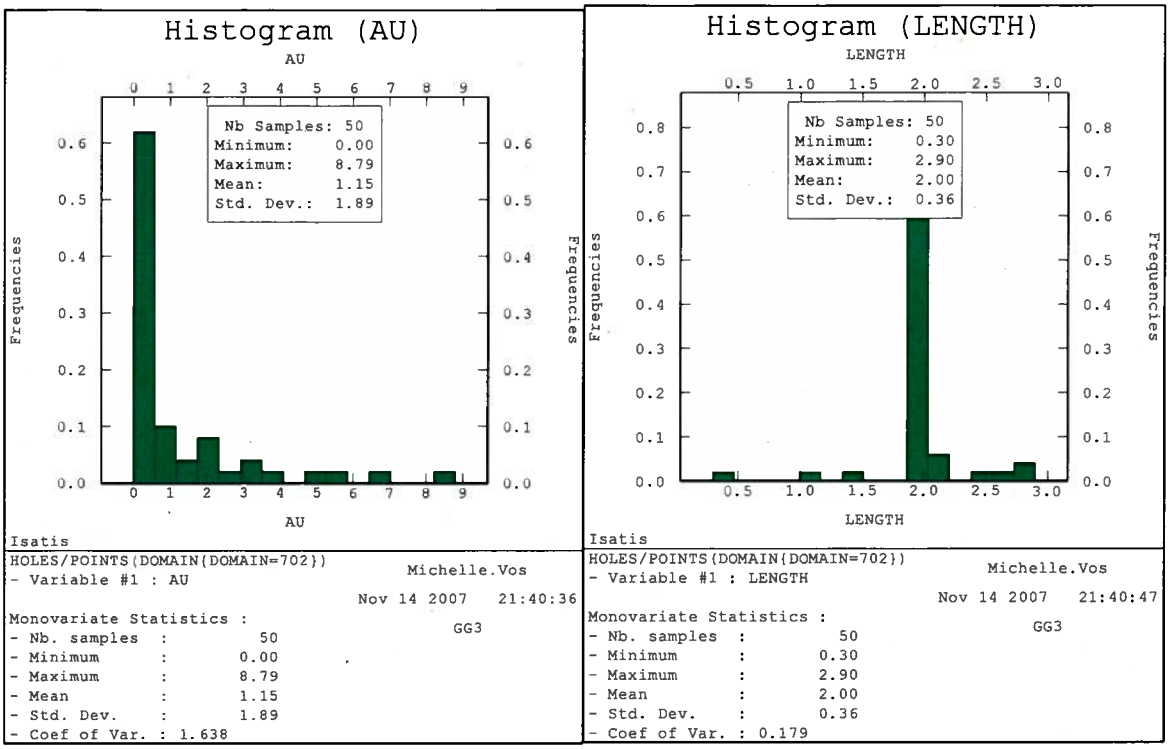

Figure 26 Histograms of Au (left) and length (right) in 2.0m composites of Domain 702.

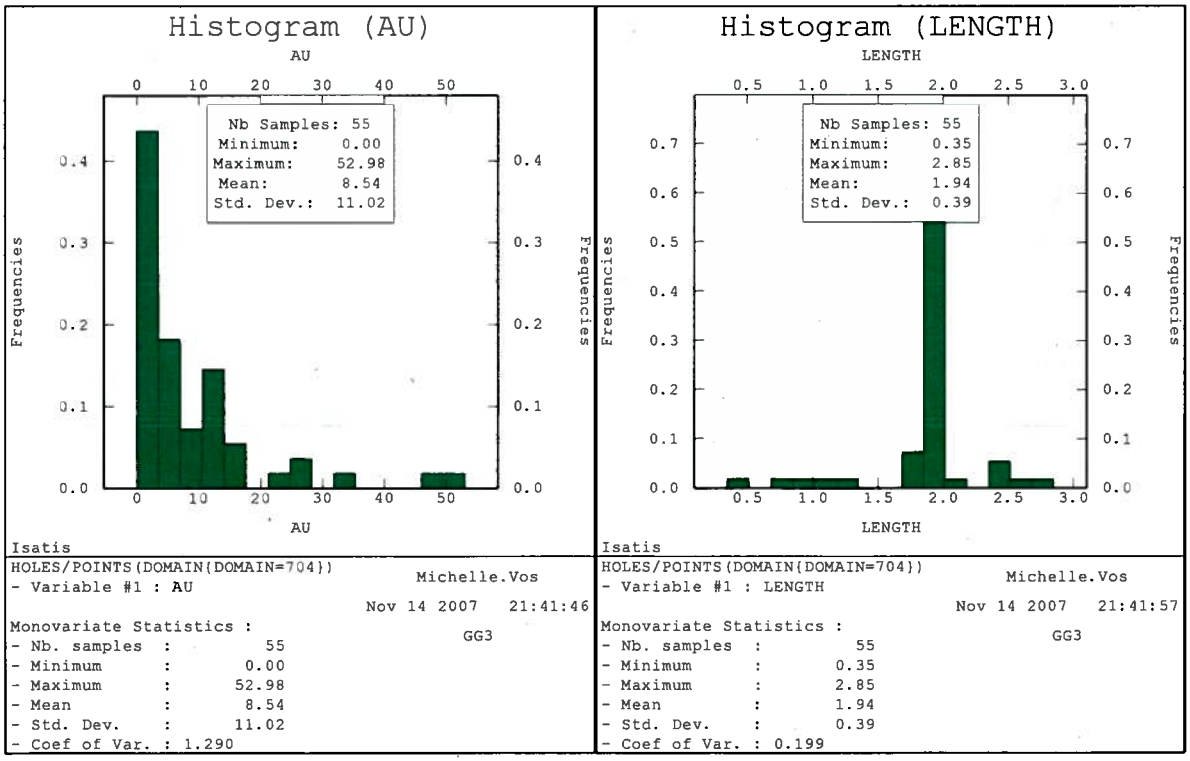

Figure 27 Histograms of Au (left) and length (right) in 2.0m composites of Domain 704.

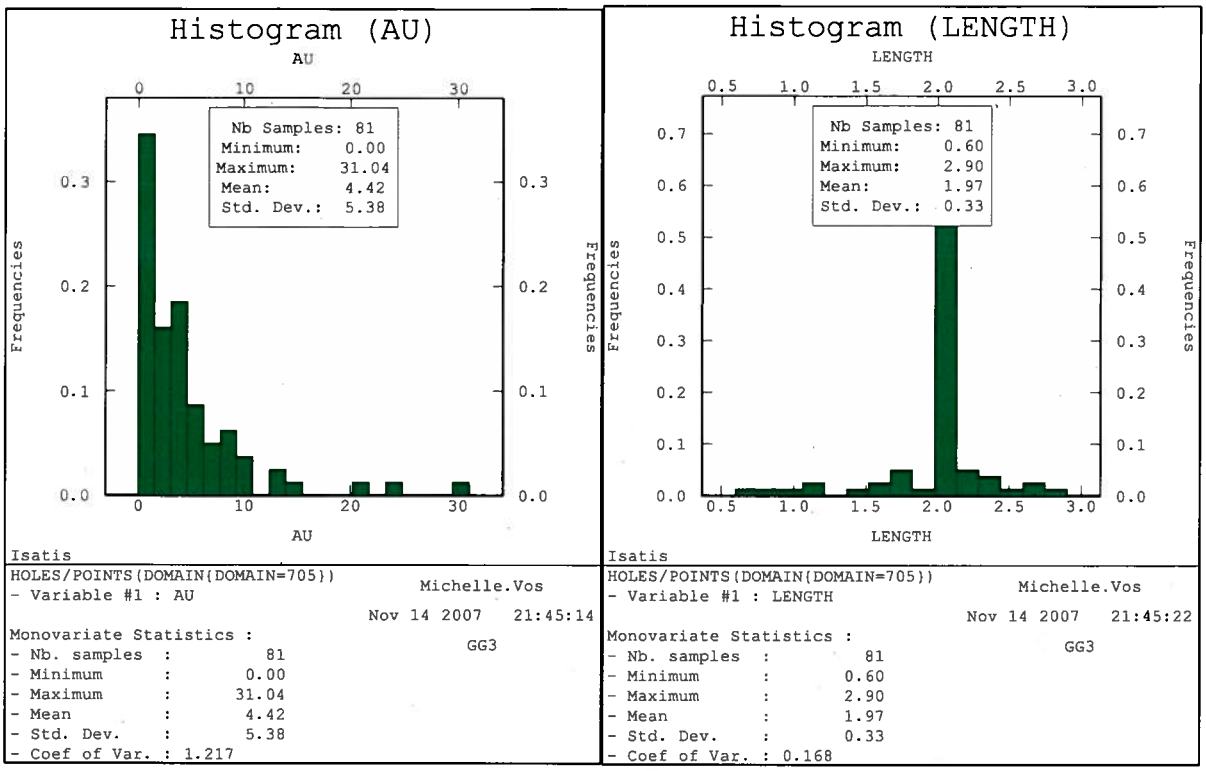

Figure 28 Histograms of Au (left) and length (right) in 2.0m composites of Domain 705.

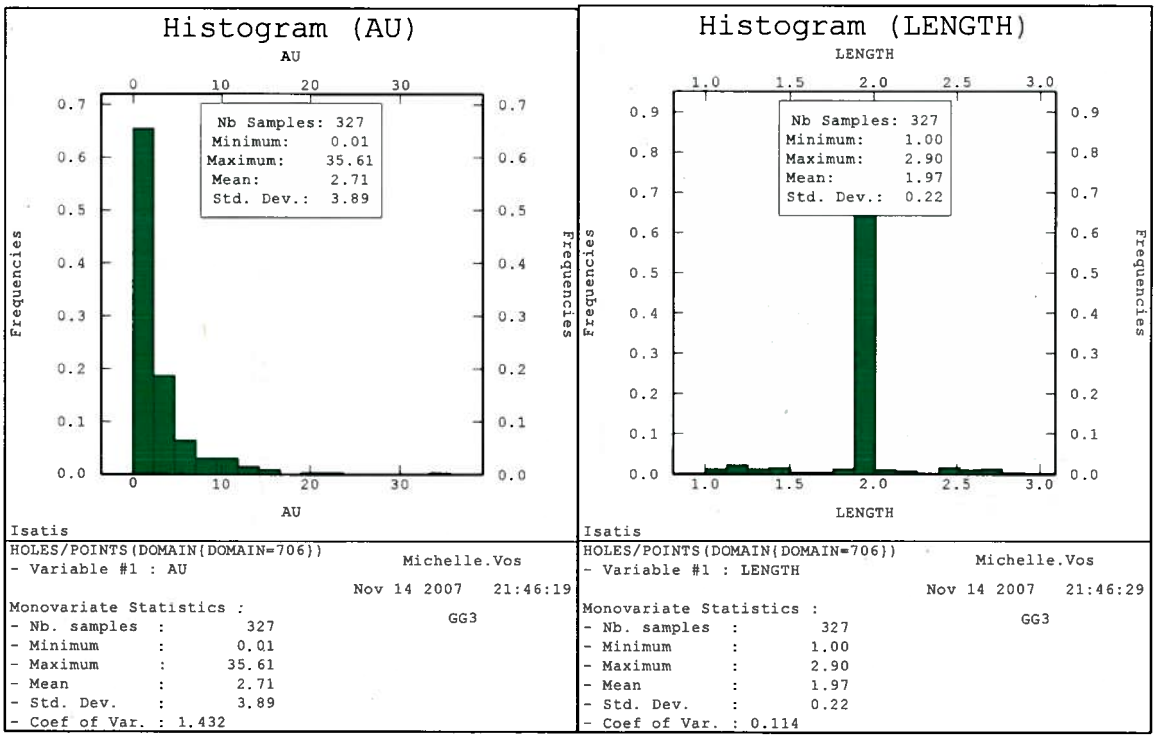

Figure 29 Histograms of Au (left) and length (right) in 2.0m composites of Domain 706.

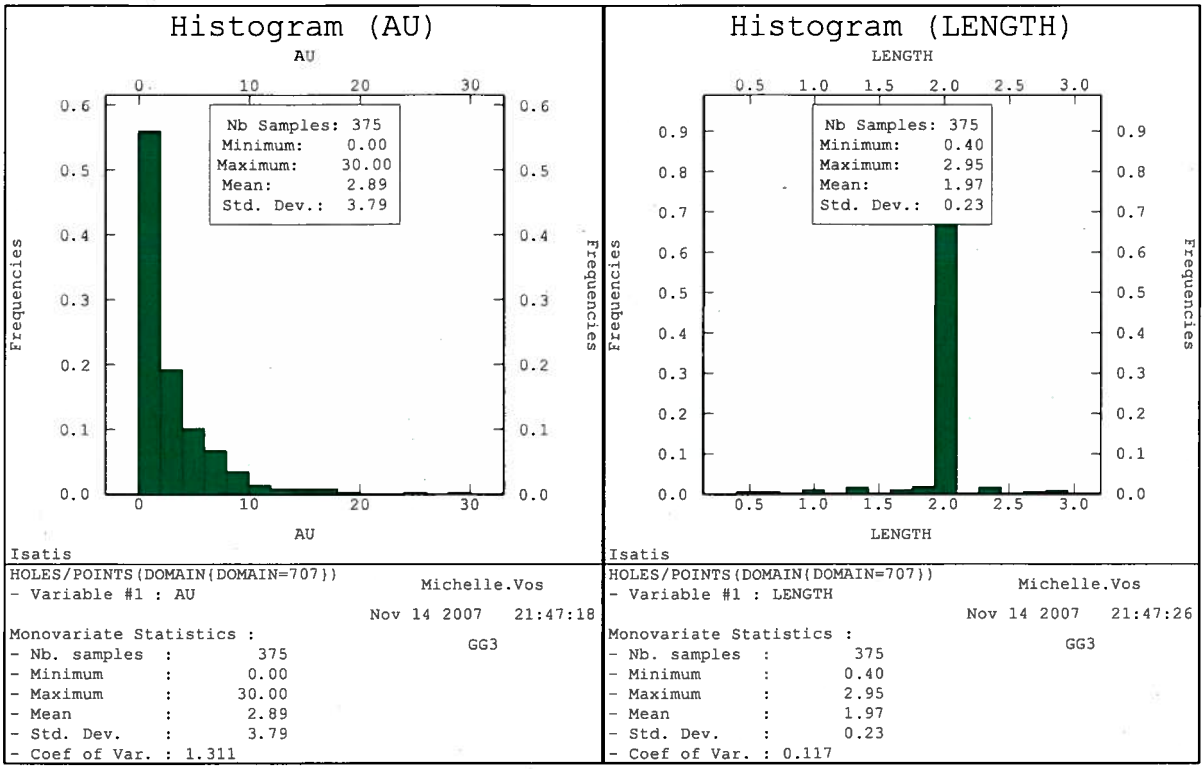

Figure 30 Histograms of Au (left) and length (right) in 2.0m composites of Domain 707.

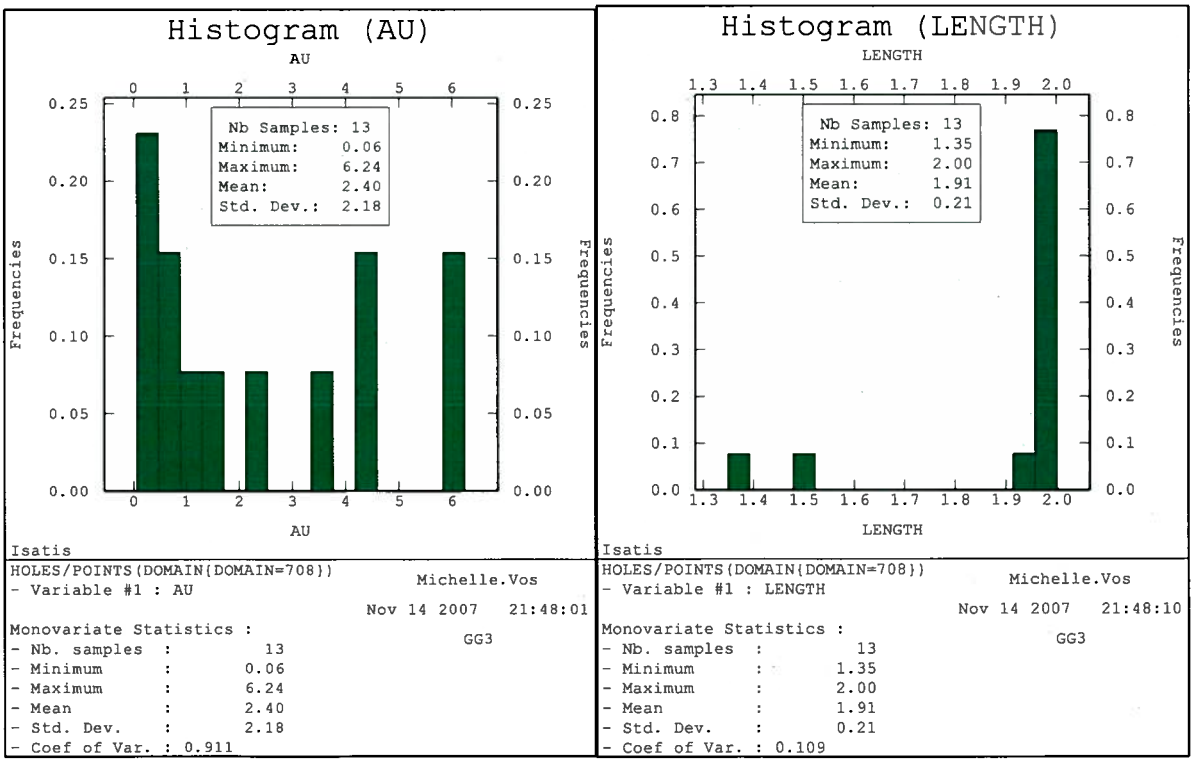

Figure 31 Histograms of Au (left) and length (right) in 2.0m composites of Domain 708.

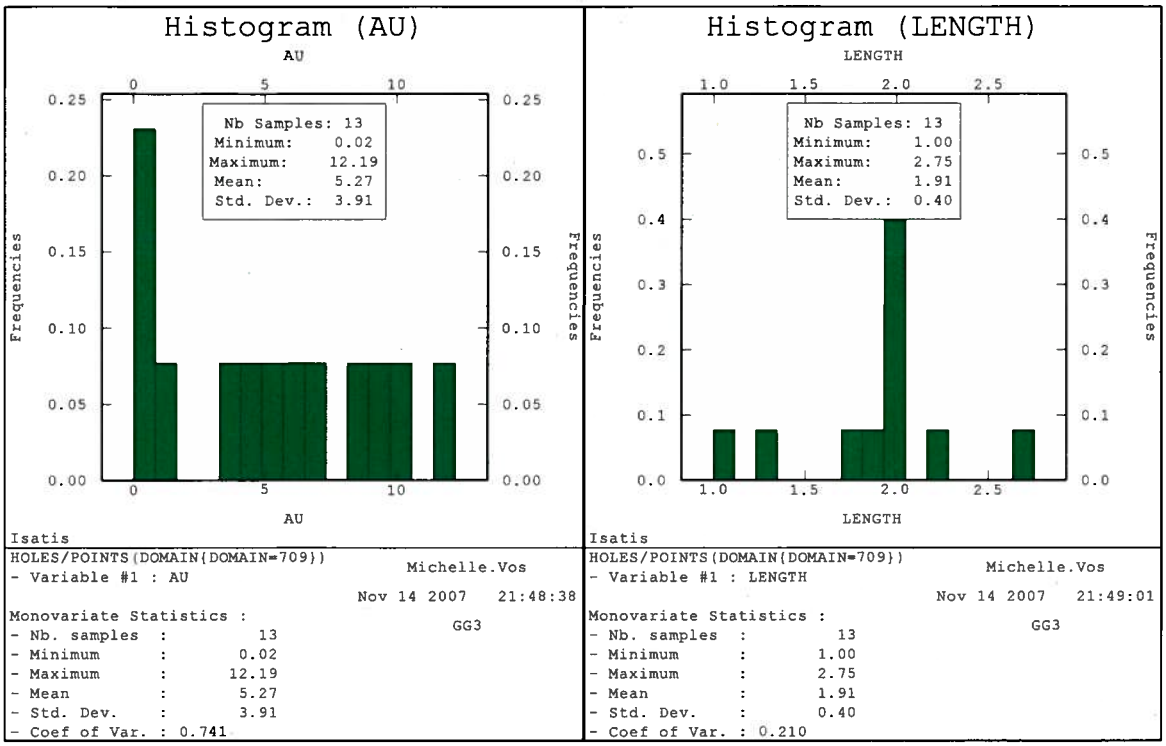

Figure 32 Histograms of Au (left) and length (right) in 2.0m composites of Domain 709.

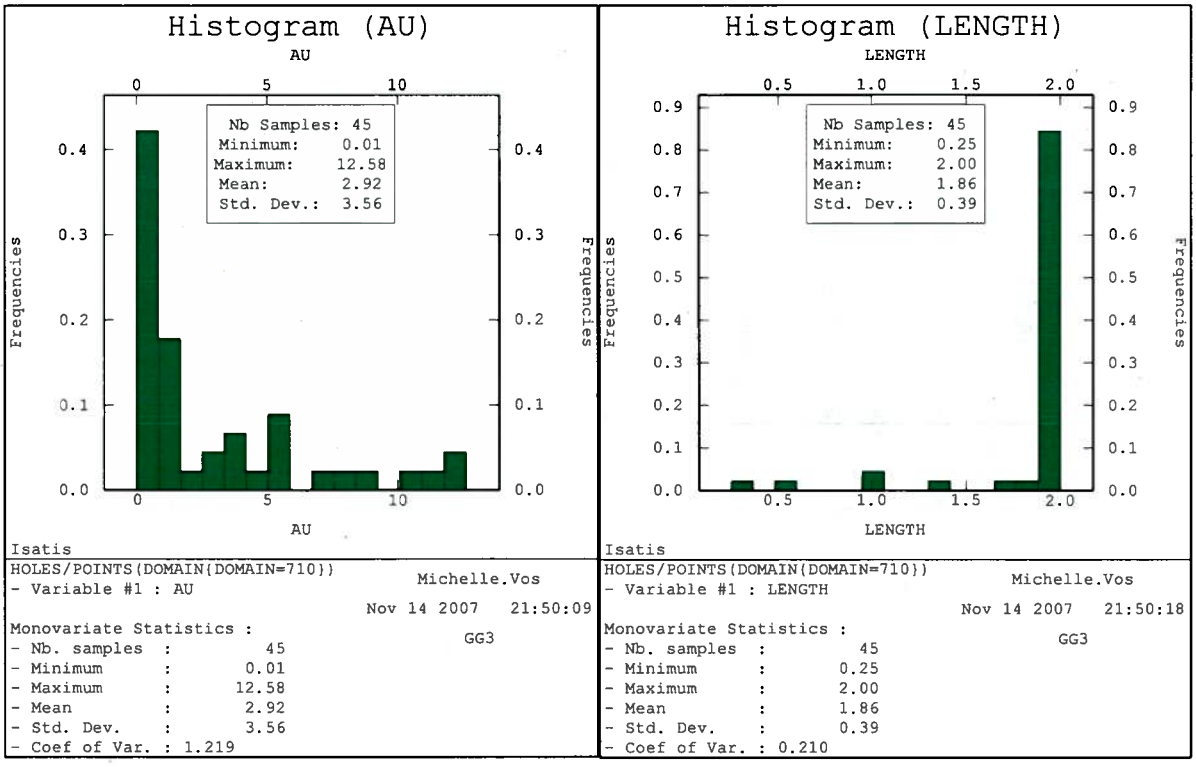

Figure 33 Histograms of Au (left) and length (right) in 2.0m composites of Domain 710.

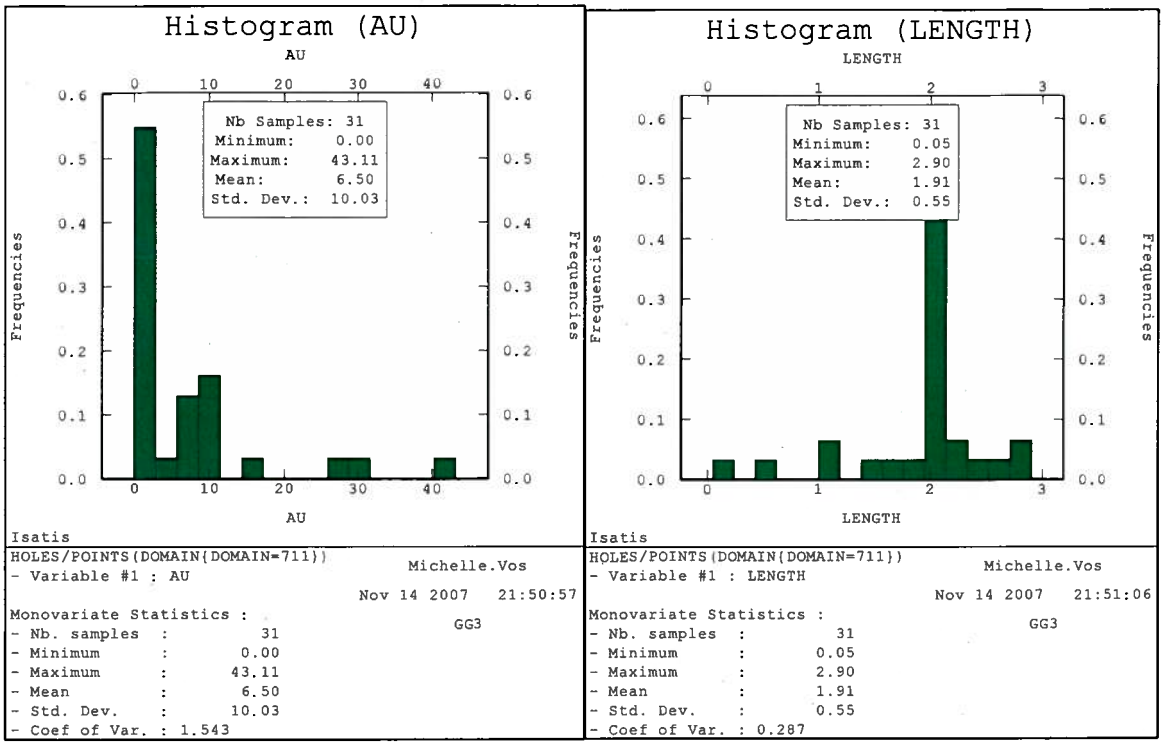

Figure 34 Histograms of Au (left) and length (right) in 2.0m composites of Domain 711.

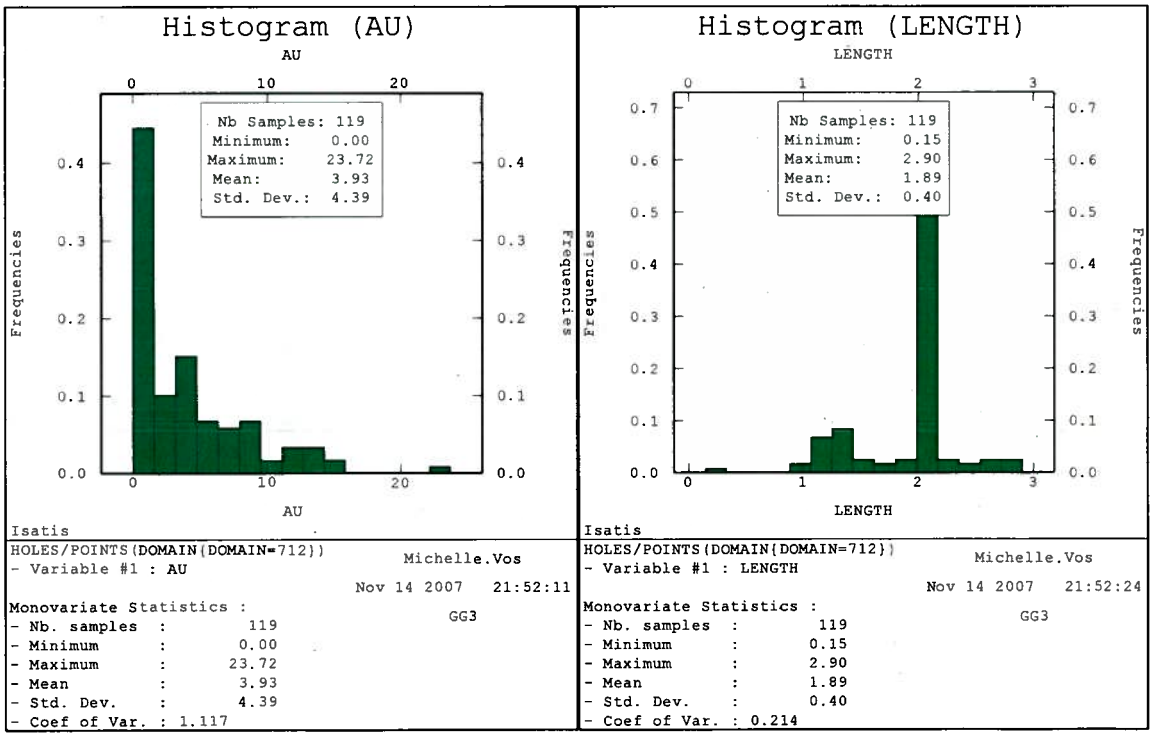

Figure 35 Histograms of Au (left) and length (right) in 2.0m composites of Domain 712.

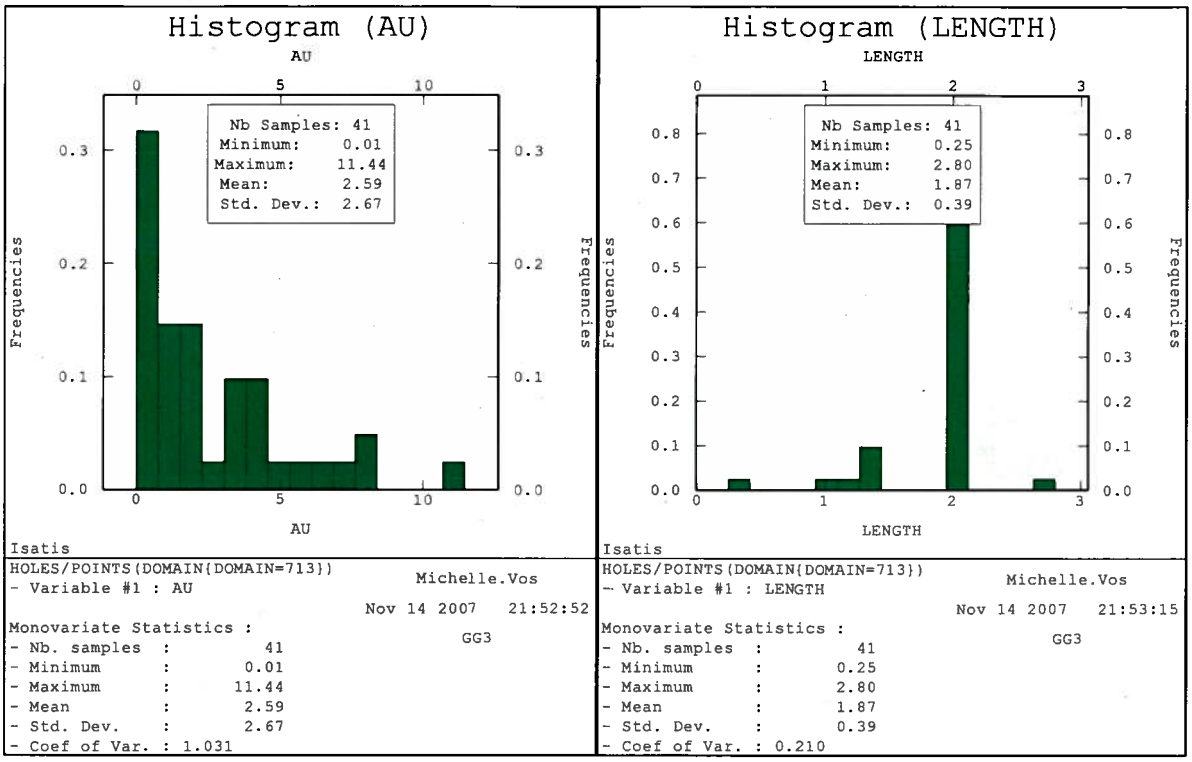

Figure 36 Histograms of Au (left) and length (right) in 2.0m composites of Domain 713.

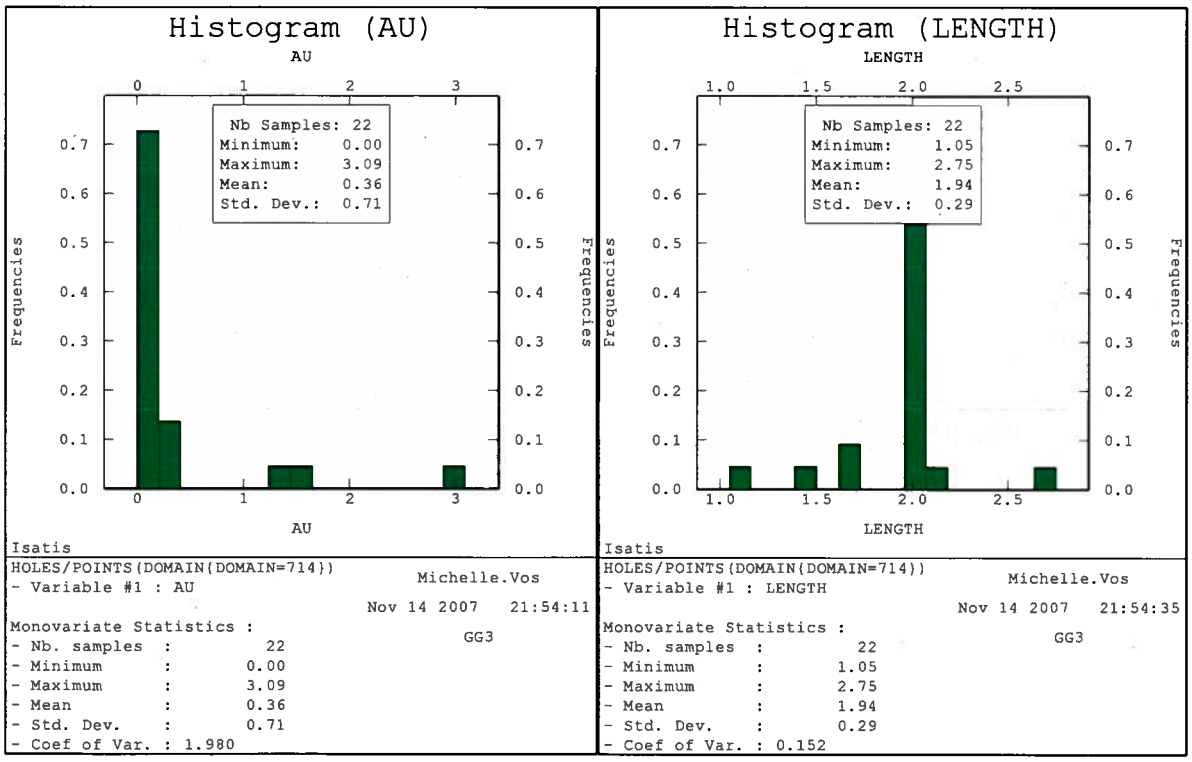

Figure 37 Histograms of Au (left) and length (right) in 2.0m composites of Domain 714.

Domain Number	Top-cut g/t
601	40
604	25
605	25
608	30
610	50
702	6
704	25
504	15

Table 2. General description of domains and top-cut applied.

5.4 Variography

Detailed variography was carried out for the May resource model update and has been applied to the November model (refer Figure 41 and Figure 42). Each domain contained a low number of samples due to the structural complexity. Hard boundaries were used for all domains. Numerous domains had too few samples to generate meaningful variograms. Each mineralisation style – Volcanogenic, Dalray HW and Dalray FW – was analysed and representative variograms were applied to each domain. Experimental variograms from Domain 501, 601 and 701 were chosen and applied to the appropriate mineralisation style. The variogram ranges derived for domain 701 were supported by the ranges seen in the experimental variogram of domains 701-709. For further detail of the experimental variograms of each of the mineralisation style refer to S:\Mining-Geology\ GoldenGift3\ Minesight\ Geology\ ResourceModelMay2007\ Documentation - GG3 Resource Model May 2007.xls - variograms. Refer to **Error! Reference source not found.** for the variogram parameter table used for block model estimation.

The variogram model for the Volcanogenic zone is shown in **Error! Reference source not found.**. The variogram axes was initially defined using the Isatis mathematical convention as 143° around the Z-axis, and 20° around the Y-axis (refer to Figure 38).

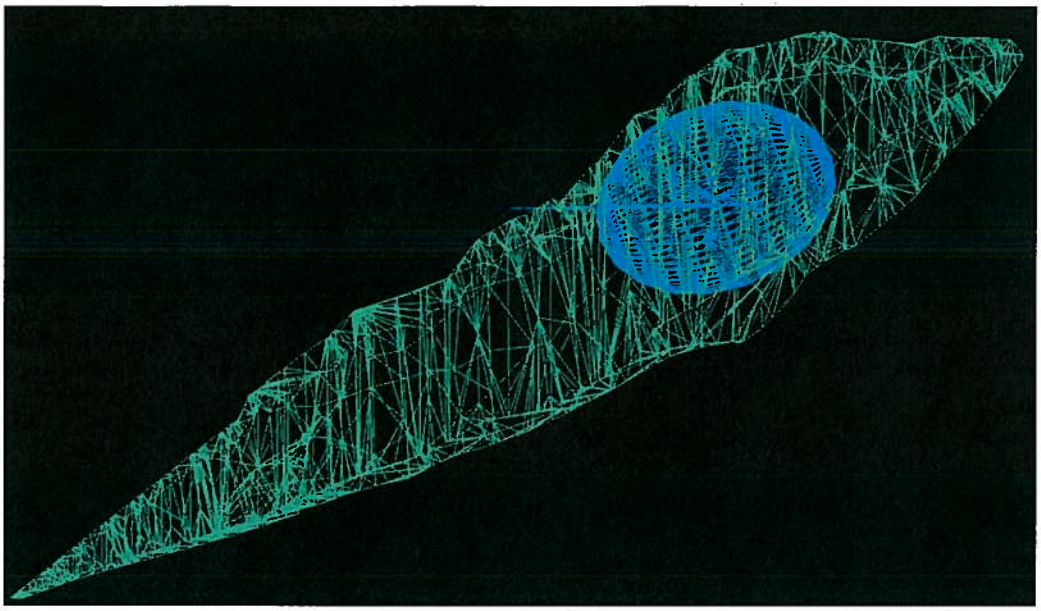

Figure 38. Minesight visualisation of Domain 501 wireframe and the search ellispse showing the orientation and major direction of the search.

The variogram model for Dalray HW is show in **Error! Reference source not found.**. The variogram axes were initially defined using the Isatis mathematical convention as 143° around the Z-axis, and 20° around the Y-axis (refer to Figure 39).

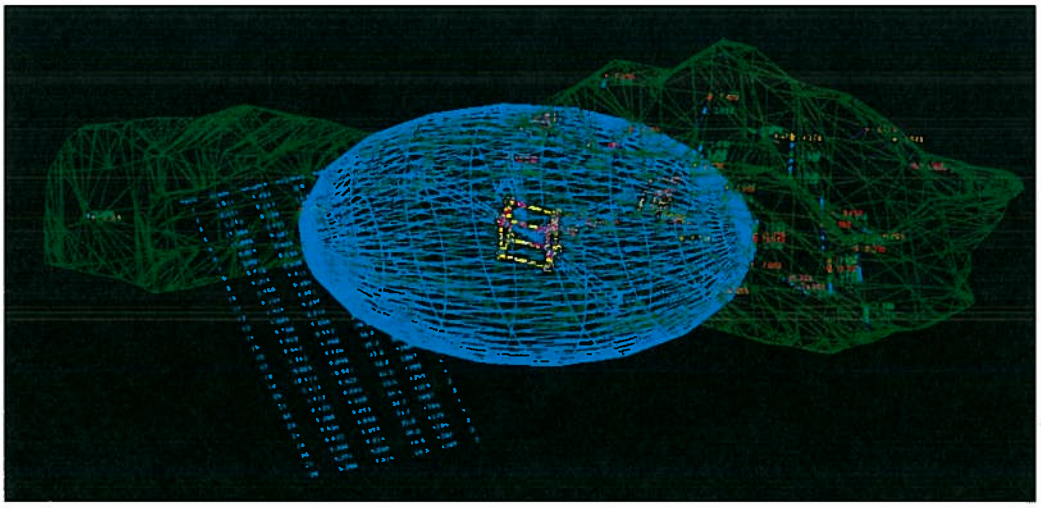

Figure 39. Minesight visualisations of Domain 601 wireframe and the search ellispse showing the orientation and major direction of the search.

The variogram model for the Dalray FW is shown in **Error! Reference source not found.**. The variogram axes were initially defined using the Isatis mathematical convention as 143° around the Z-axis, and 20° around the Y-axis (refer to Figure 40).

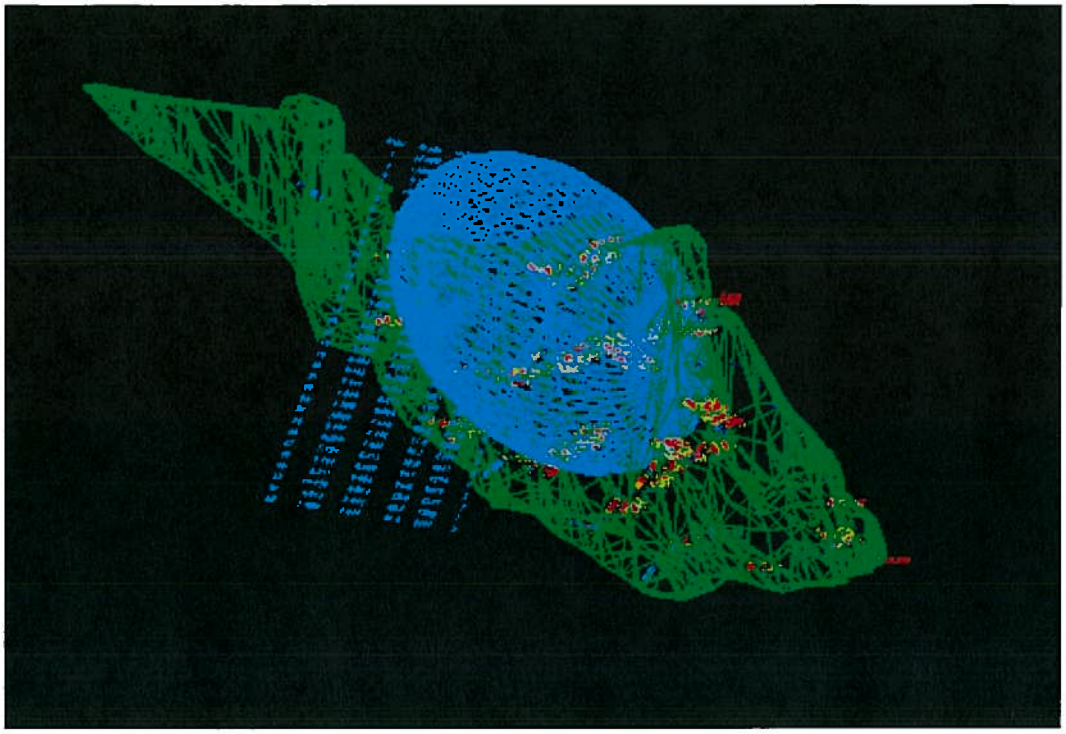

Figure 40. Minesight visualisations of Domain 701 wireframe and the search ellipse showing the orientation and major direction of the search.

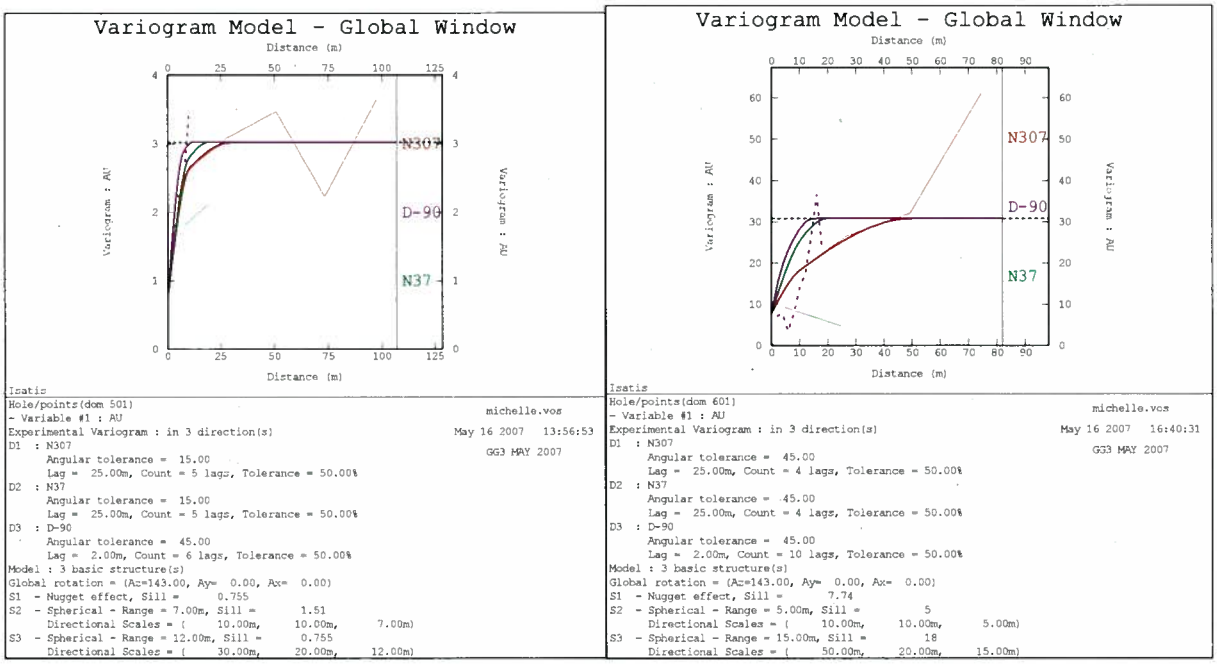

Figure 41. Variogram model fitted to the volcanogenic zone (left) and the Dalray HW (right).

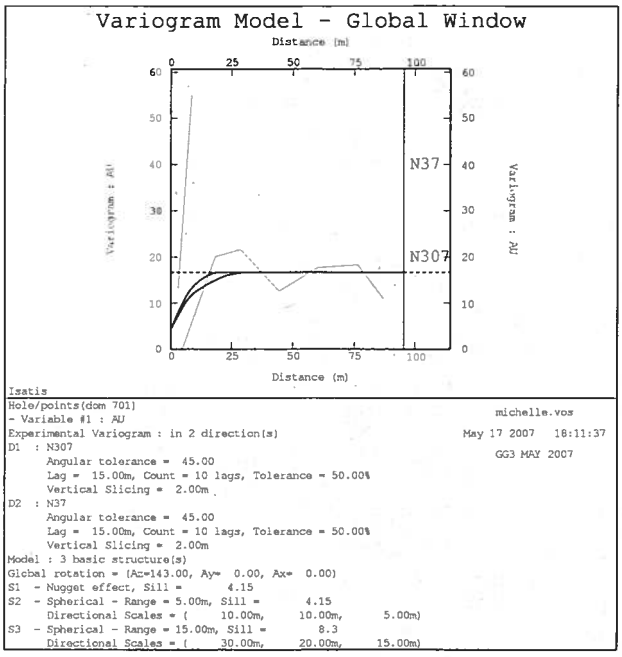

Figure 42. Variogram model fitted to the Dalray FW.

The variogram parameters used in the estimation are outlined below (refer **Error! Reference source not found.**).

PERSEVERANCE CORPORTATION LIMITED																								
GG3 RESOURCE MODEL																								
AREA	VREFNUM	axis 1	axis 2	axis 3	Nugget	1st Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	2nd Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	3rd Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	Total Variance	Nugget
		rotation																				
GG3	501	307	-20	-90	0.76	1	10	10	7	1.51	1	30	20	12	0.76						3.02	25%
GG3	503	307	-20	-90	0.76	1	10	10	7	1.51	1	30	20	12	0.76						3.02	25%
GG3	504	307	-20	-90	0.76	1	10	10	7	1.51	1	30	20	12	0.76						3.02	25%
GG3	505	307	-20	-90	0.76	1	10	10	7	1.51	1	30	20	12	0.76						3.02	25%
GG3	506	307	-20	-90	0.76	1	10	10	7	1.51	1	30	20	12	0.76						3.02	25%
GG3	507	307	-20	-90	0.76	1	10	10	7	1.51	1	30	20	12	0.76						3.02	25%
GG3	508	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	510	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	601	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	602	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	603	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	604	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	605	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	606	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	607	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	608	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	609	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	610	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	611	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	701	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	702	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	704	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	705	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	706	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	707	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	708	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	709	307	-20	-90	4.150	1	10	10	5	4.150	1	30	20	15	8.300						16.60	25%
GG3	710	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	711	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	712	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	713	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%
GG3	714	307	-20	-90	7.74	1	10	10	5	5.00	1	50	20	15	18.00						30.74	25%

Table 3. MINESIGHT variogram parameter table used for block model estimation.

6 Block Model Estimation

6.1 Neighbourhood optimisation

The neighbourhood optimisation is used to determine appropriate search dimensions and minimum and maximum number of samples during Kriging interpolation. It is done in an empirical manner by testing and spatially viewing, in ISATIS, characteristics like the distribution of negative kriging weights, the achievable theoretical regression slope, block kriging efficiency, etc, for a number of model blocks that are located in "favourable" and "unfavourable positions" with respect to the available drillholes within a domain.

The resource model the block size of 5m east, 10m north and 10m RL was used in accordance with domain geometry, data spacing and spatial continuity (refer to **Error! Reference source**

not found.). This model is a rotated model of 307° northwest. It is highly recommended that a full neighbourhood analysis of the block size, no of samples, dimensions of the search ellipse and no of discretisation points to be used in the model be completed for the next model update.

	Min	*Max*	*Cell Size*	*No. Cells*	*Origin (AMG)*
East	0	350	10	35	658315 E
North	0	620	10	62	898040 N
RL	0	340	5	68	-1010 RL

Table 4. Model Limits.

A listing of the MINESIGHT search parameters are provided in **Error! Reference source not found.**.

PERSEVERANCE CORPORTATION LIMITED								SEARCH PARAMETER TABLE												
GG3 RESOURCE MODEL																				
AREA	DOMAIN	x axis	y axis	z axis	axis 1	axis 2	axis 3	min. samples 1st search	max. samples 1st search	2nd search expansion factor	x axis	y axis	z axis	min. samples 2nd search	max. samples 2nd search	max. samples per hole	Zone	Samples used from each zone	Run file names
		search distance			rotation			sample number definition												
GG3	501	40	30	20	307	-20	-90	8	32											
GG3	503	40	30	20	307	-20	-90	8	32											
GG3	504	40	30	20	307	-20	-90	7	32											
GG3	505	40	30	20	307	-20	-90	8	32											
GG3	506	40	30	20	307	-20	-90	8	32											
GG3	507	40	30	20	307	-20	-90	8	32											
GG3	508	50	30	20	307	-20	-90	8	32											
GG3	510	50	30	20	307	-20	-90	8	32											
GG3	601	50	30	20	307	-20	-90	8	32											
GG3	602	50	30	20	307	-20	-90	8	32											
GG3	603	40	30	20	307	-20	-90	8	32											
GG3	604	50	30	20	307	-20	-90	5	32											
GG3	605	50	30	20	307	-20	-90	6	32											
GG3	606	50	30	20	307	-20	-90	3	32	1	50	30	20	8	32					
GG3	607	50	30	20	307	-20	-90	8	32											
GG3	608	50	30	20	307	-20	-90	8	32											
GG3	609	50	30	20	307	-20	-90	8	32											
GG3	610	50	30	20	307	-20	-90	8	32											
GG3	611	50	30	20	307	-20	-90	8	32											
GG3	701	40	30	20	307	-20	-90	8	32											
GG3	702	40	30	20	307	-20	-90	8	32											
GG3	704	40	30	20	307	-20	-90	8	32											
GG3	705	40	30	20	307	-20	-90	8	32											
GG3	706	40	30	20	307	-20	-90	8	32											
GG3	707	40	30	20	307	-20	-90	8	32											
GG3	708	40	30	20	307	-20	-90	8	32											
GG3	709	40	30	20	307	-20	-90	8	32											
GG3	710	50	30	20	307	-20	-90	8	32											
GG3	711	50	30	20	307	-20	-90	2	32											
GG3	712	50	30	20	307	-20	-90	8	32											
GG3	713	50	30	20	307	-20	-90	8	32											
GG3	714	50	30	20	307	-20	-90	8	32											

Table 5. Listing of MINESIGHT search parameters.

6.2 Block Estimation Parameters

All domains have been constrained by wireframes. They have been estimated using Ordinary Block Kriging. The following domains have been used for estimation (**Error! Reference source not found.**).

Fault Block	Solid	Code	Model Item	Shell Filename
Cabernet	Dalray HW	601	BAUD	601DalrayHW
Cabernet	Dalray FW	701	CAUD	701Dalray
Cabernet	Volc	501	AAUD	501volc
Eastern Steep Fault	Dalray HW	602	DAUD	602DalrayHW
Eastern Steep Fault	Dalray FW	702	EAUD	702DalrayFW
Fault A	Dalray FW	603	DAUD	603Dalray
Fault A	Volc	503	BAUD	503volc
Steep Fault	Dalray HW	604	BAUD	604DalrayHW
Steep Fault	Dalray FW	704	CAUD	704DalrayFW
Steep Fault	Volc	504	AAUD	504volc
Flat Fault	Dalray HW	605	DAUD	605DalrayHW
Flat Fault	Dalray FW	705	GAUD	705DalrayFW.
Flat Fault	Volc	505	FAUD	505volc
Pinot Splay	Dalray HW	609	AAUD	609DalrayHW
Pinot Splay	Dalray FW	709	EAUD	709DalrayFW
Pinot	Dalray HW	606	CAUD	606DalrayHW
Pinot	Dalray FW	706	HAUD	706DalrayFW
Pinot	Volc	506	BAUD	506volc
Saignee	Dalray HW/Fury	607	AAUD	607Fury
Saignee	Dalray	707	AAUD	707Dalray
Saignee	Volc	507	CAUD	507volc
New Fault	Dalray HW/Fury	608	BAUD	608Fury
New Fault	Dalray FW	708		708DalrayFW
New Fault	Dalray HW	508	DAUD	508DalrayHW
Vindaloo	Fury	610	HAUD	610Fury
Vindaloo	Dalray FW	710	AAUD	710DalrayFW
Vindaloo	Dalray HW	510	CAUD	510DalrayHW.
Tandoori	Waterloo	611	FAUD	611Waterloo
Tandoori	Volc	711	DAUD	711Volc
Korma	Dalray FW	712	BAUD	712Volc
Naan	Volc	713	HAUD	713Volc
Kamado	Volc	714	CAUD	714volc
General	Basalt	400	WAUD	basalt
General	Mineschist/Porphyry	1	WAUD	msporh

Table 6. Domains used in the Ordinary Kriged estimate.

Estimations in MINESIGHT are confined to domains (hard boundaries). The composited drill holes and block model were assigned the appropriate domain values.

The completed block model and all associated MINESITE files are stored on the Mining Geology server in the directory S:\Mining-Geology\Golden Gift 3\Minesight\Geology\Current - Resource Model November 2007. Additional details on the MINESIGHT resource model are provided in **Error! Reference source not found.**.

PERSEVERANCE CORPORTATION LIMITED	GG3 RESOURCE MODEL
Date	Nov-2007
Model compiled by	Michelle Vos/Matt Stevens
Directory Location	S:\Mining-Geology\Golden Gift 3\Minesight\Geology\Current-Resource Model November 2007
Model Name	ggnt15.dat
Previous Model Name	ggnt15.dat
Parameter Spreadsheet	GG3 Resource Model November final 07.xls
Resource Report	GG3 Resource Model Report November 07.doc
Purpose of Update	Update GG3 with new diamond drilling results for reserve estimation.

Table 7. Resource model description table.

7 Model Validation

Model validation has been completed using all tools available. To facilitate reporting, the spreadsheet GG3_Resource Modelling November 2007.xls was used.

The Slope of regression achieved in the Indicated area of the model shows acceptable block estimation, based on the available variogram model.

A comparison of the declustered composited drill hole data and mean grade of the model blocks (10mN x 5mE x 10mRL) within a domain shows an acceptable variance (refer to **Error! Reference source not found.**).

PERSEVERANCE CORPORTATION LIMITED			MODEL VALIDATION TABLE		
GG3 RESOURCE MODEL					
Date:		13/11/2007			
			Mean Grade (g/t)		
DOMAIN		Resource Category	Declust. 2.0m comp.	Model	Variance
GG3	501	Indicated	0.77	0.707	-7.6%
GG3	503	Inferred	0.660	0.660	0.0%
GG3	504	Indicated +Inferred	2.2728	2.886	27.0%
GG3	505	Indicated	1.671	1.4241	-14.8%
GG3	506	Indicated	1.16	1.160	-0.1%
GG3	507	Unclassifed	0.87	0.870	0.0%
GG3	508	Indicated	6.897	5.978	-13.3%
GG3	510	Inferred	8.07	8.07	0.0%
GG3	601	Indicated	3.1673	2.9862	-5.7%
GG3	602	Unclassifed	2.88	2.880	0.0%
GG3	603	Inferred	1.52	1.520	0.0%
GG3	604	Inferred	6.8638	9.031	31.6%
GG3	605	Inferred	8.1438	9.488	16.5%
GG3	606	Indicated +Inferred	4.461	3.370	-24.5%
GG3	607	Unclassifed	2.350	2.350	0.0%
GG3	608	Inferred	12.8786	14.178	10.1%
GG3	609	Unclassifed	11.680	11.680	0.0%
GG3	610	Inferred	19.3341	18.938	-2.0%
GG3	611	Inferred	8.329	8.747	5.0%
GG3	701	Indicated	4.8279	4.507	-6.7%
GG3	702	Inferred	1.1211	1.432	27.7%
GG3	704	Inferred	7.6894	7.501	-2.5%
GG3	705	Indicated	4.1405	4.444	7.3%
GG3	706	Indicated	2.64	2.426	-8.1%
GG3	707	Indicated	2.917	2.678	-8.2%
GG3	708	Inferred	2.49	2.490	0.0%
GG3	709	Inferred	5.208	5.210	0.0%
GG3	710	Indicated	2.967	3.166	6.7%
GG3	711	Inferred	6.867	4.8693	-29.1%
GG3	712	Indicated	4.214	3.634	-13.8%
GG3	713	Inferred	2.69	2.134	-20.7%
GG3	714	Inferred	0.424	0.420	-0.9%

Table 8. Model validation table.

Due to the low number of samples in numerous domains the variation in mean grade can be high. This has been reviewed visually in each domain and is acceptable, also in consideration with the resource classification.

8 Model Risk Assessment

The classification of Indicated and Inferred is based on geological confidence, Slope of Regression and model validation results. The model is visually compared with the drill holes to ensure smearing of high grades into areas of known low grade has not occurred and vice versa.

9 Model output

The total Modelled Inventory at a cut off grade of 4.0g/t Au is 77,400oz. A listing of the global inventory at a series of cut-off gold grades is provided in **Error! Reference source not found.** (not depleted with current mining). Note the change in interpretation and re-estimation included a volume addition.

PERSEVERANCE CORPORTATION LIMITED	GG3 RESOURCE MODEL
Resource Classification	Measured, Indicated, Inferred
Mined Status	Mined and Unmined
Other Restrictions	None
Filter Applied to Evaluation	No filter applied
Previous Model Name	ggnt15.dat
New Model Name	ggnt15.dat

RESCAT	Cutoff (g/t)	PREVIOUS MODEL			NEW MODEL			Change			Change %		
		quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)
MEASURED INDICATED INFERRED	0.0	1,960,000	2.3	147,400	2,852,800	2.1	191,200	892,800	-0.3	43,800	46%	-11%	30%
	2.0	892,000	4.0	115,700	1,008,200	4.2	134,900	116,200	0.1	19,200	13%	3%	17%
	4.0	352,700	5.8	65,400	369,900	6.5	77,400	17,200	0.7	12,000	5%	13%	18%
	5.0	190,100	6.9	42,100	211,900	7.9	53,700	21,800	1.0	11,600	11%	14%	28%
	7.0	65,000	8.9	18,600	88,300	10.7	30,300	23,300	1.7	11,700	36%	20%	63%
	9.0	27,800	10.5	9,400	47,000	13.1	19,800	19,200	2.6	10,400	69%	25%	111%
MEASURED	0.0	-	-	-	-	-	-	0	0.0	0			
	2.0	-	-	-	-	-	-	0	0.0	0			
	4.0	-	-	-	-	-	-	0	0.0	0			
	5.0	-	-	-	-	-	-	0	0.0	0			
	7.0	-	-	-	-	-	-	0	0.0	0			
	9.0	-	-	-	-	-	-	0	0.0	0			
INDICATED	0.0	1,960,000	2.3	147,400	2,563,400	1.9	155,400	603,400	-0.5	8,000	31%	-19%	5%
	2.0	892,000	4.0	115,700	850,200	3.8	103,500	-41,800	-0.2	-12,200	-5%	-6%	-11%
	4.0	352,700	5.8	65,400	294,200	5.6	52,700	-58,500	-0.2	-12,700	-17%	-3%	-19%
	5.0	190,100	6.9	42,100	152,200	6.6	32,300	-37,900	-0.3	-9,800	-20%	-4%	-23%
	7.0	65,000	8.9	18,600	41,500	8.8	11,800	-23,500	-0.1	-6,800	-36%	-1%	-37%
	9.0	27,800	10.5	9,400	15,000	10.7	5,200	-12,800	0.2	-4,200	-46%	2%	-45%
INFERRED	0.0	-	-	-	289,400	3.9	35,900	289,400	3.9	35,900			
	2.0	-	-	-	158,000	6.2	31,400	158,000	6.2	31,400			
	4.0	-	-	-	75,800	10.1	24,700	75,800	10.1	24,700			
	5.0	-	-	-	59,700	11.1	21,400	59,700	11.1	21,400			
	7.0	-	-	-	46,800	12.3	18,500	46,800	12.3	18,500			
	9.0	-	-	-	32,000	14.2	14,600	32,000	14.2	14,600			
POTENTIAL	0.0	-	-	-	-	-	-	-	0.0	-			
	2.0	-	-	-	-	-	-	-	0.0	-			
	4.0	-	-	-	-	-	-	-	0.0	-			
	5.0	-	-	-	-	-	-	-	0.0	-			
	7.0	-	-	-	-	-	-	-	0.0	-			
	9.0	-	-	-	-	-	-	-	0.0	-			

Table 9. Modelled inventory at a series of cut-off grades for a new resource model compared to the previous model.

10 Recommendations

There is a high geological risk in the current GG3 geological model. It is recommended that alternate geological models be investigated. It would be of benefit to investigate high grade areas to attempt to understand their distribution. The implementation of a thorough QAQC program is required. Further investigation is required for Kriging Neighbourhood analysis (block size, number of samples, search size and number of discretisation points). Global estimation of recoverable resources would be of value.

11 Model completion

11.1 Server storage location of model output files

The model and all accompanying Minesight, ISATIS, Microsoft Excel™ and Microsoft Word™ computer files have been stored on the Stawell Gold Mines computer server in the sub-directory S:\Mining-Geology\Golden Gift 3\Minesight\Geology\Current - Resource Model November 2007. The project directory contains a spreadsheet named "GG3_readme_first.xls" that provides a listing all basic file name information for the project (refer to Table 10).

Project Name	GG3				
Date of Study	Nov-2007				
Resource Geologist Name	MV/MS				
Directory Location	S:\Mining-Geology\Golden Gift 3\Minesight\Geology\Resource Model November 2007_msresources				
	Minesight file	**details**			
Drillhole Data					
Raw Drill Hole Data	ggnt11.ddh				
Composited and Top Cut Drill Hole File	ggnt09.2m	2.0m comp			
Fault Block	Geology Final Solids	Domain	Geocode	Grade Item	DXF File
Cabernet	Dalray HW	601	BAUD	BAUD	601DalrayHW
Cabernet	Dalray FW	701	CAUD	CAUD	701Dalray
Cabernet	Volc	501	AAUD	AAUD	501volc
Eastern Steep Fault	Dalray HW	602	DAUD	DAUD	602DalrayHW
Eastern Steep Fault	Dalray FW	702	EAUD	EAUD	702DalrayFW
Fault A	Dalray FW	603	DAUD	DAUD	603Dalray
Fault A	Volc	503	BAUD	BAUD	503volc
Steep Fault	Dalray HW	604	BAUD	BAUD	604DalrayHW
Steep Fault	Dalray FW	704	CAUD	CAUD	704DalrayFW
Steep Fault	Volc	504	AAUD	AAUD	504volc
Flat Fault	Dalray HW	605	DAUD	DAUD	605DalrayHW
Flat Fault	Dalray FW	705	GAUD	GAUD	705DalrayFW.
Flat Fault	Volc	505	FAUD	FAUD	505volc
Pinot Splay	Dalray HW	609	AAUD	AAUD	609DalrayHW
Pinot Splay	Dalray FW	709	EAUD	EAUD	709DalrayFW
Pinot	Dalray HW	606	CAUD	CAUD	606DalrayHW
Pinot	Dalray FW	706	HAUD	HAUD	706DalrayFW
Pinot	Volc	506	BAUD	BAUD	506volc
Saignee	Dalray HW/Fury	607	AAUD	AAUD	607Fury
Saignee	Dalray	707	AAUD	AAUD	707Dalray
Saignee	Volc	507	CAUD	CAUD	507volc
New Fault	Dalray HW/Fury	608	BAUD	BAUD	608Fury
New Fault	Dalray FW	708			708DalrayFW
New Fault	Dalray HW	508	DAUD	DAUD	508DalrayHW
Vindaloo	Fury	610	HAUD	HAUD	610Fury
Vindaloo	Dalray FW	710	AAUD	AAUD	710DalrayFW
Vindaloo	Dalray HW	510	CAUD	CAUD	510DalrayHW.
Tandoori	Waterloo	611	FAUD	FAUD	611Waterloo
Tandoori	Volc	711	DAUD	DAUD	711Volc
Korma	Dalray FW	712	BAUD	BAUD	712Volc
Naan	Volc	713	HAUD	HAUD	713Volc
Kamado	Volc	714	CAUD	CAUD	714volc
General	Basalt	400	WAUD	WAUD	basalt
General	Mineschist/Porphyry	1	WAUD	WAUD	msporh
Models					
Final Resource Model Name	**ggnt15.dat**				
Project Control	ggnt10.dat				

Table 10. Readme_first.xls table of file names and major model parameters.

12 References

O'Brien, V., 2005. Golden Gift 3 Resource Model. *Internal PSV document dated December 2005*

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX H

Golden Gift 5 Resource Model

June 2005

Golden Gift 5
Resource Model

Vincent O'Brien
Senior Mine Geologist

June 2005

1 Executive Summary

The new Golden Gift 5 resource model area covers the area from 280N (5600N) to 306N (6120N), and is bounded by the South Fault above. The Statesman and Empire Lode ore shoots weaken south of 288N and Statesman Lode is pinched out down dip initially by the Porphyry and then by the internal basalt. Empire Lode pinches out at depth between the internal and hangingwall basalts.

At the time of modelling there were 191 diamond drillholes intersecting the interpreted volcanogenics host rock package, which is 40% more than the number of holes used in the previous estimate. Some holes drilled previously have been incorporated into the interpretation for the first time and there were 38 new grade control in-fill holes targeting stoping blocks between the 1105 and 1145 development levels.

GG5	Tonnes	Grade	Oz's
Statesman	293,340	6.31	59,516
Empire	275,896	5.95	52,810
TOTAL	569,235	6.14	112,326

June 2005 GG5 Proven and Probable Reserves by Lode.

All resources reported for GG5 in June 2005 are from the kriged resource model with nothing reported from the 2002 Conditional Simulation model. All of the GG5 resource has been converted to Measured and Indicated status and has been included in the mineable reserve, with no remaining resources exclusive of reserves.

2 Introduction

The new Golden Gift 5 resource model area covers the same area as the previous model, from 280N (5600N) to 306N (6120N), and is bounded by the South Fault above. The Statesman and Empire Lode ore shoots weaken south of 288N and Statesman Lode is pinched out down dip initially by the Porphyry and then by the internal basalt. Empire Lode pinches out at depth between the internal and hangingwall basalts. At the time of modelling there were 191 diamond drillholes intersecting the interpreted volcanogenics host rock package. Some holes drilled previously have been incorporated into the interpretation for the first time and there were 38 new grade control in-fill holes targeting stoping blocks between the 1105 and 1145 development levels.

Many members of the SGM Geology Department conducted drillhole logging for this project, with interpretation, grade estimation and Resource Classification undertaken by Vincent O'Brien.

Project directory S:\Mining-Geology\Golden Gift 5\Minesight\Geology\Current Model

Model is a rotated model.
Rotation Parameters: 315/ 0/ 20

	Min	*Max*	*Cell Size*	*No. Cells*	*Origin (AMG)*
East	0	250	5	50	659240 E
North	0	630	15	42	897675 N
RL	0	500	10	50	-1420 RL

Table 1. Model Limits

3 Data Sources

3.1 Data added since December 2004

3.1.1 Drilling:

Since December 2004 drilling has been focused on grade control infill drilling into Empire Lode from the 1070 and 1080mRL drill cuddies, and infill drilling for Statesman and Empire Lodes from the 1105 Level cross-cut area.

The Bobcat mounted LM37 rig drilled three LTK48 (35.3mm diameter) holes from the 1085-302-85STH ore drive. Two of these holes were to help locate the Statesman Lode footwall above the porphyry whilst the third targeted Empire Lode in the hangingwall to ore development.

Since the last model grade control drill rigs have drilled total of 34 LTK60 (44.0mm diameter) holes from the 1070 and 1080 drill cuddies and the 1105 cross-cut area targeting Statesman and Empire lodes. Assays were not available at the time of estimation for two holes (MD4362 and MD4455) from the 1080mRL cuddy but the logging was used to aid interpretation.

Eight holes have been completed from the 1025 Ventlink drive targeting GG5 north of 301N. Two of these holes were complete but no assays available at the time of the December interpretation. All eight holes and assays were used for the June 2005 model.

Previous GG5 model reports have documented an inconsistency in the position of geological boundaries between holes drilled from the East Decline and those drilled from within the Golden Gift Decline system. The SGM Survey Department located the source of this error below the 900m Level, confirming that the holes collared from the East Decline were correctly located. The error was located in the Golden Gift working area and the entire dataset for this area had a survey correction applied to Surpac, Minesight and Acquire data to rectify the problem.

3.1.2 Development:

Since December 2004 the final 60m of development has been completed on the 1045mRL Level between 290N and 293N, with face sketches incorporated into the interpretation. Statesman lode development on the 1065mRL Level had been completed by December 2004 with face sketches and available mapping incorporated into the interpretation. The final 60m of development on Empire lode on the 1065mRL Level was completed with face sketches and sludge drilling used to aid interpretation.

Statesman lode development on the 1085mRL Level was complete by December 2004 with available mapping and face sketches incorporated into the interpretation. Development on Empire Lode on the 1085mRL and 1105mRL Levels had not commenced at the time of interpretation for the June 2005 model.

Development on Statesman Lode on the 1105 and 1125mRL Levels progressed south from the Tarrango fault to 299.5N in May and June 2005 with available face sketches used in the interpretation. These face sketches were critical in resolving the geometry of the top of the internal basalt nose, which layed over to be sub horizontal where present in the 1125mRL Level development.

The majority of ore development conducted in 2005 has been fibre-creted to aid in timely delivery of subsequent stoping panels. However cross-cuts and limited areas of ore development have not been fibre-creted and have been mapped.

3.2 Data collection and validation

3.2.1 QA/QC:

Prior to February 2005 all drillcore was sent to Aminya laboratories in Ballarat for assaying. Ongoing QA/QC issues with Aminya led to a change in February whereby SGM Resource Definition and Grade Control drillcore has since been sent to Genalysis in Kalgoorlie for assay.

The general procedures for QA/QC are outlined below:

- Production diamond drill core is routinely Full Core sampled.
- Routine assaying of diamond drill core samples is undertaken by fire assay (50 gram charges) with an AAS finish. Assays are reported to an LLD of 0.01ppm. For sample reporting below LLD, a value of 0.5xLLD is used in resource estimation.
- Sample QA/QC procedure incorporates routine check assays including repeats (re-assay or repeat assay), duplicate (second sample taken after

pulverisation), splits (half coarse sample split, at ratio approx 1:10 samples), and standards.

- SGM Standard reference material has been prepared by Ore Research of Melbourne, from ore samples from Magdala Mine.
- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sample ID created during logging procedure.
- Analysis of received standard values and repeat and duplicate information is used to monitor the performance of the assaying process.

3.2.2 General Validation Checks:

All data is stored within the "acQuire" Database Management System which is structured on a Data Model which applies rules of relational databases to mining and exploration data. The primary key is HoleID.

Diamond drill hole collars are surveyed and single shot down hole cameras have been used to survey the hole path. Electronic single shot instruments are now used in preference to the Eastman mechanical cameras. Magnetic intensity is recorded and used to assess validity of record. Dip +/- 1.5° and azimuth +/- 3° is highlighted and assessed.

All logging of diamond drill core is undertaken using "Datcol" software, to a defined logging scheme. It is a barcode system, eliminating errors involved in keying. Logging is both quantitative and qualitative but in both cases only valid codes or values can be recorded.

Additional validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth
- Checks are performed for overlapping samples

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (eg Minesight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted.

After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

4 Geological Interpretation and Domaining

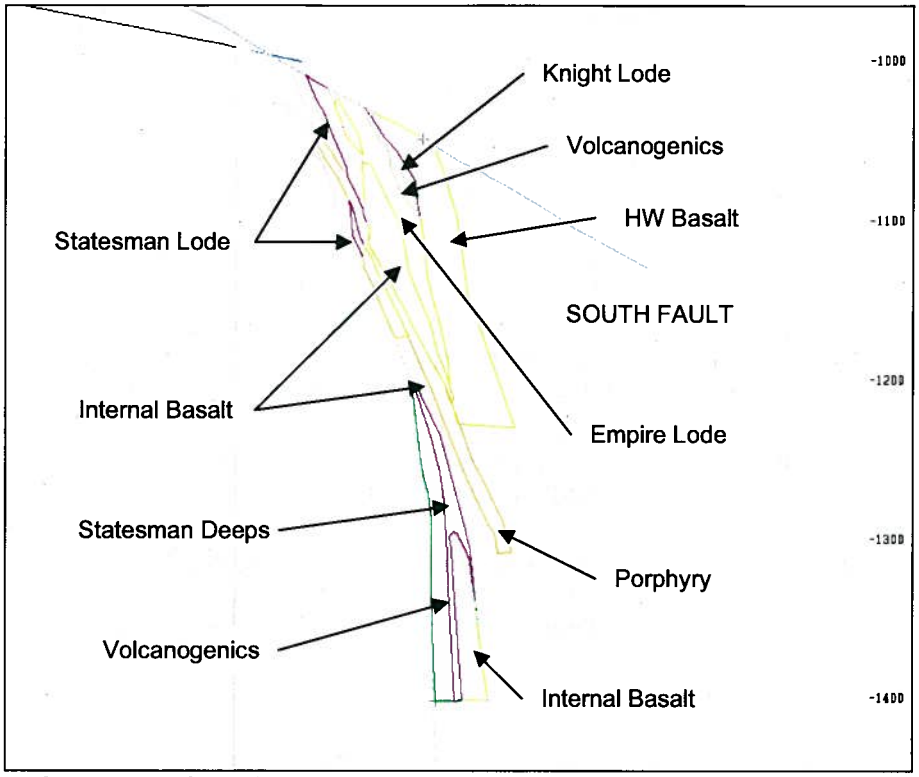

Figure 1: Cross section view at 5940N June 2005 Interpretation

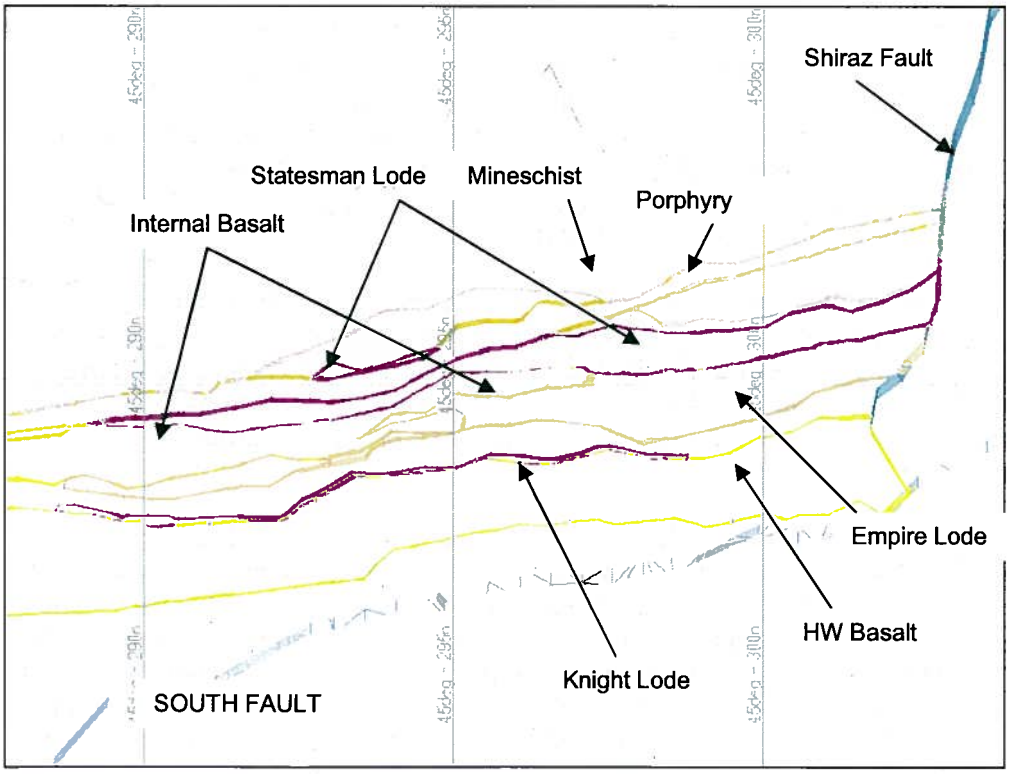

Figure 2: Plan View of June 2005 Interpretation at -1065mRL.

The June 2005 interpretation was carried out on screen in Minesight, with sectional interpretations carried out on 10m sections with interpretation strings snapped to

drillholes where available. Solids were constructed by then linking strings to form wireframe solids. Solids were overlapped, and then intersected against each other to form common surfaces. The interpretation was then sliced and validated on 10m paper sections where all available structural data could be analysed easily. No significant changes were required after this stage.

The following primary geological domains have been interpreted. Domain boundaries are based primarily on lithological or structural contacts.

Domain	Description
Statesman Lode	Lode located in footwall of Volcanogenics Unit (above and below Porphyry)
Empire Lode	Lode located in Hangingwall of Statesman Lode and Internal Basalt
Volcanogenics	Variably mineralised volcanogenics above Empire Lode
Knight Lode	Basalt contact lode on Hangingwall basalt
Porphyry	Porphyry cuts through Statesman Lode
Hangingwall Basalt	Basalt
Internal Basalt	Narrow Basalt flow, widening with depth
Footwall Basalt	Basalt flow in FW of Statesman Lode
Mineschist	Mineschist in footwall to Volcanogenics

4.1.1.1 Statesman Lode

The Statesman Lode is located at the footwall of the volcanogenic sequence, above a basalt or Mineschist contact. The hangingwall of the lode tends to be the internal basalt for most of the area modelled. Where there is no internal basalt the hangingwall position has often been logged as un-mineralised internal mineschist or Chloritic Pelite. This results in average lode true widths of between 7 and 12 metres. The lode is characterised by strong re-crystallised phyrrotite and coarse grained arsenopyrite and pyrite mineralisation. It has been logged as a stockwork zone however quartz vein percentages are typically logged between 5 and 20%. Some areas of Statesman Lode have experienced strong shearing, generally sub parallel to parallel to the strike and dip of the lode.

The interpretation has Statesman Lode cut by the Porphyry, with weaker mineralisation in the footwall to the Porphyry. Where Statesman Lode exists below the Porphyry it pinches out down dip due to widening of the internal and footwall basalts. The lower grade mineralization in Statesman Lode below the Porphyry is probably due the general grade trend within the lode with lower grades near the footwall.

Diamond drillholes drilled from the East Decline and Golden Gift South Decline test the full width of the structure, with most holes from the East Decline generally close to orthogonal to the structure. In contrast holes drilled from the 1018mRL cuddy are skewed along strike with some running down dip of Statesman Lode. Holes drilled from the 1045mRL Hangingwall cuddy range from orthogonal to Statesman Lode to skewed along strike. Many of the holes drilled from the 1060mRL, 107mRL, and 1080mRL cuddies and 1105mRL cross-cut are flat lying and perpendicular to the lodes. This may be significant given the high tenor of flat lying quartz veins intersected in the lode in some drives.

4.1.1.2 Empire Lode

Empire Lode is in a waterloo position between the Internal and Hangingwall basalts. As evident in Figure 1, Empire Lode continues up dip from this waterloo position to be situated in the immediate hangingwall of Statesman Lode.

Gold grade tends to be higher in the waterloo position and immediately adjacent to the nose of the internal basalt. Empire Lode mineralization is significantly variable compared to Statesman Lode. Gold grades and sulphide contents can vary greatly across the lode as well as along strike and up and down dip. Re-crystallised phyrrotite is by far the dominant sulphide present, with coarse grained Arsenopyrite associated with higher grade mineralisation. Empire Lode was often logged as a stockwork zone, however it has a lower quartz content than Statesman Lode, generally between 5 to 10%.

Drilling from the 1045 Hangingwall Cuddy targeting Statesman Lode passed through Empire Lode with surprising results between 297N and 299N below 1055mRL. Empire Lode in this area had broad drill coverage (approximately 25 x 30m) with no indication of significant mineralisation. However the steep holes from the 1045mRL returned full width grades of 5.9 to 11.3g/t in this area, delineating a flat lying ore shoot associated with the nose of the internal basalt. This ore shoot was followed up with a program of short holes from the 1065mRL Statesman Lode ore development. These holes closed off the shoot but also indicated the internal grade distribution to be erratic.

Empire Lode was reviewed with the aim of sub-domaining to improve the quality of variograms produced and tailor estimation parameters. This led to Empire Lode being split into three domains: Nose Domain, Open Flank Domain and Top Domain. The Nose Domain was characterised by highly variable, high grade mineralization situated over the top of the broad Internal Basalt nose with large widths and limited vertical extent. The mineralization has diffuse margins with the upper and southern margins defined broadly using a 3.5g/t cut-off.

The Open Flank Domain defined the area on the hangingwall of the Internal Basalt and is generally located in a waterloo position between the Internal and Hangingwall Basalts as seen in Figure 3. This domain is characterised by elevated grades near both basalt contacts with mineralization extending from footwall to hangingwall as the waterloo narrows. The Top domain is a low grade domain with no economic mineralization.

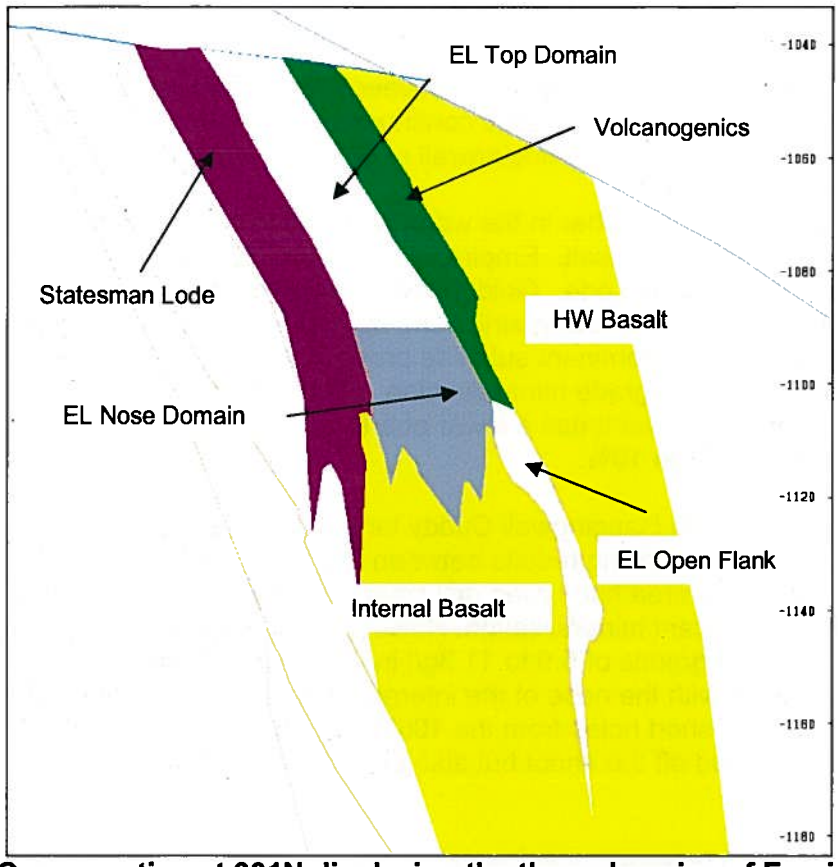

Figure 3: Cross section at 301N displaying the three domains of Empire Lode.

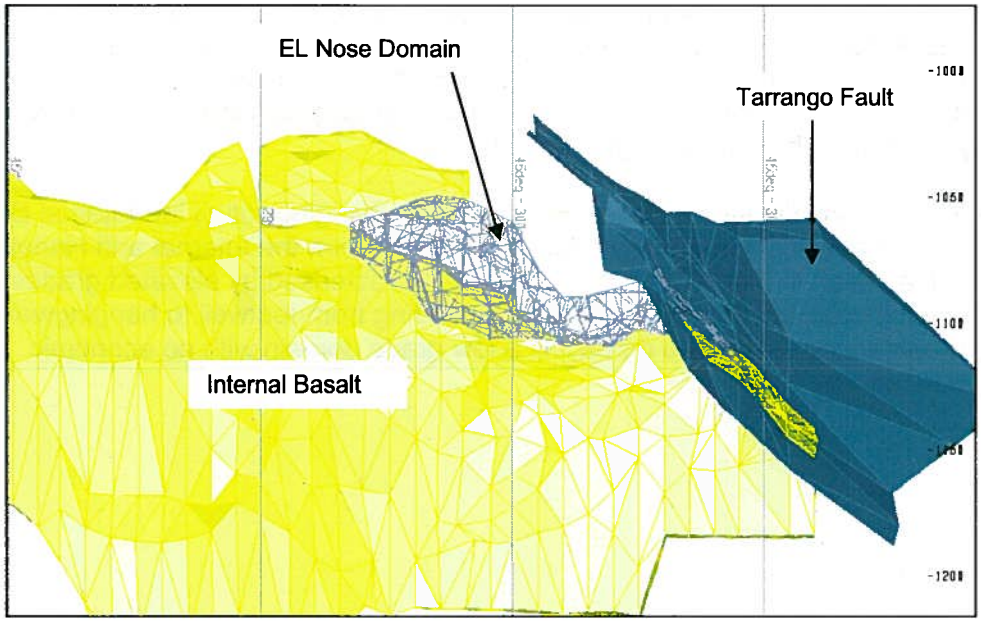

Figure 4: Long-section view displaying location of Nose Domain relative to the north plunging Internal Basalt nose.

4.1.1.3 Volcanogenics

The volcanogenics domain incorporates all volcanogenics in the hangingwall to Empire Lode and in the footwall to Knight Lode

4.1.1.4 Knight Lode

The Knight Lode appears to be a typical thin (<2m) basalt contact style lode. The main feature of Knight Lode is the consistent presence of Siliceous sediments on the contact with the Hangingwall basalt.

5 Drillhole Data

Type of data	Assay	Survey	Composite	Description
Diamond drillholes, coded by geological domain.	ggnt11.ddh	ggnt12.ddh	ggnt09.ddh	2m ddh composites, all geologic domains. Composites <0.9m excluded
	ggnt11.1m	ggnt12.1m	ggnt09.1m	1m composites used for Knight Lode domain. Composites <0.5m excluded
	ggnt11.5m	ggnt12.5m	ggnt09.5m	5m composites used for Empire Lode Nose domain. Composites <2.2m excluded

Table2. Drillhole Data Files

Diamond drill data was dumped from Acquire and loaded to Medsystem files ggnt11.ddh and ggnt12.ddh. Given the large proportion of existing drilling the drillhole coding was imported from December 2004 and all new holes were manually coded. Coding of all holes was then visually inspected in Minesight and manually re-coded where necessary.

Previous GG5 model reports have documented an inconsistency in the position of geological boundaries between holes drilled from the East Decline and those drilled from within the Golden Gift Decline system. The SGM Survey Department located the source of this error below the 900m Level, confirming that the holes collared from the East Decline were correctly located. The error was located in the Golden Gift working area and the entire dataset for this area had a survey correction applied to Surpac, Minesight and Acquire data to rectify the problem. Following this correction very few holes required manual coding due to not matching the geological interpretation.

6 Geological Coding

Drillholes, composites and block model utilize the same geological coding for domains. Codes values are:

Domain	Model Code	DXF Filename
Statesman Lode	600	states
Empire Lode	602	Empire
Knight Lode	604	Knight
Volcanogenics	501	Volcs
Porphyry	1	porph
Mineschist	1	Ms
Basalt	400	bas
Empire Nose Domain	Flag1=3	
Empire Top Domain	Flag1=1	
Empire Open Flank	Flag1=0	

Table 3. Geological Domains

7 Compositing

A 2m composite length was used which is suitable to the broad lode widths, except for Knight Lode and parts of Empire Lode. Knight Lode required a 1m composite length due to its narrow width, whilst the Empire Lode Nose Domain utilized a 5m composite due to the highly variable mineralization and broad widths above the basalt nose. It was decided not to use the 5m composites for the Open Flank domain due to the consistent presence of narrow basalt contact mineralization within the lode and the fact the waterloo pinches out down dip.

Investigation of the composite data indicated that a cut-off of composites greater than or equal to 0.9m in length was suitable for all domains except for Knight Lode which used a 0.5m cut-off and Empire Lode Nose Domain which used a 2.2m cut-off. These values resulted in metal losses due to compositing of 2.4% for Statesman Lode, 2.3% for Empire Lode Open Flank Domain, 1.8% for Empire Lode Nose Domain and 1.7% for Knight Lode. The Minesight software compositing routine has recently been improved to allow these small samples to be incorporated into the adjoining composite and this function will be used for the next resource estimate.

Domain	#	Total Length (m)	Mean Au	Std Dev.	CV	Cut value	# cut	Cut Mean	% metal cut
Statesman Lode	898	1732m	7.1	7.2	1.0	39	2	7.1	0.2%
Empire Lode Open Flank Domain	720	1394m	3.5	5.9	1.7	**34**	2	4.2	1.5%
Empire Lode Top Domain	173	338m	1.9	2.7	1.4	NA	0	1.9	0
Volcs	479	925m	0.5	1.3	2.7	NA	0	0.5	0

Table 4.June 2005 Composite Statistics for 2m composites, above 0.9m in length

Domain	#	Total Length (m)	Mean Au	Std Dev.	CV	Cut value	# cut	Cut Mean	% metal cut
Empire Lode Nose Domain	233	1076m	6.4	6.0	0.9	**27**	2	6.4	1.0%
Knight Lode	97	88m	5.0	6.8	1.4	**25**	1	4.8	4.3%

Table 5.June 2005 Composite Statistics for Empire Lode Nose Domain 5m composites, above 2.2m in length and for Knight Lode 1m composites above 0.5m Length

All individual drillhole samples from recently drilled holes have been weighed with a SG value calculated based on this weight, sample length and core diameter. Given the high sulphide content of Statesman Lode the relative SG values of composites for the geological domains were reviewed. It was decided to apply different SG values for each domain based on these statistics, rather than the standard SGM practice of applying a default SG of 2.85.

Domain	#	Min	Max	Mean	SD	Assigned SG
Statesman Lode	897	2.02	3.87	**2.90**	0.19	**2.9**
Empire Lode	1233	2.11	3.7	**2.88**	0.20	**2.85**
Knight Lode	88	2.16	3.86	**2.8**	0.29	**2.85**
Volcs	**448**	**2.17**	**3.54**	**2.75**	**0.17**	**2.75**
Basalt	783	2.23	3.82	**2.79**	0.16	**2.8**
Porphyry & Mineschist	**395**	**2.1**	**3.79**	**2.7**	**0.19**	**2.7**

Table 6. SG Value Composite Statistics for 2m composites. Note that values outside the 2 – 4 range were excluded.

8 Variography

Good normal downhole variograms were achieved for all domains. All variograms were modelled using Isatis software with normal variograms. The December 2004 Statesman Lode and Volcanogenics variograms were re-scaled and applied. The 5m composites from Empire Lode Nose Domain produced good, robust variograms in the orientation of the plunge of Internal Basalt. The 2m composites from the Empire Lode Open Flank Domain produced variograms with long flat ranges along strike and a reduced range up and down dip, reflecting the flat lying shoots that appear to occur in this domain.

Domain	Orientation of principal axis			Nugget	1st sill	2nd sill	Principal axis		Intermediate axis		Minor axis	
	Strike	Plunge	Dip	S0	S1	S2	R1	R2	R1	R2	R1	R2
Statesman Lode	315	0	-70 (E)	16.4	16.4	19.2	35	42	20	26	10	10
Volcanogenics	315	0	-65 (E)	0.6	0.3	0.7	26	90	26	50	2	10
Empire Lode Open Flank	315	0	-70 (E)	11	10	14.7	24	52	18	23	7	12
Empire Lode Nose Domain	315	-25(N)	0	13	10	12.5	24	52	24	52	7	15
Empire Lode Top Domain	315	0	-70 (E)	11	10	14.7	24	52	18	23	7	12
Knight Lode	315	0	-65 (E)	8	11.5	13	40	52	10	14	2	3

Table 7. Summary of Variography

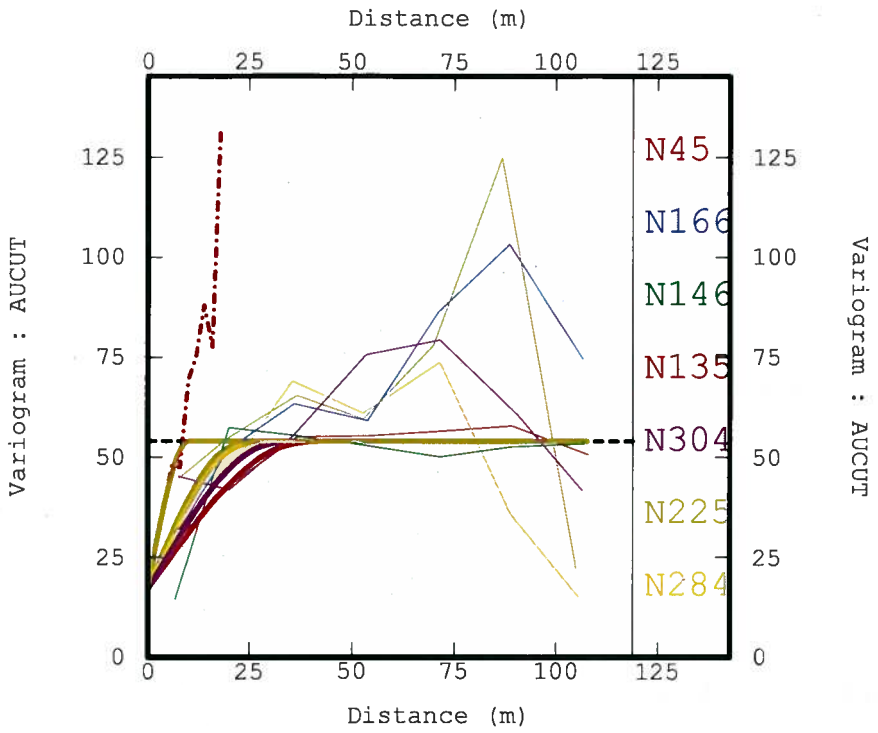

Figure 5: Statesman Lode Normal Variograms and variogram models. The variogram fan is generated in a plane formed by rotating East -45° around RL and then -70° (down) around the East axis. The solid red line represents the variogram in the plane of the mineralization. The dotted red line represents the variogram perpendicular to the plane of mineralization.

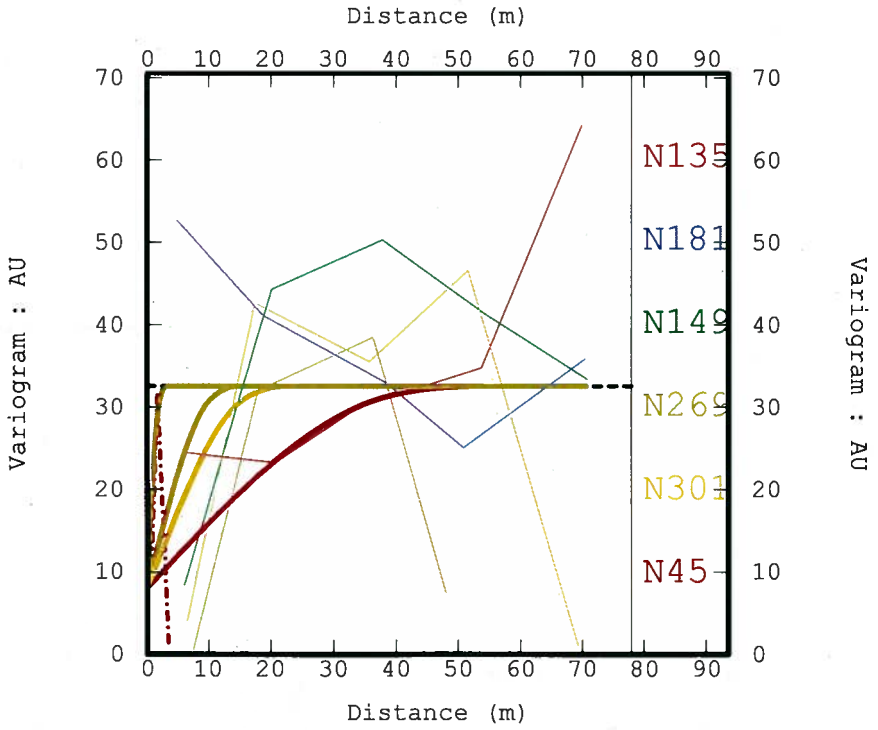

Figure 7: Knight Lode Normal Variograms and variogram models. The variogram fan is generated in a plane formed by rotating East -45° around RL and then -65° (down) around the East axis. The solid red line represents the variogram in the plane of the mineralization. The dotted red line represents the variogram perpendicular to the plane of mineralization

9 Block Model

Block model was coded directly from 3d geological solids. Geology code and % of block inside solid were stored. Each geological domain "family" has its own set of block model items – this is the simplest way to account for multiple geological domains within a single block in the underground situation. The alternative methodology of accounting for block partials in order of decreasing volume within a block is problematic underground. Coding was visually checked within Minesight.

Structure	Model Item	Code
Statesman Lode	SGEO	600
Volcanogenics	VGEO	501
Knight Lode	CGEO	604
Basalt	WGEO	400
Empire Lode	BGEO	602
Empire Nose Domain	BFLAG	3
Empire Top Domain	BFLAG	1
Empire Open Flank	BFLAG	0

Table 8. Domain codes

Block model is rotated into the plane of the structure, so that the rotated Z axis lies along strike, the rotated x axis lies up dip and the z direction is perpendicular to the orebody. Block dimensions are:

Strike	10.0m
RL	15.0m
Cross- Dip	5.0m

Block dimension has been determined with respect to data density, the variogram model and desired selectivity. Drilling density across the model area is generally to grade control standard (15 x 20m) so block dimensions are approximately 2/3 of the drilling spacing.

Block Model file – ggnt15.dat

Label	Min	Max	Prec.	Round	Description
TOPO	0	100	1	0.5	Default
Empire Lode Items					
B%	0	100	1	0.5	Empire % of block
BACCM	0	200	0.01	0.005	N/A Kriged accumulation
BAUBC	0	100	0.01	0.005	Kriged grade - Top Domain
BAUD	0	100	0.01	0.005	Kriged grade – all domains combined
BAUS	0	100	0.01	0.005	Kriged grade – Nose Domain
BCAT	0	10	1	0.5	N/A. Resource Category
BDIST	0	50	0.1	0.05	Distance to closest composite
BFLAG	0	10	1	0.5	Secondary code for EL Domain
BGEO	1	999	1	0.5	Geology code item
BSG	0	10	0.01	0.005	Bulk density
BSLRA	0	2	0.01	0.005	N/A Slope of regression - accumulation
BSLRD	0	2	0.01	0.005	Slope of regression - diamond
BSLRS	0	2	0.01	0.005	N/A Slope of regression - sludge
BTHK	0	999	0.01	0.005	N/A Thickness
BWGTA	0	2	0.01	0.005	N/A. Weight of mean from SK - accumulation

BWGTD	0	2	0.01	0.005	N/A. Weight of mean from SK - diamond
BWGTS	0	2	0.01	0.005	N/A. Weight of mean from SK - sludge
Statesman Lode Items					
S%	0	100	1	0.5	Statesman % of block
SAUD	0	100	0.01	0.005	Kriged grade - diamond data
SAUS	0	100	0.01	0.005	N/A. Kriged grade - sludge data
SCAT	0	10	1	0.5	N/A. Resource Category
SFLAG	0	10	1	0.5	
SGEO	1	999	1	0.5	Geology code item
SSG	0	10	0.01	0.005	Bulk density
SSLRD	0	2	0.01	0.005	Slope of regression - diamond
SSLRS	0	2	0.01	0.005	N/A. Slope of regression - sludge
SSPAR	0	999	0.01	0.005	Spare item
SWGTD	0	2	0.01	0.005	Weight of mean from SK - diamond
SWGTS	0	2	0.01	0.005	Weight of mean from SK - sludge
Knight Lode Items					
C%	0	100	1	0.5	Lode % of block
CAUD	0	100	0.01	0.005	Kriged grade - diamond data
CAUS	0	100	0.01	0.005	N/A. Kriged grade - sludge data
CCAT	0	10	1	0.5	Resource Category
CFLAG	0	10	1	0.5	
CGEO	1	999	1	0.5	Geology code item
CSG	0	10	0.01	0.005	Bulk density
CSLRD	0	2	0.01	0.005	Slope of regression - diamond
CSLRS	0	2	0.01	0.005	N/A. Slope of regression - sludge
CSPAR	0	999	0.01	0.005	Spare item
CWGTD	0	2	0.01	0.005	N/A. Weight of mean from SK - diamond
CWGTS	0	2	0.01	0.005	N/A. Weight of mean from SK - sludge
Volcanogenics Items					
V%	0	100	1	0.5	Lode % of block
VAUD	0	100	0.01	0.005	Kriged grade - diamond data
VAUS	0	100	0.01	0.005	N/A. Kriged grade - sludge data
VCAT	0	10	1	0.5	Resource Category
VFLAG	0	10	1	0.5	
VGEO	1	999	1	0.5	Geology code item
VSG	0	10	0.01	0.005	Bulk density
VSLRD	0	2	0.01	0.005	Slope of regression - diamond
VSLRS	0	2	0.01	0.005	N/A. Slope of regression - sludge
VSPAR	0	999	0.01	0.005	Spare item
VWGTD	0	2	0.01	0.005	N/A. Weight of mean from SK - diamond
VWGTS	0	2	0.01	0.005	N/A. Weight of mean from SK - sludge
Waste Items					
WAU	0	100	0.01	0.005	Default waste grade
WGEO	1	999	1	0.5	Geology code item
WSG	0	10	0.01	0.005	Bulk density
Extra items					
XTRA1	0	10	1	0.5	
XTRA2	0	999	0.01	0.005	

Table 9: Model Fields and descriptions

10 Resource Estimation

Resource estimation method used is Ordinary Kriging. For explanation of the appropriateness and implementation of this estimation method the reader is referred to the following QG reports:

Vann, J., 2001. Review of Geological Interpretation, Domain Decision-Making and Kriging Plan at the Stawell Gold Mine, Victoria, Australia. *Confidential consultant report by Quantitative Geoscience Pty Ltd. for Stawell Gold Mine.*

Bertolli, O., 2001. Proposed Improvements for In-situ Resource Estimates and Recoverable Resource Estimation at the Stawell Gold Mine, Victoria, Australia. *Confidential consultant report by Quantitative Geoscience Pty Ltd. for Stawell Gold Mine. September 2001*

Jackson. S, 2004. Comments on Resource Estimate at Stawell. *Confidential consultant report by Quantitative Geoscience Pty Ltd. for Stawell Gold Mine. August, 2004.* QG Project MPI 00420.

10.1.1.1 Quantified Kriging Neighborhood Analysis and Block Estimation

Selection of appropriate kriging plans builds on review of similar resource models and work carried out on Kriging Neighborhood Analysis. Five to seven blocks were analysed for each domain for this model with results correlating with previous analysis for these parameters.

The M624 run-files used in interpolation are attached in Appendix A.

10.1.1.2 Statesman Lode

Data Selection

Data selection is controlled using an isotropic ellipsoidal search of 80x80x25. A minimum of 10 composites, a maximum of 25 composites, a maximum of 6 composites per drillhole, and maximum distance to closest composite of 35m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 8. (file r624b.600)

10.1.1.3 Empire Lode Nose Domain

Data Selection

Data selection of 5m composites was controlled using an isotropic ellipsoidal search of 80x80x25. Soft boundaries with other Empire Lode domains were used, with 5m composites from all Empire Lode domains used in estimating the Nose Domain. A minimum of 11 composites, a maximum of 40 composites, a maximum of 7 composites per drillhole, and maximum distance to closest composite of 35m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 10. (file r624c.602)

10.1.1.4 Empire Lode Open Flank Domain

Data Selection

Data selection of 2m composites was controlled using an isotropic ellipsoidal search of 80x80x25. A soft boundary was used, with 2m composites in the Nose Domain used to estimate the Open Flank Domain. A minimum of 21 composites,

a maximum of 80 composites, a maximum of 7 composites per drillhole, and maximum distance to closest composite of 35m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 20. (file r624a.602)

10.1.1.5 Empire Lode Top Domain

Data Selection

Data selection of 2m composites is controlled using an isotropic ellipsoidal search of 80x80x25. A hard boundary was used against the Nose Domain to prevent the smearing of high grades above the ore shoot boundary. Data selection is controlled using an isotropic ellipsoidal search of 80x80x25. A minimum of 21 composites, a maximum of 80 composites, a maximum of 7 composites per drillhole, and maximum distance to closest composite of 35m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 20. (file r624b.602)

10.1.1.6 Volcanogenics

Data Selection

Data selection is controlled using an isotropic ellipsoidal search of 80x80x25. A minimum of 10 composites, a maximum of 45 composites, no maximum composites per drillhole was applied, and maximum distance to closest composite of 35m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 12. (file r624a.501)

10.1.1.7 Knight Lode

Data Selection (NB Parameters based on Dukes Lode in USF)

Data selection is controlled using an isotropic ellipsoidal search of 60x60x25. A minimum of 8 composites, a maximum of 32 composites, no maximum composites per drillhole was applied, and maximum distance to closest composite of 30m are used to control interpolation, along with a rotated quadrant search, with a maximum per quadrant of 8. (file r624a.604)

11 Validation

11.1 Test Kriged Blocks

Approximately 5 to 10 blocks were test kriged for each of the domains for each set of search parameters trialled. Test kriging generates the search ellipse, and reports the number of composites used, kriging weights, minimum and maximum distances to samples and the slope of regression. The search ellipse and weighting strings were then visualized in Minesight to ensure all suitable composites were used.

Slope of regression values appear closely related to the distance of the closest sample. Blocks at the periphery of the data set generally showed much lower slope of regression, and longer search distances to data.

12 Reporting

Reporting from the model should be carried out in S:\Mining-Geology\Golden Gift 5\Minesight\Geology\Current Model using ggnt15.dat. The domain boundary .dxf's and grade and slope fields are listed below.

Structure	Model Item	Code	Grade Field	SG	Slope Field	.dxf filename
Statesman Lode	SGEO	600	SAUD	SSG	SSLRD	States
Volcanogenics	VGEO	501	VAUD	VSG	VSLRD	Volcs
Empire Lode	BGEO	602	BAUD	BSG	BSLRD	empire
Knight Lode	CGEO	604	CAUD	CSG	CSRLD	knight
Basalts	WGEO	1	WAU	WSG		Bas
Mineschist & Por	WGEO	400	WAU	WSG		mspor

Table 10: Fields and files to use in sub-block reporting routine.

Group Minesight
Procedure pkcode.sgm
Name Subblock reserves reporting

Small discrepancies have been found using the sub-block routine so all resources reported here have been sourced using the Detailed Reserves (pitres.met) procedure in Compass software.

12.1.1.1 Reserves and Resources at 30-6-05

GG5	Tonnes	Grade	Oz's
Statesman	293,340	6.31	59,516
Empire	275,896	5.95	52,810
TOTAL	569,235	6.14	112,326

Table 11: June 2005 GG5 Proven and Probable Reserves by Lode.

All resources reported for GG5 in June 2005 are from the kriged resource model with nothing reported from the 2002 Conditional Simulation model. All of the GG5 resource has been converted to Measured and Indicated status and has been included in the mineable reserve, with no remaining resources exclusive of reserves. The GG5 Reserve at December 2004 was 551,278 @ 6.75g/t for 119,612Oz. The reduction in ounces is due to the mining of high grade Statesman Lode resources and replacing it with lower grade Empire Lode resources.

12.1.1.2 Modelled Resources prior to mining depletion

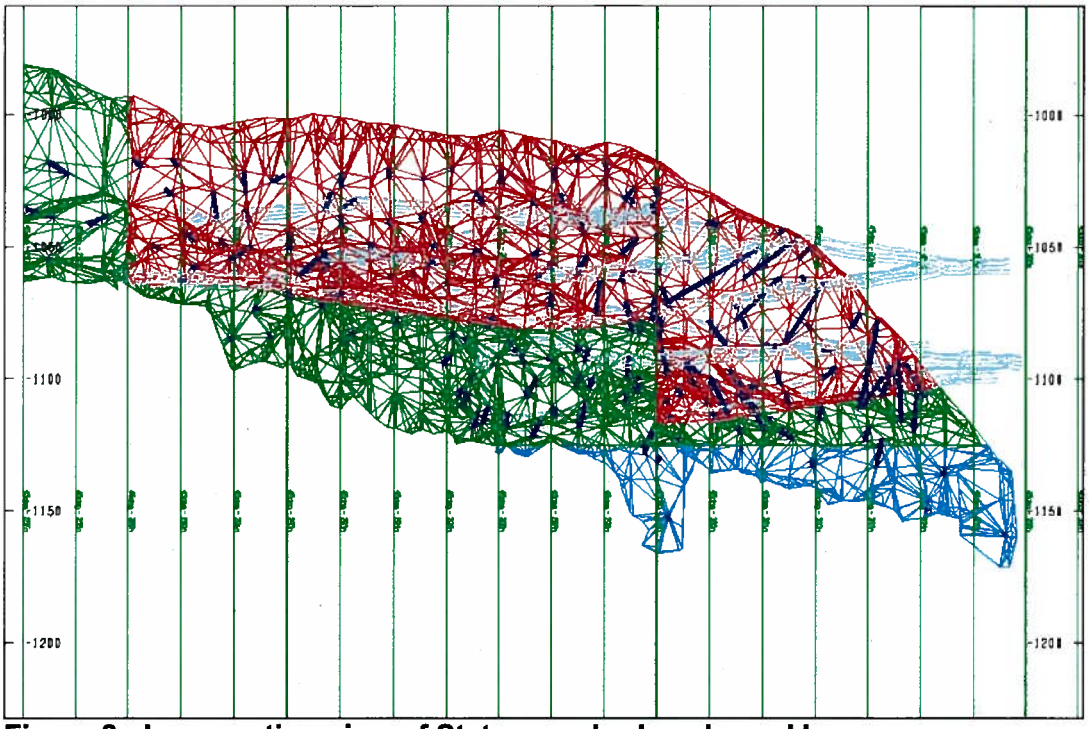

Figure 8: Longsection view of Statesman Lode coloured by resource classification (Blue = Inferred, Green = Indicated, Red = Measured) with drillhole intercepts in dark blue and development in light blue.

Statesman Lode's slope of regression values have benefited from the recent infill drilling. Measured Resources were delineated from well informed areas which have been developed. This includes all Statesman Lode resources from the 1045, 1065 and 1085 Levels and the 1105 and 1125 Levels north of 299.5N.

	Tonnes	Grade	Oz's	Slope
0.0g/t cutoff				
Measured	589,384	6.79	128,589	0.89
Indicated	205,407	5.32	35,100	0.78
Inferred	26,035	7.04	5,889	0.69
Total	820,826	6.43	169,578	
4.0g/t cutoff				
Measured	434,588	8.32	116,222	0.93
Indicated	115,666	7.75	28,824	0.85
Inferred	26,035	7.04	5,889	0.72
Total	**576,289**	**8.15**	**150,935**	

Table 12: June 2005 Statesman Lode resources (prior to mining depletion)

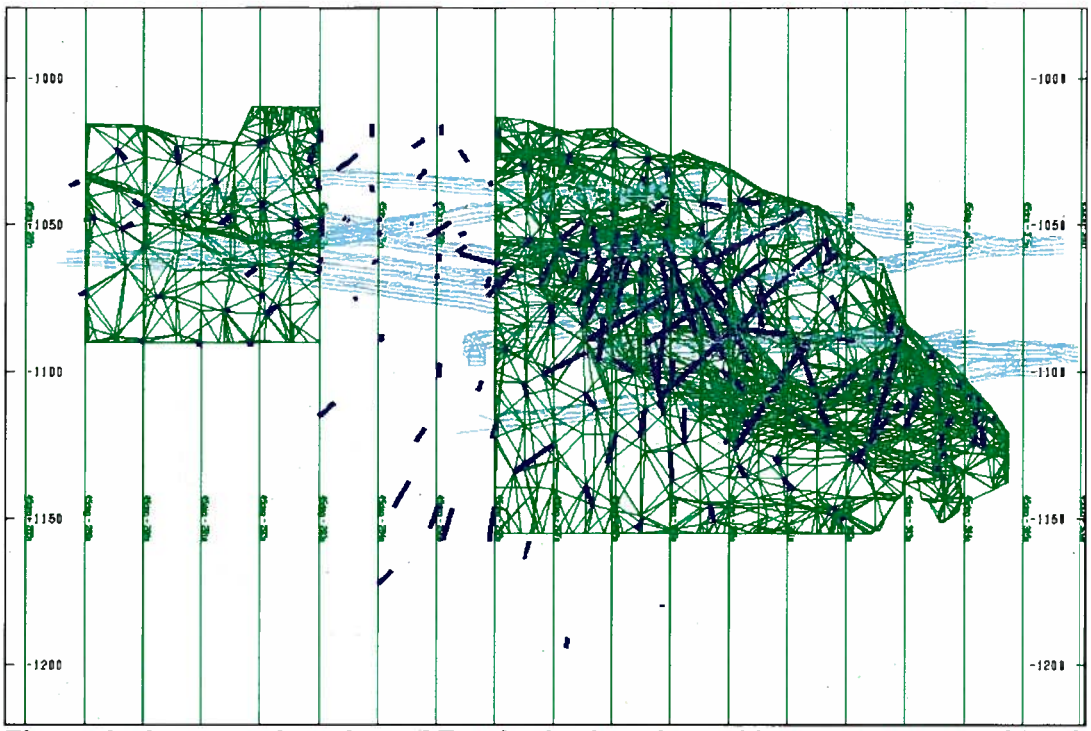

Figure 9: Longsection view of Empire Lode coloured by resource classification (Green = Indicated) with drillhole intercepts in dark blue and development in light blue.

Indicated Resources for Empire Lode were determined by evaluating drilling coverage and slope of regression values. The area between 293N and 296N is well informed but not classified, as material in the area is below the 4.0g/t cut-off grade. The proposed 1165 Level on Empire Lode was later included in the Indicated Reserve based on the high slope values and the additional information provided by hole MD4362 in the area, whose assays (4.8m @ 8.5g/t) were returned after the model was complete.

	Tonnes	Grade	Oz's	Slope
0.0g/t cutoff				
Indicated	665,555	4.53	96,912	0.78
Inferred	136,501	7.14	31,313	0.24
Total	802,056	4.97	128,225	
4.0g/t cutoff				
Indicated	354,654	6.24	71,162	0.75
Inferred	134,085	7.20	31,047	0.24
Total	**488,739**	**6.50**	**102,210**	

Table 13: June 2005 Empire Lode resources (prior to mining depletion).

	Tonnes	Grade	Oz's
4.0g/t cutoff			
Indicated	35,590	5.47	6,259

Table 14: June 2005 Knight Lode resources. Note these resources were to be classified as Indicated but are not economic to extract with the current mining plan so no resource was reported for Knight Lode.

Declustering Cell Size	Mean Au Statesman	Mean Au EL Nose Domain	Mean Au EL Open Flank	Mean Au Knight
5*10*15	6.7 to 6.9	5.9 to 6.2	4.4 to 4.5	4.6 to 4.9
10*20*30	6.3 to 6.8	6.3 to 6.7	4.3 to 4.6	4.1 to 4.6
30*30*30	5.9 to 6.3	6.5 to 7.0	4.1 to 4.7	4.2 to 4.4
Clustered Composites	7.1	6.4	4.8	4.9
Model (0.0g/t cut-off)	5.9	6.3	4.7	4.5

Table 15: Comparison of block model gold grades with de-clustered composite gold grades.

12.1.1.3 Comparison to Previous Model

	Tonnes	Grade	Oz's
0.0g/t cutoff			
Jun-05	1,604,948	5.57	287,309
Dec-04	1,378,347	5.66	251,000
Difference	226,601	4.98	36,309
4.0g/t cutoff			
Jun-05	986,455	7.58	240,494
Dec-04	849,075	7.61	207,632
Difference	**137,380**	**7.44**	**32,863**

Table 16: Comparison of Measured and Indicated resources from June 2005 and December 2004. Excluding Knight Lode.

The considerable diamond drilling into GG5 since December 2004 has converted virtually all the resource from December 2004 into Measured and Indicated status. The combined Measured, Indicated and Inferred resource for Statesman and Empire Lodes only increased by 11,329 Oz during this time.

Recent drilling has increased the confidence in the mineralization continuity in Empire Lode allowing a large increase in Indicated resources and improved conversion to Probable Reserve. All classifications of both lodes have higher slope of regression (confidence) values than for the previous model.

13 Reconciliations

	Tonnes	Grade	Oz's
Mill Reconciled			
Development	87,019	6.5	18,153
Stoping	192,220	6.8	42,148
Total	**279,238**	**6.7**	**60,301**

Table 17: GG5 Resources depleted from GG5 by all development and stoping to June 2005.

Table 18 displays the reconciliation of GG5 stope reserves and truck haul grades for all GG5 stopes completely mined to 30-06-05. Most of these reserves were evaluated against the December 2004 model with those mined in June against the June 2005 model.

14 Grade Shells

Grade shells, clipped against the geology solids, were generated for each of the surfaces. The following cutoff grades were used for grade shell generation:

Statesman Lode and Empire Lode	Volcanogenics
>4.0 g/t au	>2.0 g/t
>6.0 g/t au	
> 9.0 g/t au	

APPENDIX A: Estimation run files

```
MEDS-624V1  10=ggnt10.dat  9=ggnt09.ddh  15=ggnt15.dat;
MEDS-624V1  3=rpt624.600  30=states.var
** KRIGING of 3-D block values for SAUD **

RUN = RESET

USR =  /  Vincent Jun05 2m comps

IOP2 = 8. / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM       4=Split Quadrant.
COM       Note: Negative#=use rotated search.  By Default,
COM       octants & quadrants are defined using the project's axes.
IOP16= 25 / Max.# of composites for interpolating a block
IOP19= 6 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 80. / Limiting search distance
PAR7 = 35. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 5 7 5 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH 80. 80. 25. 315. 0. -70.

ITM1 = SAUD AUCUT CALC KRIGE
ITM2 = SSLRD Block error
ITM3 = SGEO Block limit
ITM4 = GEOCD RANGE 600. 600.
CMD = Block limit SGEO 600
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

```
MEDS-624V1  10=ggnt10.dat  9=ggnt09.ddh  15=ggnt15.dat;
MEDS-624V1  3=rpt624.502  30=empire.var
** KRIGING of 3-D block values for BAUD **

RUN = RESET

USR =  /  Vincent JUN05 2m comps EL OPEN FLANK

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 11 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM       4=Split Quadrant.
COM       Note: Negative#=use rotated search.  By Default,
COM       octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 7 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 80. / Limiting search distance
PAR7 = 35. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 5 7 5 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH 80. 80. 25. 315. 0. -70.

ITM1 = BAUD AUCUT CALC KRIGE
ITM2 = BSLRD Block error
ITM3 = BGEO Block limit
ITM4 = BFLAG Block limit
ITM5 = GEOCD RANGE 602. 602.
CMD = Block limit BGEO 602
CMD = Block limit BFLAG 0
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

```
MEDS-624V1  10=ggnt10.dat  9=ggnt09.ddh  15=ggnt15.dat;
MEDS-624V1  3=rpt624.502  30=empire.var
** KRIGING of 3-D block values for BAUBC **

RUN = RESET

USR =  /  Vincent JUN05 2m comps EL TOP DOMAIN

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 11 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM       4=Split Quadrant.
COM       Note: Negative#=use rotated search.  By Default,
COM       octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 7 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 80. / Limiting search distance
PAR7 = 35. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 5 7 5 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH 80. 80. 25. 315. 0. -70.

ITM1 = BAUBC AUCUT CALC KRIGE
ITM2 = BSLRA Block error
ITM3 = BGEO Block limit
ITM4 = BFLAG Block limit
ITM5 = FLAG1 RANGE 3. 3.
CMD = Block limit BGEO 602
CMD = Block limit BFLAG 3
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

```
MEDS-624V1  10=ggnt10.dat  9=ggnt09.5m  15=ggnt15.dat;
MEDS-624V1   3=rpt624.502  30=elnose.var
** KRIGING of 3-D block values for BAUS **

RUN = RESET

USR =   /  Vincent JUN05 5m comps ABOVE NOSE DOMAIN

IOP2 = 10 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 11 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM        4=Split Quadrant.
COM        Note: Negative#=use rotated search.  By Default,
COM        octants & quadrants are defined using the project's axes.
IOP16= 40 / Max.# of composites for interpolating a block
IOP19= 7 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 80. / Limiting search distance
PAR7 = 35. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 5 7 5 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH 80. 80. 25. 315. -25. 0.

ITM1 = BAUS AUCUT CALC KRIGE
ITM2 = BSLRS Block error
ITM3 = BGEO Block limit
ITM4 = BFLAG Block limit
ITM5 = GEOCD RANGE 602. 602.
CMD = Block limit BGEO 602
CMD = Block limit BFLAG 1
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

```
MEDS-624V1  10=ggnt10.dat  9=ggnt09.1m  15=ggnt15.dat;
MEDS-624V1  3=rpt624.604  30=knight.var
** KRIGING of 3-D block values for CAUD **

RUN = RESET

USR =  /  Vincent Dec04 1m comps

IOP2 = 8 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 8 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM       4=Split Quadrant.
COM       Note: Negative#=use rotated search.  By Default,
COM       octants & quadrants are defined using the project's axes.
IOP16= 32 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 100. / Limiting search distance
PAR7 = 30. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 5 7 5 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH 60. 60. 25. 315. 0. -65.

ITM1 = CAUD AUCUT CALC KRIGE
ITM2 = CSLRD Block error
ITM3 = CGEO Block limit
ITM4 = GEOCD RANGE 604. 604.
CMD = Block limit CGEO 604
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

```
MEDS-624V1  10=ggnt10.dat  9=ggnt09.ddh  15=ggnt15.dat;
MEDS-624V1  3=rpt624.501  30=volc2.var
** KRIGING of 3-D block values for VAUD **

RUN = RESET

USR =  /  Vincent Jun05 2m comps

IOP2 = 12 / Max # of composites per octant or quadrant
IOP3 = 1. / 1=3-D spherical search; 0=2-D search
IOP4 = 0 / 0=No geologic matching; 1=Match one item
IOP6 = 1 / 0=Use true dist.; 1=Use anisotropic dist.
IOP7 = 10 / Min # of comps for interp
IOP11= 00 / Row # for extended output
IOP12= -2 / 0=All; 1=Octant; 2=Quadrant search; 3=Split Octant;
COM       4=Split Quadrant.
COM       Note: Negative#=use rotated search.  By Default,
COM       octants & quadrants are defined using the project's axes.
IOP16= 45 / Max.# of composites for interpolating a block
IOP19= 0 / Max.# of composites per hole (DEFAULT=No limit)
IOP20= 8 / 0=Store variance; 1=STD DEV; 2=Rel. variance

PAR1 = 100. / Primary X-search distance
PAR2 = 100. / Primary Y-search distance
PAR3 = 100. / Primary Z-search distance
PAR4 = 65. / Limiting search distance
PAR7 = 35. / Max distance to closest point
PAR8 = 0. / Max distance to project single composite
PAR27= 5 7 5 / Block discretization in X,Y,Z

IOP17 = 0   /0=Ordinary kriging, 5=Simple
PAR30 = 0.   /Mean grade of the area to be simple kriged

CMD = SEARCH PROJECT
CMD = VGM PROJECT
CMD = SEARCH 80. 80. 25. 315. 0. -65.

ITM1 = VAUD AUCUT CALC KRIGE
ITM2 = VSLRD Block error
ITM3 = VGEO Block limit
ITM4 = GEOCD RANGE 501. 501.
CMD = Block limit VGEO 501
CMD = ELEV ZMID

I-O = 0 / Debug level
END
0 0 0 0 0 0
```

HOLE ID	FROM	GEOCD
MD2640W1	860.15	602
MD3289	386.85	602
MD3289W1	347.65	604
MD3301	293.95	602
MD3380	320.8	602
MD3384	380.1	501
MD3384W1	345.65	602
MD3408	394.25	600
MD3409	245.7	604
MD3410	448	600
MD3414	369	501
MD3415	266.85	604
MD3416	246.65	602
MD3798	252.55	602
MD3798W1	262.2	602
MD3799	246.5	602
MD3800	245.4	602
MD3801	254.4	602
MD3802	265.55	602
MD3803	220.6	604
MD3804	269.4	602
MD3804W1	262.15	602
MD3804W2	265.7	602
MD3823	247.3	602
MD3841	262.8	602
MD3842W2	259.3	602
MD3843	232.9	604
MD3871	249.6	602
MD3871W1	257.4	602
MD3891	249.5	604
MD3907	251.9	604
MD3908	265.8	602
MD3935	148.5	602
MD3937	121.4	602
MD3938	98.1	602
MD3939	173.7	604
MD3951	126.9	602
MD3952	251.65	604
MD3953	163.3	604
MD3954	146.65	602
MD3955	119.6	602
MD3956	146.2	602
MD3957	275.65	602
MD3962	165.5	602
MD3963	172.55	602
MD3964	259.45	604
MD3965	270.6	604
MD3966	283.25	604
MD3967	362.4	600
MD3968	38.6	602
MD3969	116.4	602
MD3970	98.6	602
MD3971	64.1	604
MD3972	33	602
MD3973	111	602
MD3974	71.1	602
MD3975	45.5	602
MD3976	58.55	602
MD3977	78.9	602
MD3978	89.5	604
MD3998	28.4	600
MD4005	311.65	501
MD4038	305.2	602
MD4052	74	604
MD4053	43.4	602
MD4054	38.8	600
MD4055A	85.45	604
MD4056	60.2	604
MD4057	65.1	604
MD4058	46.9	602
MD4059	80	604
MD4060	57.65	602
MD4061	70.3	604
MD4062	43.7	602
MD4063	76.9	604
MD4064	79	604
MD4065	58.6	602
MD4066	46.6	602
MD4067	72.65	604
MD4068	86.85	604
MD4069	65.5	604
MD4070	48.2	602
MD4071	73.85	604
MD4078	317.4	501
MD4080	91	604
MD4081	92.05	602
MD4082	114	604
MD4085	103.8	604
MD4088	101.1	602
MD4109	289.6	602
MD4141	16.8	604
MD4142	18.75	604
MD4142A	20.95	604
MD4143	17.7	604
MD4144	39.9	602
MD4145	46	602
MD4146	14.8	604
MD4148	52.2	602
MD4149	13.5	604
MD4150	52.9	602
MD4151	61.4	602
MD4152	18.15	604
MD4172	67.95	602
MD4173	42.6	602
MD4174	51.45	602
MD4175	15.6	604
MD4180	49.2	602

MD4181	42.3	604
MD4182	27	602
MD4183	41.9	604
MD4184	25.6	600
MD4185	33.6	602
MD4186	26.8	602
MD4187	15.7	602
MD4188	27.7	602
MD4190	33.3	602
MD4191	12.7	602
MD4192	50.65	602
MD4193	27	604
MD4194	162.9	602
MD4195	164.5	602
MD4196	156.4	602
MD4197	165	602
MD4198	155.1	602
MD4199	157.25	602
MD4200	111.5	604
MD4201	109.6	604
MD4202	101.75	604
MD4203	75.15	600
MD4204	86.2	602
MD4205	77.65	600
MD4206	114.4	604
MD4207	23.4	604
MD4208	38.2	604
MD4209	33	604
MD4216	300.75	602
MD4217	21.3	604
MD4218	22	604
MD4219	41.5	604
MD4220	17.4	604
MD4221	47.05	604
MD4222	17.3	604
MD4223	98.4	604
MD4224	107.2	604
MD4225A	116.3	604
MD4226	124.7	604
MD4236	101.9	600
MD4237	106.7	600
MD4238	97.35	600
MD4239	119.15	602
MD4240	141.6	602
MD4241	96.25	602
MD4242	139.7	602
MD4243	87.95	602
MD4244	97.65	602
MD4281	117.5	604
MD4283	114.7	602
MD4320	282.05	602
MD4321	6.2	600
MD4322	5.4	600
MD4324	22.45	602
MD4325	141.8	602
MD4326	145.8	602
MD4341	120.2	602
MD4342	104.3	602
MD4345	101.8	602
MD4346	201.9	600
MD4356	128.9	602
MD4357	81.4	600
MD4358A	83.1	600
MD4359	83.2	602
MD4360	96.15	602
MD4361	120.15	602
MD4370	160.6	602
MD4376	74.2	602
MD4377	64.3	602
MD4378	96.35	602
MD4379	92.8	602
MD4380	52	600
MD4384	74.4	602
MD4385	28.15	600
MD4386	72.3	602
MD4387	98.7	602
MD4388	49.95	602
MD4393	80.9	602
MD4443	116.1	602
MD4444	107.2	602
MD4445	106.6	602
MD4446	102.9	602
MD4447	101	602
MD4448	112.4	602

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX I

Golden Gift 5 Lower Geology and Resource Model Report

November 2007

Stawell Gold Mines PTY LTD

Golden Gift 5 Lower Geology and Resource Model Report

November 2007

SUMMARY

The Golden Gift 5 Statesman Deeps resource model covers the area from 291N (897500N) to 302N (898500N) above -1450mRL. The Statesman Lode in GG5 is bound by the Low Fault and is situated between the Internal Basalt and Mineschist contact. The Statesman Lode does not intersect Statesman Deeps at any point, despite being located in a similar structural position. The Statesman Deeps Lode has a significantly steeper dip.

At the time of modelling there were 119 diamond drillholes intersecting the orebody mainly on a 15 x 15m intercept grid. The current interpretation differs from the previous with the inclusion of two flat fault structures through the ore block. These offset across these faults is observed to be relatively minor. The southern end of the ore body is still open but recent drilling has delineated the southern ore geometries to a greater extent. The extra drilling has also identified a second ore zone at the northern end of GG5L which is located between the (internal) basalt hangingwall and the schist. This ore zone is referred to as the Open Flank lode and will be the focus of future drilling.

During 2007 a thorough QAQC program was implemented at SGM which highlighted that the global laboratory basis for 2007 was a negative 3% for gold. Due to the large increase in data for this model changes to the variography from the last model were noted.

The total Indicated Modelled Inventory at a cut off grade of 4.0g/t Au is 109,300oz. A listing of the global inventory at a series of cut-off gold grades for the previous model and the November Model Inventory is provided below.

Leviathan Resources - Stawell Gold Mines	GG5L
Resource Classification	**Modelled Inventory**
Mined Status	**Small amount depleted**
Other Restrictions	**AREA='GG5L'**
Filter Applied to Evaluation	**Not Applied**
Previous Model Name	**gg5l15.new**
New Model Name	**gg5l15.dat**

RESCAT	Cutoff (g/t)	PREVIOUS MODEL			NEW MODEL			Change			Change %		
		quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)
MEASURED INDICATED INFERRED	**0.0**	**425,700**	**10.0**	**136,300**	**950,400**	**5.3**	**161,300**	524,700	-4.7	25,000	123%	-47%	18%
	2.0	**425,700**	**10.0**	**136,300**	**562,500**	**8.2**	**148,000**	136,800	-1.8	11,700	32%	-18%	9%
	4.0	**420,900**	**10.0**	**135,700**	**456,800**	**9.4**	**138,700**	35,900	-0.6	3,000	9%	-6%	2%
	5.0	**412,800**	**10.1**	**134,500**	**409,400**	**10.0**	**131,700**	-3,400	-0.1	-2,800	-1%	-1%	-2%
	7.0	**372,600**	**10.6**	**126,500**	**313,700**	**11.2**	**113,400**	-58,900	0.7	-13,100	-16%	6%	-10%
	9.0	**271,000**	**11.5**	**100,100**	**216,900**	**12.7**	**88,500**	-54,100	1.2	-11,600	-20%	10%	-12%
MEASURED	**0.0**	-	-	-	-	-	-	0	0.0	0			
	2.0	-	-	-	-	-	-	0	0.0	0			
	4.0	-	-	-	-	-	-	0	0.0	0			
	5.0	-	-	-	-	-	-	0	0.0	0			
	7.0	-	-	-	-	-	-	0	0.0	0			
	9.0	-	-	-	-	-	-	0	0.0	0			
INDICATED	**0.0**	298,300	10.2	98,000	313,300	10.9	109,500	15,000	0.7	11,500	5%	6%	12%
	2.0	298,300	10.2	98,000	313,300	10.9	109,500	15,000	0.7	11,500	5%	6%	12%
	4.0	295,200	10.3	97,700	310,700	10.9	109,300	15,500	0.6	11,600	5%	6%	12%
	5.0	289,400	10.4	96,800	306,600	11.0	108,600	17,200	0.6	11,800	6%	6%	12%
	7.0	266,700	10.8	92,200	269,300	11.7	101,300	2,600	0.9	9,100	1%	9%	10%
	9.0	200,100	11.6	74,800	205,900	12.8	85,000	5,800	1.2	10,200	3%	10%	14%
INFERRED	**0.0**	127,400	9.3	38,300	637,000	2.5	51,800	509,600	-6.8	13,500	400%	-73%	35%
	2.0	127,400	9.3	38,300	249,100	4.8	38,500	121,700	-4.5	200	96%	-49%	1%
	4.0	125,700	9.4	38,100	146,100	6.3	29,400	20,400	-3.2	-8,700	16%	-34%	-23%
	5.0	123,400	9.5	37,700	102,800	7.0	23,100	-20,600	-2.5	-14,600	-17%	-27%	-39%
	7.0	105,900	10.1	34,200	44,500	8.5	12,100	-61,400	-1.6	-22,100	-58%	-16%	-65%
	9.0	70,900	11.1	25,300	11,000	10.1	3,600	-59,900	-1.0	-21,700	-84%	-9%	-86%
POTENTIAL	**0.0**	-	-	-	-	-	-	-	0.0	-			
	2.0	-	-	-	-	-	-	-	0.0	-			
	4.0	-	-	-	-	-	-	-	0.0	-			
	5.0	-	-	-	-	-	-	-	0.0	-			
	7.0	-	-	-	-	-	-	-	0.0	-			
	9.0	-	-	-	-	-	-	-	0.0	-			

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1 Introduction

Since the last Mineral Resource and Ore Reserve update the GG5 Lower area has been accessed by decline and is in ore development. As part of the preparation for mining, the diamond drill spacing has been progressively infilled to approximately 15m along strike by 15m up and down dip. In addition to this drilling, mapping information from development completed on the 1265mRL and 1285mRL levels has provided additional geological control for domaining.

This model update includes all of this data however has relied upon kriging neighbourhood evaluation that was completed for earlier models. Brief validation of this work has been carried out as part of this update to support the parameters used.

1.1 Location

The Golden Gift 5 Statesman Deeps resource model covers the area from 291N (897500N) to 302N (898500N) above -1450mRL. The Statesman Lode in GG5 is bound by the Low Fault and is situated between the Internal Basalt and Mineschist contact. The Statesman Lode does not intersect Statesman Deeps at any point, despite being located in a similar structural position. The Statesman Deeps Lode has a significantly steeper dip.

At the time of modelling there were 119 diamond drillholes that have been coded as part of the geological modelling process. These drillholes are mostly on a grid of 15m along strike and 15m up and down dip although on the northern and southern limits of the orebody this spacing increases to approximately 30m along strike and 30m up and down dip.

1.2 Geology

1.2.1 Regional setting

The GG5 Lower ore body sits within the lower reaches of the Golden Gift mineralised system on the western flank of the Magdala Basalt dome, between the -1225 and -1400RL. Spatially the ore block is located directly beneath GG5 (290N to 310N on the 45° mine grid) as shown in Figure 1, and is separated from the main G5 ore body above by an East dipping - NW trending porphyry.

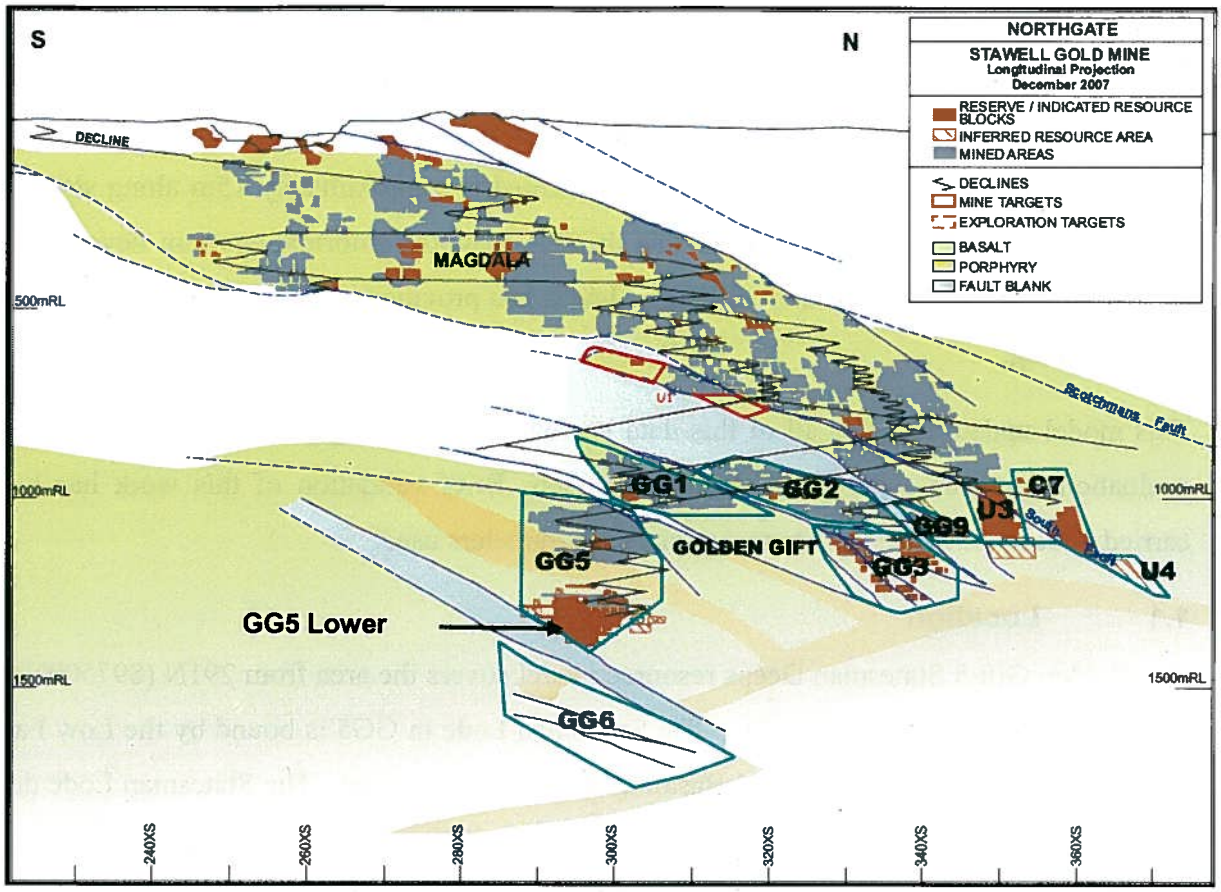


Figure 1 Long projection showing the location of the GG5 Lower area relative to other Stawell orebodies.

1.2.2 Local setting and geology

Geologically, GG5 Lower shares the same characteristics as the other Golden Gift orebodies, comprising a main sub-vertical ore lode adjacent to a basalt hangingwall contact; a mineschist footwall; and a low-grade volcanogenic package between the ore and the schist. A detailed cross section showing the relationship between the various geological components is shown in Figure 2. The main Statesman ore lode presents as a high grade waterloo structure hosted by "internal" basalt noses along the hangingwall basalt contact – similar to the geological setting encountered in GG5 upper. Both at depth and to the north, this waterloo pinches out as the internal basalt bodies join with the main hangingwall basalt. Peripheral to the main orebody, is a second zone of mineralisation that occurs as a skin between the (internal) basalt hangingwall and the schist (Open Flank lode).

Unique to GG5 Lower is an unmineralised footwall zone between the low-grade volc and the true mineschist, comprising a bleached sedimentary unit intercalated with altered, sheared basalt. (refer to Figure 2).

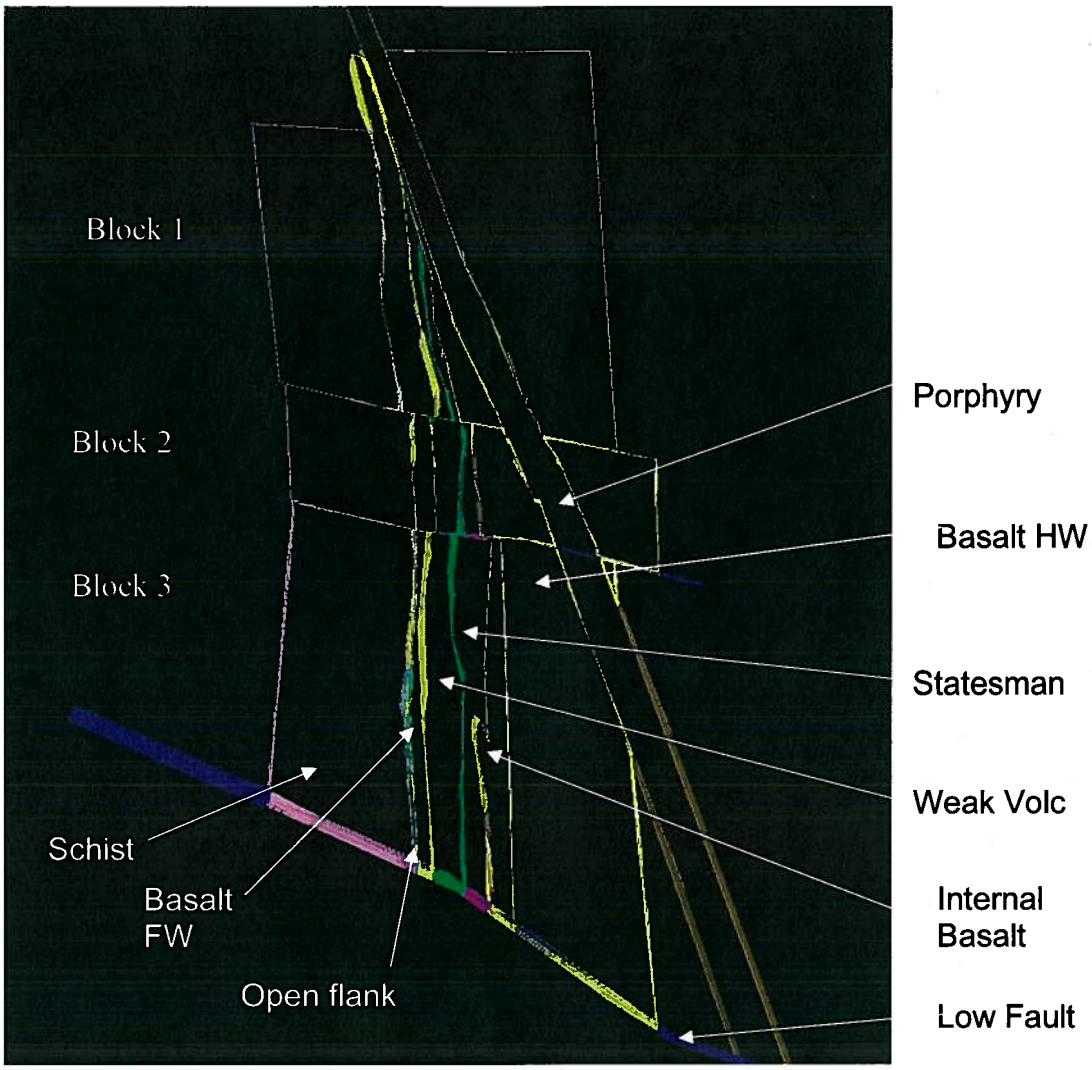


Figure 2 Typical cross-section through GG5L showing current interpreted geological setting.

Structurally, GG5 Lower is relatively simple, containing only one major fault – the easterly dipping Low Fault which cuts the lode off at depth (the offset portion of which forms the GG6 lode to the east). A futher two minor faults (running sub-parallel to the Low fault) have been interpreted, dividing the orebody into three blocks. The offsets observed across these faults from drilling to date are relatively minor.

1.2.3 Mineralisation styles and domaining

The mineralised zones of GG5 Lower have been divided into three separate lodes, each of which exhibit distinct mineralisation styles: Statesman Lode (Volc (statesman) fnl, domain 601); Weak Volc (Weak Volc fnl, domain 501); and Open Flank (Volc (open flank) fnl, domain 502).

Statesman Lode encompasses the highest grade material adjacent to the basalt hangingwall. In drill core, this presents as wide zones of green-grey stockwork volc with strong silica, chlorite

and sulphide alteration (recrystallised pyrrhotite and cubic arsenopyrite). It is visually distinctive from the dark green weak volc – which typically exhibits stronger chlorite and weaker sulphide alteration (weak pyrrhotite veining). In contrast, the Open Flank ore is characterised by narrow, sheared zones of grey-green, silica altered volc, generally with weaker chlorite alteration than both the statesman and weak volc lodes.

All three lodes have been further divided into three sub-domains, bounded by the Low fault and the two sub-vertical, easterly dipping faults, which are interpreted to cut the orebody into three blocks. While offsets across these faults appear to be minimal, they do represent potential grade boundaries, as well as effecting slight changes in the orientation of the mineralised surface within each domain.

1.2.4 Further Work

Ongoing grade control diamond drill programs into the GG5 Lower area will provide greater levels of confidence in regards to the presence and orientation of minor internal faults, and if present, better define their role as grade boundaries. Lower angle drill holes will also provide more reliable ore widths and help to enhance structural and geometrical understanding of the orebody at depth thus leading to better understanding of the controls on mineralisation.

Additional drilling is planned for the Open Flank north of 300N. Current drill coverage of the Open Flank is sparse and does not indicate strong continuity of grade along strike or down dip.

2 Drilling, Logging and Sampling

Drilling for the definition of the GG5 Lower area commenced in 2000 with exploration of the Golden Gift area identifying mineralised intercepts in drillholes MD 2353W1 and MD2353W3. Subsequent drilling has been in several phases that have been completed over the period 2004 – 2007. This drilling has progressively increased the drilling density within the project area as commitments to mine this section deposit area have been made.

Figure 3 shows the number of drillholes completed by period. The vast majority of drilling has been completed in 2007 which is predominantly grade control drilling.

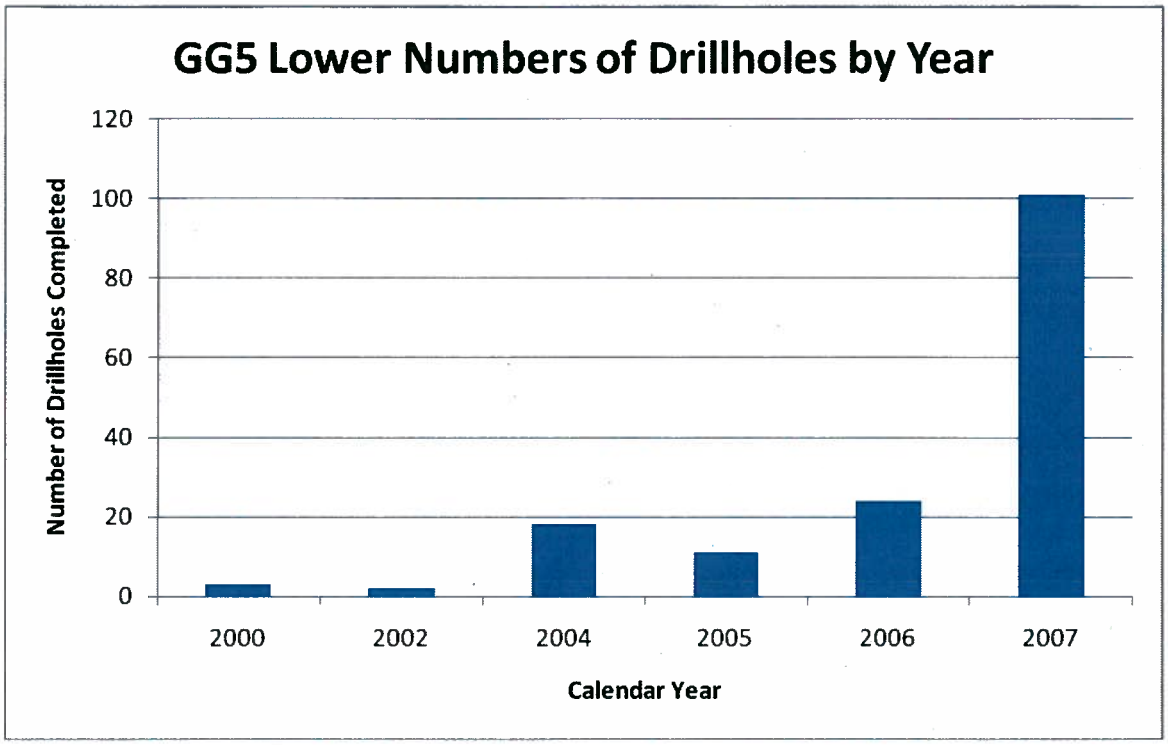


Figure 3 Plot showing the number of diamond drill holes drilled into the GG5 area by year

Following the commitment to access this area with the extension of the GG5 decline the mine geology team have completed extensive grade control diamond drill programs from the diamond drill cuddies located off the main decline. Much of this drilling has been conducted during 2007.

Initial exploration and resource definition diamond drill programs have been completed using NQ2 drillcore sizes recovered by wireline methods. Grade control diamond drill programs have been completed with LTK60 sized diamond drill core. Drilling has been conducted by contract diamond drill personnel using modern LM75 and LM90 style diamond drill rigs. The drilling has been conducted under the supervision of SGM personnel who retain the responsibility for setting out and monitoring the drill hole path.

Drill core is places in core trays provided by SGM personnel with the down hole depths of every core run completed marked on specially provided core blocks by the drill crews.

A regime of regular rig audits and inspections are also used to assist with maintaining the high level of core presentation and sample quality.

2.1 Logging

All diamond drill core is logged by the site geological teams using a standardised logging legend. The data is captured electronically at the point of collection using a barcode logging "Datcol" software system. This system was developed on site in the mid 1990's and has remained the standard process since that time with the key tables for lithology, alteration, and structure and geotechnical information are populated during the logging process.

As part of the standard SGM geological procedures all core collected from diamond drill holes are photographed and a complete record of core photographs are available to assist in the geological interpretation process.

Core recovery in the GG5 Lower area is very good and generally 100% of the core is recovered, particularly within and adjacent to the ore zones. Where core is lost it is usually associated with faulting and in the case of the GG5 Lower area the Low Fault. Lost core is identified in the logging as "LOST" and as such there are very few if any assay intervals utilised in the estimate where core recovery is less than 100%.

2.2 Sampling and Sample Preparation

Sampling of diamond drillcore follows one of two methods the details of which are shown in flow sheet format in Figure 4.

- Resource Definition – NQ2 drill programs
    - Core is logged and geological derived intervals are marked up for sampling
    - Sample intervals are matched to geological boundaries are fall within the range of 0.15m – 1.5 m. The average sample interval is approximately 1.0m
    - For Resource Definition drilling programs 1 in 5 holes is cut with a diamond saw prior to sampling and one half of the core sent for assay. The remaining half core is retained as a record within the core library. All other holes are sampled as whole core which is sent for sample preparation and assay as per the flowsheet shown in Figure 4.
- Grade Control – LTK60 drill programs
    - Core is logged and geological derived intervals are marked up for sampling

- Sample intervals are matched to geological boundaries are fall within the range of 0.15m – 1.5 m. The average sample interval is approximately 1.0m
- For grade control drilling programs drill holes are sampled as whole core with the entire sample sent for sample preparation and assay as per the flowsheet shown in Figure 4.

Sample preparation prior to assay has been completed on site at the SGM laboratory. By retaining responsibility for this work through the existing site based facility SGM has flexibility in sending the pulps only to a variety of assaying laboratories and also retain the coarse rejects on site for ongoing metallurgical testwork programs.

2.3 Assaying and QA/QC

Prior to February 2005 all drillcore was sent to Aminya laboratories in Ballarat for assaying. Ongoing QA/QC issues with Aminya led to a change in February 2005 whereby SGM Resource Definition and Grade Control drillcore has since been sent to Genalysis in Kalgoorlie, Amdel in Adelaide and in recent times to ALS laboratories in Orange NSW.

The general flowsheet for the sample preparation and assaying is shown in figure 4:

- Production diamond drill core is routinely Full Core sampled with one in five Resource Definition sent as half core.
- Routine assaying of diamond drill core samples has been undertaken utilising fire assay methodologies (recently changed from 50gram to 30 gram charges) with an AAS finish.
- Assays are reported to an LLD of 0.01ppm. For sample reporting below LLD, a value of 0.5xLLD is used in resource estimation.
- Sample QA/QC procedure incorporates routine check assays including repeats (re-assay or repeat assay), duplicate (second sample taken after pulverisation), splits (half coarse sample split, at ratio approx 1:10 samples), and standards.

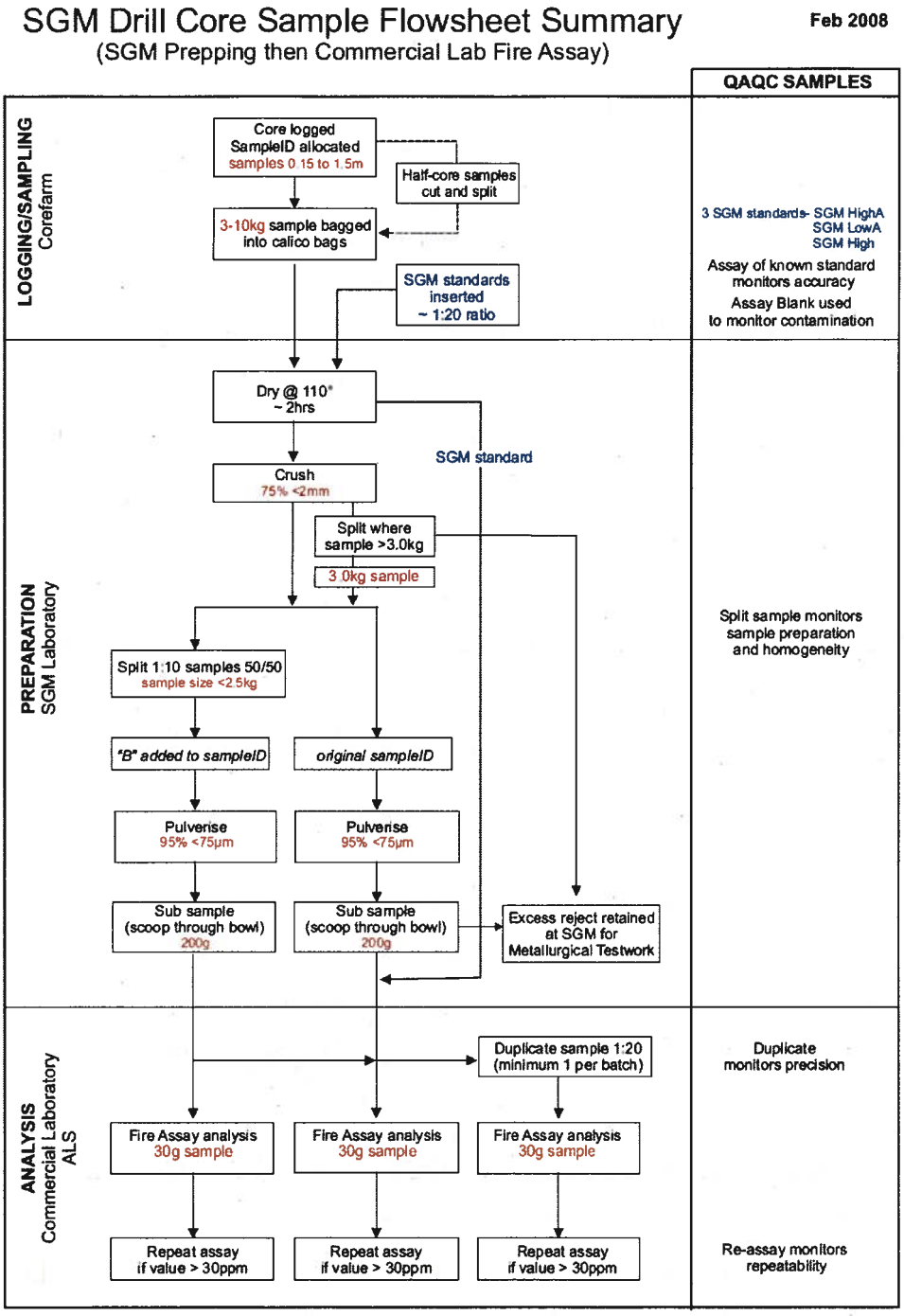


Figure 4 Stawell Gold Mines drill core sample preparation, assay and QA/QC flowsheet.

- SGM Standard reference material has been prepared by Ore Research of Melbourne, from ore samples from Magdala Mine. Three carefully prepared and certified standards are currently being utilised – SGM High A, SGM Low A, and SGM High. Certification certificates are available in SGM records.

- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sample ID created during logging procedure.
- Analysis of received standard values and repeat and duplicate information is used to monitor the performance of the assaying process.

2.3.1 Assay QA/QC data

For this update and analysis of the QA/QC data returned for the period 1 January 2007 to 30 December 2007 has been completed. This analysis is not specifically for the GG5 Lower drilling but encompasses all QA/QC data returned to SGM during that period and serves to demonstrate a responsible and ongoing approach to managing assay data quality is maintained at SGM.

A range of checks are in place to monitor the QA/QC of the Stawell data set. The first checks are regular submission of standards and blanks at the sampling stage (refer to Figure 4). This is followed by the use of splits of the crushed stage as well as the use of laboratory splits and assay duplicates and repeats (refer to Figure 7 and Figure 8). Monitoring of these checks are done within two days of the sample batch return and actioned generally no later than sevens days after the return date. Any actions taken during the monitoring process are recorded in the on site QA/QC diary which was set up in September 2007 (see Figure 6). When monitoring theses checks the following guidelines are followed (full flow sheet shown in Figure 5):

- If the standard assay is outside ±4 standard deviations from the known value then ~50% of that batch is automatically repeated. Entry is made into the on site QA/QC diary to indicate which sample IDs have been repeated.
- Repeated assays are compared to initial assays with any action to be taken entered into the on site QA/QC diary to indicate what actions respond to Batch and sample IDs.

During 2007, 197 initial assays where replaced by repeat assays as a result of the QA/QC monitoring process.

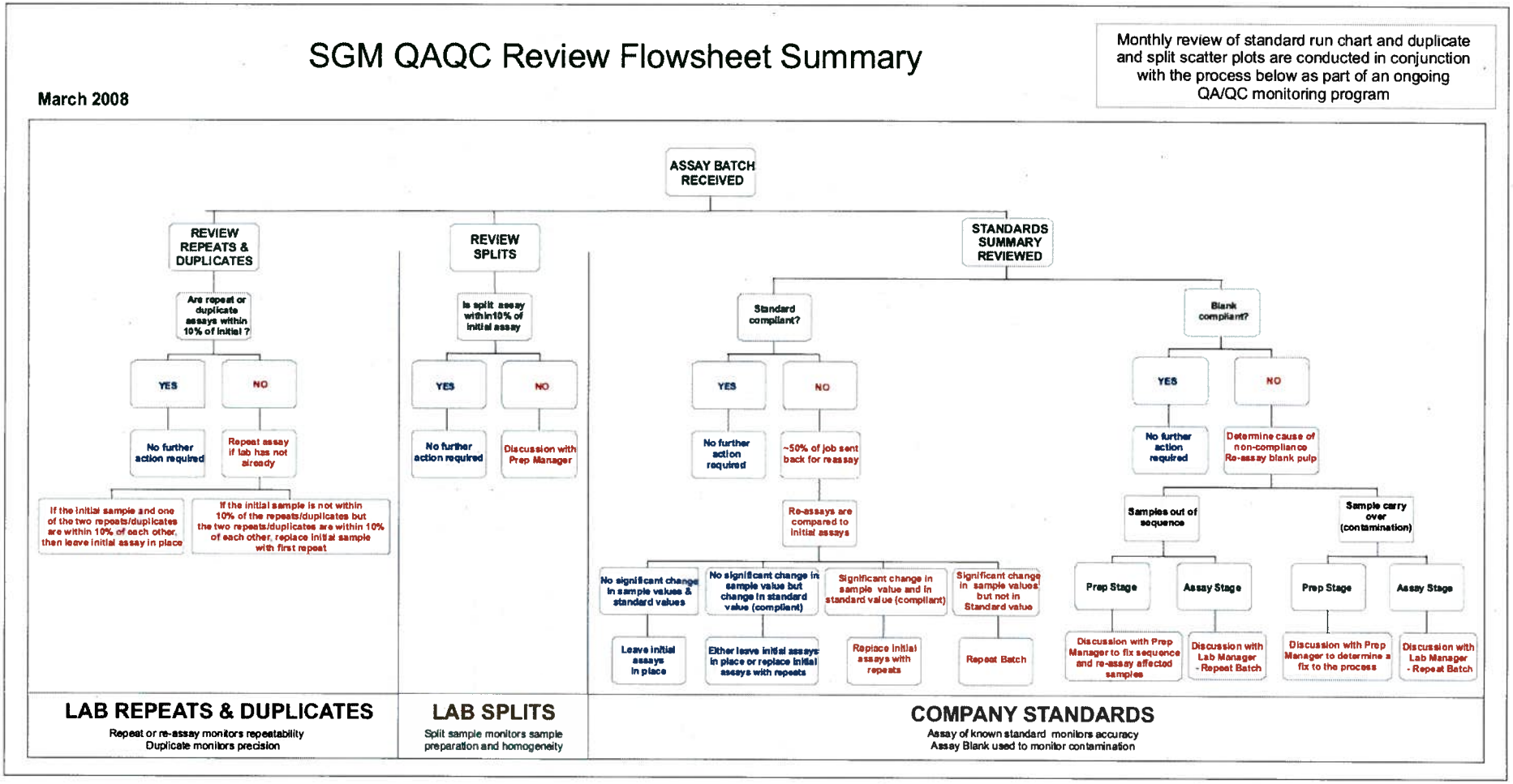


Figure 5 SGM QA/QC review flow sheet

Stawell Gold Mines

QAQC Request Record

Mine lease U/G drilling

Date Re-assay request	Requested by	HoleID	Orebody	Non-compliant sampleID	Lab Name	Lab Job No.	Repeated samples	Date Re-assays reported	Repeats Lab Job No.	Date loaded to DB	Reviewed by	comments	Action requested	Actioned by
?	DH -> JC	MD5062		MD5062-022/022B	ALS	OR07093156	MD5062-020 to -023	06-Sep-07	OR07098050	06-Sep-07		diff between sample and split 2.03/26.3		
18/09/2007	DH -> JC	MD5090		MD5090-042	ALS	OR07099872	MD5090-034 to -047	26-Sep-07	OR07107140	05-Oct-07	DH	non-compliant standard	Change database values to re-assay	CD
?	DH -> JC	MD5089		MD5089-021	ALS	OR07102944	MD5089-016 to -028	27-Sep-07	OR07107141	05-Oct-07	DH	non-compliant standard	Change database values to re-assay	CD
12/10/2007	DH -> JC	MD5116		MD5116-008	ALS	OR07113517	MD5116-004 to -011	21-Nov-07	OR07109857	26-Nov-07		non-compliant standard		
12/10/2007	DH -> JC	MD5116		MD5116-053	ALS	OR07113517	MD5116-050 to -056	21-Nov-07	OR07109857	26-Nov-07	DH	non-compliant standard	Change database values to re-assay	CD
12/10/2007	DH -> JC	MD5111		MD5111-012	ALS	OR07112313	MD5111-005 to -015	21-Nov-07	OR07109858	26-Nov-07	DH	non-compliant standard	Change database values to re-assay	CD
22/10/2007	DH -> JC	MD5100		MD5100-021	ALS	OR07106068	MD5100-015 to -030	21-Nov-07	OR07116775	26-Nov-07	DH	non-compliant standard. Assays repeated MD5096-014 to -028	Change database values to re-assay	CD
23/10/2007	DH -> JC	MD5122		MD5122-020	ALS	OR07117694	MD5122-015 to -024	26-Nov-07	OR07116776	26-Nov-07	DH	Assays repeated MD5110-014 to -022	Leave first assay batch as priority 1	
24/10/2007	DH -> JC	MD5124		MD5124-018	ALS	OR07116740	MD5124-008 to -019	21-Nov-07	OR07116777	26-Nov-07	DH	Assays repeated MD5114-008 to -019	Leave first assay batch as priority 1	
05/11/2007	DH -> JC	MD5132		MD5132-021	ALS	OR07123244	MD5132-001 to -068	21-Nov-07	OR07129002	26-Nov-07	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5142		MD5142-028	ALS	OR07139007	MD5142-017 to -030	17-Dec-07	OR07143758	09-Jan-08	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5152		MD5152-015	ALS	OR07139007	MD5152-003 to -017	17-Dec-07	OR07143758	09-Jan-08	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5158		MD5158-019	ALS	OR07139007	MD5158-015 to -024	17-Dec-07	OR07143758	09-Jan-08	DH		Leave first assay batch as priority 1	–
03/12/2007	DH -> JC	MD5145		MD5145-022	ALS	OR07139008	MD5145-017 to -023	17-Dec-07	OR07143757	09-Jan-08	DH	holeID is MD5145 but samples for repeat are MD5143-017 to -023	Leave first assay batch as priority 1	
03/12/2007	DH -> JC	MD5153		MD5153-021	ALS	OR07139008	MD5153-016 to- 028	17-Dec-07	OR07143757	09-Jan-08	DH		Change database values to re-assay	CD
03/12/2007	DH -> JC	MD5135		L039800	ALS	OR07135650	L039795 to L039809	17-Dec-07	OR07143759	09-Jan-08	DH		Change database values to re-assay	
05/12/2007	DH -> JC	WD164		L039362	ALS	OR07127816	L039358 to L039370							
05/12/2007	DH -> JC	WD119		L039423	ALS	OR07127816	L039418 to L039430							
05/12/2007	DH -> JC	MD5131		MD5131-012	ALS	OR07127621	MD5131-10 to -028							
13/12/2007	DH -> JC	MD5155		MD5155-042	ALS	OR07142500	MD5155-033 to -045	18-Dec-07	OR07147701	09-Jan-08	DH		Change database values to re-assay	CD

Figure 6 QA/QC diary entries for 2007

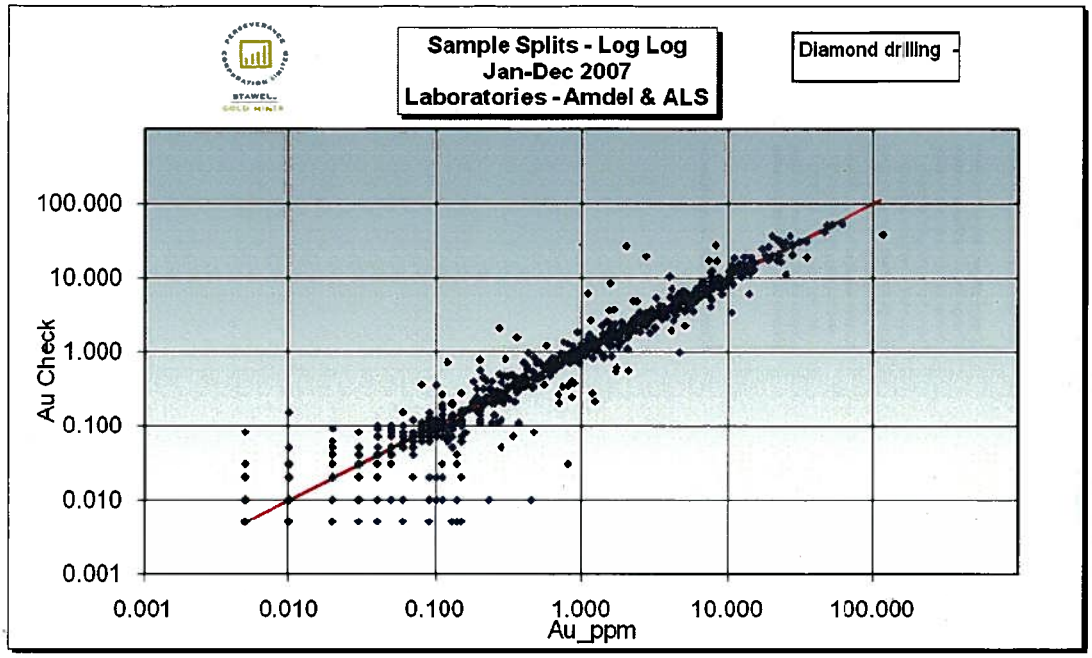


Figure 7 Sample Splits comparison for 2007

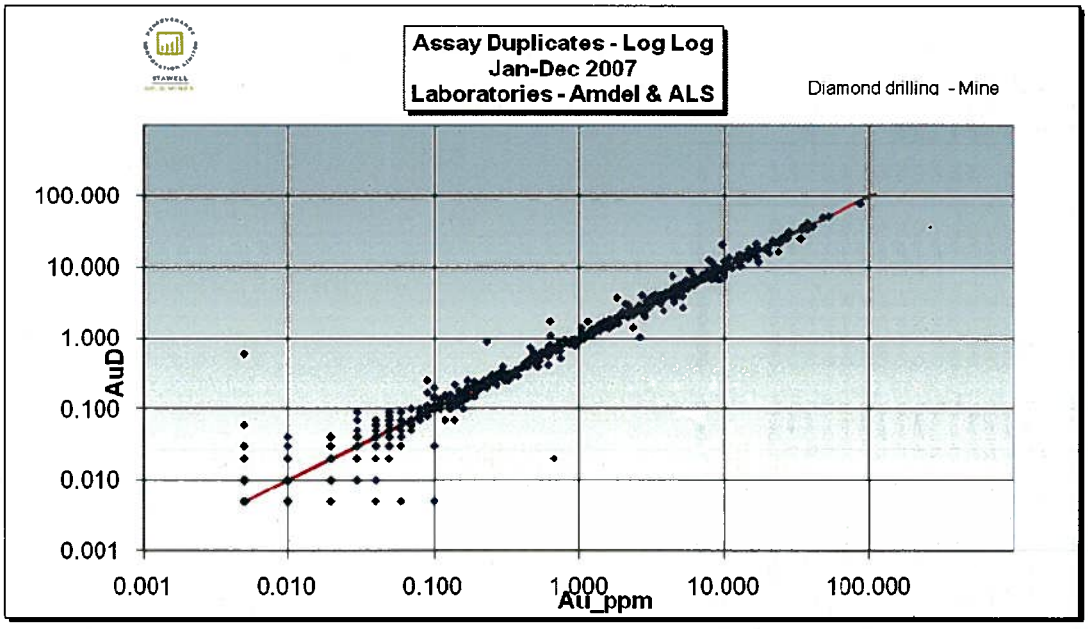


Figure 8 Assay duplicate comparison for 2007

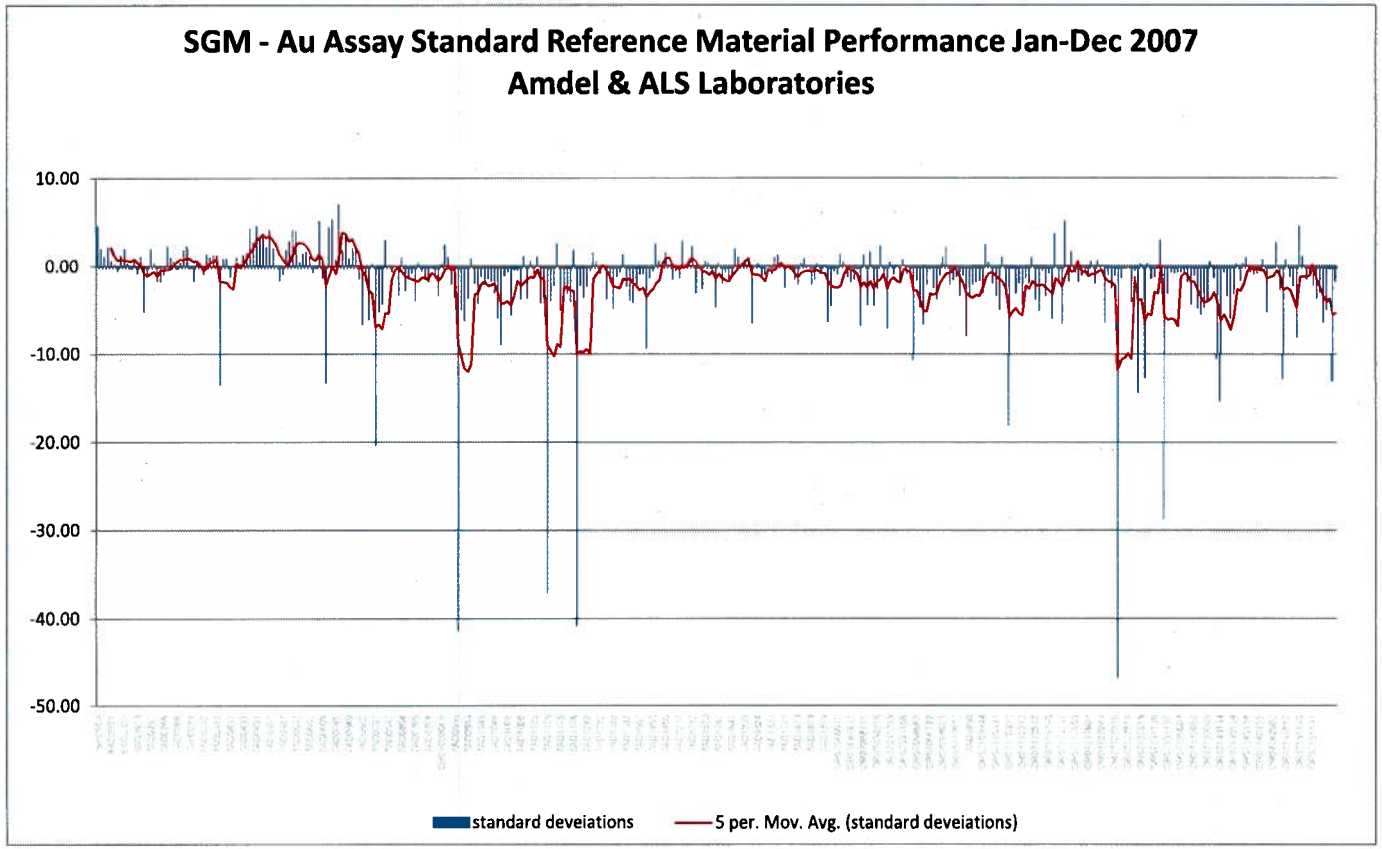


Figure 9 showing 2007 Original (first) Assay Standard performance.

Two external laboratories were used during 2007 for fire assay of all SGM drill core. These were Amdel in Adelaide SA and ALS laboratories in Orange NSW. During the first half of the year Amdel was used for Stawell fire assay work with 40g charge weights. As seen in Figure 9 cycling could be seen with the standard assaying causing concerns with the laboratory quality. Limited work was conducted between SGM and Amdel to improve the assaying process. Towards the end of SGM and Amdel's relationship the current SGM QA/QC system was being developed but the turn around times for samples sent to Amdel in some cases were out to 60 days. This prolonged turnaround time impeded QA/QC interaction but also severely impacted on production requirements. Due to this in August 2007 SGM changed to ALS laboratories in Orange NSW for all fire assay work.

The same cyclic nature to the standard assaying was seen by ALS at first. ALS sample turnaround averaged two weeks which made QA/QC dialog a smoother process. To react to the cyclic nature of the Standard assaying ALS in November 2007 added extra litharge to the flux. The extra litharge was implemented due to concerns that the high sulphide component to the Stawell ore could be interfering with the flux during the fusion process. This proved to have limited success so towards the end of December 2007 ALS changed the charge size from 50g down to 30g as another attempt to combat the high sulphide nature of the Stawell ore. The success of this reduced charge size is currently under review. ALS also highlighted that the Standards were much finer than the sample product and could pose issues in mixing with the flux.

Overall for 2007 based of the original (first) assay received the laboratories have been running at a negative 4% bias. However the QA/QC process identifies batches that require check assays. During 2007 a number of check assays on batches were completed with 10 Standards previously out of specification having to be replaced. From this it is estimated that the laboratory bias for 2007 is negative 3% for gold.

2.4 Surveying

Diamond drillhole collars are surveyed by SGM Survey personnel. Survey control points are maintained in the underground decline by SGM Survey personnel and these locations provide the control for all mark out and pick-up surveying that is conducted in the underground environment. On conclusion of drilling and hole grouting diamond drilling personnel will

insert wooden wedge labelled with the drill hole ID into the collar of the hole. This provides permanent identification of the drillhole collar to ensure matching of surveying information to the correct drill hole collars. The collar survey information is entered in the SGM database by data managers.

Single shot down hole cameras have been used to survey the drillhole path. Electronic single shot instruments are have progressively been used in preference to the Eastman mechanical cameras since 2002 at Stawell and the vast majority of the downhole surveys of diamond drillholes utilised in this estimation have been made with Electronic single shot cameras. Magnetic intensity is recorded and used to assess validity of record. Dip +/- 1.5° and azimuth +/- 3° is highlighted and assessed.

Where clear issues are identified with the validity of the survey information adjustements to the surveys are made in the database and the suspect data given a lower priority in the database system. A record of survey methods and or adjustments are maintained in the main acQuire database.

A single drill hole with inaccurate down hole locations, (MD 2353W1), has been identified using updated geological contact locations and subsequently excluded from this resource model.

2.4.1 Database Storage and Integrity

All SGM drilling data is stored within the "acQuire" Database Management System. The database operates in an SQL framework and data security is established by having various levels of user access rights. SGM maintain a security access system where loading and manipulation of data is only conducted by one of two data managers. All geological personnel have access to the database for read only purposes.

Data validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth

- Checks are performed for overlapping samples

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (e.g. Minesight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted.

After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

3 Bulk Density

All whole core sample intervals that are sent for assay are bagged and weighed prior to leaving site. The weights of the core are stored in the drillhole database and an apparent dry bulk density for each interval is calculated based upon a theoretical volume for the specific core diameter for each sample interval. This apparent density measurement includes some residual moisture which by measurement has been estimated at approximately 2%.

Analysis of the Bulk Density data set has been completed by domain and following removal of low and high outliers gives the values as shown in Table 1.

Table 1. GG5 Lower bulk density analysis

Zone	Number of Intervals	Length Weighted Average	Value Applied to Model
Statesman Lode GEOCD 601	732	2.99	2.95
Weak Volcanics GEOCD 501	785	2.88	2.85
Open Flank GEOCD 502	151	2.83	2.85

4 Metallurgical Test Work

The global metallurgical recovery for GG5L is 90.1%.

5 Geological Modelling

Geology modelling has been carried out by Catherine Smith the area Mine Geologist. Interpretations have been completed on cross section and or within the 3 dimensional framework provided by Minesight software. The interpretations completed for this model incorporate face mapping information from the 1265mRL and 1285mRL development drives that been completed during the last 5 months of the 2007 calender year.

5.1 Estimation Domains

The objectives of the interpretaion process are to define the major mineralisaed domains and faults as discussed in Section 1 of this report such as to provide suitable estimation domain solids.

The following domain solids have been created

- Statesman Lode (Volc (statesman) fnl, domain 601); This zone has further been subdivided into fault blocks 601, 602, 603 (using SFLAG item).
- Weak Volc (Weak Volc fnl, domain 501);
- Open Flank (Volc (open flank) fnl, domain 502).
- Basalt bodies, Footwall and Hanging Wall (400)
- Mine Schist and Porphyry (1)

Coding of the drillhole assays and block model have been carried out using these domain solids.

6 Model Statistics

6.1 Compositing and Descriptive Statistics

The raw sample results have been composited to 2.0m intervals using the MineSight compositing procedures. The compositing process creates 2.0m composites of the primary assay intervals in a downhole direction honouring the coded geological domains. The MineSight software downhole compositing routine provides an option to accumulate short

intervals into the preceding interval. This option has been used to prevent a number of smaller intervals from forming on the downhole margins of estimation domains and as such all intervals can be used in the estimation process. A lower composite threshold of 0.75m was chosen and as such the maximum composite interval would be 2.75m.

Analysis of the statistics for composite intervals show the average to be close to 2.0m for all zones with only a small number of longer and shorter intervals for each domain.

A complete listing of descriptive statistics for all domains is provided in Table 2. The composite sets selected for the resource interpolation are highlighted in yellow.

Log Probability plots of the 2.0m composite lengths and the gold assay values are shown FigureFigure 10Figure 11Figure 12. A top-cut value of 35.0g/t Au was applied to domain 601 to 603 (Statesman Lode), 15g/t Au for domain 501 and 18g/t Au for domain 502. A general description of the domains and the top-cut applied is outlined in Table 3.

Top cuts applied are based on a review of the Log Probability plots shown in figures 10 – 12. For the main Statesman Lode areas the top cuts result in the cutting of a small number of composites which is consistant with the relatively smooth high grade tail to population.

Leviathan Resources - Stawell Gold Mines						
GG5L RESOURCE MODEL						
Date:	Nov-2007					
Variable	Data Type(s)	Number of Samples	Minimum g/t	Maximum g/t	Mean g/t	Coeff of Var
Domain 601 - Statesman						
Au Raw	DD		0.01	75.30	11.35	0.950
Au 2.0m Composites	DD	307	0.12	46.65	11.28	0.740
TC35	DD	307	0.12	35.00	11.23	0.718
Length 2.0m Composites	DD		0.75	2.70	1.95	
Domain 602 - Statesman						
Au Raw	DD		0.04	61.80	10.34	0.960
Au 2.0m Composites	DD	83	0.14	39.20	10.37	0.710
TC 35	DD	83	0.14	35.00	10.32	0.703
Length 2.0m Composites	DD		1.00	2.70	1.92	
Domain 603 - Statesman						
Au Raw	DD		0.01	24.10	7.65	0.840
Au 2.0m Composites	DD	41	0.04	24.10	7.25	0.700
TC 35	DD	41	0.04	24.10	7.26	0.703
Length 2.0m Composites	DD		0.75	2.70	1.77	
Domain 501- Weak volc						
Au Raw	DD		0.01	60.60	1.34	2.771
Au 2.0m Composites	DD	424	0.01	30.30	1.37	2.161

TC 15	DD	426	0.01	15.00	1.33	1.758
Length 2.0m Composites	DD		0.75	2.75	1.89	
Domain 502 - Open flanks						
Au Raw	DD		0.01	50.40	3.92	1.732
Au 2.0m Composites	DD	88	0.01	26.30	3.77	1.340
TC 18	DD	89	0.01	18.00	3.84	1.273
Length 2.0m Composites	DD		0.75	2.65	1.83	

Table 2 Descriptive statistics of the 2.0m composites per domain.

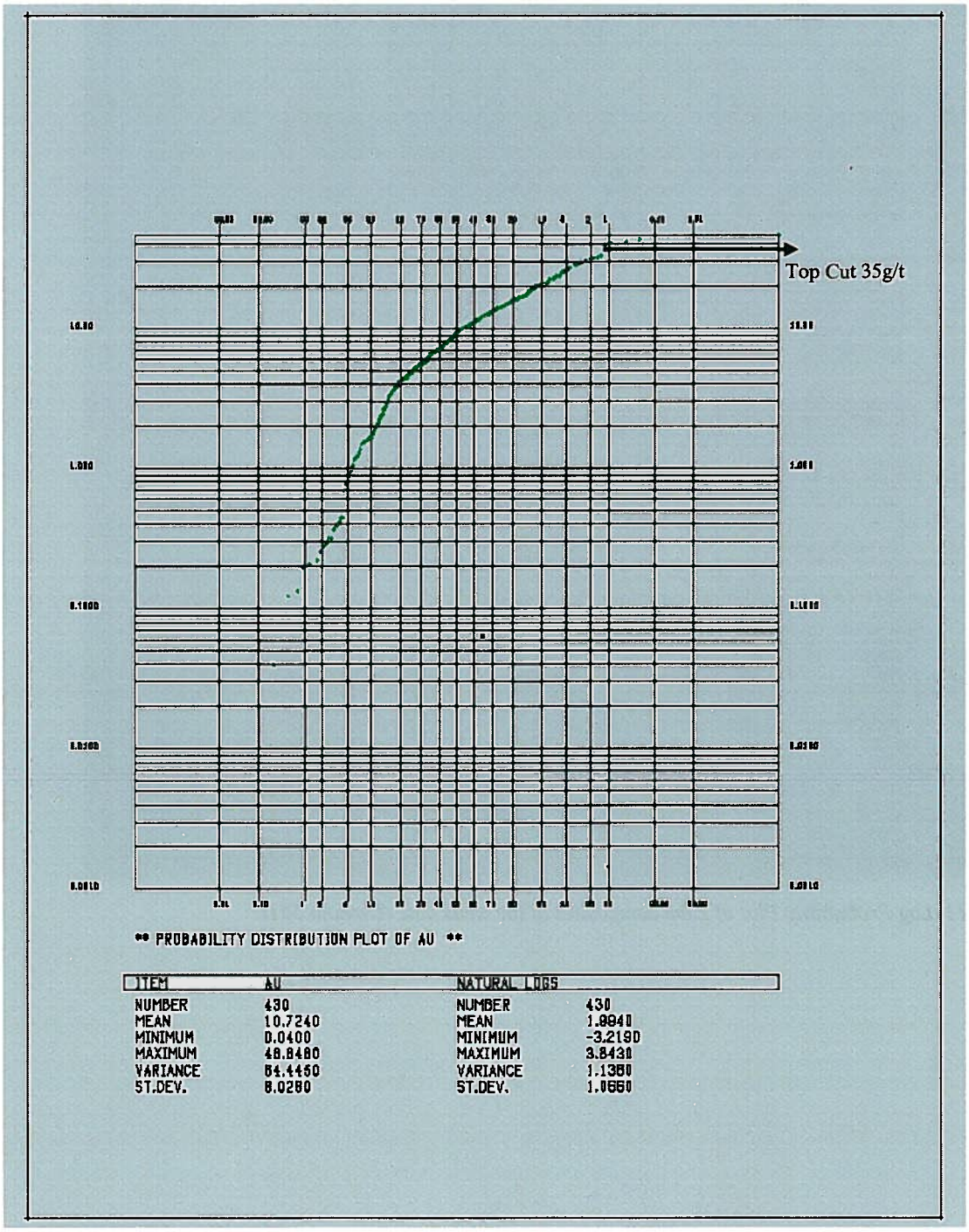


Figure 10. Log Probability plot of 2.0m composites of the statesman lode (Domain 601, 602, 603).

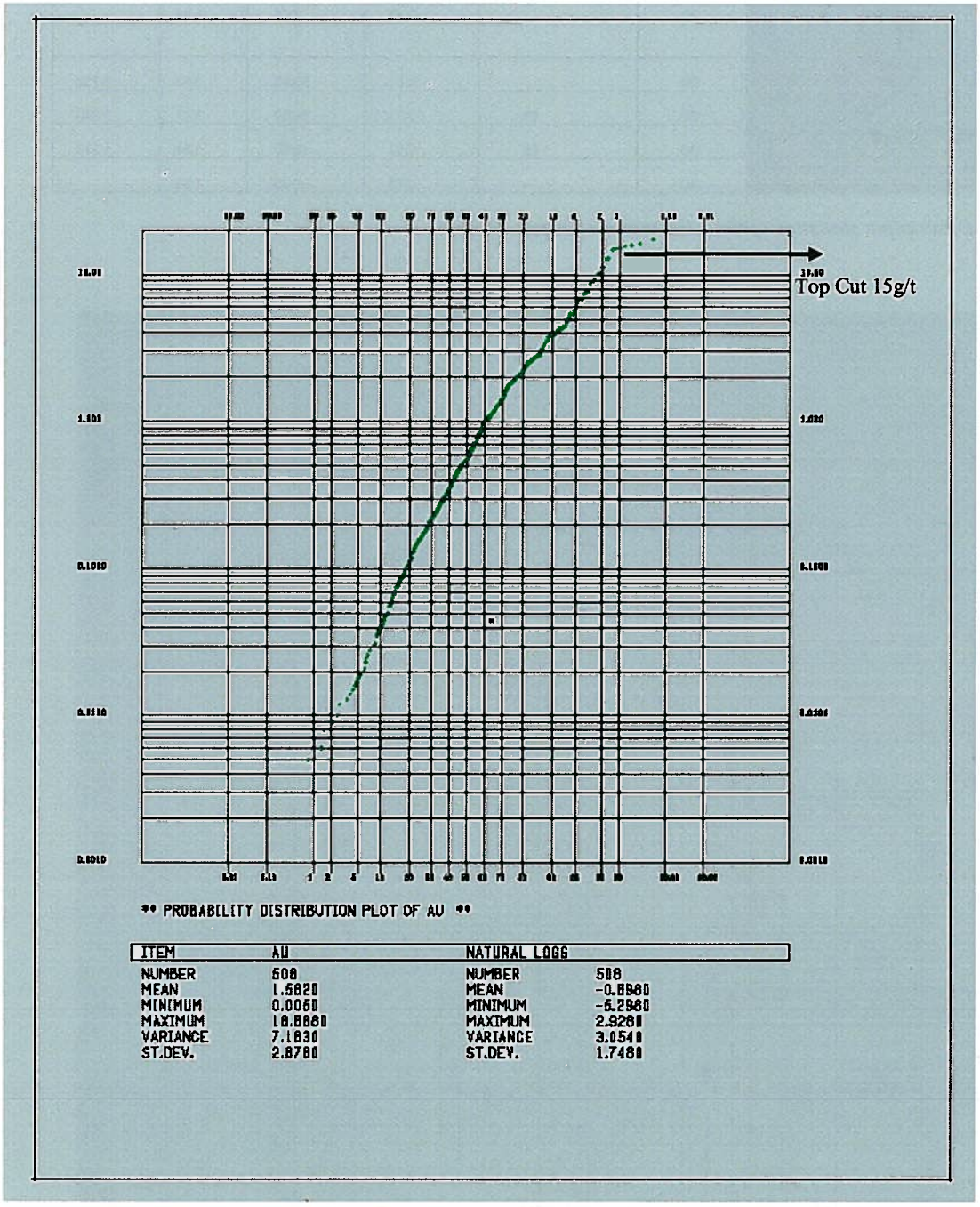


Figure 11 Log Probability Plot of 2.0m composites of the weak volc (Domain 501).

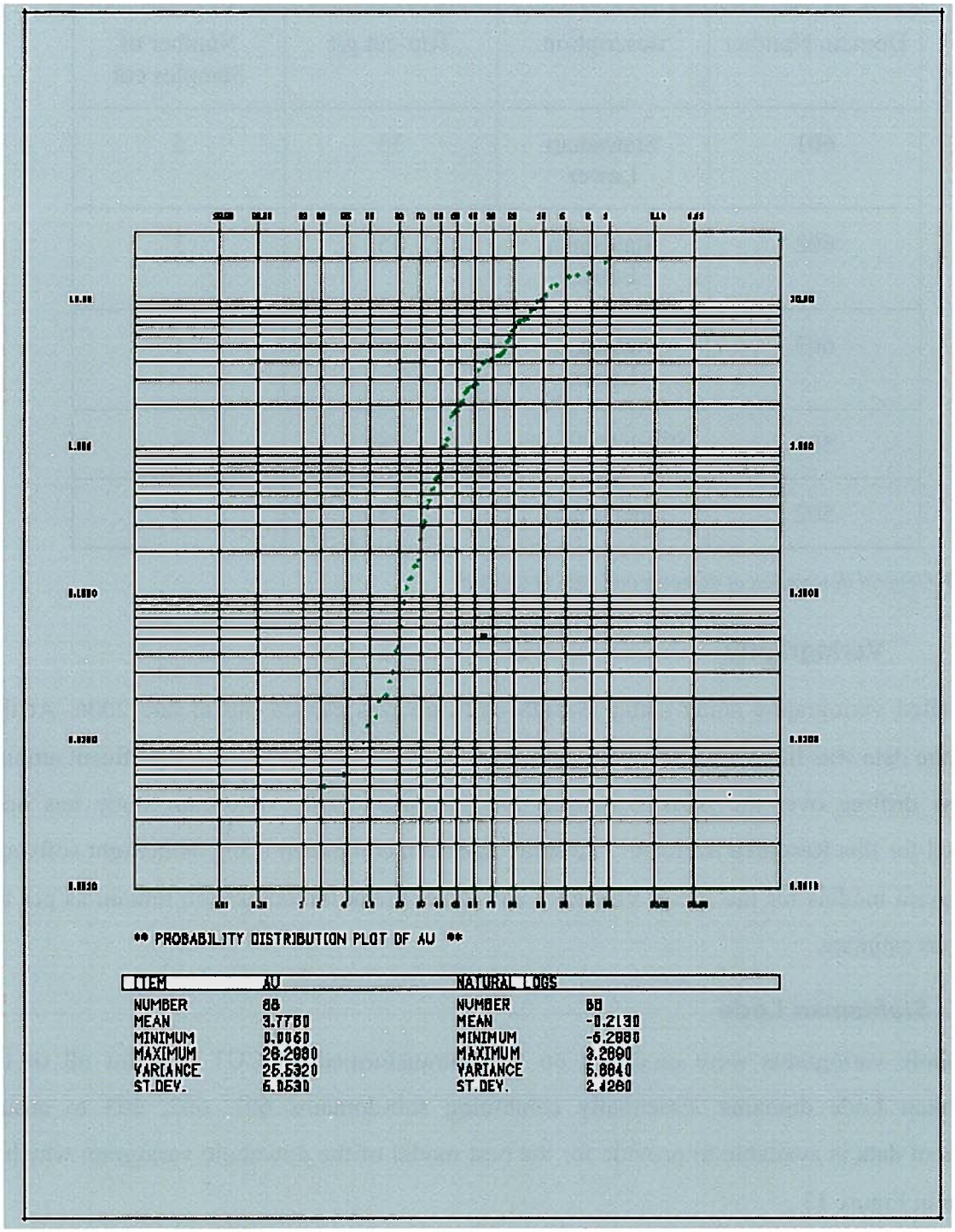


Figure 12 Log Probability plot of 2.0m composites of the open flank (Domain 502).

Domain Number	description	Top-cut g/t	Number of Samples cut
601	Statesman Lower	35	5
602	Statesman Lower	35	3
603	Statesman Lower	35	1
501	Weak Volcanic	15	6
502	Open Flank	18	2

Table 3 General description of domains and top-cut applied.

6.2 Variography

A detailed variography study using ISATIS software was carried out in late 2006. At this stage the data was limited to the 30m X 30m Resource drilling. Given the significant amount of new drilling over the past 12 months the Statesman Lode variogram study has been updated for this Resource estimate. The study has been completed using MineSight software. Variogram models for the Weak Volcanics and Open Flank mineralisation remain as per the previous estimate.

6.2.1 Statesman Lode

Downhole variograms were modelled on the untransformed AUCUT data for all of the Statesman Lode domains. Essentially combining sub-domains 601, 602, 603 to ensure sufficient data is available to provide for the best model of the downhole variogram which is shown in Figure 13.

Subsequent directional variograms were completed also using Minesight routines (Figure 14 and Figure 15). The variograms were modelled using relative experimental variograms on AUCUT for the combined domain. The variograms were calculated in the plan of the structure as essentially 2D models and then transformed back into true 3D space. A rotation (MEDS) of

ROTN = 305, DIPN = 0, DIPE = -82 was used to define the plane of the Statesman Lode

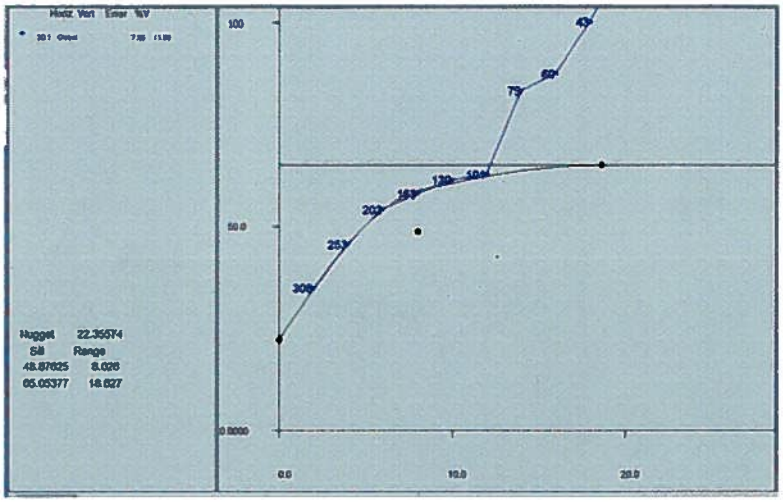


Figure 13 Downhole raw experimental variogram for AUCUT and the resulting variogram model

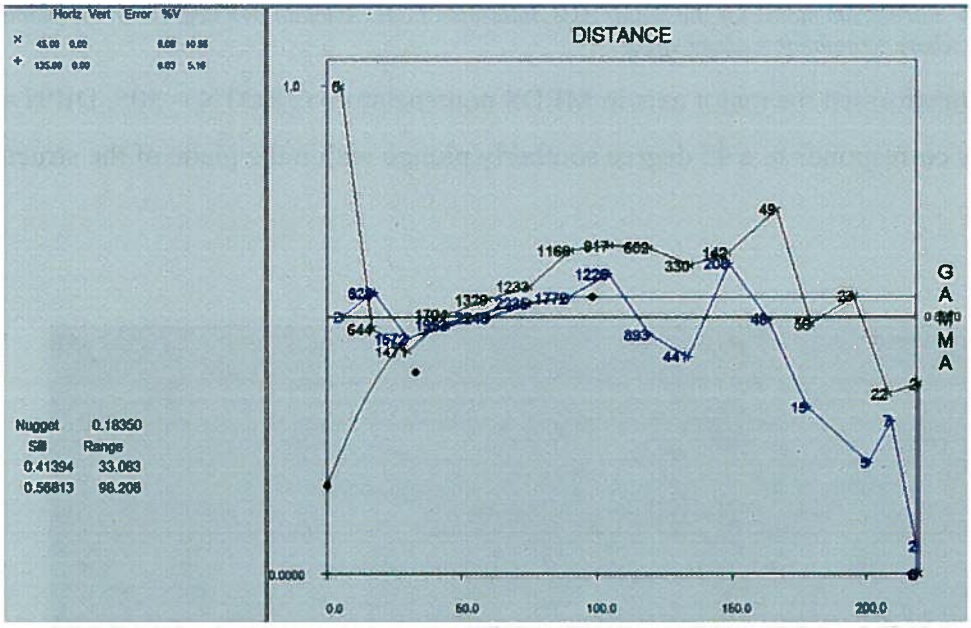


Figure 14 Variogram model for the Major Axis Statesman Lode. Azimuth 135 degrees in the plane of the structure where Azimuth 0 = along strike.

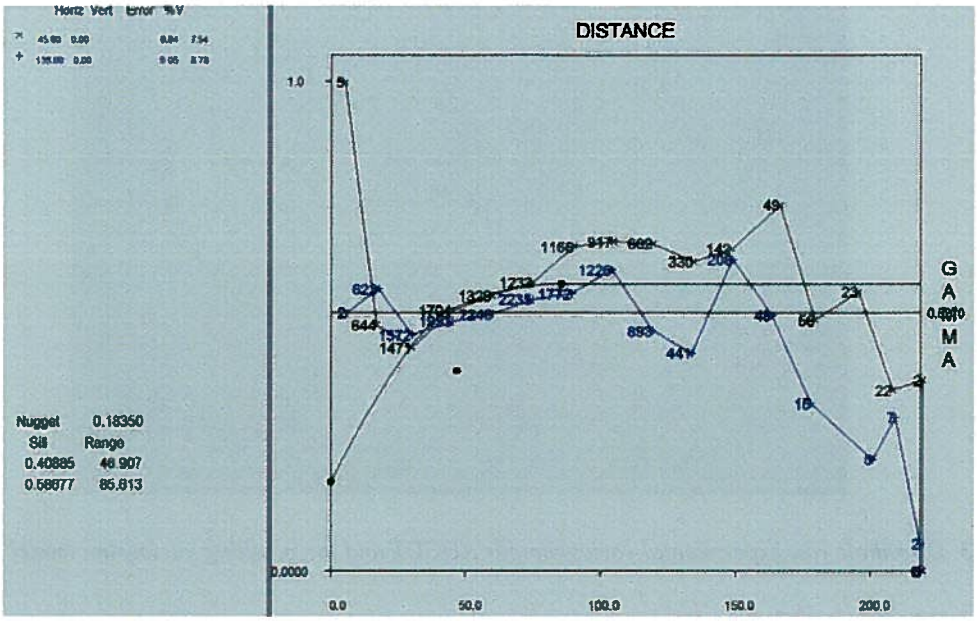


Figure 15 Variogram model for the Major Axis Statesman Lode. Azimuth 045 degrees in the plane of the structure where Azimuth 0 = along strike.

When transformed the major axis in MEDS nomenclature is ROTN = 305, DIPN = 45, DIP -85. This corresponds to a 45 degree southerly plunge within the plane of the structure (Figure 16).

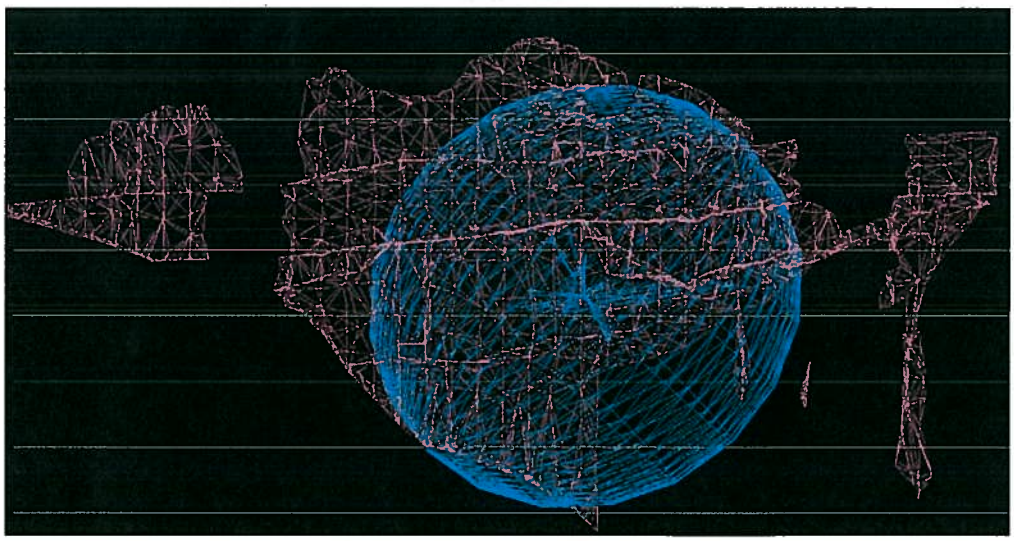


Figure 16. Minesight visualisation of the statesman lode wireframe and the search ellispse showing the orientation and major direction of the search.

6.2.2 Weak Volcanogenics

The variogram model for the weak volcanogenics (domain 501) is show in Figure 17. The variogram axes was initially defined using the Isatis mathematical convention as 150° around the Z-axis, and 80° around the X-axis (refer Figure 18).

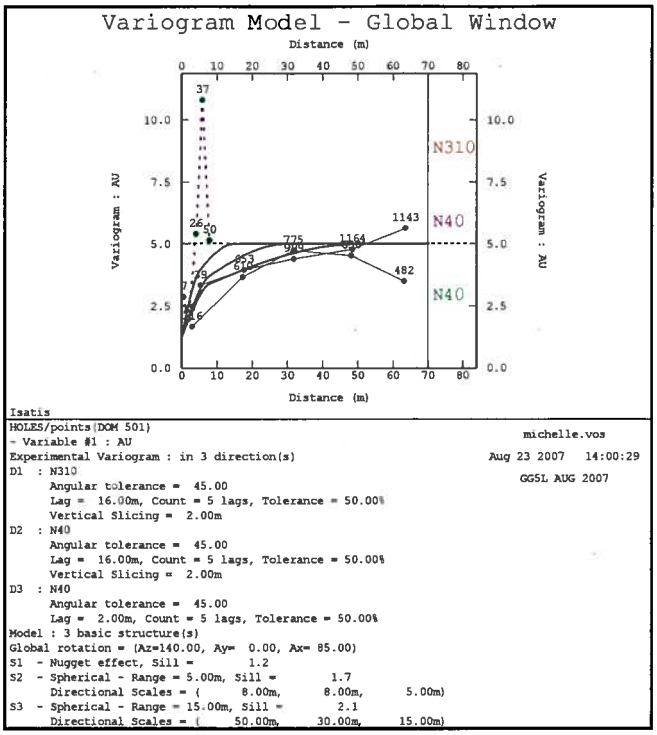


Figure 17 Variogram model fitted to the weak volcanogenic mineralisation (domain 501).

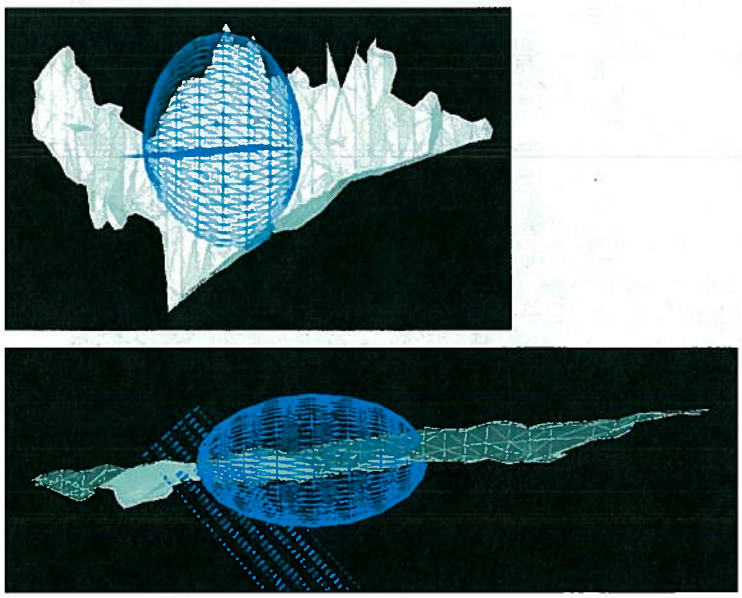


Figure 18. Minesight visualisations of the weak volcanogenic wireframe and the search ellispse showing the orientation and major direction of the search.

6.2.3 Open Flank

The variogram model for the open flank (domain 502) is shown in Figure 19. The variogram axes was initially defined using the Isatis mathematical convention as 140° around the Z-axis, and 85° around the X-axis (refer to **Error! Reference source not found.**).

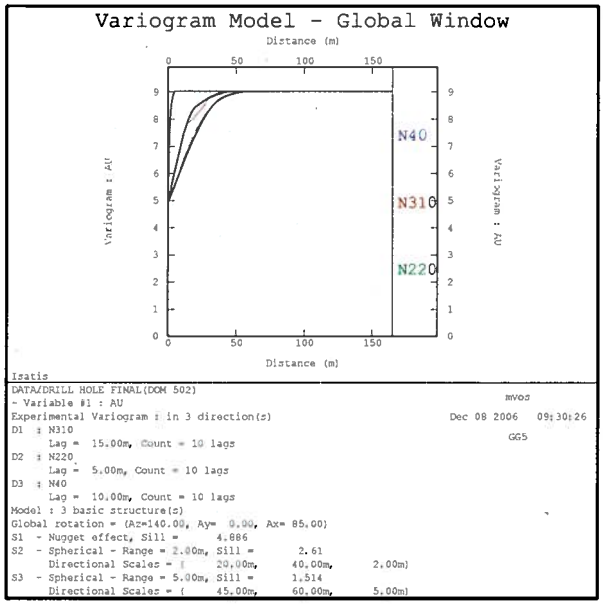


Figure 19 Variogram model fitted to the open flank mineralisation (domain 502).

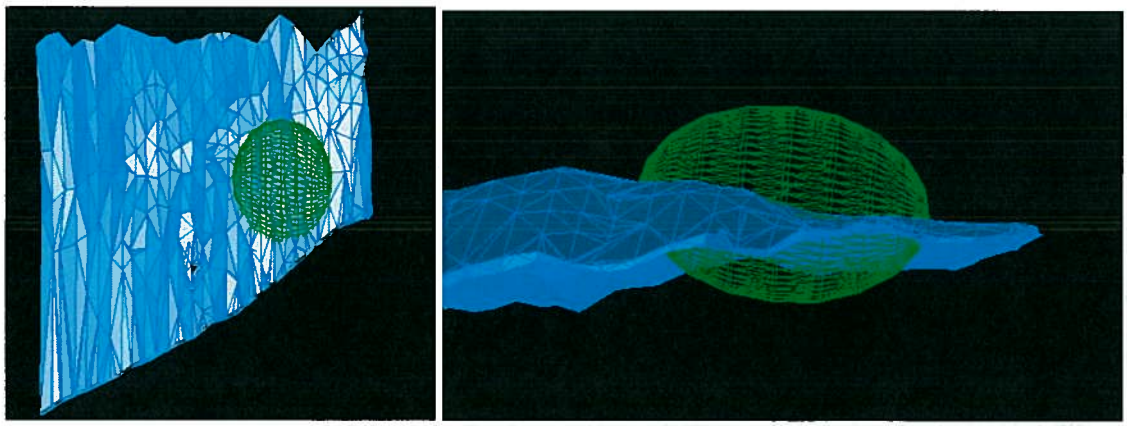


Figure 20 Minesight visualisations of the open flank mineralisation wireframe and the search ellispse showing the orientation and major direction of the search.

6.2.4 Model Variogram Parameters

The variogram parameters used in the estimation are outlined below (refer Table 4)

Stawell Gold Mines											VARIOGRAM PARAMETER TABLE						
GG5L RESOURCE MODEL																	
AREA	VREFNUM	axis 1	axis 2	axis 3	Nugget	1st Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	2nd Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	Total Variance	Nugget
		rotation															
GG5L	601	300	45	-85	22	1	33	47	7	26	1	98	85	12	17	65.00	34%
GG5L	602	300	45	-85	22	1	33	47	7	26	1	98	85	12	17	65.00	34%
GG5L	603	300	45	-85	22	1	33	47	7	26	1	98	85	12	17	65.00	34%
GG5L	501	310	0	-85	1.2	1	8	8	5	1.7	1	50	30	15	2.1	5.00	24%
GG5L	502	310	0	-85	14.15	1	10	10	5	14	1	50	45	15	23	51.15	28%

Table 4. MINESIGHT variogram parameter table used for block model estimation.

7 Block Model Estimation

7.1 Neighbourhood optimisation

The neighbourhood optimisation is used to determine appropriate search dimensions and minimum and maximum number of samples during Kriging interpolation. It is done in an empirical manner by testing and spatially viewing, in ISATIS and in Minesight characteristics like the distribution of negative kriging weights, the achievable theoretical regression slope, block kriging efficiency, etc, for a number of model blocks that are located in "favourable" and "unfavourable positions" with respect to the available drillholes within a domain.

A resource model the block size of 5m east, 10m north and 10m RL the details of which are shown in Table 5. This model is a rotated model of 310° northwest with a dip of and 85° to the east.

	Min	*Max*	*Cell Size*	*No. Cells*	*Origin (AMG)*
East	0	350	5	75	659160 E
North	0	630	10	63	897700 N
RL	0	650	10	65	-1650 RL

Table 5. Model Limits

A listing of the MineSight search parameters are provided in Table 6.

Leviathan Resources- Stawell Gold Mines												
GG5L RESOURCE MODEL												
AREA	DOMAIN	x axis	y axis	z axis	axis 1	axis 2	axis 3	min. samples 1st search	max. samples 1st search	Search Type	Max Samps per Quad	max. samples per hole
		search distance			rotation			sample number definition				
GG5L	601	80	72	9	300	45	-82	8	32	Quad	10	5
GG5L	602	80	72	9	300	45	-82	8	32	Quad	10	5
GG5L	603	80	72	9	300	45	-82	8	32	Quad	10	5
GG5L	501	50	30	20	310	0	-85	8	32	Quad	10	N/A
GG5L	502	50	45	30	303	0	-80	8	40	Quad	10	N/A

Table 6. Listing of MineSight search parameters.

7.2 Block Estimation Parameters

All domains have been constrained by wireframes. They have been estimated using Ordinary Block Kriging. The following domains have used for estimation;

- Domain 601, (Volc (statesman) fnl) Lower Statesman Lode.
- Domain 602, (Volc (statesman) fnl) Middle Statesman Lode.
- Domain 603, (Volc (statesman) fnl) Upper Statesman Lode.
- Domain 501, (Weak Volc fnl), Weak Volcanogenics.
- Domain 502, (volc (Open flank) fnl)) Open Flank.

Standard estimation settings in MineSight were used in the estimation of the model:

- Estimation is confined to domains.
- The maximum number of samples allowed to be selected per drillhole has been restricted in the estimation to 5 composites per hole.

The composited drill holes and block model were assigned the appropriate domain values.

The completed block model and all associated MineSight files are stored on the Mining Geology server in the directory S:\Mining-Geology\GoldenGift 5\Minesight\Geology\ Statesman Deeps Nov 2007. Additional details on the MineSight resource model are provided in Table 7.

Leviathan resources - Stawell Gold Mines	GG5L RESOURCE MODEL
Date	**Nov-2007**
Model compiled by	**DF/DH/CS**
Directory Location	**S:\Mining-Geology\Golden Gift 5\Minesight\Geology\Statesman Deeps Model Nov 07**
Model Name	**gg5l15.new**
Previous Model Name	**gg5l15.dat**
Parameter Spreadsheet	**GG5_Resource_Modelling.xls**
Resource Report	**GG5 Model Report November 07.doc**
Purpose of Update	**Update GG5L with new drilling results for underground optimisation.**

Table 7. Resource model description table.

8 Model Validation

Model validation has been completed using a visual assessment of the kriged block grades relative to the input composites. A manually compiled long projection of undiluted drillhole intercepts has been overlain with the kriged grade shells at a number of cutoffs. These indicate generally good agreement. In addition a plot of the Slope of Regression of the Estimate is used to indicate kriging confidence. This plot is shown as Figure 21. Blocks with values above 0.8 are well supported by the data and a high level of confidence in the kriged outcome can be attached to these blocks.

Given that the significant majority of metal above a 4.0g/t cutoff occurs within the Statesman's Lode similar plots have not been presented for the other domains.

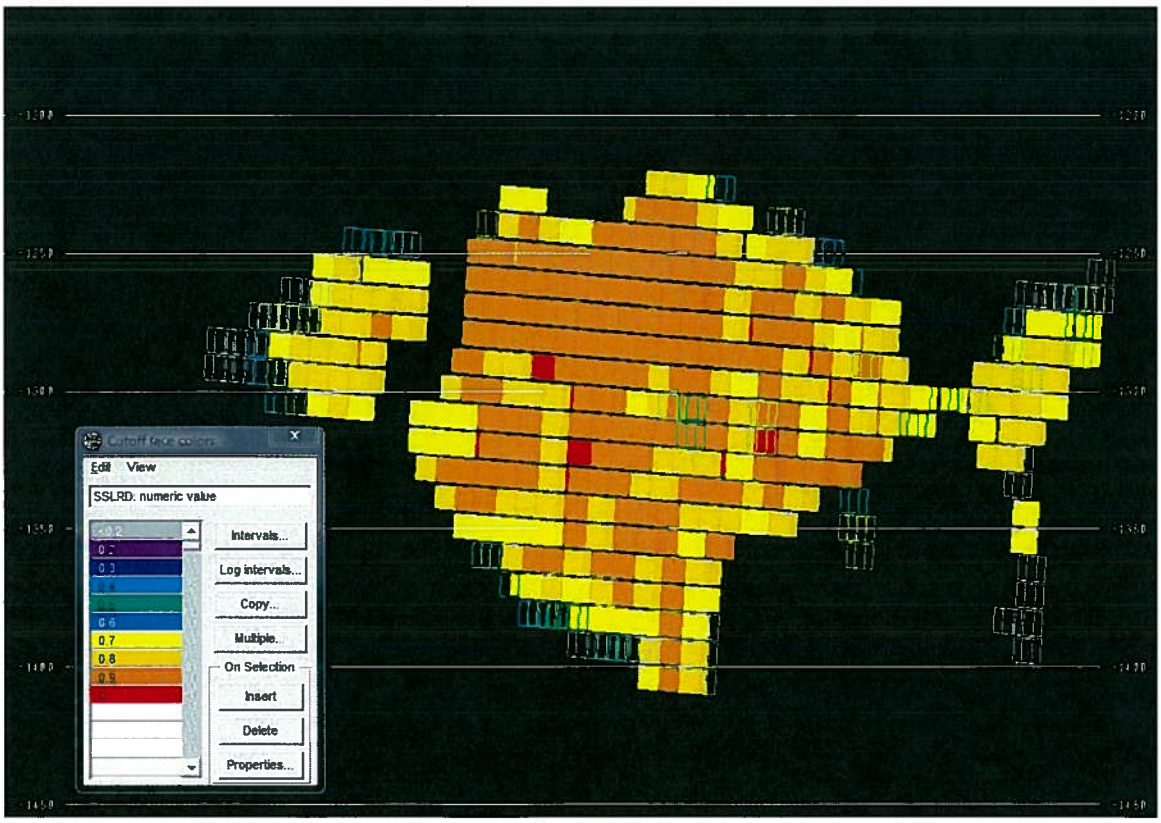


Figure 21 Statesman Lode resource blocks coloured by slope of Regression of the Estimate. The legend is for Slope is shown on the left of the figure.

A comparison of the declustered composited drill hole data and mean grade of the model blocks (15mN x 5mE x 210mRL) within a domain shows an acceptable variance (refer Table 8).

Stawell Gold Mines		MODEL VALIDATION TABLE		
GG5L RESOURCE MODEL				
Date:				
	Volume (m3)	Mean Grade (g/t)		
DOMAIN	Wireframe	Declust. 2.0m comp. 20X20x10	Model	Variance
Domain 601 - Statesman - block 1		10.79	10.98	1.8%
Domain 602 - Statesman - block 2		9.43	9.97	5.7%
Domain 603 - Statesman - block 3		7.07	7.95	12.4%
Statesman block		10.35	10.34	-0.1%
Domain 501- Weak volc		3.32	3.96	19.3%
Domain 502- Open Flank		1.56	1.43	-8.3%

Table 8. Model validation table.

9 Resource Classification

The classification of Indicated and Inferred is based on geological confidence, Slope of Regression and model validation results. Indicated/Inferred material for Statesman domain is illustrates in Figure 22.

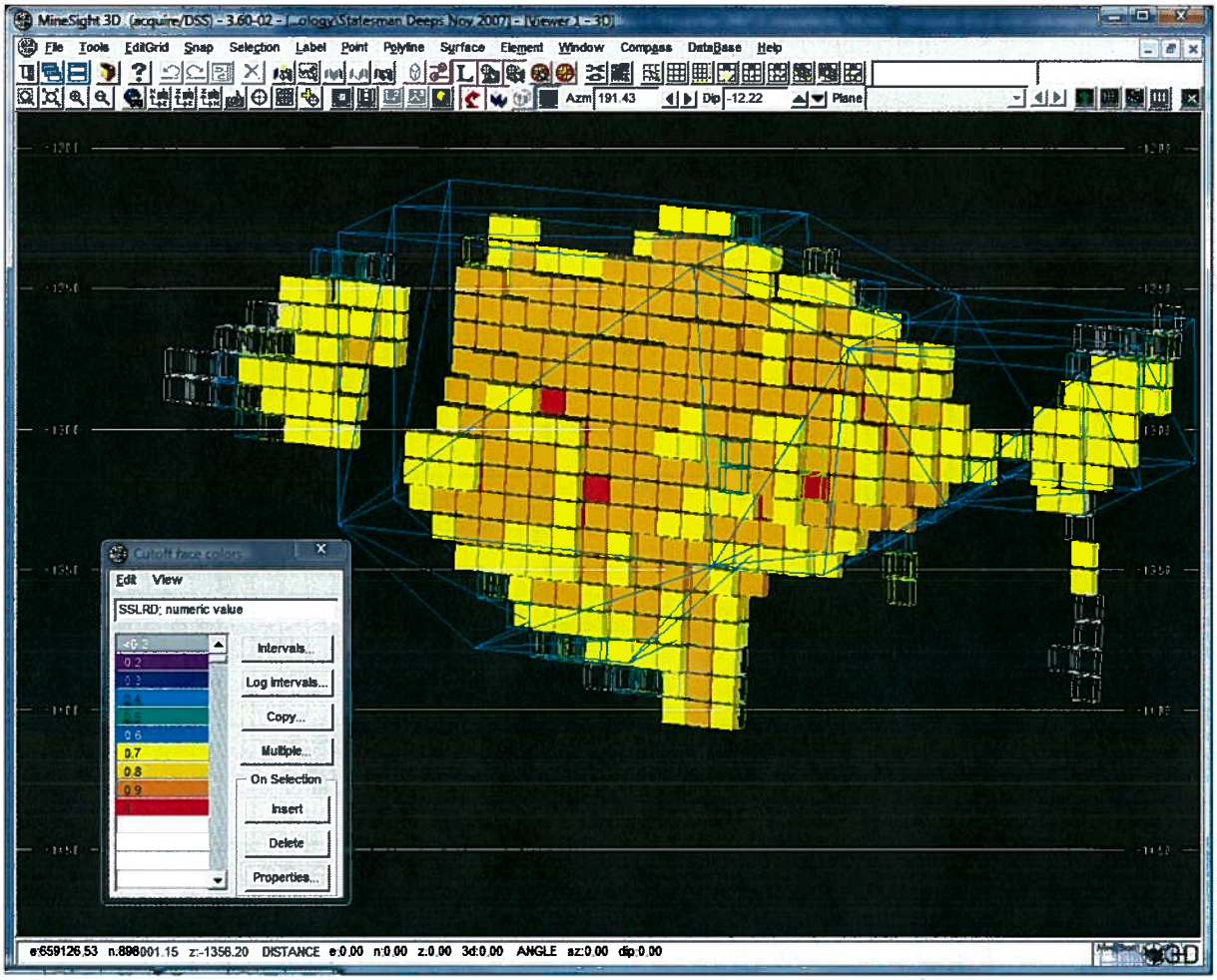


Figure 22 MineSight screen capture of the resource classification of Statesman Lode. Indicated is within the blue wireframe. Blocks are coloured by Slope.

10 Model output

The total Indicated Modelled Inventory at a cut off grade of 4.0g/t Au is 310,000 tonnes at 10.9 g/t for 109,300oz. A listing of the global inventory at a series of cut-off gold grades is provided in Table 9.

Leviathan Resources - Stawell Gold Mines	GG5L
Resource Classification	**Modelled Inventory**
Mined Status	**Small amount depleted**
Other Restrictions	**AREA='GG5L'**
Filter Applied to Evaluation	**Not Applied**
Previous Model Name	**gg5l15.new**
New Model Name	**gg5l15.dat**

RESCAT	Cutoff (g/t)	PREVIOUS MODEL			NEW MODEL			Change			Change %		
		quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)
MEASURED INDICATED INFERRED	**0.0**	**425,700**	**10.0**	**136,300**	**950,400**	**5.3**	**161,300**	524,700	-4.7	25,000	123%	-47%	18%
	2.0	**425,700**	**10.0**	**136,300**	**562,500**	**8.2**	**148,000**	136,800	-1.8	11,700	32%	-18%	9%
	4.0	**420,900**	**10.0**	**135,700**	**456,800**	**9.4**	**138,700**	35,900	-0.6	3,000	9%	-6%	2%
	5.0	**412,800**	**10.1**	**134,500**	**409,400**	**10.0**	**131,700**	-3,400	-0.1	-2,800	-1%	-1%	-2%
	7.0	**372,600**	**10.6**	**126,500**	**313,700**	**11.2**	**113,400**	-58,900	0.7	-13,100	-16%	6%	-10%
	9.0	**271,000**	**11.5**	**100,100**	**216,900**	**12.7**	**88,500**	-54,100	1.2	-11,600	-20%	10%	-12%
MEASURED	**0.0**	-	-	-	-	-	-	0	0.0	0			
	2.0	-	-	-	-	-	-	0	0.0	0			
	4.0	-	-	-	-	-	-	0	0.0	0			
	5.0	-	-	-	-	-	-	0	0.0	0			
	7.0	-	-	-	-	-	-	0	0.0	0			
	9.0	-	-	-	-	-	-	0	0.0	0			
INDICATED	**0.0**	298,300	10.2	98,000	313,300	10.9	109,500	15,000	0.7	11,500	5%	6%	12%
	2.0	298,300	10.2	98,000	313,300	10.9	109,500	15,000	0.7	11,500	5%	6%	12%
	4.0	295,200	10.3	97,700	310,700	10.9	109,300	15,500	0.6	11,600	5%	6%	12%
	5.0	289,400	10.4	96,800	306,600	11.0	108,600	17,200	0.6	11,800	6%	6%	12%
	7.0	266,700	10.8	92,200	269,300	11.7	101,300	2,600	0.9	9,100	1%	9%	10%
	9.0	200,100	11.6	74,800	205,900	12.8	85,000	5,800	1.2	10,200	3%	10%	14%
INFERRED	**0.0**	127,400	9.3	38,300	637,000	2.5	51,800	509,600	-6.8	13,500	400%	-73%	35%
	2.0	127,400	9.3	38,300	249,100	4.8	38,500	121,700	-4.5	200	96%	-49%	1%
	4.0	125,700	9.4	38,100	146,100	6.3	29,400	20,400	-3.2	-8,700	16%	-34%	-23%
	5.0	123,400	9.5	37,700	102,800	7.0	23,100	-20,600	-2.5	-14,600	-17%	-27%	-39%
	7.0	105,900	10.1	34,200	44,500	8.5	12,100	-61,400	-1.6	-22,100	-58%	-16%	-65%
	9.0	70,900	11.1	25,300	11,000	10.1	3,600	-59,900	-1.0	-21,700	-84%	-9%	-86%
POTENTIAL	**0.0**	-	-	-	-	-	-	-	0.0	-			
	2.0	-	-	-	-	-	-	-	0.0	-			
	4.0	-	-	-	-	-	-	-	0.0	-			
	5.0	-	-	-	-	-	-	-	0.0	-			
	7.0	-	-	-	-	-	-	-	0.0	-			
	9.0	-	-	-	-	-	-	-	0.0	-			

Table 9. Modelled Global insitu inventory at a series of cut-off grades for a new resource model compared to the previous model.

Minining over the period has depleted the Mineral Resource with development on the 1265 and 1285 levels and the first stope from the 1285 Level.

Reconciliaton of mined Ore blocks to the end of January 2008 are shown in Table 10. These reconciliation suggest significant additional metal has been produced from these stopes relative to the Ore Reserve estimate. During the period that these stopes were mined the overall mine reconciliation has been very positive and it is thought much of the extra metal can be attributable to material mined in the GG3 area. Even with a significant allocation of metal back to these sources it can be reasonable assumed the reconciliation of the material mined to date from the Statesman Lode is positive.

11 Recommendations

Many of the recommendation made in previous documents have been completed over successive updates. At the time of this update additional geological evaluation work is required to

- ensure that currently modelled faults are accurately represented in the model
- geological information collected form development headings is incorporated into the upcoming model updates
- additional evaluation of the geological controls and domain boundaries for material above 1265mRL level is completed
- define the lower sections of the main shoot by ongoing grade control drilling and
- complete Resource Definition and Grade Control drilling north of 300N to define any shoots in the open flank position

12 Model completion

12.1 Server storage location of model output files

The model and all accompanying Minesight, ISATIS, Microsoft Excel™ and Microsoft Word™ computer files have been stored on the Stawell Gold Mines computer server in the sub-directory S:\Mining-Geology\Golden Gift 5\Minesight\Geology\Statesman deeps November2007.

Stope	Tonnes				Grade				Variance to Ore Reserve		Contained Ounces					
				Variance to O'R											Variance to Ore Reserve	
	O' Rve	haul & mill	June07 OR	haul (Mill)	O' Rve	haul	mill	June07 OR	haul	Mill	O' Rve	haul	mill	Jun-07	haul	Mill
GG3																
1116-329-88	18,400	17,004	18,400	92%	6.8	8.2	9.1	6.8	121%	133%	4,023	4,483	4,948	4,023	111%	123%
1116-330-88	14,890	16,455	16,700	111%	6.1	8.2	9.3	6.0	135%	152%	2,920	4,359	4,904	3,222	149%	168%
1116-330-96	9,356	9,499	6,100	102%	5.5	10.7	14.4	6.3	195%	262%	1,654	3,268	4,398	1,236	198%	266%
1116-331-96	9,034	9362	7,156	104%	8.1	9.1	10.0	5.4	112%	123%	2,353	2,727	3,010	1,242	116%	128%
Sub Total	51,680	52,320	48,356	101%	6.6	8.8	10.3	6.3	134%	156%	10,950	14,837	17,259	9,722	135%	158%
GG5L																
1285-297-86	8,450	9,175	10,800	109%	5.8	5.6	6.2	6.9	96%	107%	1,576	1,646	1,832	2,396	104%	116%
1285-297.5-86	6,000	10,491	9,595	175%	4.3	5.6	6.8	5.2	130%	159%	829	1,882	2,304	1,604	227%	278%
1285-298-86	4,700	7619	10845	162%	5.0	6.6	7.4	4.8	132%	148%	756	1,614	1,818	1,674	214%	241%
Sub Total	19,150	27,285	31,240	142%	5.1	5.9	6.8	5.6	114%	132%	3,161	5,142	5,953	5,674	163%	188%
GG9																
1059-342-89	5,500	7,238	4,000	132%	4.0	2.8	3.0	4.7	69%	76%	707	642	707	604	91%	100%
1059-343-89	1,800	2,495	3,100	139%	4.2	3.7	5.0	4.0	88%	119%	243	298	400	399	122%	165%
1079-343.5-89	5,400	5,374	5,400	100%	4.6	5.1	5.7	4.6	110%	124%	799	876	987	799	110%	124%
1079-343-89	4,200	4,381	3,800	104%	4.2	3.3	3.6	4.8	78%	86%	567	459	507	586	81%	89%
1099-342.5-89	5,400	4,791	4,400	89%	4.6	4.0	4.9	4.6	87%	106%	799	619	749	651	78%	94%
1099-342-89	3,800	4,273		112%	4.0	6.0	6.6	8.0	150%	166%	489	824	912	0	169%	187%
1099-344-89	5,100	2,759	5,100	54%	3.7	4.5	5.1	3.7	121%	136%	607	398	448	607	66%	74%
Sub Total	31,200	31,311	25,800	100%	4.2	4.1	4.7	4.4	97%	111%	4,210	4,117	4,710	3,646	98%	112%
Total Stoping	102,030	110,916	105,396	109%	5.6	6.8	7.8	5.6	121%	140%	18,321	24,096	27,923	19,041	132%	152%

Table 10 Stope reconciliation table for the operation for the period October2007 – January 2008

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX J

Golden Gift 9 Geology and Resource Model Report

May 2007

Golden Gift 9 Geology and Resource Model Report

May 2007

Technical Note TN 004

SUMMARY

The Golden Gift 9 resource model covers the area from 6710N to 6985N (mine grid) and between -980mRL to -1240mRL. The ore body is bound by the Merlot Fault above and to the east, the Durif Fault below and the Wildcat porphyry at depth to the north. The main lodes are the Kingston Lode which occurs as a basalt contact Lode, and the Delta Lode which occurs as a waterloo within the Kingston basalt and is interpreted to pinch out at depth. The Lodes are offset by four shallow North West dipping faults. The faults converge at the southern extent of the model making a highly faulted and complex zone.

At the time of modelling there were 31 new diamond drillholes intersecting the ore body mainly on a 15m x 15m intercept grid however drilling at depth is limited. Further drilling of the GG9 ore body is planned for the near future to test for extensions. The current interpretation differs from the previous interpretation with the addition of a number of faults between the main Merlot and Durif faults.

It is recommended that high grade zones be domained separately in the next model. The implementation of a thorough QAQC program is required. Further investigation is required for Kriging Neighbourhood analysis (block size, number of samples, search size and number of discretisation points). Global estimation of recoverable resources would be of value. The variography will require further investigation.

The total Indicated Modelled Inventory at a cut off grade of 4.0g/t Au is 16,300oz. A listing of the global inventory at a 4g/t cut-off gold grade is provided in Table 11 (not depleted with current mining).

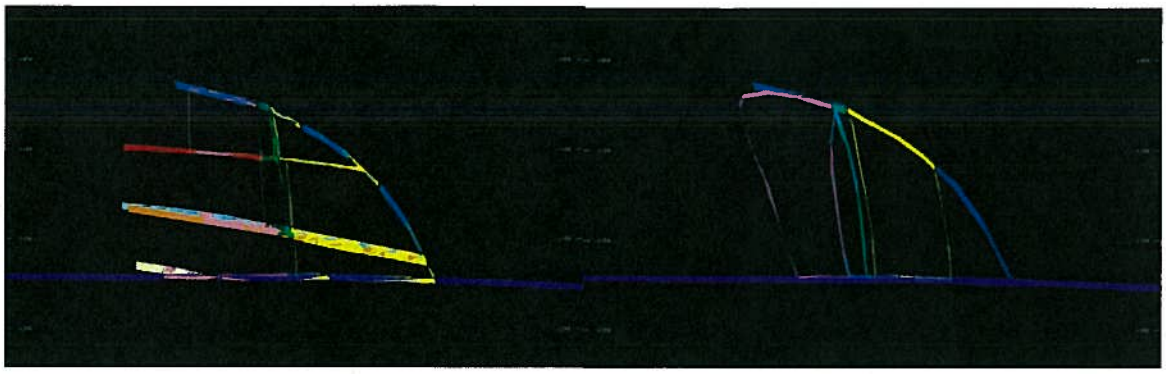
Leviathan Resources - Stawell Gold Mines	GG9
Resource Classification	Modelled Inventory
Mined Status	
Other Restrictions	AREA='GG9'
Filter Applied to Evaluation	MSTATUS=0
Previous Model Name	gg0915.dat
New Model Name	gg0915.dat

RESCAT	Cutoff (g/t)	PREVIOUS MODEL			NEW MODEL			Change			Change %		
		quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)
MEASURED INDICATED INFERRED	0.0	-	-	-	-	-	-	0	0.0	0			
	2.0	-	-	-	-	-	-	0	0.0	0			
	4.0	110,500	6.0	21,200	84,300	6.0	16,300	-26,200	0.1	-4,900	-24%	1%	-23%
	5.0	-	-	-	-	-	-	0	0.0	0			
	7.0	-	-	-	-	-	-	0	0.0	0			
	9.0	-	-	-	-	-	-	0	0.0	0			

Table 1. Modelled inventory at a series of cut-off grades for a new resource model compared to the previous model.

The Indicated model inventory of the new model is less than that reported in November 2005. At a cut-off of 4.0g/t the Indicated tonnes have decreased by 4,900oz, the grade has remained

the same. This change is due to a reinterpretation of the geological model which now more accurately represents the structural complexity. Refer to the screen captures below.



Minesight screen capture of section 6910N (left) May 2007 Model (right) November 2006 Model.

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1 INTRODUCTION

Considerable activity has occurred in GG9 since the November 2005 resource estimation including diamond drilling, sludge sampling (not used for estimation but to define geological boundaries) and face and backs mapping. This information has been used to improve the geological model and also to complete a resource model by May. The objective was to complete a reserve update for production.

1.1 Location

The Golden Gift 9 resource model covers the area from 6710N to 6985N (mine grid) and between -980mRL to -1240mRL. The ore body is bound by the Merlot Fault above and to the east, the Durif Fault below and the Wildcat porphyry at depth to the north. The main lodes are the Kingston Lode which occurs as a basalt contact Lode, and the Delta Lode which occurs as a waterloo within the Kingston basalt and is interpreted to pinch out at depth. The Lodes are offset by four shallow North West dipping faults. The faults converge at the southern extent of the model making a highly faulted and complex zone.

At the time of modelling there were 31 new diamond drillholes intersecting the ore body mainly on a 15m x 15m intercept grid however drilling at depth is limited. Further drilling of the GG9 ore body is planned for the near future to test for extensions. The current interpretation differs from the previous interpretation with the addition of a number of faults between the main Merlot and Durif faults.

1.2 Alteration Styles

The alteration observed at Golden Gift 9 is dominated by classic volcanogenic alteration (chlorite/carbonate/silicate/sulphide) as seen elsewhere in the golden gift ore body.

1.3 Structural Geology

Significant faulting occurs within the GG9 geological model. The ore body is bound by the major fault structures of the Merlot at the top which rolls over to the east, the Durif below and the Wildcat porphyry at depth to the north. Between the major faults are a series of four North-West dipping faults which are from top to bottom the Matt Fault, Fault 3, Fault 2 and Fault 1. The faults converge at the south end of the model where the Merlot and Durif faults intersect forming an extremely faulted zone with significant variation in the fault offsets. As the faults separate to the North the offsets become more consistent. Offsets vary considerably between each fault and the distance between faults increases from South to North (refer to Figure 4 and Figure 5).

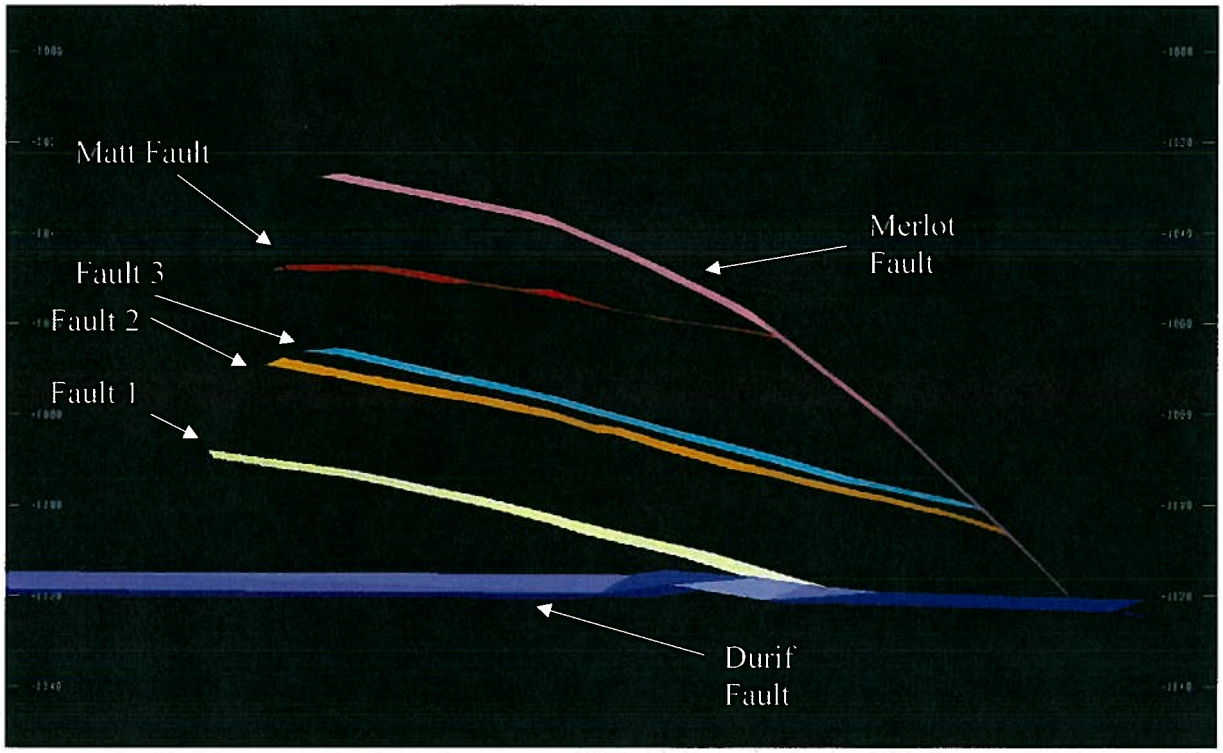

Figure 1 Cross section showing faulting within the central GG9 ore body (Section 6832.5mN).

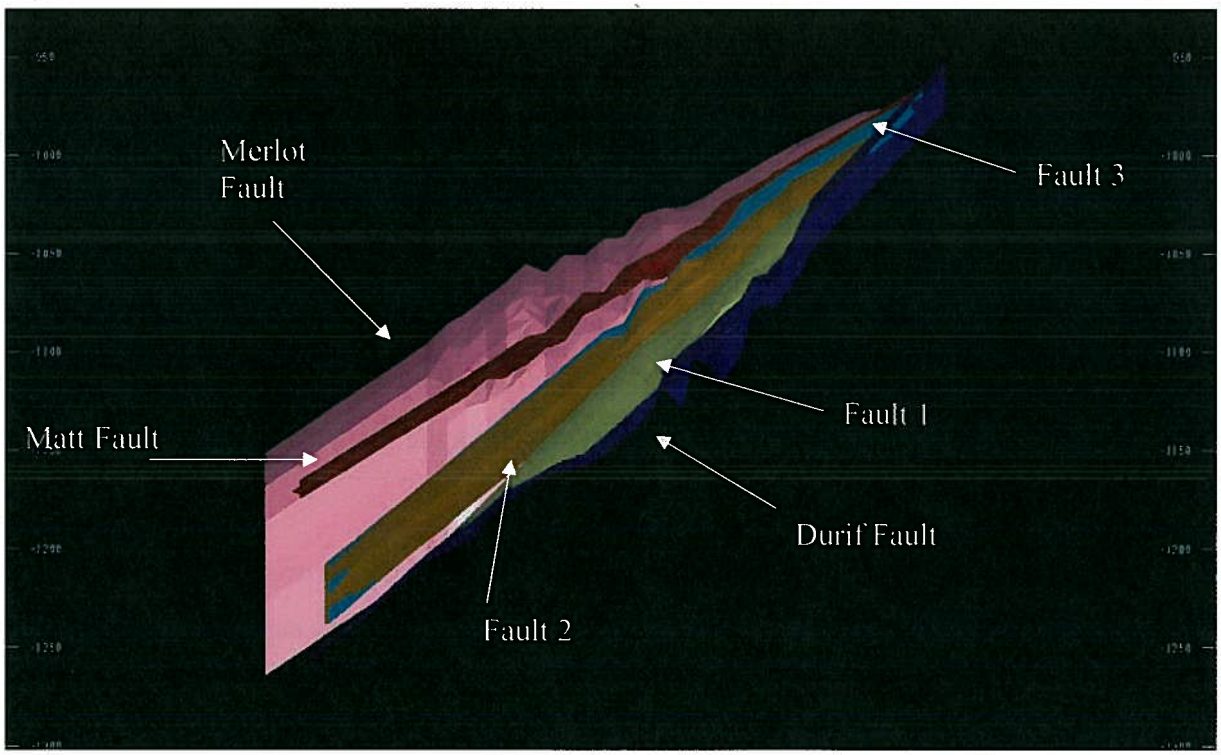

Figure 2 Long section toward 045° showing faulting within the GG9 ore body.

2 Drilling and Sampling Quality

The majority of drill intercepts used in grade estimation were drilled from the 1079 stockpile and the 1099 drive. Holes used were drilled with LTK60 equipment, with core diameters of 44mm.

The average drillhole spacing in the Indicated areas of the Model Inventory is 15mN x 15mE and 20mN x 15mE.

Diamond drilling data was used to update the Model Inventory.

Geological logging of diamond drill core is completed to the detail required for resource modelling using the standard SGM geological legend. Lithology, alteration, and structural characteristics known to affect mineralisation are collected. In addition, diamond holes are photographed.

Diamond drilling is assumed to provide 100% sample recovery unless recorded otherwise. The ore zones are generally competent and sample recovery is high. Core loss is recorded by the logging geologist and can be queried in the geological database. Regular rig audits and inspections are also used to assist with maintaining the high level of sample quality.

Drill core is prepared by cutting to intervals specified by the geologist. Sampling is within geological boundaries and intervals of one metre are desired. Predominantly the diamond core was submitted as full core for analysis. The metallurgical test work was performed on pulp samples.

2.1 QAQC Assessment

2.1.1 Quality Management Measures

Sampling of the diamond drill core is performed by samplers at the core farm.

Prior to February 2005 all drillcore was sent to Aminya laboratories in Ballarat for assaying. Ongoing QA/QC issues with Aminya led to a change in February whereby SGM Resource Definition and Grade Control drillcore has since been sent to Genalysis in Kalgoorlie and Amdel in South Australia for assay.

The general procedures for QA/QC are outlined below:

- Production diamond drill core is routinely Full Core sampled with one in five sent as half core.
- Routine assaying of diamond drill core samples is undertaken by fire assay (50 gram charges) with an AAS finish. Assays are reported to an LLD of 0.01ppm. For sample reporting below LLD, a value of 0.5xLLD is used in resource estimation.
- Sample QA/QC procedure incorporates routine check assays including repeats (re-assay or repeat assay), duplicate (second sample taken after pulverisation), splits (half coarse sample split, at ratio approx 1:10 samples), and standards.
- SGM Standard reference material has been prepared by Ore Research of Melbourne, from ore samples from Magdala Mine.
- Analysis results are received from laboratory in fixed digital format. The load routine imports assay data matching against sample ID created during logging procedure.
- Analysis of received standard values and repeat and duplicate information is used to monitor the performance of the assaying process.

2.1.2 Location of Data Points

Diamond drillhole collars are surveyed by the Survey Department and single shot down hole cameras have been used to survey the drillhole path. Electronic single shot instruments are now used in preference to the Eastman mechanical cameras. Magnetic intensity is recorded and used to assess validity of record. Dip +/- 1.5° and azimuth +/- 3° is highlighted and assessed.

All data is stored within the "acQuire" Database Management System which is structured on a Data Model which applies rules of relational databases to mining and exploration data. The primary key is HoleID.

The drill holes with inaccurate hole collar locations have been excluded from the resource model.

2.1.3 Database Integrity

All logging of diamond drill core is undertaken using "Datcol" software, to a defined logging scheme. It is a barcode system, eliminating errors involved in keying. Logging is both quantitative and qualitative but in both cases, only valid codes or values can be recorded.

Additional validation occurs during upload of data to database using the acQuire DBMS. Checks include:

- All alphanumeric codes (eg lithology) are valid and not duplicated
- All numeric fields are within acceptable limits and not duplicated
- Sample from-to depths cannot be greater than the maximum hole depth
- Checks are performed for overlapping samples

Alpha analysis codes are stored as logged and/or reported eg NS (Not Sampled), IS (Insufficient Sample), <0.01. The database MetaAssayExport table records equivalent values which are substituted by client software (eg Minesight). The convention for defined values is a numeric value half detection limit for results at LLD, and for all other codes, -1 is substituted.

After data compilation is complete, it is critically reviewed by geologists with on-going scrutiny using logs, section/plan plotting and 3D modelling.

3 Bulk Density

The SG values used in this model have been carried over from the previous GG9 model, 2.85g/cm3. It is intended that SG values specific to the current GG9 model that incorporate recent drilling be formulated in the near future.

4 Metallurgical Test Work

The global metallurgical recovery for GG9 is 88.8%.

5 Geological Modelling

The re-interpretation of GG9 in quarter one of 2007 was initially undertaken on paper 10m spaced cross sections and 20m spaced level plans. Interpretation of the southern end of the model involved 2.5m spaced paper cross sections used in conjunction with Minesight. The paper sections were digitized and imported into Minesight where the strings were snapped to drillholes where available. Solids were constructed by then linking strings to form wireframe solids. Solids were overlapped, and then clipped against each other to form common surfaces.

The following general geological domains have been interpreted (refer to Figure 3). These domains have been split further into individual fault blocks for estimation purposes, due to offsets across fault boundaries, but the general geological characteristics remain the same throughout.

Domain	Description
HG Kingston Lodes	High grade lode located on the Kingston basalt contact
LG Kingston Lodes	Low grade lode located between the high grade Kingston Lode and the Mineschist.
Delta Lode	Located in the waterloo position within the Kingston basalt, bound by basalt on both sides.
Kingston Basalt	Basalt
Mineschist	Mineschist

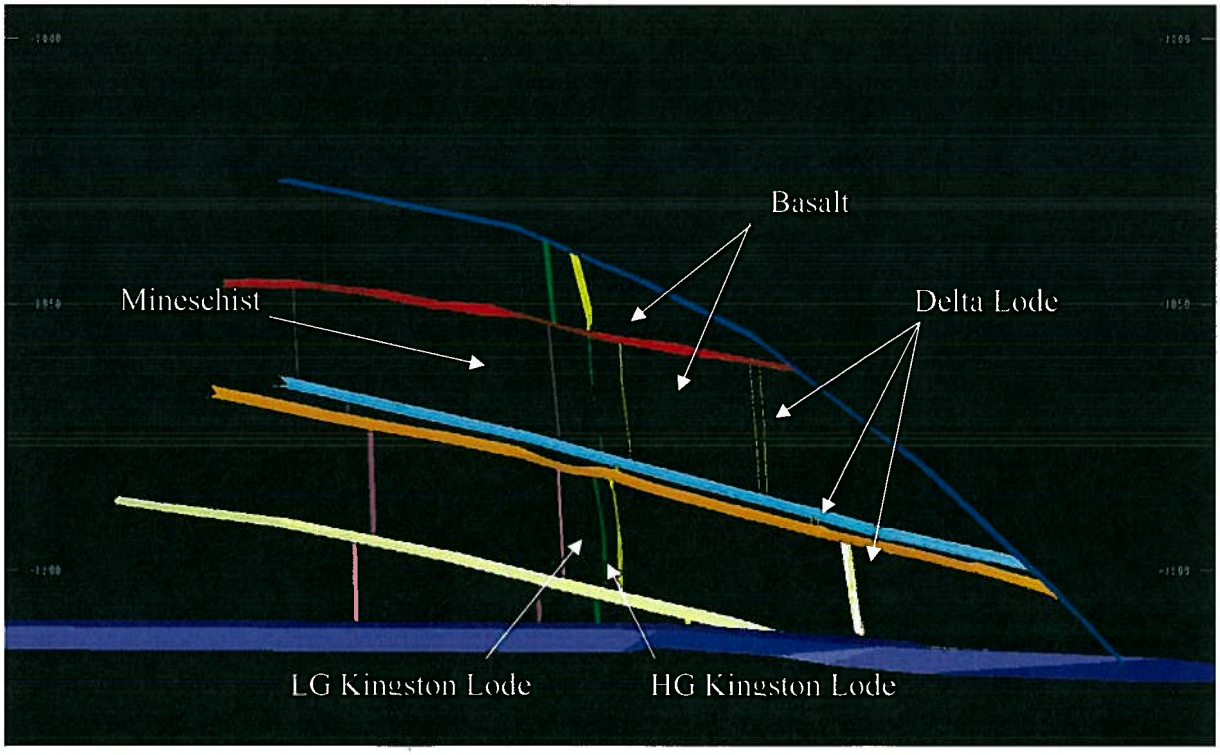

Figure 3 Cross section at 6830mN highlighting the different geological domains.

5.1 Sub-Domaining

No further sub-domaining of the major mineralised surfaces has been carried out. The geological solids and domain numbers are in Table 2.

Geology	Geology Final Solids	Domain
Kingston Ore Volc 1	Kingston Ore Volc 1 revised	601
Kingston Ore Volc 2	Kingston Ore Volc 2 revised	602
Kingston Ore Volc 3	Kingston Ore Volc 3 revised	603
Kingston Ore Volc 4	Kingston Ore Volc 4 revised	604
Kingston Ore Volc 5	Kingston Ore Volc 5 revised	605
Kingston Volc Weak 1	Kingston Volc Weak 1 revised	606
Kingston Volc Weak 2	Kingston Volc Weak 2 revised	607
Kingston Volc Weak 3	Kingston Volc Weak 3 revised	608
Kingston Volc Weak 4	Kingston Volc Weak 4 revised	609
Kingston Volc Weak 5	Kingston Volc Weak 5 revised	610
Delta Volc 1 revised	Delta Volc 1 revised	611
Delta Volc 2 revised	Delta Volc 2 revised	612
Delta Volc 3 revised	Delta Volc 3 revised	613
Delta Volc 4 revised	Delta Volc 4 revised	614

Table 2 Geological domains of GG9.

5.2 Domaining and Descriptive Statistics

As described previously based on the current drillhole information, the geological model is divided into fourteen separate areas. As a general practice, the raw sample results have been composited to 2.0m intervals and a top-cut applied, if deemed necessary, after compositing.

A listing of descriptive statistics is provided in Table 3. The composite sets selected for the resource interpolation are highlighted in yellow. Histograms of the 2.0m composite lengths and the gold assay values are shown in Figure 4 to Figure 17. A top-cut value of 15.0g/t Au was applied to domain 601, 15g/t Au for domain 603, 15g/t Au for domain 607, 15g/t Au for domain 611, 10g/t Au for domain 613 and 10g/t Au for domain 614. A general description of the domains and the top-cut applied is outlined in Table 4.

A 15g/t top-cut for domain 601 affects 2% of the sample population and decreases the coefficient of variation to 1.065. A 15g/t top-cut for domain 603 affects 11.76% of the sample population and decreases the coefficient of variation to 0.572. A 15g/t top-cut for domain 604 affects 5.78% of the sample population and decreases the coefficient of variation to 1.006. A 15g/t top-cut for domain 605 affects 6.52% of the sample population and decreases the coefficient of variation to 1.387. A 15g/t top-cut for domain 607 affects 5.48% of the sample population and decreases the coefficient of variation to 1.549. A 15g/t top-cut

for domain 611 affects 36.36% of the sample population and decreases the coefficient of variation to 1.179. A 15g/t top-cut for domain 611 affects 36.36% of the sample population and decreases the coefficient of variation to 1.179. A 10g/t top-cut for domain 613 affects 10% of the sample population and decreases the coefficient of variation to 1.379. A 10g/t top-cut for domain 614 affects 19.23% of the sample population and decreases the coefficient of variation to 1.275.

Leviathan Resources - Stawell Gold Mines	DESCRIPTIVE STATISTICS OF INPUT DATA								
GG9 RESOURCE MODEL									
Date:	May-2007								
Variable	Data Type(s)	Number of Samples	Count used	Minimum g/t	Maximum g/t	Mean g/t	Std Dev g/t	Variance g/t²	Coeff of Var
Domain 601									
Au Raw	DD	115	115	0.01	39.800	4.102	5.790	33.524	1.412
Au 2.0m Composites	DD	49	49	0.02	23.004	4.102	4.609	21.244	1.124
Au TC 15 - 1 sample	DD	49	49	0.02	15.000	3.964	4.130	17.056	1.042
Length Raw	DD	118	118	0.15	2.850	0.872			
Length 2.0m Composites	DD	49	49	0.55	2.550	1.950			
Domain 602									
Au Raw	DD	130	130	0.01	87.700	6.270	9.175	84.171	1.463
Au 2.0m Composites	DD	59	59	0.17	20.734	6.270	5.604	31.409	0.894
Length Raw	DD	132	132	0.20	4.950	0.971			
Length 2.0m Composites	DD	59	59	1.05	3.000	2.040			
Domain 603									
Au Raw	DD	38	38	0.07	21.710	8.676	6.508	42.349	0.750
Au 2.0m Composites	DD	17	17	0.07	20.920	8.676	5.418	29.351	0.624
Au TC 15 - 2sample	DD	17	17	0.07	15.000	8.236	4.592	21.086	0.558
Length Raw	DD	38	38	0.25	1.500	0.857			
Length 2.0m Composites	DD	17	17	1.40	2.100	1.910			
Domain 604									
Au Raw	DD	153	153	0.01	175.000	6.464	14.494	210.079	2.242
Au 2.0m Composites	DD	69	69	-	54.240	6.464	9.431	88.949	1.459
TC to 15 - 4 samples	DD	69	69	-	15.000	5.109	4.935	24.351	0.966
Length Raw	DD	156	156	0.15	5.500	0.884			
Length 2.0m Composites	DD	69	69	0.45	2.950	1.890			
Domain 605									
Au Raw	DD	90	90	0.01	55.500	3.357	7.088	50.240	2.111
Au 2.0m Composites	DD	46	46	0.10	27.000	3.357	5.508	30.335	1.641
TC to 15 - 3 samples	DD	46	46	0.10	15.000	2.929	4.087	16.704	1.395
Length Raw	DD	95	95	0.30	3.600	0.987			
Length 2.0m Composites	DD	46	46	1.00	2.650	1.850			
Domain 601-605									
Au Raw	DD	526	526	0.01	175.000	5.512	10.080	101.606	1.829
Au 2.0m Composites	DD	240	240	0.01	54.240	5.512	6.868	47.168	1.246
Length Raw	DD	539	539	0.15	5.500	0.919			
Length 2.0m Composites	DD	240	240	0.45	3.000	1.930			
Domain 606									
Au Raw	DD	62	62	0.01	5.480	0.774	1.332	1.775	1.721
Au 2.0m Composites	DD	28	28	0.01	3.036	0.774	0.942	0.888	1.218
Length Raw	DD	67	67	0.20	2.000	0.886			
Length 2.0m Composites	DD	28	28	1.15	2.900	1.970			
Domain 607									
Au Raw	DD	142	142	0.01	49.260	2.870	6.455	41.664	2.249
Au 2.0m Composites	DD	73	73	0.01	31.658	2.870	5.833	34.022	2.032
TC to 15 - 4 samples	DD	73	73	0.01	15.000	2.348	3.633	13.196	1.547
Length Raw	DD	146	146	0.15	2.300	1.016			
Length 2.0m Composites	DD	73	73	0.80	2.800	1.960			
Domain 608									
Au Raw	DD	19	19	0.01	7.690	0.716	1.519	2.306	2.121
Au 2.0m Composites	DD	10	10	0.01	3.810	0.716	1.083	1.172	1.511
Length Raw	DD	19	19	0.20	1.600	1.045			
Length 2.0m Composites	DD	10	10	1.20	295.000	1.990			

Domain 609									
Au Raw	DD	139	139	0.01	9.180	0.279	0.663	0.440	2.376
Au 2.0m Composites	DD	69	69	0.01	1.923	0.279	0.374	0.140	1.341
Length Raw	DD	142	142	0.15	2.100	0.945			
Length 2.0m Composites	DD	69	69	0.30	2.950	1.860			
Domain 610									
Au Raw	DD	117	117	0.01	20.400	1.160	2.394	5.730	2.064
Au 2.0m Composites	DD	61	61	0.01	9.027	1.160	1.807	3.267	1.558
Length Raw	DD	119	119	0.45	3.000	1.041			
Length 2.0m Composites	DD	61	61	0.90	2.950	1.950			
Domain 606-610									
Au Raw	DD	497	497	0.01	49.260	1.376	3.961	15.693	2.879
Au 2.0m Composites	DD	241	241	0.01	31.658	1.376	3.536	12.501	2.569
Length Raw	DD	493	493	0.15	3.000	0.985			
Length 2.0m Composites	DD	241	241	0.30	2.950	1.930			
Domain 611									
Au Raw	DD	23	23	0.02	63.200	8.162	16.623	276.317	2.037
Au 2.0m Composites	DD	11	11	0.03	30.330	8.162	11.680	136.425	1.431
TC to 10 - 4 samples		11	11	0.02	10.000	3.674	4.516	20.397	1.229
Length Raw	DD	23	23	0.30	1.500	0.930			
Length 2.0m Composites	DD	11	11	1.30	2.400	1.950			
Domain 612									
Au Raw	DD	100	100	0.01	35.420	2.090	5.354	28.666	2.562
Au 2.0m Composites	DD	47	47	0.01	15.300	2.090	3.776	14.260	1.806
Length Raw	DD	100	100	0.20	1.500	0.936			
Length 2.0m Composites	DD	47	47	1.05	2.900	1.990			
Domain 613									
Au Raw	DD	21	21	0.01	28.750	4.417	8.235	67.809	1.864
Au 2.0m Composites	DD	10	10	0.02	21.540	4.417	7.326	53.670	1.659
TC to 10 - 1 samples	DD	10	10	0.02	10.000	2.880	3.827	14.648	1.329
Length Raw	DD	21	21	0.45	1.400	1.055			
Length 2.0m Composites	DD	10	10	1.60	2.950	2.220			
Domain 614									
Au Raw	DD	60	60	0.01	65.130	4.757	10.701	114.518	2.250
Au 2.0m Composites	DD	26	26	0.07	30.810	4.757	8.141	66.276	1.711
TC to 10 - 5 samples	DD	26	26	0.07	10.000	2.789	3.632	13.189	1.302
Length Raw	DD	60	60	0.20	1.400	0.911			
Length 2.0m Composites	DD	26	26	1.70	2.950	2.100			
Domain 611-614									
Au Raw	DD	204	204	0.01	65.130	3.796	9.330	87.047	2.458
Au 2.0m Composites	DD	94	94	0.01	30.805	3.796	7.069	49.967	1.862
Length Raw	DD	204	204	0.20	1.500	0.940			
Length 2.0m Composites	DD	94	94	1.05	2.950	2.040			

Table 3. Descriptive statistics of the 2.0m composites per domain.

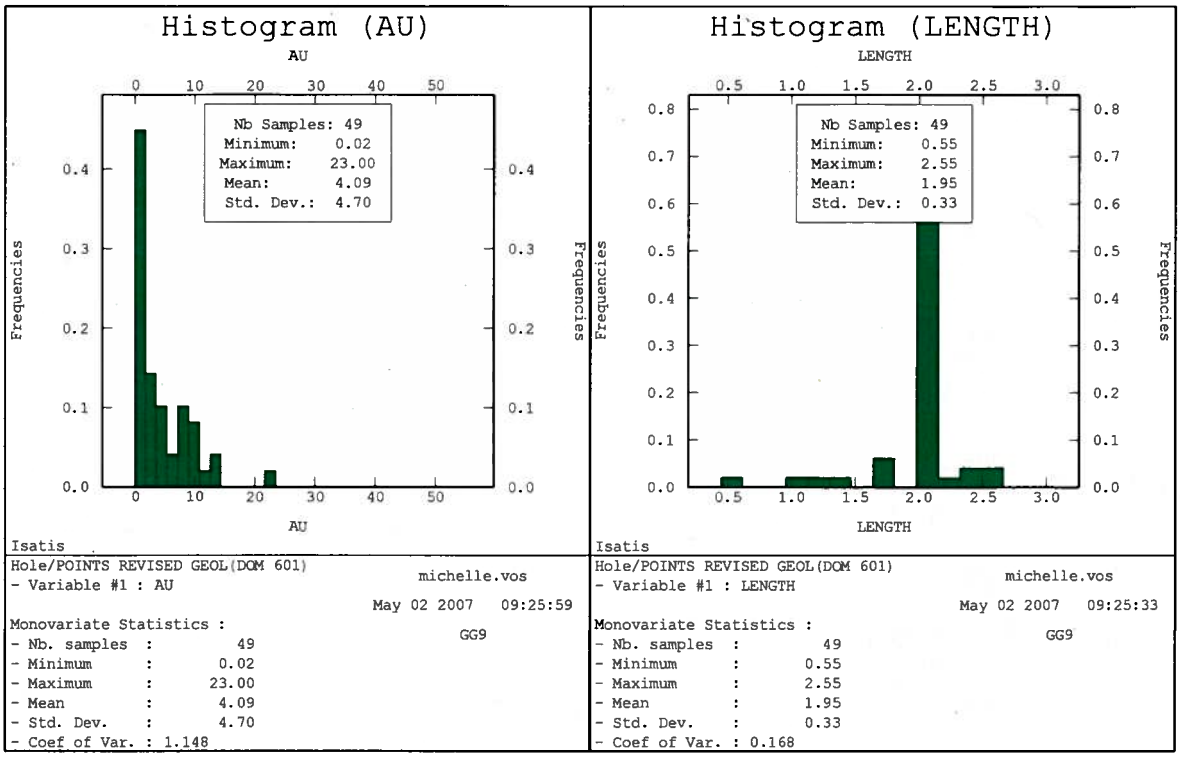

Figure 4. Histograms of Au (left) and length (right) in 2.0m composites of the Domain 601.

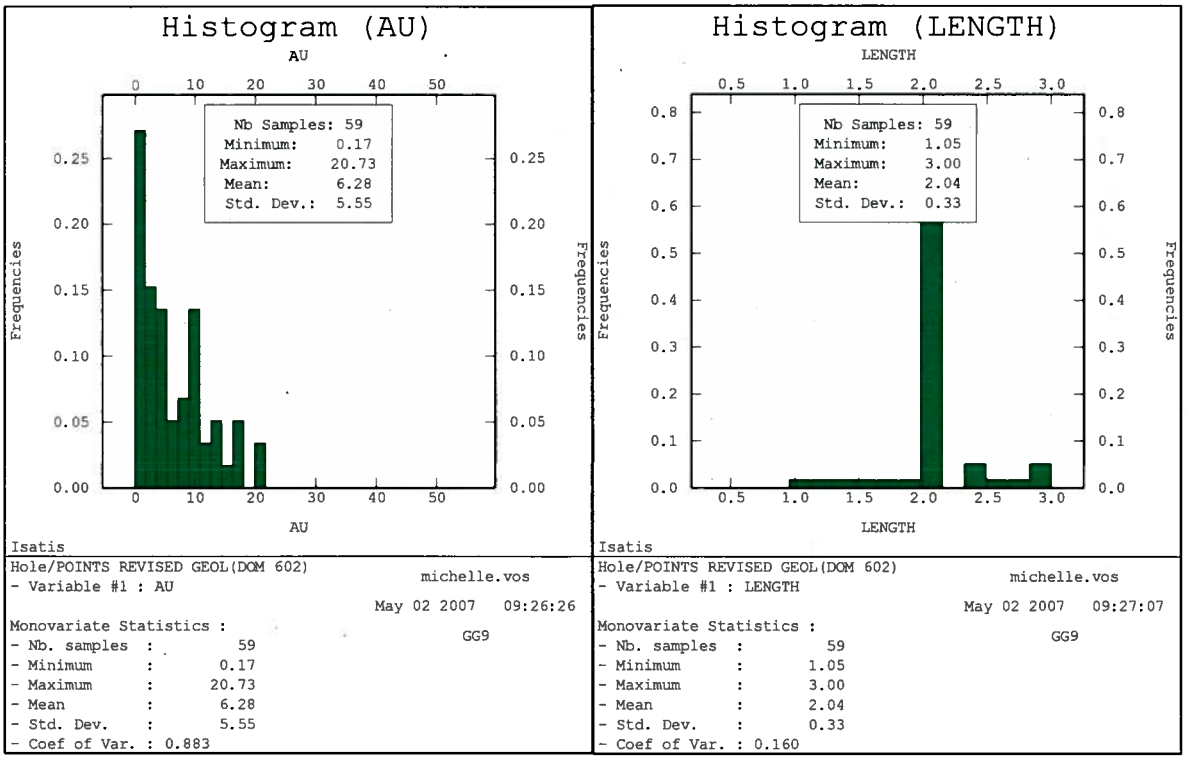

Figure 5. Histograms of Au (left) and length (right) in 2.0m composites of the Domain 602.

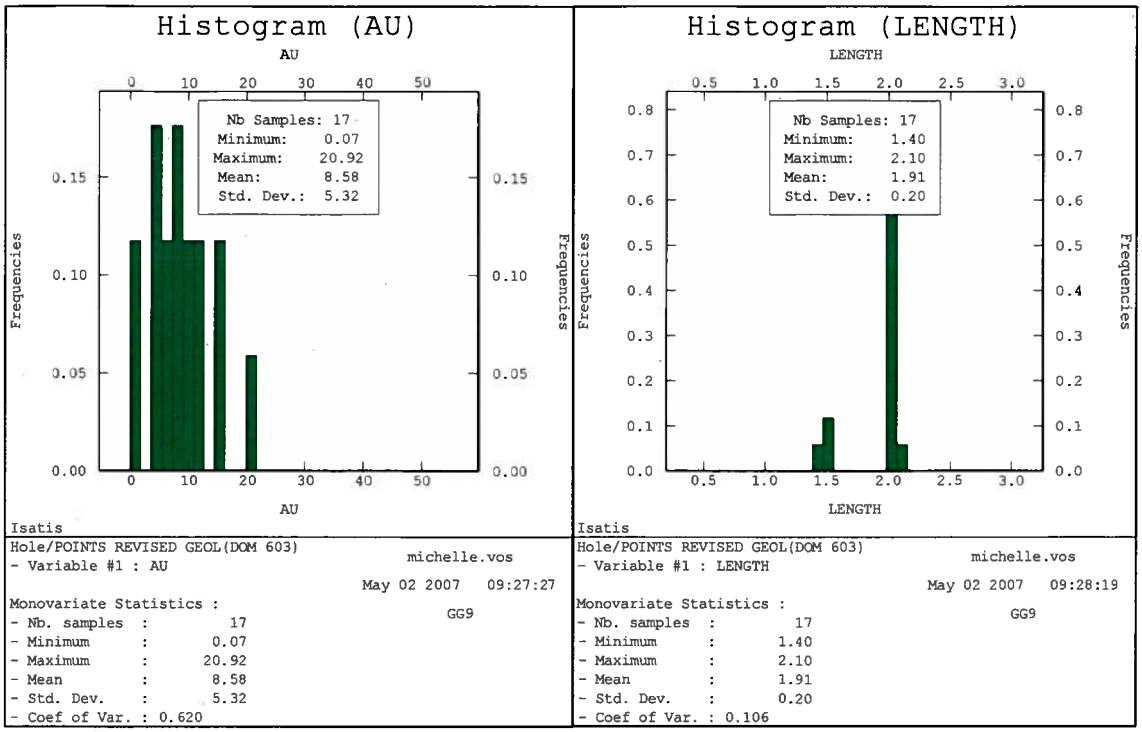

Figure 6. Histograms of Au (left) and length (right) in 2.0m composites of Domain 603.

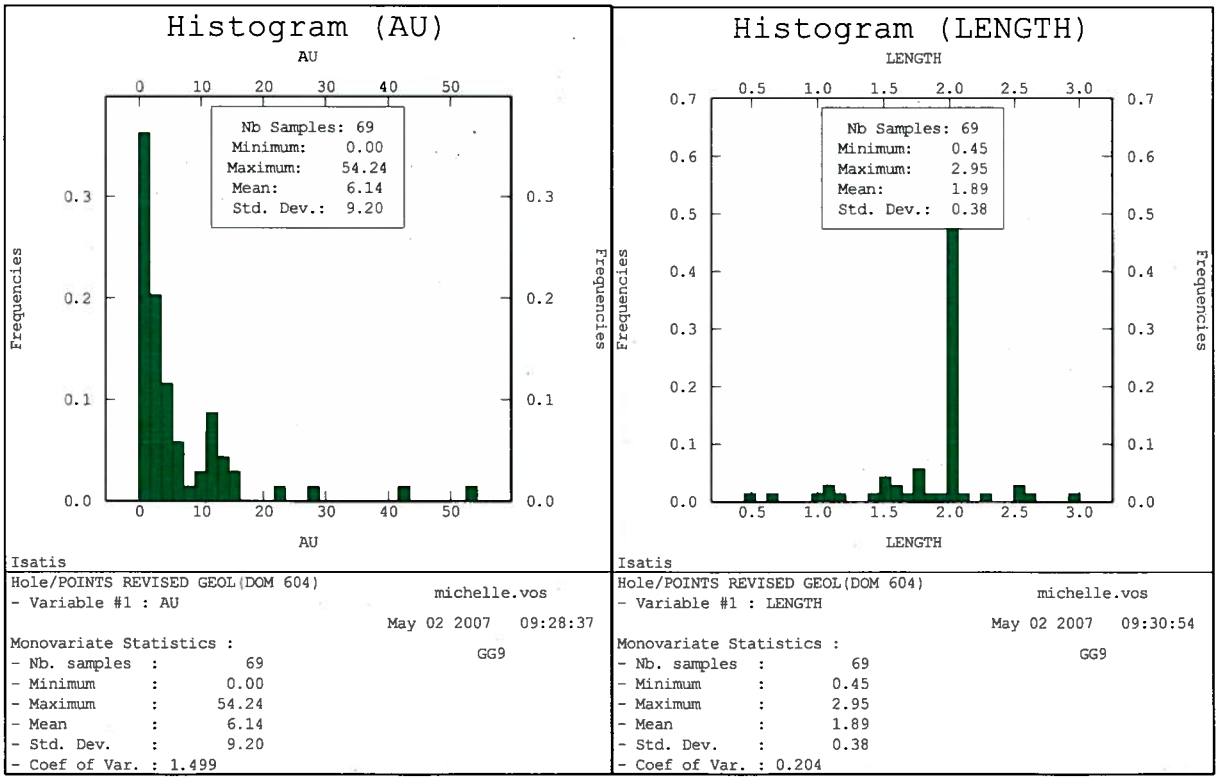

Figure 7. Histograms of Au (left) and length (right) in 2.0m composites of Domain 604.

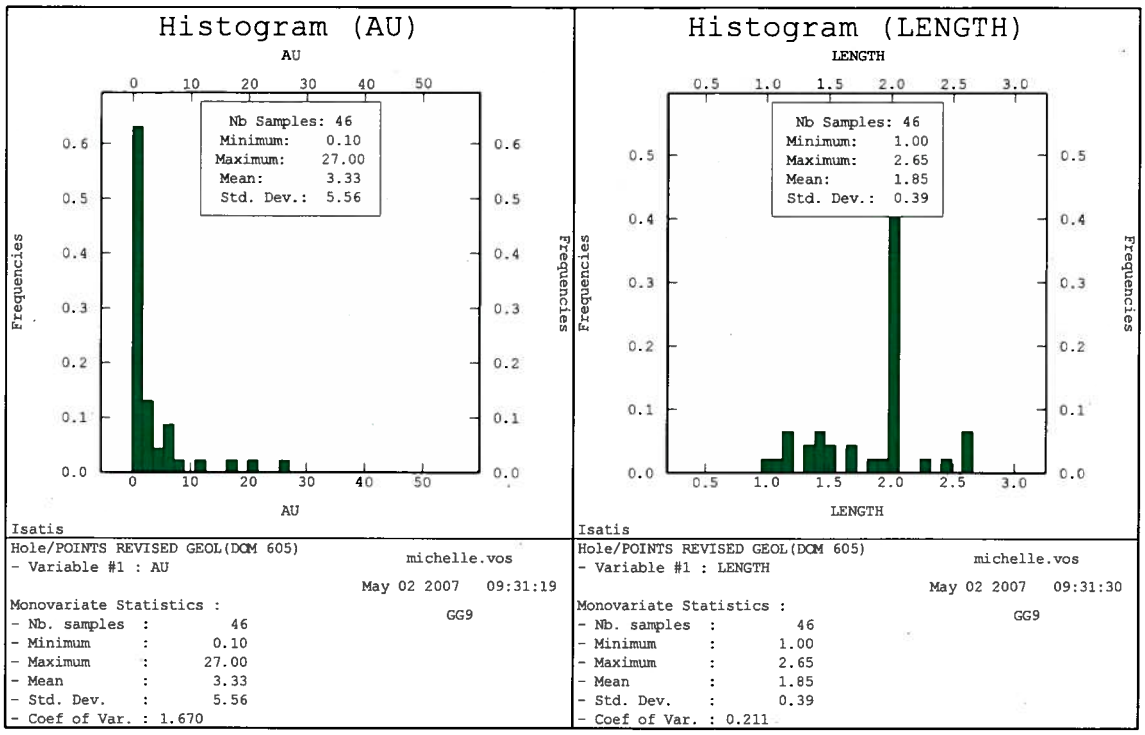

Figure 8. Histograms of Au (left) and length (right) in 2.0m composites of Domain 605.

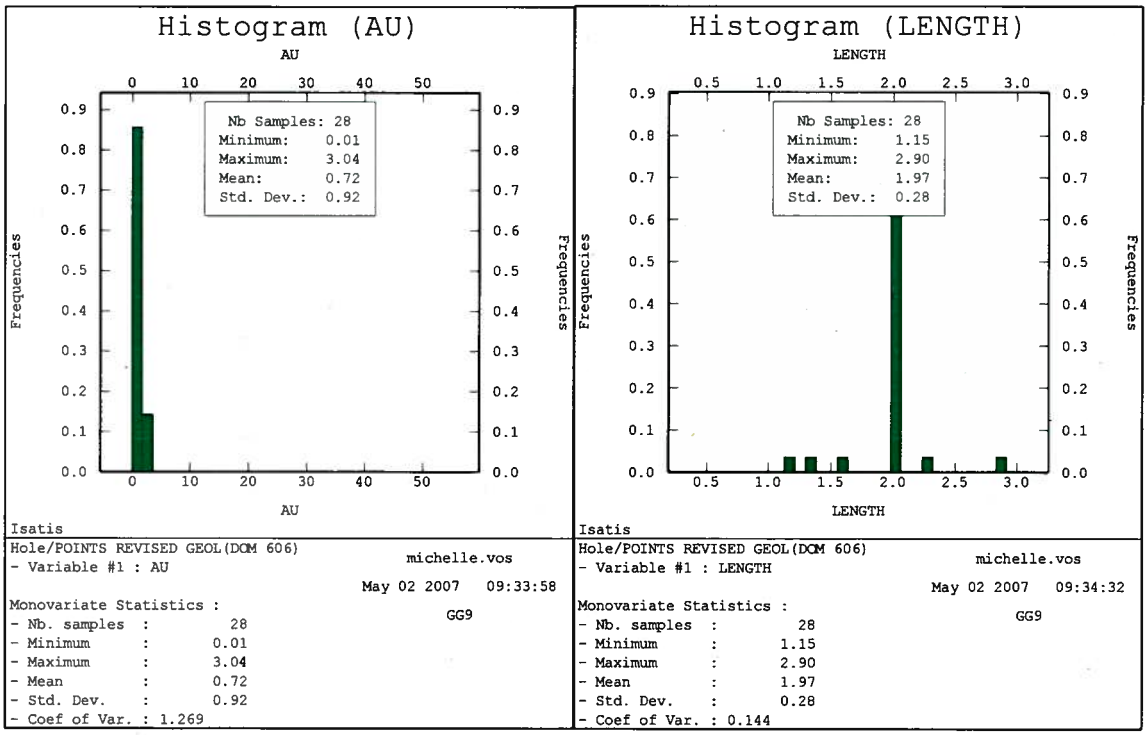

Figure 9. Histograms of Au (left) and length (right) in 2.0m composites of Domain 606.

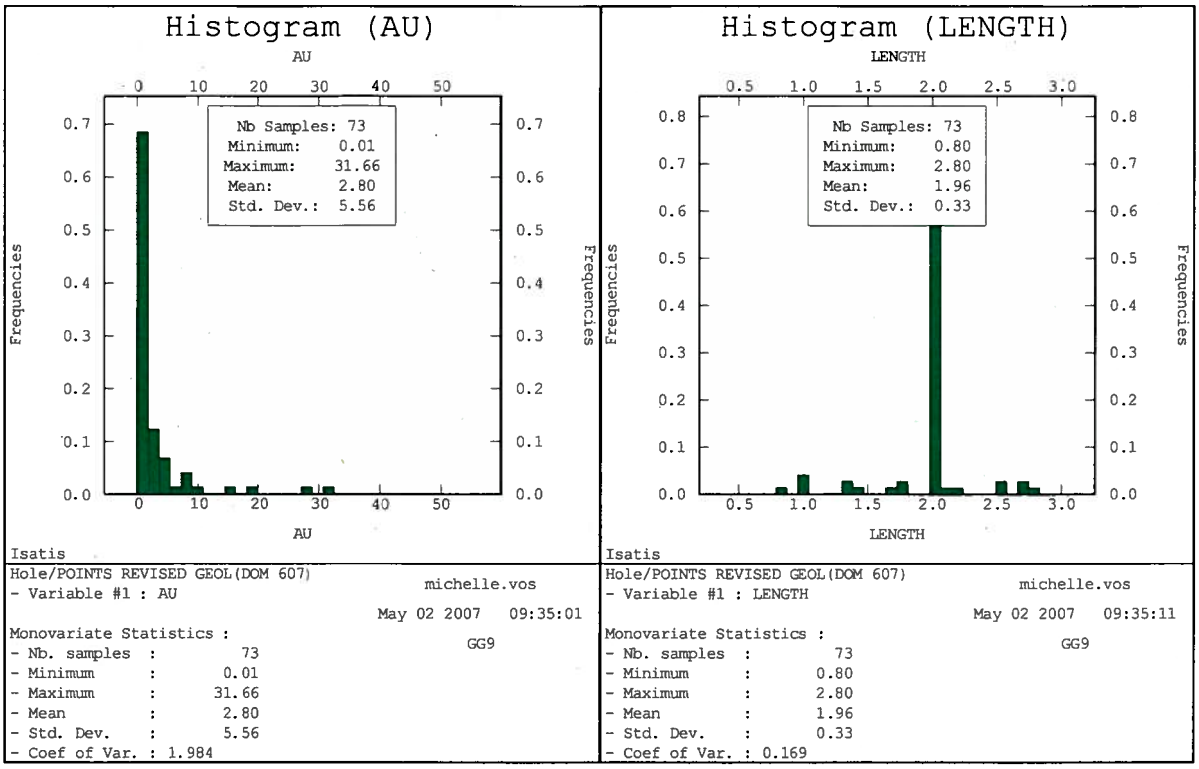

Figure 10. Histograms of Au (left) and length (right) in 2.0m composites of Domain 607.

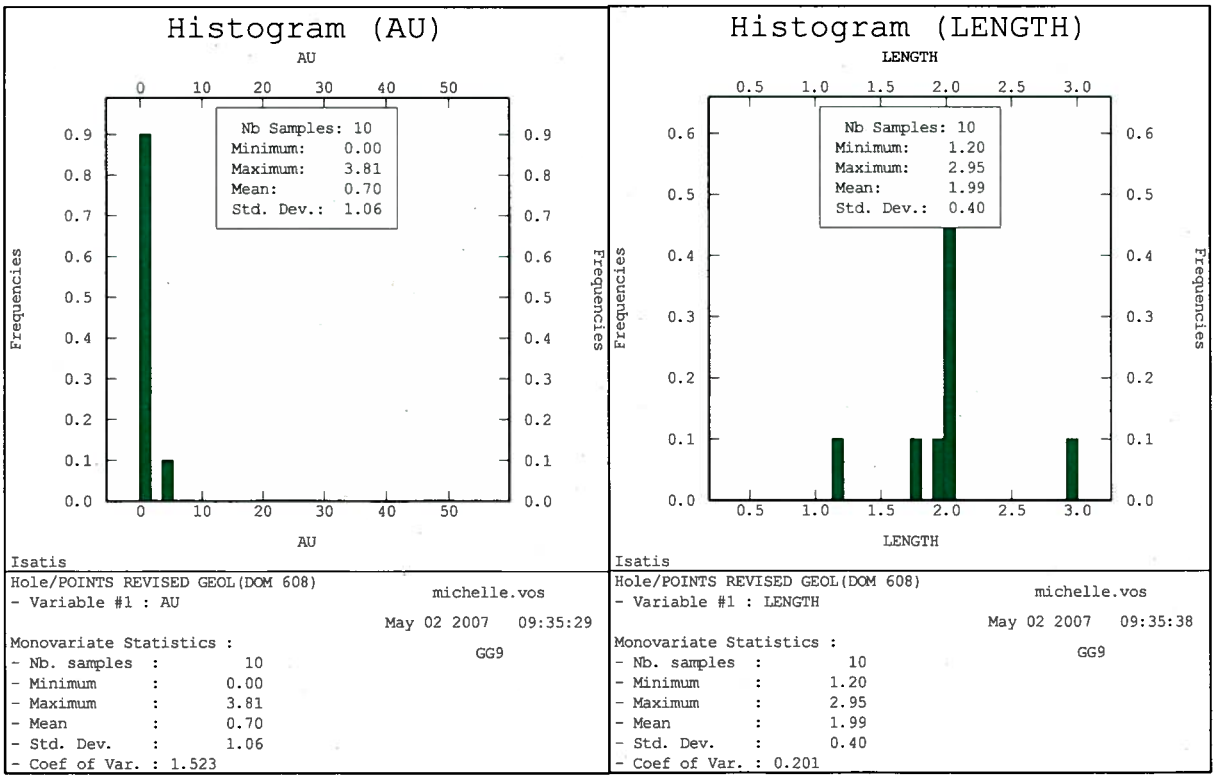

Figure 11. Histograms of Au (left) and length (right) in 2.0m composites of Domain 608.

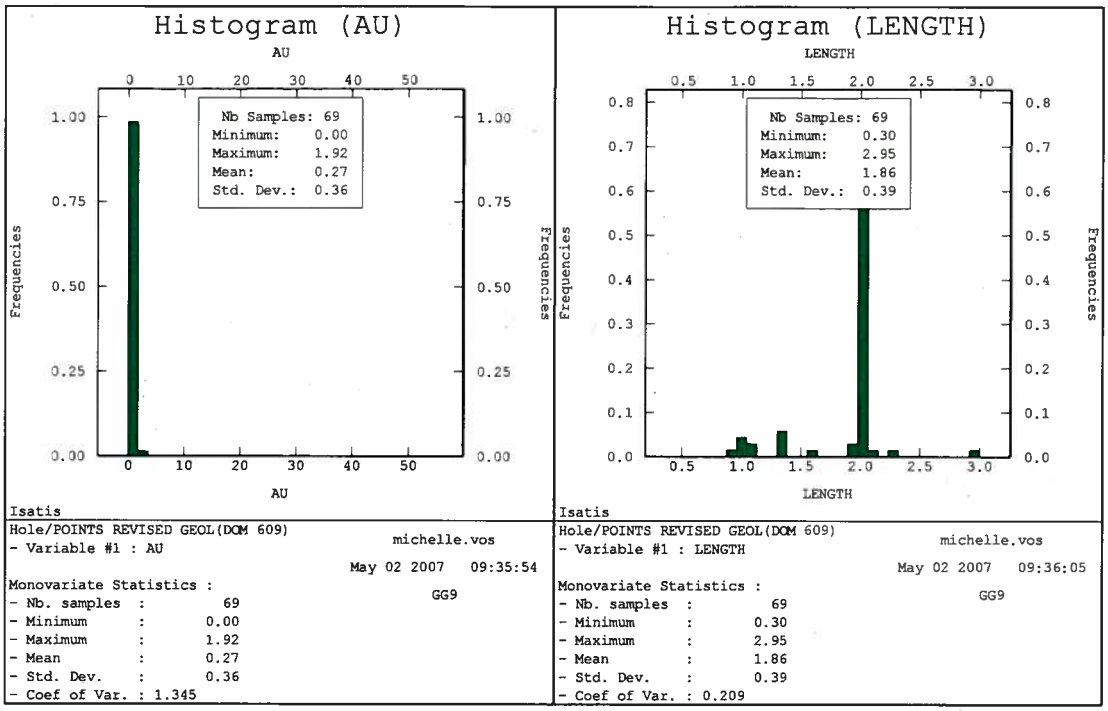

Figure 12. Histograms of Au (left) and length (right) in 2.0m composites of Domain 609.

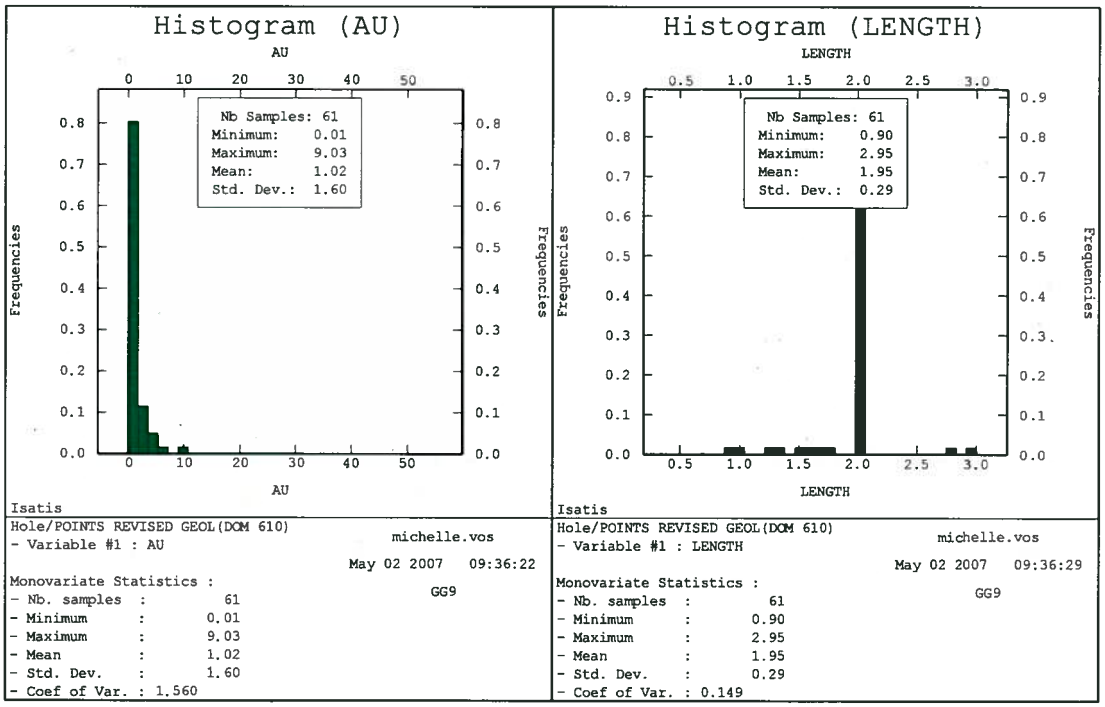

Figure 13. Histograms of Au (left) and length (right) in 2.0m composites of Domain 610.

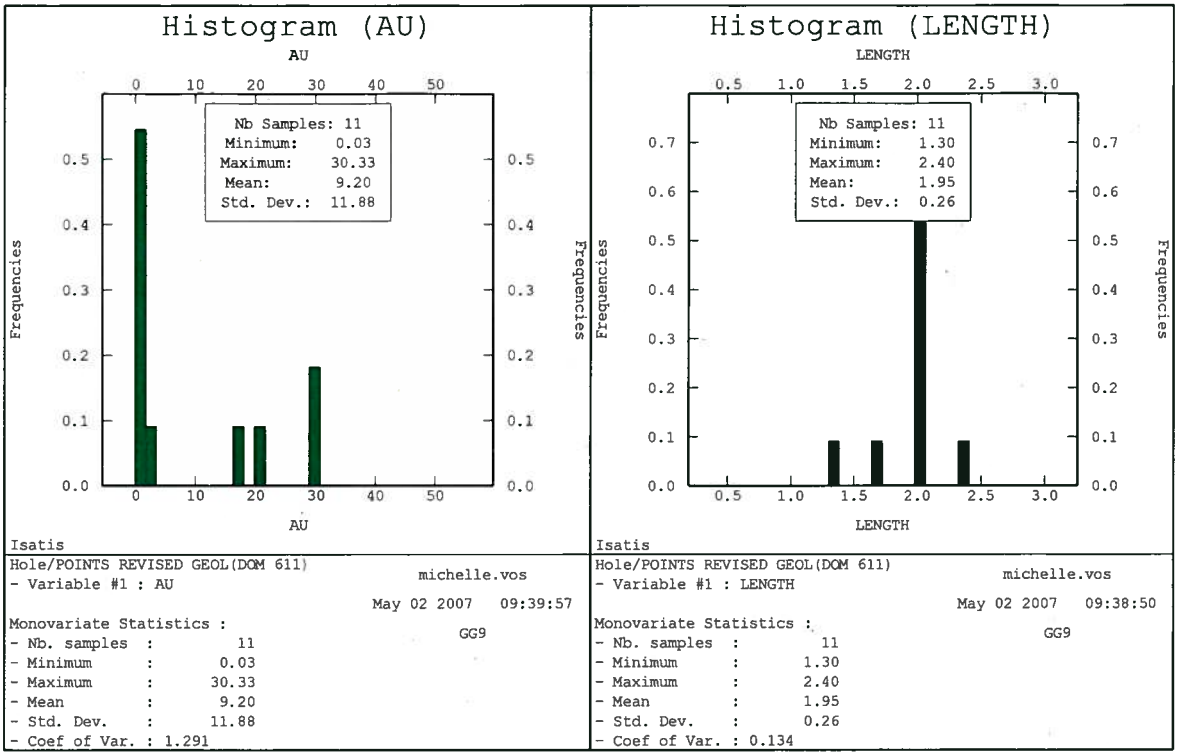

Figure 14. Histograms of Au (left) and length (right) in 2.0m composites of Domain 611.

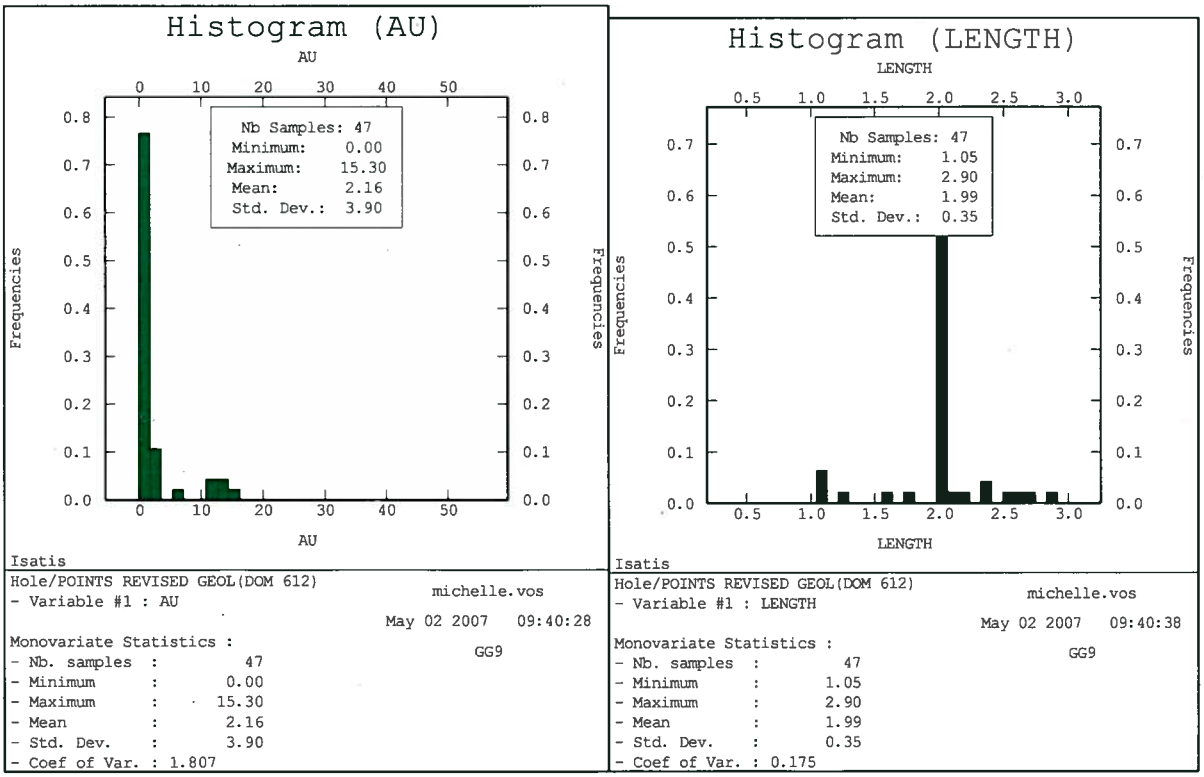

Figure 15. Histograms of Au (left) and length (right) in 2.0m composites of Domain 612.

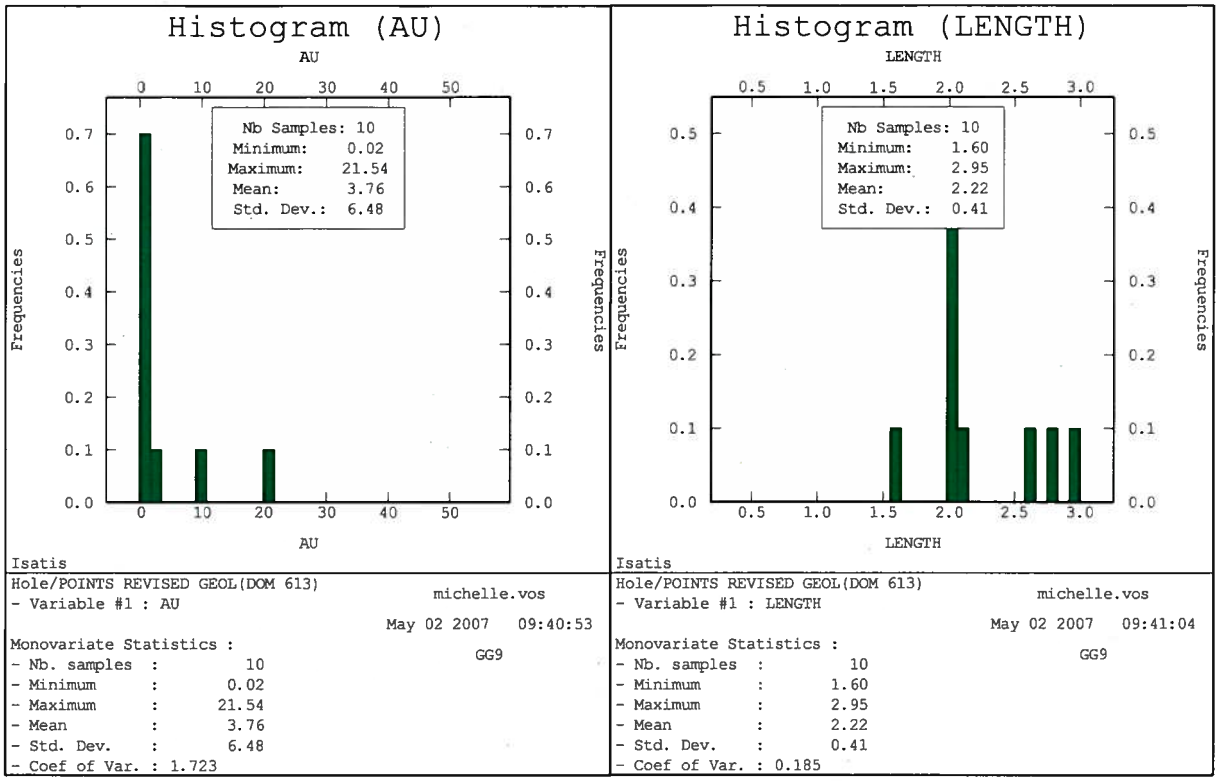

Figure 16. Histograms of Au (left) and length (right) in 2.0m composites of Domain 613.

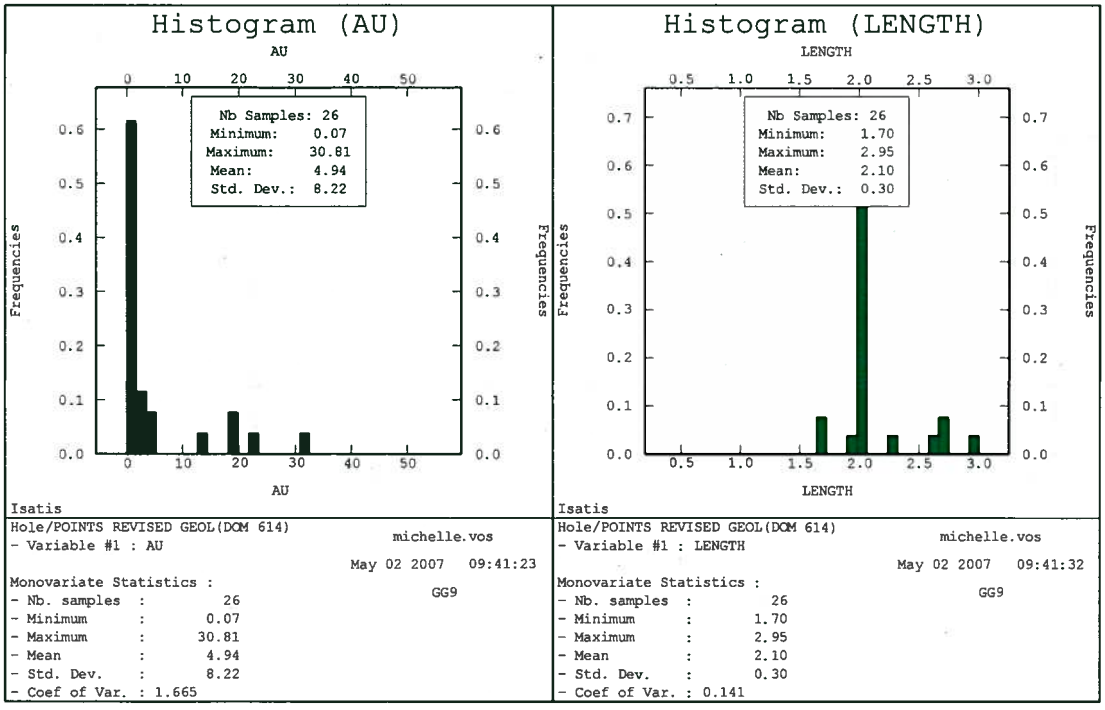

Figure 17. Histograms of Au (left) and length (right) in 2.0m composites of Domain 614.

Domain Number	Description	Top-cut g/t	Number of sample cut
601	Kingstone ore volc	15	1
604	Kingstone ore volc	15	4
603	Kingstone ore volc	15	2
605	Kingstone ore volc	15	3
607	Kingstone weak volc	15	4
611	Delta Lode	10	4
613	Delta Lode	10	1
614	Delta Lode	10	5

Table 4. General description of domains and top-cut applied.

5.3 Variography

Detailed variography was carried out using ISATIS. The number of samples per domain is low due to the structural complexity of the geological model. Ideally unfolding the geometry and generating experimental variograms would be the best method of analysis. Minesight and Isatis are not capable of this function.

Valid variograms were difficult to obtain, omnidirectional variograms of combined domains 600 to 605 are not reliable as the paired data maybe structurally offset. Omnidirectional variograms of domains 600-605, domain 602 and domain 609 suggested ranges in the order of 20 to 40m, this is seen elsewhere along strike in the golden gift domains (refer to S:\Mining-Geology\Golden Gift 9\Minesight\Geology\Resource Model May 2007\documents – GG9_Resource Model 2007.xls - variogram analysis). A shorter range of 8-15m is commonly seen across strike. Due to the lack of confidence in the variography, a range of 30m along strike, 20m across strike and 15m near vertical will be used in the estimation for all domains. The near vertical range was estimated from downhole variography of domains 600-605 (combined), refer to Figure 8. The sill and nugget values were calculated using the total variance and a 30% nugget (refer to Table 5).

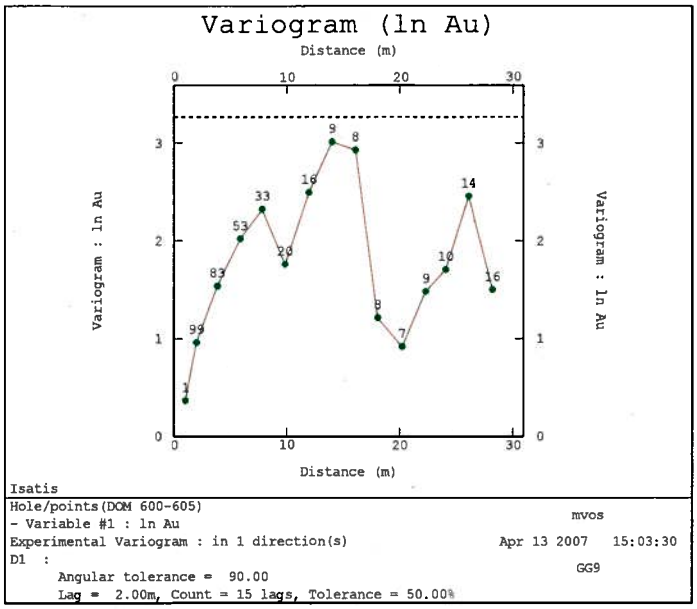

Figure 18. Experimental variogram of domain 600 -605.

Perseverance - Stawell Gold Mine						VARIOGRAM PARAMETER TABLE																	
AREA	VREFNUM	axis 1	axis 2	axis 3	Nugget	1st Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	2nd Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	3rd Structure Spherical	pri. axis (x)	sec. axis (y)	ter. axis (z)	variance	Total Variance	Nugget
		rotation																				
GG9	601	315	-35	0	6.76	1	8	8	5	5.63	1	30	20	15	10.134						22.52	30%
GG9	602	315	-35	0	9.39	1	8	8	5	7.83	1	30	20	15	14.09						31.31	30%
GG9	603	315	-35	0	8.49	1	8	8	5	7.075	1	30	20	15	12.735						28.30	30%
GG9	604	315	-35	0	7.2	1	8	8	5	6.003	1	30	20	15	10.805						24.01	30%
GG9	605	315	-35	0	4.75	1	8	8	5	3.96	1	30	20	15	7.128						15.84	30%
GG9	606	315	-35	0	0.26	1	8	8	5	0.22	1	30	20	15	0.396						0.88	30%
GG9	607	315	-35	0	9.39	1	8	8	5	7.825	1	30	20	15	14.085						31.30	30%
GG9	608	315	-35	0	0.34	1	8	8	5	0.28	1	30	20	15	0.504						1.12	30%
GG9	609	315	-35	0	0.039	1	8	8	5	0.033	1	30	20	15	0.059						0.13	30%
GG9	610	315	-35	0	0.77	1	8	8	5	0.64	1	30	20	15	1.152						2.56	30%
GG9	611	315	-35	0	14.37	1	8	8	5	11.97	1	30	20	15	21.55						47.89	30%
GG9	612	315	-35	0	4.56	1	8	8	5	3.803	1	30	20	15	6.844						15.21	30%
GG9	613	315	-35	0	3.89	1	8	8	5	3.24	1	30	20	15	5.83						12.96	30%
GG9	614	315	-35	0	4.02	1	8	8	5	3.35	1	30	20	15	6.03						13.40	30%

Table 5. MINESIGHT variogram parameter table used for block model estimation.

6 Block Model Estimation

6.1 Neighbourhood optimisation

The neighbourhood optimisation is used to determine appropriate search dimensions and minimum and maximum number of samples during Kriging interpolation. It is done in an empirical manner by testing and spatially viewing, in ISATIS, characteristics like the distribution of negative kriging weights, the achievable theoretical regression slope, block kriging efficiency, etc, for a number of model blocks that are located in "favourable" and "unfavourable positions" with respect to the available drillholes within a domain.

Due to the tight deadline of the resource model the block size of 5m east, 10m north and 10m RL was used in accordance with domain geometry, data spacing and spatial continuity (refer to Table 6). This model is a rotated model of 315° Northwest. It is highly recommended that a full neighbourhood analysis of the block size, no of samples, dimensions of the search ellipse and no of discretisation points used in the model be completed for the next model update.

	Min	*Max*	*Cell Size*	*No. Cells*	*Origin (AMG)*
East	0	250	5	50	658010 E
North	0	630	10	45	898040 N
RL	0	400	10	45	-1400 RL

Table 6.. Model Limits.

A listing of the MINESIGHT search parameters are provided in Table 7.

Leviathan Resources- Stawell Gold Mines													
GG3 RESOURCE MODEL													
AREA	DOMAIN	x axis	y axis	z axis	axis 1	axis 2	axis 3	min. samples 1st search	max. samples 1st search	2nd search expansion factor	min. samples 2nd search	max. samples 2nd search
		search distance			rotation			sample number definition				
GG9	601	40	20	15	315	-40	0	8	32			
GG9	602	40	20	15	315	-40	0	8	32			
GG9	603	40	20	15	315	-40	0	8	32			
GG9	604	40	30	20	315	-40	0	7	32			
GG9	605	40	30	25	315	-40	0	7	32	1	4	32
GG9	606	40	20	15	315	-40	0	8	32			
GG9	607	40	20	15	315	-40	0	8	32			
GG9	608	40	20	15	315	-40	0	6	32			
GG9	609	40	20	15	315	-40	0	8	32			
GG9	610	40	30	20	315	-40	0	8	32			
GG9	611	40	20	15	315	-40	0	8	32			
GG9	612	40	20	15	315	-40	0	8	32			
GG9	613	40	20	15	315	-40	0	8	32			
GG9	614	40	30	20	315	-40	0	8	32			

Table 7. Listing of MINESIGHT search parameters.

6.2 Block Estimation Parameters

All domains have been constrained by wireframes. They have been estimated using Ordinary Block Kriging. The domains used for estimation are outlined in Table 8.

Geology	Geology Final Solids	Domain
Kingston Ore Volc 1	Kingston Ore Volc 1 revised	601
Kingston Ore Volc 2	Kingston Ore Volc 2 revised	602
Kingston Ore Volc 3	Kingston Ore Volc 3 revised	603
Kingston Ore Volc 4	Kingston Ore Volc 4 revised	604
Kingston Ore Volc 5	Kingston Ore Volc 5 revised	605
Kingston Volc Weak 1	Kingston Volc Weak 1 revised	606
Kingston Volc Weak 2	Kingston Volc Weak 2 revised	607
Kingston Volc Weak 3	Kingston Volc Weak 3 revised	608
Kingston Volc Weak 4	Kingston Volc Weak 4 revised	609
Kingston Volc Weak 5	Kingston Volc Weak 5 revised	610
Delta Volc 1 revised	Delta Volc 1 revised	611
Delta Volc 2 revised	Delta Volc 2 revised	612
Delta Volc 3 revised	Delta Volc 3 revised	613
Delta Volc 4 revised	Delta Volc 4 revised	614

Table 8. Resource model description table.

Estimations in MINESIGHT are confined to domains (hard boundaries). The composited drill holes and block model were assigned the appropriate domain values.

The completed block model and all associated MINESITE files are stored on the Mining Geology server in the directory S:\Mining-Geology\Golden Gift 9\ Minesight\ Geology\ Resource Model May 2007. Additional details on the MINESIGHT resource model are provided in Table 9.

Leviathan resources - Stawell Gold Mines	GG9 RESOURCE MODEL
Date	**May-2007**
Model compiled by	**MV/MS/DH**
Directory Location	S:\Mining-Geology\Golden Gift 9\Minesight\Geology\Resource Model May 2007
Model Name	**gg0915.dat**
Previous Model Name	**gg0915.dat**
Parameter Spreadsheet	**GG9_Resource Model May 2007.xls**
Resource Report	**GG9_Resource Model Report May 2007.doc**
Purpose of Update	**This model was updated with recently completed grade control drilling.**

Table 9. Resource model description table.

7 Model Validation

Model validation has been completed using all tools available. It is highly recommended that the Kriging Efficiency be built into the validation process. To facilitate reporting, the spreadsheet GG9_Resource_ Modelling_ May 2007.xls was used.

The Slope of regression achieved in the Indicated/Inferred areas of the model shows acceptable block estimation, based on the available variogram model (refer to Figure 19 to Figure 33). Each Figure illustrates the slope of regression, Indicated/Inferred wireframe and drill holes.

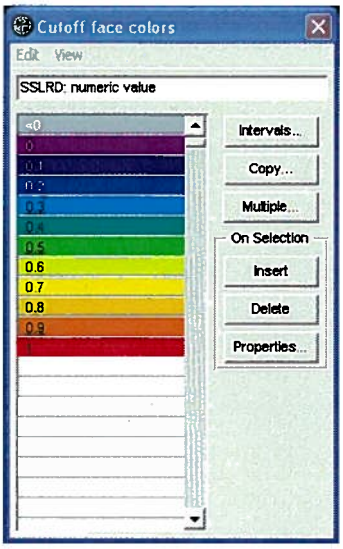

Figure 19. Slope of Regression legend

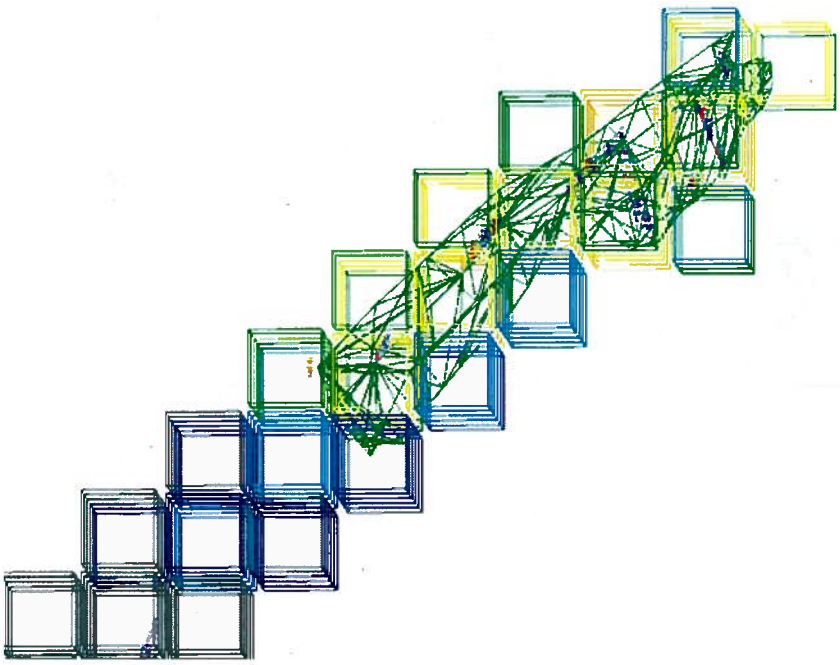

Figure 20. Slope of Regression for Domain 601.

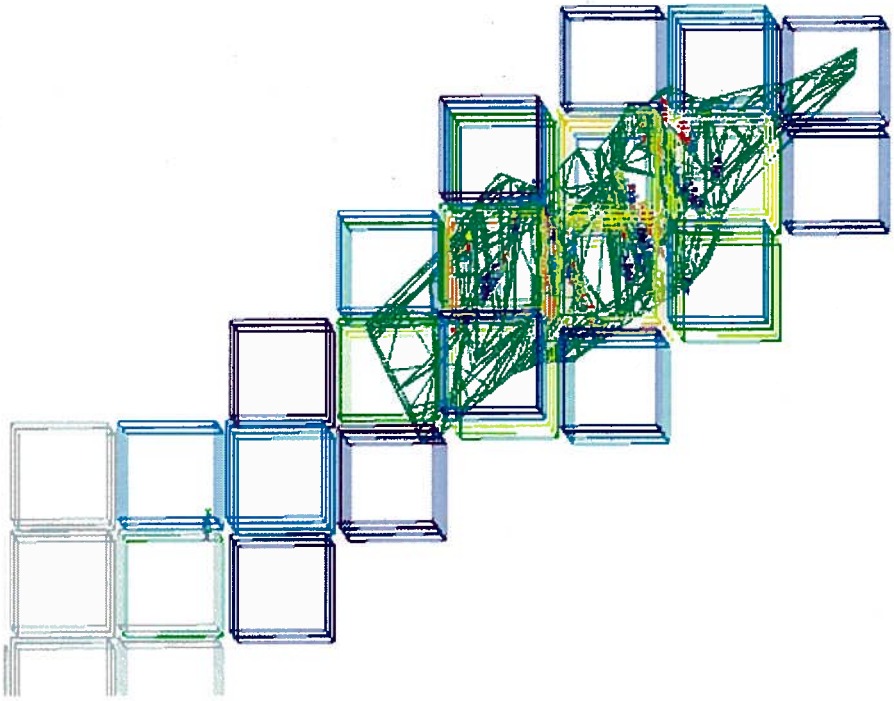

Figure 21. Slope of Regression for Domain 602.

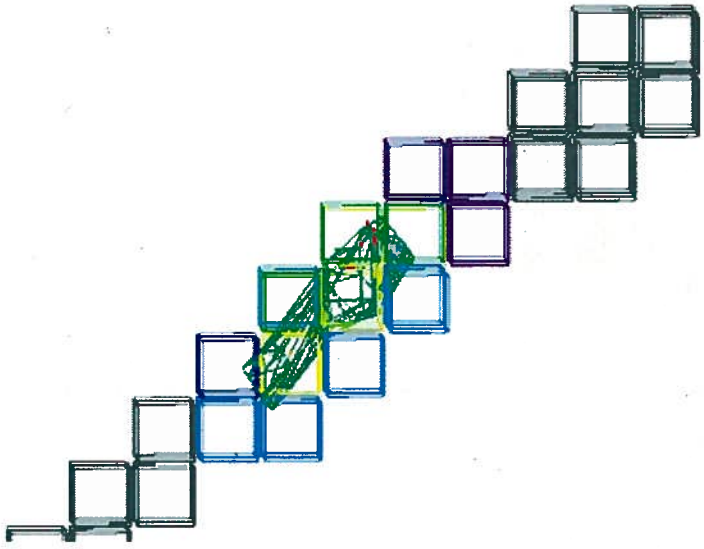

Figure 22. Slope of Regression for Domain 603.

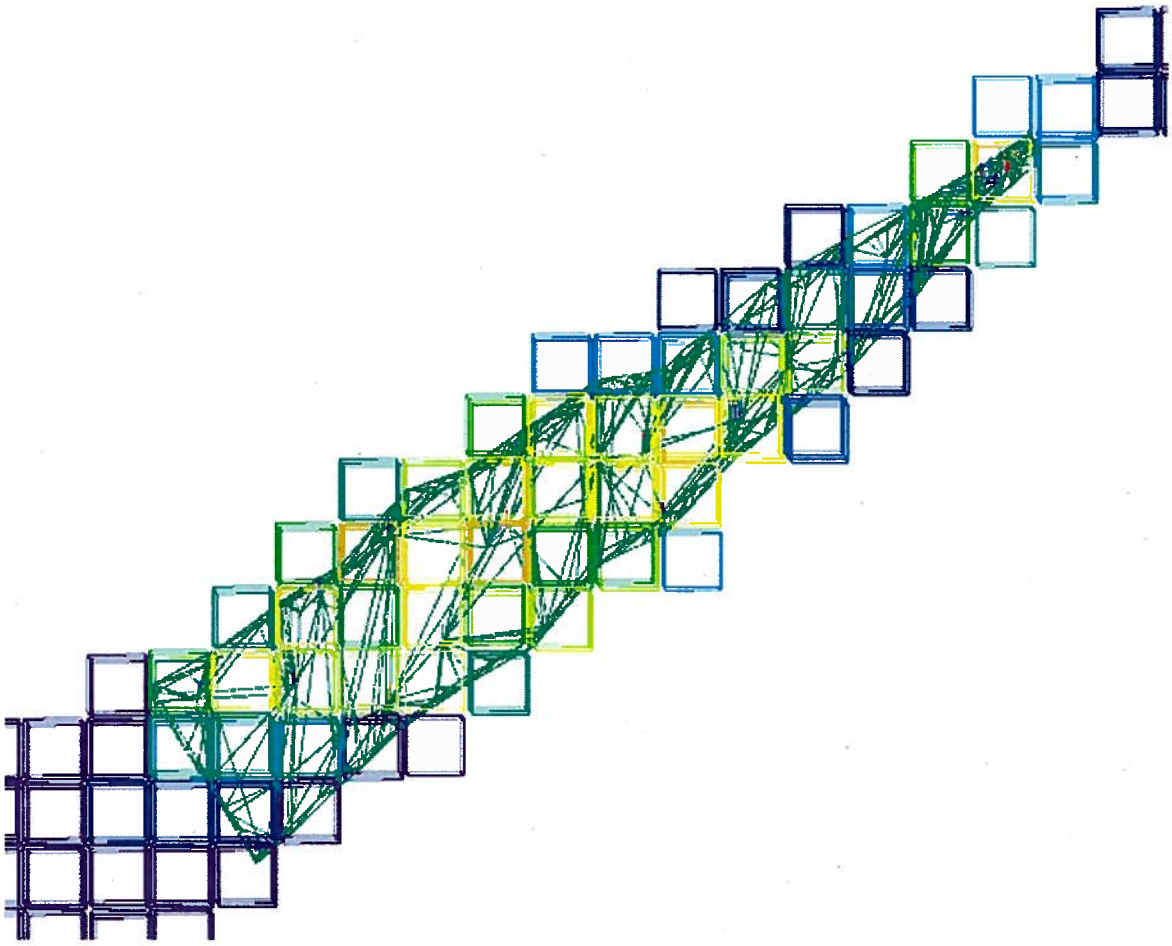

Figure 23. Slope of Regression for Domain 604.

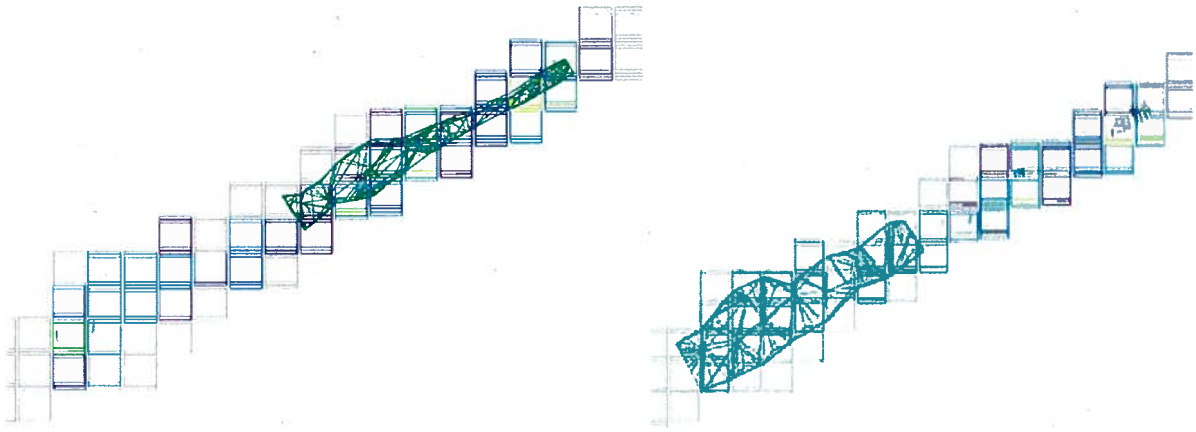

Figure 24. Slope of Regression for Domain 605 (left) Indicated and (right) Inferred.

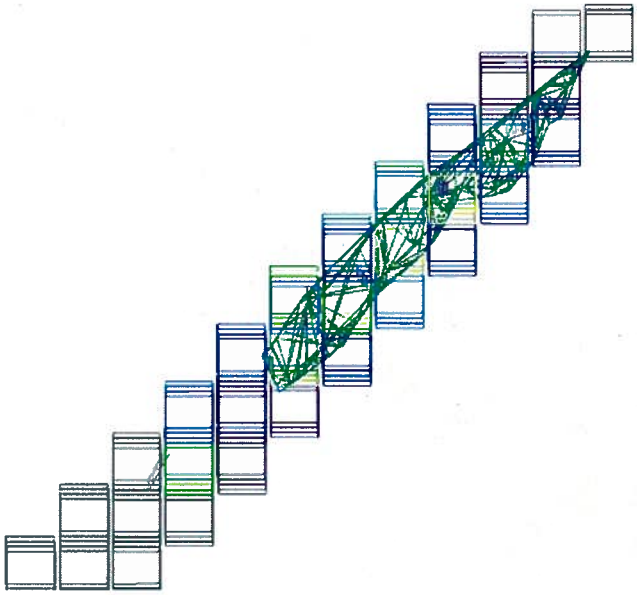

Figure 25. Slope of Regression for Domain 606.

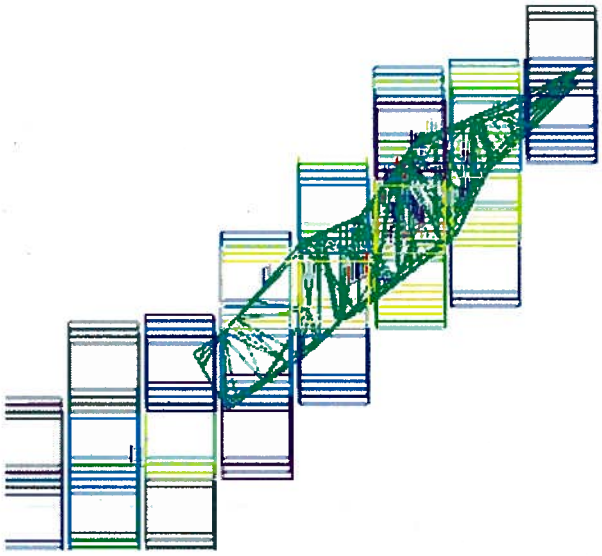

Figure 26. Slope of Regression for Domain 607.

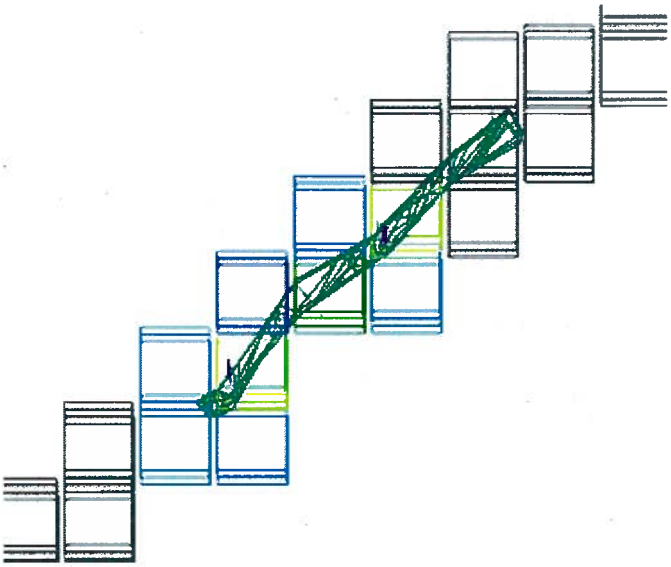

Figure 27. Slope of Regression for Domain 608.

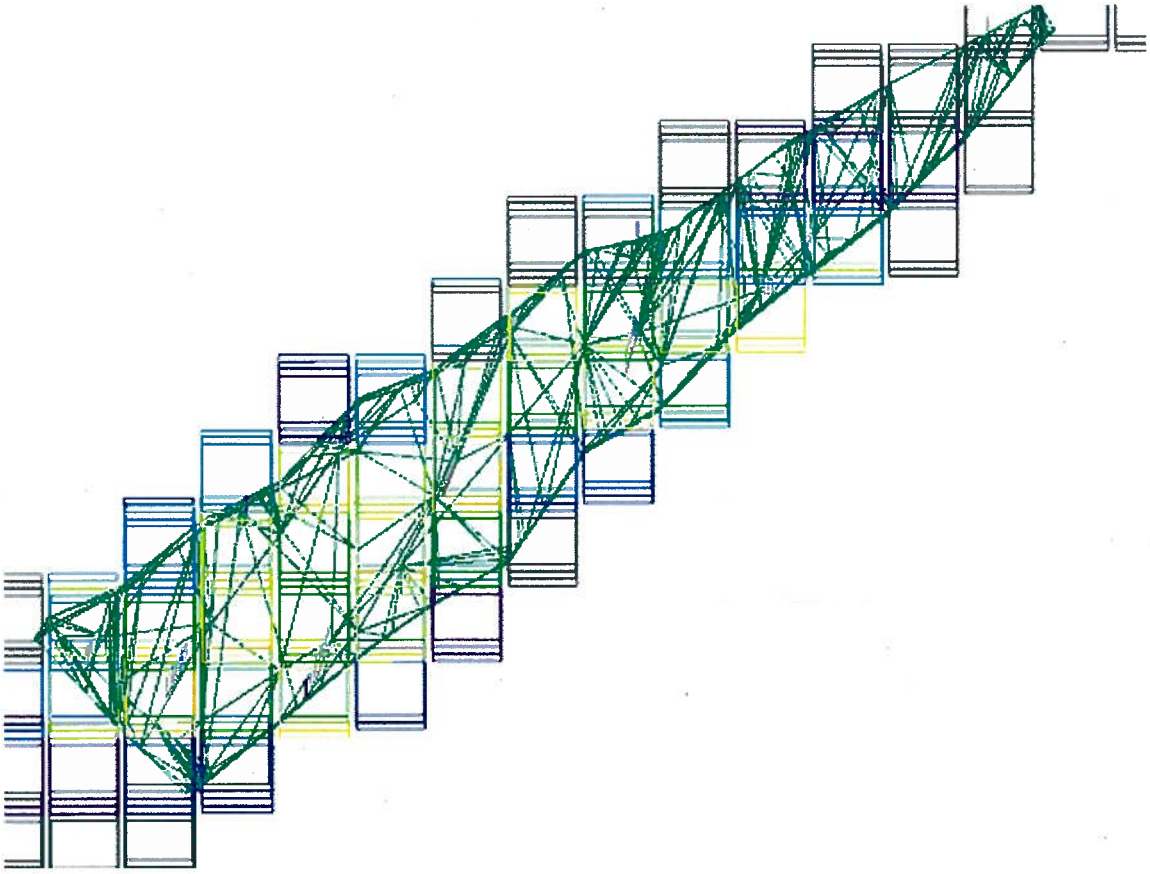

Figure 28. Slope of Regression for Domain 609.

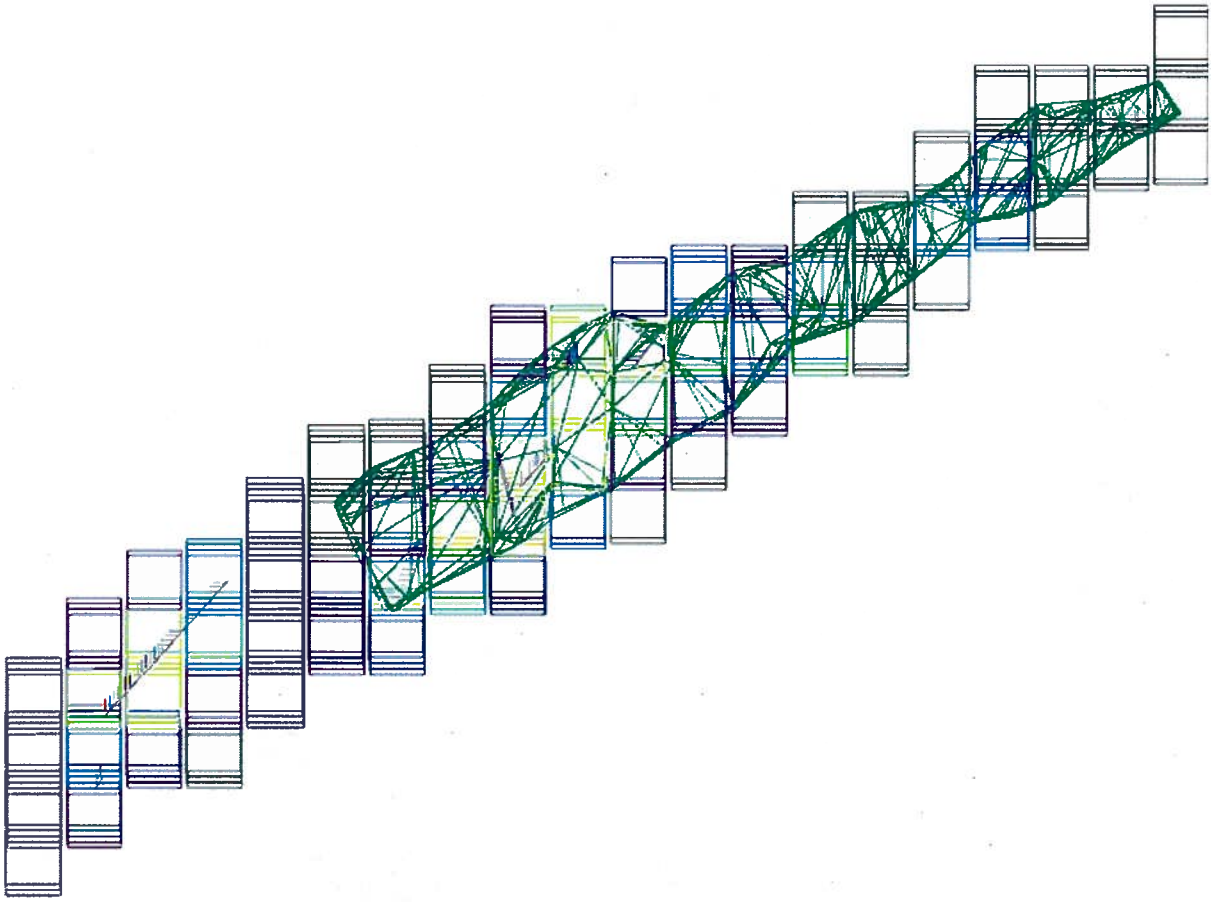

Figure 29. Slope of Regression for Domain 610.

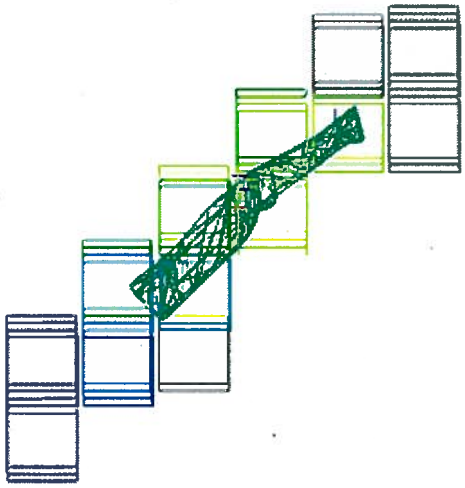

Figure 30. Slope of Regression for Domain 611.

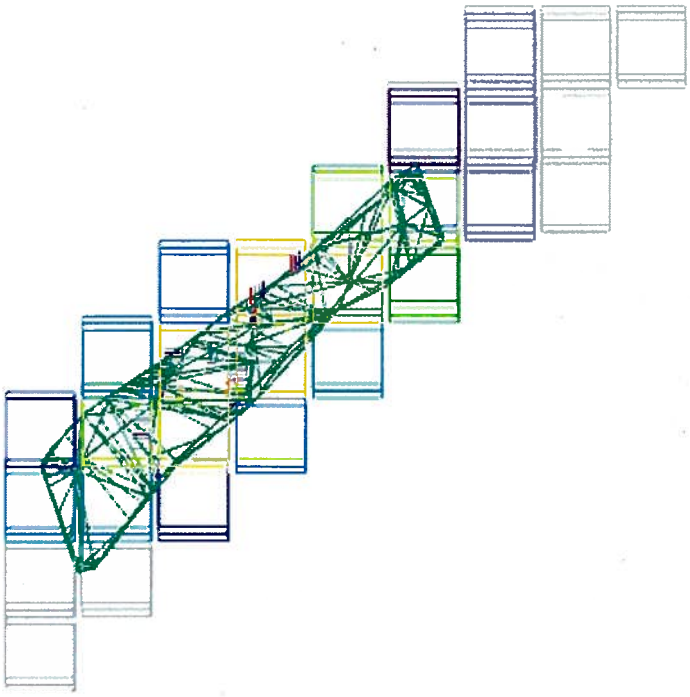

Figure 31. Slope of Regression for Domain 612.

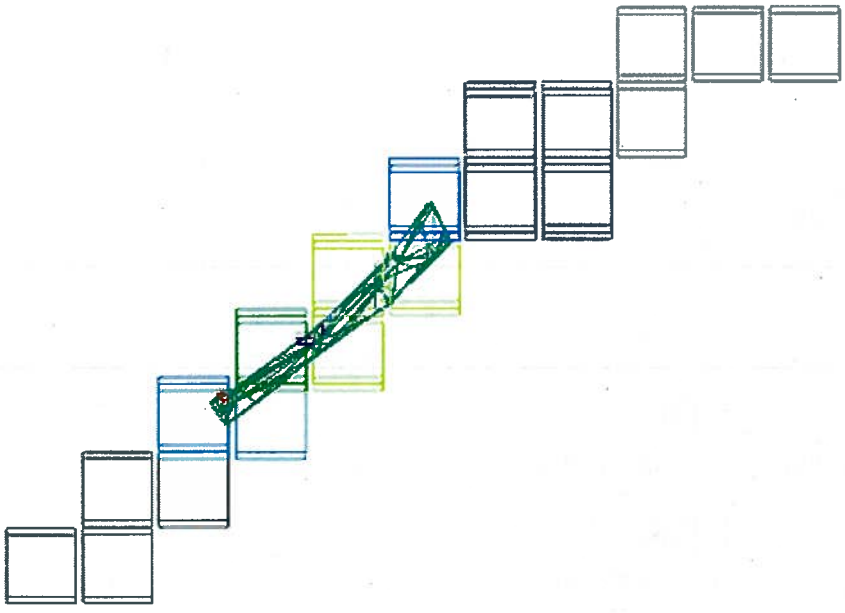

Figure 32. Slope of Regression for Domain 613.

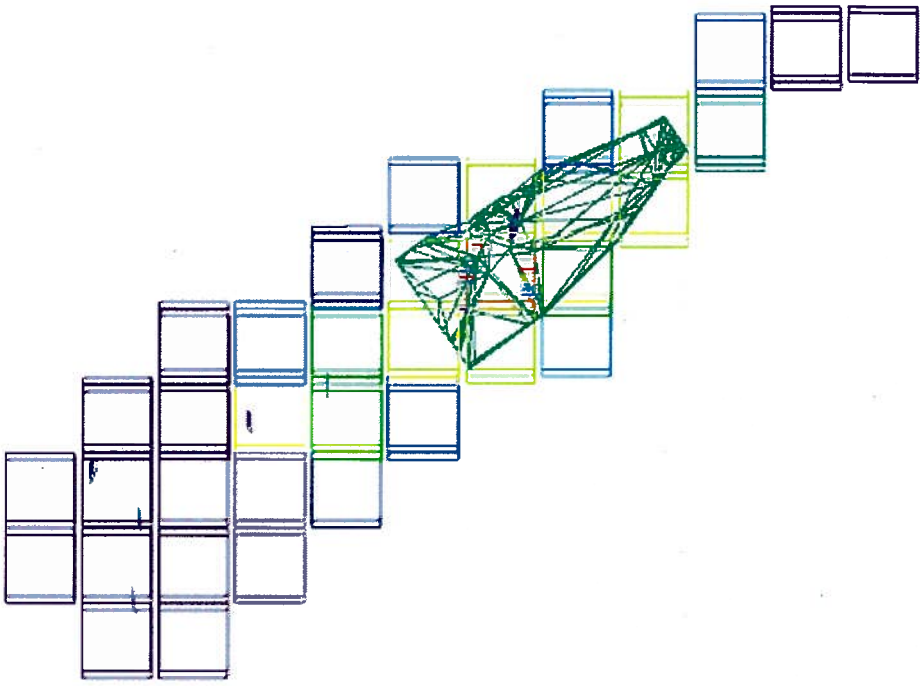

Figure 33. Slope of Regression for Domain 614.

A comparison of the declustered composited drill hole data and mean grade of the model blocks (10mN x 5mE x 10mRL) within a domain shows an acceptable variance (refer Table 10).

Leviathan Resources - Stawell Gold Mines	**MODEL VALIDATION TABLE**			
GG9 RESOURCE MODEL				
Date:	Indicated			
	Mean Grade (g/t)			
DOMAIN	**Declust. 2.0m comp.**	**2.0m comp.**	**Model**	**Variance**
Domain 601	3.80	3.96	4.129	8.7%
Domain 602	5.46	6.27	5.759	5.5%
Domain 603	7.430	8.236	7.329	-1.4%
Domain 604	5.85	5.11	5.754	-1.6%
Domain 605	5.75	2.93	4.910	-14.6%
Domain 606	0.880	0.774	0.895	1.8%
Domain 607	2.63	2.35	2.045	-22.3%
Domain 608	0.87	0.72	0.903	3.8%
Domain 609	0.320	0.279	0.309	-3.3%
Domain 610	0.980	1.160	0.994	1.4%
Domain 611	5.870	3.674	3.635	-38.1%
Domain 612	2.360	2.090	1.905	-19.3%
Domain 613	3.780	2.880	3.072	-18.7%
Domain 614	3.1	2.789	3.2282	4.1%

Table 10. Model validation table.

8 Model Risk Assessment

The classification of Indicated and Inferred is based on geological confidence and drill hole spacing, Slope of Regression and model validation results. The above Figure 20 to Figure 33 illustrate the slope of regression, Indicated/Inferred wireframes and drill holes.

9 Model output

The total Indicated Modelled Inventory at a cut off grade of 4.0g/t Au is 16,300oz. A listing of the global inventory at a 4g/t cut-off gold grade is provided in Table 11.

Leviathan Resources - Stawell Gold Mines	GG9
Resource Classification	Modelled Inventory
Mined Status	
Other Restrictions	AREA='GG9'
Filter Applied to Evaluation	MSTATUS=0
Previous Model Name	gg0915.dat
New Model Name	gg0915.dat

RESCAT	Cutoff (g/t)	PREVIOUS MODEL			NEW MODEL			Change			Change %		
		quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)	quantity (t)	grade (g/t)	metal (oz)
MEASURED INDICATED INFERRED	0.0	-	-	-	-	-	-	0	0.0	0			
	2.0	-	-	-	-	-	-	0	0.0	0			
	4.0	110,500	6.0	21,200	84,300	6.0	16,300	-26,200	0.1	-4,900	-24%	1%	-23%
	5.0	-	-	-	-	-	-	0	0.0	0			
	7.0	-	-	-	-	-	-	0	0.0	0			
	9.0	-	-	-	-	-	-	0	0.0	0			

Table 11. Modelled inventory at a series of cut-off grades for a new resource model compared to the previous model.

The Indicated model inventory of the new model is less than that reported in November 2005. At a cut-off of 4.0g/t the Indicated tonnes have decreased by 4,900oz, the grade has remained the same. This change is due to a reinterpretation of the geological model which now more accurately represents the structural complexity (refer to Figure 34).

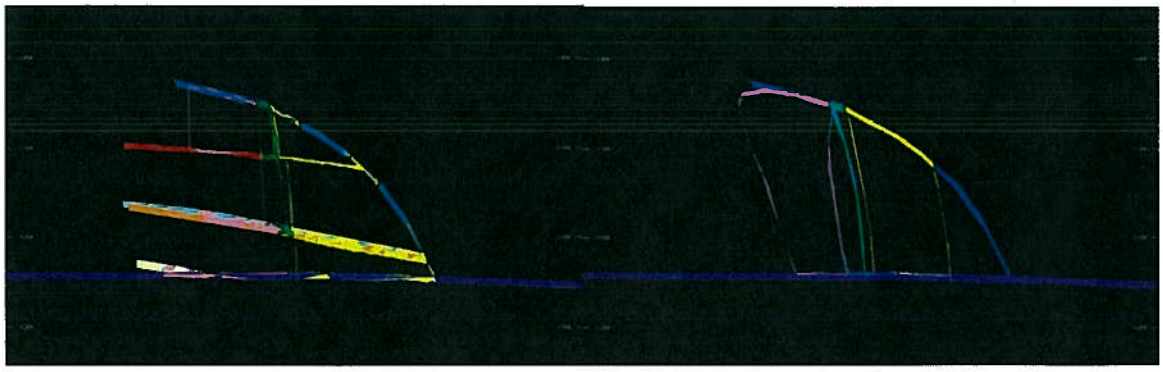

Figure 34. Minesight screen capture of section 6910N (left) May 2007 Model (right) November 2006 Model.

10 Recommendations

It is recommended that high grade zones be domained separately in the next model. The implementation of a thorough QAQC program is required. Further investigation is required for Kriging Neighbourhood analysis (block size, number of samples, search size and number of discretisation points). Global estimation of recoverable resources would be of value. The variography will require further investigation. Negative Kriging weights and Kriging Efficeny is recommended to be included in the standard Protocol.

11 Model completion

11.1 Server storage location of model output files

The model and all accompanying Minesight, ISATIS, Microsoft Excel™ and Microsoft Word™ computer files have been stored on the Stawell Gold Mines computer server in the sub-directory S:\Mining-Geology\Golden Gift 9\Minesight\Geology\Resource Model May 2007. The project directory contains a spreadsheet named "GG9_read_me_first.xls" that provides a listing all basic file name information for the project (refer to Table 12).

Project Name	GG9			
Date of Study	Dec-2006			
Resource Geologist Name	MV/MS/DH			
Directory Location	S:\Mining-Geology\Golden Gift 9\Minesight\Geology\Resource Model May 2007_ms\resources			
	Minesight file	**details**		
Drillhole Data				
Raw Drill Hole Data	gg0911.ddh			
Composited and Top Cut Drill Hole File	gg0909.2m	2.0m comp		
Geology	Geology Final Solids	Domain	Geocode	Grade Item
Kingston Ore Volc 1	Kingston Ore Volc 1 revised	601	AGEO	AAUD
Kingston Ore Volc 2	Kingston Ore Volc 2 revised	602	BGEO	BAUD
Kingston Ore Volc 3	Kingston Ore Volc 3 revised	603	CGEO	CAUD
Kingston Ore Volc 4	Kingston Ore Volc 4 revised	604	DGEO	DAUD
Kingston Ore Volc 5	Kingston Ore Volc 5 revised	605	BGEO	BAUD
Kingston Volc Weak 1	Kingston Volc Weak 1 revised	606	EGEO	EAUD
Kingston Volc Weak 2	Kingston Volc Weak 2 revised	607	FGEO	FAUD
Kingston Volc Weak 3	Kingston Volc Weak 3 revised	608	AGEO	AAUD
Kingston Volc Weak 4	Kingston Volc Weak 4 revised	609	EGEO	EAUD
Kingston Volc Weak 5	Kingston Volc Weak 5 revised	610	FGEO	FAUD
Delta Volc 1 revised	Delta Volc 1 revised	611	AGEO	AAUD
Delta Volc 2 revised	Delta Volc 2 revised	612	BGEO	BAUD
Delta Volc 3 revised	Delta Volc 3 revised	613	CGEO	CAUD
Delta Volc 4 revised	Delta Volc 4 revised	614	DGEO	DAUD
Basalt 1	Basalt 1 revised	400		
Basalt 2	Basalt 2 revised	400		
Basalt 3	Basalt 3 revised	400		
Basalt 4	Basalt 4 revised	400		
Basalt 5	Basalt 5 revised	400		
Mine Schist 1	Mine Schist 1 revised	1		
Mine Schist 2	Mine Schist 2 revised	1		
Mine Schist 3	Mine Schist 3 revised	1		
Mine Schist 4	Mine Schist 4 revised	1		
Mine Schist 5	Mine Schist5 revised	1		
Durif	Durif			
Merlot	Merlot			
Matt's Fault	Fault Matt			
Fault 1	Fault 1			
Fault 2	Fault 2			
Fault 3	Fault 3			
Models				
Final Resource Model Name	gg0915.dat			
Project Control	gg0910.dat			

Table 12. Readme_first.xls table of file names and major model parameters.

TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX K

Wonga Gold Deposit

Pit Limit Optimisation Study

Initial Results & Notes

June 2007

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Perseverance Corporation Limited

Wonga Gold Deposit

Pit Limit Optimisation Study

Initial Results & Notes

Prepared by Tamer Dincer

26 June 2007

Wonga Pit Optimisation Results – 16 June 2007

The tables and figures referred below have been attached to the end of this document (4 pages of text).

1. Resource Model

The following data was received from PCL for the construction of the pit optimisation model:

- Resource block model file including the items:
  - Gold grade (AUPPM)
  - Proportion of resource in each block (ORE%),
  - Bulk density (SG),
  - Oxide classification (OXID=oxide/transitional/Fresh),
  - Resource category (RESCAT= Indicated/ Inferred),
  - Proportion of blocks below existing topography (TOPO), and
  - Mined proportion of blocks in old U/G workings (VOID%)
- Surface data for the old air leg stopes
- Surface data for the existing open pit as mined before the waste backfill
- Surface data for the existing topography, including waste dumps and backfill
- Waste dump and backfill density of 1.9t/bcm.

Based on the above data, the pit optimisation model was created as follows:

- The air leg stope surfaces were used to calculate the proportion of blocks within 3m vertical thickness above the surfaces. This thickness might be considered conservative but the air leg mining would target the higher grade proportion of the blocks selectively (which would normally affect the grade estimates in the model).
- Although the string data was also received for some additional air leg stopes (not a significant volume), these could not be used due to the string format provided.
- The calculated air leg stope proportions of the blocks were added to the VOID% received in the resource model.
- An ore percentage item (ORE2%) was calculated by deducting the VOID% from the ORE% item. Although this approach was conservative in calculating the resource tonnes, it was considered again that the underground workings would mine the higher grade proportion of the resources selectively.
- The TOPO% item (the proportion of the blocks below topographic surface) was adjusted for the mined proportion of the bocks (VOID%). This allowed the estimation of the remaining block volumes after underground mining.
- The ORE2% item was also adjusted for the change in TOPO item so that the ore (and waste) volumes were calculated in the optimisation model correctly.
- The block densities were adjusted for the backfill waste proportion of the blocks in the mined pit and waste dump areas.

- The resource blocks reported within the backfill material in the mined pit was ignored by adjusting the ORE2% item.
- Based on the memorandum on density test work (D. Coventry, 14 June 2007), the fresh ore density (2.84t/bcm) in the received resource model was revised to 2.70t/bcm.
- The resource types for Whittle pit optimisation program (WHIT item) was calculated as a combination of the oxidisation and confidence categories.
- The mining costs were set in the mining model by levels, which was followed by the adjustment of the mining costs for the backfilled or dumped volumes (lower cost).

The global resources reported from the pit optimisation model above 0.5g/t cut-off were as follows:

- Indicated Resources: 1.18Mt @ 2.44g/t
- Inferred Resources: 2.12Mt @ 2.13g/t
- Total Resources: 3.30Mt @ 2.24g/t

2. Pit Optimisation Input Parameters

The input parameters used in the optimisations have been summarised in Table 1. The basis for the input parameters is as follows:

- A base case gold price of $840/oz was used in the optimisations. Sensitivity optimisations were carried out for ±40 and ±80/oz variations in the base case gold price.
- As requested by PCL, the largest optimal pit shell was reported at $1,200/oz gold price to define the envelope for global resource reporting.
- The preliminary mine supervision and grade control costs were estimated by MSC based on experience in similar operations. Note that these costs largely depend on the ore and waste quantities to be mined.
- The crusher loading cost was provided by PCL. The processing cost for transitional resource ($12.50/t) was interpolated based on the oxide ($10/t) and primary ($15/t) ore processing costs received from PCL.
- Processing recoveries were provided by PCL. The transitional resource recovery was interpolated based on the recoveries for oxide and primary ore types.
- It was assumed that general site and site administration costs (overheads) were included in the processing costs (Please confirm this assumption or the basis for the exclusion of general and administration costs).
- Break-even cut-off grade for each ore type was calculated using the base $840/oz gold price, throughput costs and process recoveries. Note that the throughput cost in the COG calculation includes the ore - waste mining cost difference (~$0.25/t).
- The cut-off grades for the resource model include the effect of mining dilution and ore loss.
- A mining dilution of 10% and ore loss of 5% were assumed in the pit optimisations and reporting of the optimal pit resources.

- According to JORC code, the estimation of the ore reserves should be based on Indicated resources only. The Inferred resources were included separately to see the potential of the project for a resource upside case.

3. Mining Costs by Levels

Table 2 lists the ore and waste mining cost estimates by levels used in the optimisations. The estimation was based on the experience in similar operations, which was also supported by the reference costs provided by PCL from a nearby operation.

The operation of an equipment fleet based on Cat 777 trucks was assumed in the cost estimates. The load & haul costs were increased by 20% for mining below 80m depth, assuming that the 10m wide existing ramp on the western wall can be utilised after changing to smaller trucks.

The equipment mobilisation, contractor establishment and any other capital costs were not included in the pit optimisations. These costs should be considered separately in addition to the operating values reported from the pit optimisations.

4. Pit Slopes

The overall slope angles used in the pit optimisations were provided by PCL. The preliminary wall slope designs in Table 3 were proposed after the review of the existing pit slopes.

It was assumed in the calculation of the overall slope angles that the temporary access ramp will be on the eastern wall initially. The temporary ramp will be mined later as the access to the lower levels is established through the existing ramp on the western wall.

The pit slope design for the west wall will be variable naturally depending on the footwall dip of the orebody (average 37degrees). Similar to the hangingwall, a maximum slope angle of 42.8 degrees was assumed for the west wall as a limiting angle for pit optimisation purposes.

5. Pit Optimisation Results

Pit optimisation results for the studied cases have been summarised in Table 4. The result details for the $840/oz gold price can be seen in Table 5 for the Indicated resources only case and in Table 6 for the Indicated + Inferred resources case.

The following figures have been provided for the plan views of the optimal pit shells:

- Figure 1 Existing Topographic Surface
- Figure 2 Optimal Pit Shell for Indicated Resources ($840/oz)
- Figure 3 Optimal Pit Shell for Indicated & Inferred Resources ($840/oz)
- Figure 4 Larger Pit Shell ($1,200/oz) for Indicated & Inferred Resources (for resource reporting)
- Figure 5 Planar Section View of the Optimal Pit Shells at -50mRL

Based on the tables and figures described above, the optimisation results can be summarised as follows:

- For a gold price of $840/oz, the potential expansion of the existing Wonga pit is limited to the relatively shallow resource along the strike of the mineralisation.
- Based on the Indicated resources only, the area to the south of the existing pit along strike has the most potential to support a reserve estimate (Run01c Shell 15). However, (1) this area is located in the existing rubbish dump? and (2) the ramp design to access the relatively small cutback will require significant amount of additional waste mining proportionally.
- In a pit design for reserve estimation purposes, the backfilling of the southern section of the existing pit might be a feasible option (1) to establish a backfill ramp cost effectively and (2) to minimise the waste dumping costs.
- Based on the Indicated and Inferred resources, pit optimisation results indicate the area to the north of the existing pit along strike as the secondary potential cutback area (Run02c Shell 15). Optimal pit shell limits in this area indicate a mining width of 15-40m, which is quite narrow for the considered mining equipment and establishment of the access ramp cost effectively.
- Based on the Indicated and Inferred resource optimisation results, the break-even gold price for a 50-60m wide significant cutback development at the south and southern half of the eastern wall is ~$1,000/oz (Run02c Shell 19).
- Based on the Indicated and Inferred resource optimisation results, the break-even gold price for a 50-60m wide full length cutback development along the south, east and north walls of the existing pit is ~$1,250/oz (Run02c Shell 24).
- Compared to the global model resources, the grades reported in the optimal pit shells are relatively lower due to the shallower nature of the optimal pit developments. The significant proportion of the in-pit resources reported are lower grade (1.3-1.5g/t) oxide ore.
- The price sensitivity results show that the optimal pit limits for the Indicated resources only case is not sensitive to a change in the gold price within ~$\pm$10% range. This indicates that the variation in the other parameters (operating costs & slopes) would also not affect he optimal pit limits significantly

6. Further Work

At this stage of the project, the pit optimisation results do not indicate a large pit design and reserve estimate potential for the Wonga Deposit.

The slope assumptions and mining costs may need to be revised for a small cutback design at the south of the existing pit depending on the access ramp options. This would depend on the issues and the costs associated with the relocation of the rubbish dump at the south.

Further work for the finalisation of a reserve estimate will depend on the feed back from PCL.

Table 1

Wonga Pit Optimisation

Input Parameters Summary

Throughput Costs		**Oxide**	**Trans.**	**Primary**	
Mine Supervision	$/t ore	1.00	1.00	1.00	Preliminary MSC estimate
Grade Control	$/t ore	1.50	1.80	2.10	Preliminary MSC estimate
Crusher Loading	$/t ore	0.62	0.62	0.62	C. Hamilton's email
Processing	$/t ore	10.00	12.50	15.00	C. Hamilton's email
General & Administration	$/t ore	Included in processing?			
Rehabilitation	$/t ore	0.25	0.25	0.25	For the pit / cutback to be mined
Total Throughput Cost	$/t ore	13.37	16.17	18.97	

Process Recovery	%	88%	85%	82%

Break-even Cut-off Grade ($840/oz)	Au g/t	0.57	0.65	0.74
Pre-dilution Resource Model Cut-off	Au g/t	0.66	0.76	0.86

Gold Price and Royalty		**a**	**b**	**c**	**d**	**e**	**f**
Gold Price	$/oz	760	800	840	880	920	1200
	$/gm	24.43	25.72	27.01	28.29	29.58	38.58
Government Royalty	%						
Other Royalty							
Gold Price after Royalty	$/oz	760.0	800.0	840.0	880.0	920.0	1200.0
	$/gm	24.43	25.72	27.01	28.29	29.58	38.58

Resource Model	
Mining dilution & ore loss	10% dilution, 5% ore loss (in addition to the internal dilution? in the model)
Classification used:	Indicated only, Inferred included as an upside case for sensitivity analysis

Table 2
Wonga Pit Optimisation
Mining Costs by Levels

Levels		Ore Mining Costs							Waste Mining Costs						
From mRL	To mRL	L&H $/bcm	D&B $/bcm	Dewater $/bcm	Other $/bcm	Total $/bcm	Density t/bcm	Total $/tonne	L&H $/bcm	D&B $/bcm	Dewater $/bcm	Other $/bcm	Total $/bcm	Density t/bcm	Total $/tonne
Surf	-10	3.65	2.40		0.26	6.31	2.25	2.80	3.90	2.40		0.27	6.57	2.25	2.92
-10	-20	3.90	2.40		0.27	6.57	2.25	2.92	4.15	2.40		0.29	6.84	2.25	3.04
-20	-30	4.15	2.40		0.29	6.84	2.25	3.04	4.40	2.40		0.31	7.11	2.25	3.16
-30	-40	4.80	2.80		0.34	7.94	2.63	3.02	5.05	2.80		0.35	8.20	2.63	3.12
-40	-50	5.12	2.80		0.36	8.27	2.63	3.15	5.37	2.80		0.38	8.54	2.63	3.25
-50	-60	5.86	3.17		0.41	9.45	2.84	3.33	6.11	3.17		0.43	9.71	2.70	3.60
-60	-70	6.21	3.17		0.43	9.81	2.84	3.45	6.46	3.17		0.45	10.08	2.70	3.73
-70	-80	6.55	3.17		0.46	10.17	2.84	3.58	6.80	3.17		0.48	10.44	2.70	3.87
-80	-90	8.13	3.17		0.48	11.78	2.84	4.15	8.42	3.17		0.50	12.08	2.70	4.47
-90	-100	8.39	3.17		0.50	12.06	2.84	4.25	8.68	3.17		0.52	12.36	2.70	4.58
-100	-110	8.66	3.17		0.52	12.34	2.84	4.35	8.94	3.17		0.54	12.65	2.70	4.68
-110	-120	8.92	3.17		0.54	12.63	2.84	4.45	9.20	3.17		0.56	12.93	2.70	4.79
-120	-130	9.18	3.17		0.56	12.91	2.84	4.55	9.47	3.17		0.58	13.21	2.70	4.89
-130	-140	9.44	3.17		0.58	13.19	2.84	4.65	9.73	3.17		0.60	13.49	2.70	5.00
-140	-150	9.71	3.17		0.60	13.47	2.84	4.74	9.99	3.17		0.62	13.78	2.70	5.10
-150	-160	9.97	3.17		0.62	13.76	2.84	4.84	10.25	3.17		0.64	14.06	2.70	5.21
-160	-170	10.23	3.17		0.64	14.04	2.84	4.94	10.52	3.17		0.66	14.34	2.70	5.31
-170	-180	10.49	3.17		0.66	14.32	2.84	5.04	10.78	3.17		0.68	14.62	2.70	5.42
-180	-190	10.76	3.17		0.68	14.60	2.84	5.14	11.04	3.17		0.70	14.91	2.70	5.52
-190	-200	11.02	3.17		0.70	14.89	2.84	5.24	11.30	3.17		0.72	15.19	2.70	5.63
-200	-210	11.28	3.17		0.72	15.17	2.84	5.34	11.57	3.17		0.74	15.47	2.70	5.73
-210	-220	11.54	3.17		0.74	15.45	2.84	5.44	11.83	3.17		0.76	15.75	2.70	5.84

Notes:

Other category includes dayworks, area preparation and other costs.

Densities are based on the average depth of weathering surfaces.

Ore based mining costs (supervision and grade control) are listed separately as part of the throughput costs.

Equipment mobilisation and contractor's establishment costs need to be considered separately as a capital cost.

No dewatering cost was allowed due to the drainage in to the underground.

Waste backfill and dump cost assignec 1.50 $/t for dump, and 2.30 $/t pit backfill, above and below level -30mRL respectively.

Below level -80mRL, 13% increase in total cost for smaller equipment (approx 20% of average load & haul cost).

Table 3
Wonga Pit Optimisation
Overall Slope Calculation

Depth From m	Depth To m	Model Density t/bcm	Batter Height m	Batter Slope deg	Berm With m	Cum. OSA deg
West Wall (200-345deg Az)						
DEPENDING ON THE DIP OF FOOTWALL, VARIABLE DESIGN						
-40	-50	2.25	10	45	6	32.0
-50	-70	2.25	20	45	6	35.5
-70	-90	2.63	20	45	6	**36.3**
-90	-110	2.84	20	45	6	37.6
-110	-130	2.84	20	45	6	37.6
-130	-150	2.84	20	45	16	34.3
-150	-170	2.84	20	45	6	35.1
-170	-190	2.84	20	45		**36.7**
				Overall Slope (deg)		**36.6**
East Wall (15-135deg Az)						
-40	-50	2.25	10	50	6	34.8
-50	-70	2.25	20	50	6	38.9
-70	-90	2.63	20	50	6	**39.8**
-90	-110	2.63	20	50	6	41.3
-110	-130	2.84	20	55	6	43.1
-130	-150	2.84	20	55	6	43.7
-150	-170	2.84	20	55	6	44.0
-170	-190	2.84	20	55		**45.9**
				Overall Slope (deg)		**43.7**

Depth From m	Depth To m	Model Density t/bcm	Batter Height m	Batter Slope deg	Berm With m	Cum. OSA deg
North Wall 345-15deg Az)						
-40	-50	2.25	10	45	6	32.0
-50	-70	2.25	20	50	6	37.7
-70	-90	2.63	20	50	6	**39.1**
-90	-110	2.70	20	50	6	41.3
-110	-130	2.84	20	50	6	41.3
-130	-150	2.84	20	50	6	41.3
-150	-170	2.84	20	50	6	41.3
-170	-190	2.84	20	50		**42.8**
				Overall Slope (deg)		**41.5**
South and Southeast Wall (135-200deg Az)						
-40	-50	2.25	10	50	6	34.8
-50	-70	2.25	20	50	6	38.9
-70	-90	2.63	20	50	6	**39.8**
-90	-110	2.70	20	55	6	45.0
-110	-130	2.84	20	60	6	46.8
-130	-150	2.84	20	60	16	42.7
-150	-170	2.84	20	60	6	44.1
-170	-190	2.84	20	60		**46.7**
				Overall Slope (deg)		**44.2**

Notes:

Slopes are based on the existing pit slopes and the general condition of the walls as seen in the two photographs for east and west walls.

Assumed ramp width is 10m on the western and southern walls as per the existing pit. A temporary ramp will be used initially to access the eastern cutback.

The slope of the western wall is based on the orebody footwall dipping ~37 degrees east.

Prepared by Mining Solutions, 26 June 2007

Table 4

Wonga Pit Optimisation Results Summary

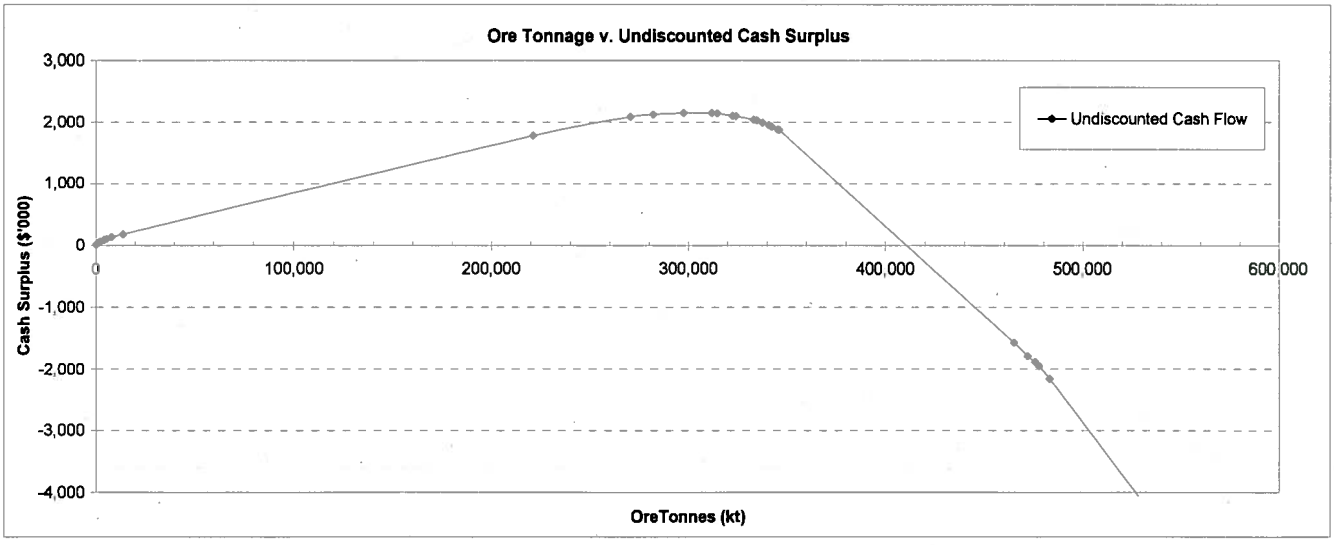
			Physicals													Economics					
			Oxide		Transitional		Primary		Ore		Waste	Total	W/O	Contn'd	Recv'd	Recv.	Mining	Th'put	Reven.	Undisc'd	Unit	Unit
Run ID	Case Description	Pit No	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Total Tonnes	Au g/t	Tonnes	Tonnes	Ratio t/t	Ounces koz	Ounces koz	%	Cost $'000	Cost $'000	$'000	C'Flow $'000	Cost $/oz	Cost $/t Ore
run01a	$760/oz Price, Indicated Resources	12	183,653	1.24	74,749	1.92	22,692	2.07	281,094	1.49	1,041,742	1,322,836	3.71	13,434	11,592	86.3%	3,489	4,132	8,809	1,188	657	27.1
Run01b	$800/oz Price, Indicated Resources	12	184,123	1.23	74,749	1.92	22,692	2.07	281,564	1.48	1,041,272	1,322,836	3.70	13,442	11,600	86.3%	3,489	4,138	9,280	1,653	658	27.1
Run01c	$840/oz Price, Indicated Resources	13	188,449	1.23	78,621	1.91	30,438	2.02	297,508	1.49	1,128,853	1,426,361	3.79	14,270	12,294	86.2%	3,775	4,409	10,329	2,145	666	27.5
Run01d	$880/oz Price, Indicated Resources	14	196,181	1.22	80,256	1.91	35,428	2.09	311,865	1.49	1,214,737	1,526,602	3.90	14,984	12,896	86.1%	4,055	4,636	11,348	2,657	674	27.9
Run01e	$920/oz Price, Indicated Resources	15	197,227	1.22	80,974	1.90	36,485	2.08	314,686	1.49	1,231,768	1,546,454	3.91	15,122	13,013	86.1%	4,111	4,682	11,972	3,179	676	27.9
Run02c	$840/oz Price, Indicated & Inferred R	14	214,392	1.47	143,532	1.96	45,402	2.34	403,326	1.74	1,584,778	1,988,104	3.93	22,603	19,413	85.9%	5,468	6,104	16,310	4,738	596	28.7
Run02f	$1,200/oz Price, Indicated & Inferred	22	278,937	1.30	237,324	1.92	333,087	2.38	849,348	1.90	6,431,021	7,280,369	7.57	51,771	43,593	84.2%	21,594	14,053	52,310	16,663	818	42.0

Table 5

Wonga Pit Optimisation Results - Run01c

$840/oz Price, Indicated Resources only

Pit No	Physicals: Oxide Tonnes	Oxide Au g/t	Transitional Tonnes	Transitional Au g/t	Primary Tonnes	Primary Au g/t	Total Resource Tonnes	Total Resource Au g/t	Waste Tonnes	Total Tonnes	W/O Ratio t/t	Contn'd Ounces koz	Recv'd Ounces koz	Recv. %	Economics: Mining Cost $'000	Th'put Cost $'000	Revenue $'000	Operating Surplus $'000	Unit Cost $/oz	Marg. Cost $/oz
1	144	2.51					144	2.51	197	341	1.37	12	10	88.0%	1	2	9	6	293	293
2	288	2.51	124	3.14			412	2.70	854	1,266	2.07	36	31	87.0%	3	6	26	17	290	288
3	686	1.98	247	3.14			933	2.28	1,210	2,143	1.30	68	60	86.9%	6	13	50	31	319	352
4	945	1.88	330	3.05			1,275	2.18	1,581	2,856	1.24	89	78	86.9%	8	18	65	39	334	384
5	2,002	1.70	330	3.05			2,332	1.89	2,644	4,976	1.13	142	124	87.3%	14	32	104	58	372	435
6	3,085	1.61	971	2.03			4,056	1.71	4,012	8,068	0.99	223	194	87.1%	22	57	162	83	407	468
7	4,073	1.56	1,187	1.94	107	2.60	5,367	1.67	6,341	11,708	1.18	288	250	87.0%	33	76	210	101	435	534
8	5,986	1.54	1,861	1.69	107	2.60	7,954	1.59	12,369	20,323	1.56	407	354	87.1%	56	113	297	128	477	578
9	9,018	1.47	4,517	1.62	107	2.60	13,642	1.53	26,228	39,870	1.92	670	582	86.9%	114	197	489	178	534	623
10	162,832	1.25	56,097	1.99	2,214	1.98	221,143	1.44	766,825	987,968	3.47	10,262	8,914	86.9%	2,560	3,152	7,488	1,776	641	648
11	178,803	1.24	72,884	1.93	18,904	2.16	270,591	1.49	1,004,400	1,274,991	3.71	12,961	11,191	86.3%	3,358	3,963	9,402	2,081	654	707
12	184,711	1.23	74,749	1.92	22,692	2.07	282,152	1.48	1,040,684	1,322,836	3.69	13,453	11,610	86.3%	3,489	4,146	9,754	2,119	658	750
13	188,449	1.23	78,621	1.91	30,438	2.02	297,508	1.49	1,128,853	1,426,361	3.79	14,270	12,294	86.2%	3,775	4,409	10,329	2,145	666	802
14	196,181	1.22	80,256	1.91	35,428	2.09	311,865	1.49	1,214,737	1,526,602	3.90	14,984	12,896	86.1%	4,055	4,636	10,834	2,143	674	843
15	197,227	1.22	80,974	1.90	36,485	2.08	314,686	1.49	1,231,768	1,546,454	3.91	15,122	13,013	86.1%	4,111	4,682	10,932	2,139	676	872
16	198,156	1.22	83,983	1.89	40,254	2.14	322,393	1.51	1,314,112	1,636,505	4.08	15,636	13,440	86.0%	4,376	4,816	11,291	2,099	684	933
17	199,141	1.22	84,423	1.89	40,681	2.13	324,245	1.51	1,325,988	1,650,233	4.09	15,715	13,508	86.0%	4,413	4,845	11,349	2,091	685	968
18	201,431	1.22	85,767	1.89	45,926	2.05	333,124	1.50	1,383,615	1,716,739	4.15	16,114	13,842	85.9%	4,595	4,999	11,628	2,034	693	1007
19	202,495	1.21	85,767	1.89	46,457	2.05	334,719	1.50	1,393,665	1,728,384	4.16	16,175	13,894	85.9%	4,625	5,024	11,673	2,024	694	1053
20	202,830	1.22	86,223	1.89	48,651	2.06	337,704	1.51	1,434,685	1,772,389	4.25	16,375	14,060	85.9%	4,751	5,078	11,813	1,984	699	1085
21	203,965	1.21	86,337	1.89	50,639	2.04	340,941	1.51	1,458,985	1,799,926	4.28	16,513	14,175	85.8%	4,829	5,134	11,909	1,946	703	1168
22	204,035	1.21	86,877	1.89	51,532	2.03	342,444	1.51	1,475,658	1,818,102	4.31	16,595	14,244	85.8%	4,883	5,160	11,966	1,923	705	1169
23	205,516	1.21	87,059	1.89	52,970	2.02	345,545	1.51	1,506,074	1,851,619	4.36	16,733	14,360	85.8%	4,974	5,211	12,063	1,878	709	1222
24	[illegible]																			
25	205,916	1.21	101,316	1.96	157,653	2.21	464,885	1.71	3,830,003	4,294,888	8.24	25,610	21,673	84.6%	12,317	7,469	18,208	-1,578	913	1313
26	206,077	1.21	101,587	1.96	164,128	2.22	471,792	1.72	3,965,396	4,437,188	8.40	26,117	22,089	84.6%	12,753	7,600	18,557	-1,796	921	1362
27	206,189	1.21	102,064	1.96	167,388	2.20	475,641	1.72	4,015,554	4,491,195	8.44	26,313	22,251	84.6%	12,912	7,673	18,694	-1,891	925	1432
28	206,212	1.21	102,428	1.95	168,906	2.20	477,546	1.72	4,052,388	4,529,934	8.49	26,442	22,358	84.6%	13,033	7,708	18,783	-1,958	928	1466
29	206,959	1.21	102,463	1.96	173,447	2.21	482,869	1.73	4,166,384	4,649,253	8.63	26,832	22,679	84.5%	13,412	7,806	19,052	-2,166	936	1485
30	207,191	1.21	102,765	1.95	218,045	2.22	528,001	1.77	5,173,871	5,701,872	9.80	30,074	25,338	84.3%	16,681	8,675	21,287	-4,069	1001	1556
31	207,669	1.21	102,799	1.95	224,032	2.22	534,500	1.78	5,313,441	5,847,941	9.94	30,514	25,700	84.2%	17,130	8,798	21,591	-4,337	1009	1582
32	207,695	1.21	102,799	1.95	254,078	2.17	564,572	1.78	5,863,653	6,428,225	10.39	32,291	27,157	84.1%	18,963	9,378	22,815	-5,526	1044	1656
33	207,695	1.21	103,026	1.95	259,140	2.18	569,861	1.79	6,009,262	6,579,123	10.55	32,704	27,496	84.1%	19,430	9,479	23,099	-5,810	1051	1676
34	207,719	1.21	103,067	1.95	259,229	2.18	570,015	1.79	6,012,277	6,582,292	10.55	32,713	27,504	84.1%	19,441	9,482	23,107	-5,816	1052	1786
35	207,719	1.21	103,084	1.95	260,075	2.18	570,878	1.79	6,037,790	6,608,668	10.58	32,782	27,561	84.1%	19,523	9,499	23,154	-5,868	1053	1757



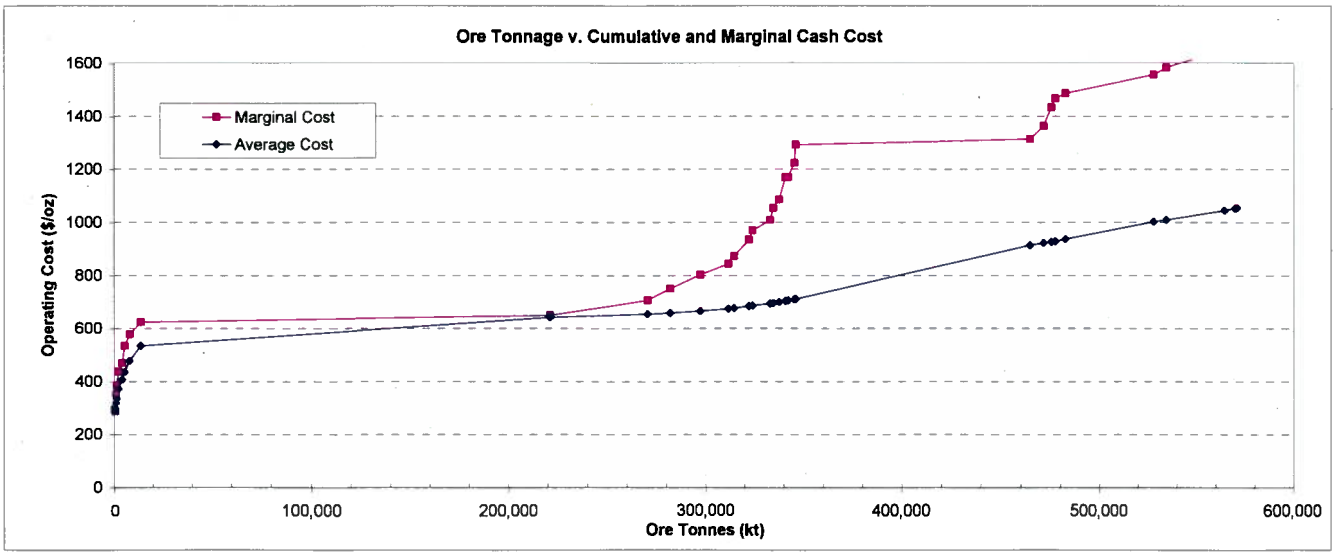
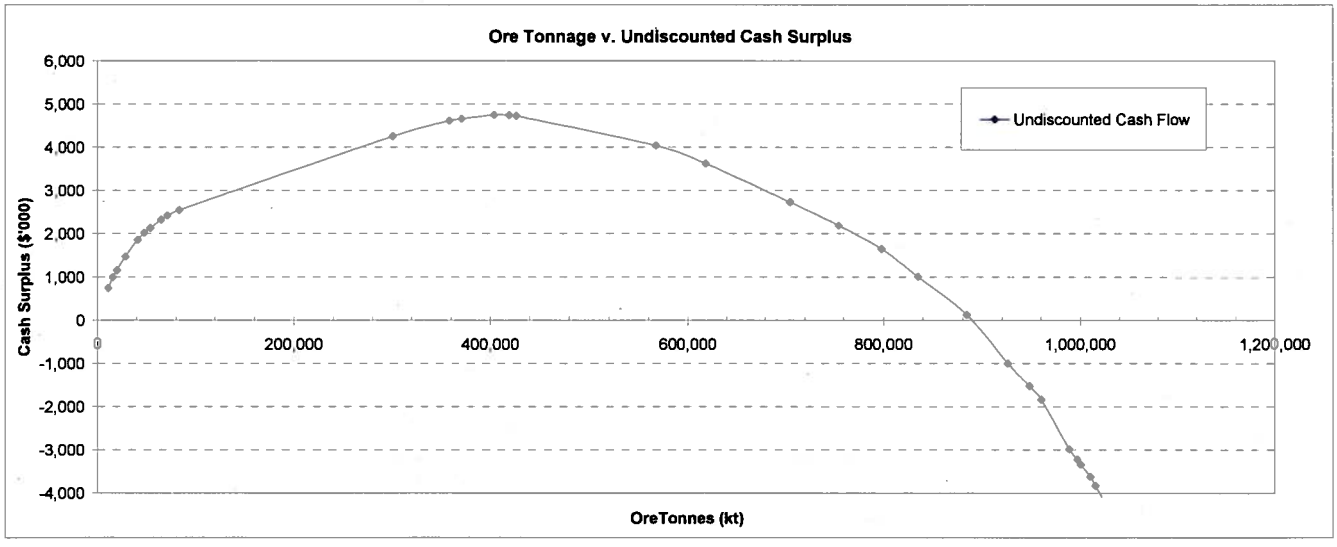


Table 6

Wonga Pit Optimisation Results - Run02c

$840/oz Price, Indicated & Inferred Resources

Pit No	Physicals														Economics					
	Oxide		Transitional		Primary		Total Resource		Waste	Total	W/O Ratio	Contn'd Ounces	Recv'd Ounces	Recv.	Mining Cost	Th'put Cost	Revenue	Operating Surplus	Unit Cost	Marg. Cost
	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Tonnes	t/t	koz	koz	%	$'000	$'000	$'000	$'000	$/oz	$/oz
1	11,011	3.75					11,011	3.75	24,682	35,693	2.24	1,327	1,168	88.0%	102	149	982	731	215	215
2	15,702	3.58					15,702	3.58	34,532	50,234	2.20	1,810	1,593	88.0%	144	212	1,338	982	224	247
3	19,710	3.38	124	3.14			19,834	3.38	43,377	63,211	2.19	2,153	1,895	88.0%	182	268	1,592	1,142	238	311
4	22,047	3.29	6,357	2.66			28,404	3.15	59,321	87,725	2.09	2,879	2,518	87.4%	255	401	2,116	1,460	261	331
5	23,923	3.24	16,866	2.56			40,789	2.96	92,179	132,968	2.26	3,881	3,374	86.9%	391	597	2,834	1,846	293	388
6	25,447	3.16	22,361	2.45			47,808	2.82	104,767	152,575	2.19	4,341	3,767	86.8%	449	707	3,164	2,008	307	427
7	26,530	3.09	26,948	2.37	315	3.55	53,793	2.73	118,526	172,319	2.20	4,718	4,088	86.7%	508	802	3,434	2,124	320	479
8	29,267	2.98	33,807	2.27	1,498	3.52	64,572	2.62	159,609	224,181	2.47	5,441	4,704	86.5%	662	974	3,952	2,316	348	530
9	31,571	2.89	38,122	2.24	1,498	3.52	71,191	2.56	186,320	257,511	2.62	5,849	5,055	86.4%	760	1,075	4,247	2,412	363	567
10	34,738	2.75	46,426	2.13	2,210	3.54	83,374	2.43	224,852	308,226	2.70	6,501	5,611	86.3%	912	1,267	4,714	2,535	388	619
11	188,628	1.52	104,217	2.04	7,826	2.97	300,671	1.74	1,022,833	1,323,504	3.40	16,784	14,520	86.5%	3,563	4,391	12,198	4,244	548	648
12	204,599	1.48	127,189	1.99	26,686	2.48	358,474	1.74	1,311,018	1,669,492	3.66	20,030	17,255	86.1%	4,541	5,344	14,495	4,610	573	706
13	210,507	1.47	129,829	1.98	30,578	2.37	370,914	1.72	1,351,520	1,722,434	3.64	20,570	17,714	86.1%	4,688	5,542	14,882	4,652	578	752
14	214,392	1.47	143,532	1.96	45,402	2.34	403,326	1.74	1,584,778	1,988,104	3.93	22,603	19,413	85.9%	5,468	6,104	16,310	4,738	596	789
15	222,124	1.45	146,273	1.96	50,392	2.36	418,789	1.74	1,676,775	2,095,564	4.00	23,372	20,062	85.8%	5,769	6,349	16,854	4,736	604	842
16	223,241	1.45	149,853	1.96	53,341	2.35	426,435	1.74	1,745,667	2,172,102	4.09	23,852	20,465	85.8%	5,998	6,479	17,193	4,716	610	890
17	241,660	1.39	194,050	1.91	131,728	2.39	567,438	1.80	3,094,795	3,662,233	5.45	32,842	27,937	85.1%	10,480	8,960	23,470	4,030	696	932
18	247,518	1.38	210,050	1.96	160,767	2.47	618,335	1.86	3,825,971	4,444,306	6.19	36,961	31,361	84.9%	12,869	9,859	26,346	3,618	725	960
19	250,108	1.37	214,452	1.96	240,538	2.44	705,098	1.92	4,955,424	5,660,522	7.03	43,417	36,668	84.5%	16,577	11,505	30,805	2,723	766	1009
20	254,575	1.36	218,362	1.95	281,892	2.41	754,829	1.92	5,504,626	6,259,455	7.29	46,664	39,343	84.3%	18,444	12,427	33,051	2,180	785	1043
21	266,026	1.34	227,947	1.95	304,130	2.39	798,103	1.91	5,952,892	6,750,995	7.46	49,106	41,382	84.3%	19,958	13,167	34,765	1,640	800	1105
22	271,009	1.33	232,109	1.95	332,075	2.38	835,193	1.92	6,445,176	7,280,369	7.72	51,512	43,369	84.2%	21,594	13,841	36,434	999	817	1163
23	277,356	1.31	241,260	1.93	365,835	2.35	884,451	1.91	7,020,324	7,904,775	7.94	54,316	45,690	84.1%	23,539	14,727	38,384	118	838	1220
24	[illegible]																			
25	281,232	1.31	248,337	1.92	418,629	2.38	948,198	1.94	8,212,886	9,161,084	8.66	59,162	49,682	84.0%	27,362	15,914	41,738	-1,538	871	1306
26	281,870	1.30	250,739	1.92	427,439	2.37	960,048	1.94	8,393,951	9,353,999	8.74	59,889	50,282	84.0%	27,959	16,132	42,242	-1,849	877	1359
27	282,031	1.30	251,121	1.92	455,696	2.41	988,848	1.97	9,183,530	10,172,378	9.29	62,620	52,522	83.9%	30,432	16,685	44,123	-2,994	897	1351
28	282,143	1.30	253,054	1.91	461,943	2.41	997,140	1.97	9,308,544	10,305,684	9.34	63,103	52,921	83.9%	30,848	16,839	44,459	-3,228	901	1428
29	282,619	1.30	253,891	1.91	464,303	2.40	1,000,813	1.97	9,369,466	10,370,279	9.36	63,317	53,098	83.9%	31,049	16,904	44,608	-3,345	903	1503
30	283,372	1.30	255,046	1.91	472,107	2.39	1,010,525	1.97	9,508,270	10,518,795	9.41	63,841	53,531	83.8%	31,521	17,084	44,972	-3,633	908	1506
31	283,827	1.30	256,392	1.91	475,469	2.39	1,015,688	1.96	9,614,698	10,630,386	9.47	64,166	53,800	83.8%	31,857	17,177	45,197	-3,837	911	1593
32	284,775	1.30	258,605	1.90	490,848	2.38	1,034,228	1.96	9,971,267	11,005,495	9.64	65,321	54,752	83.8%	33,050	17,522	45,997	-4,575	924	1616
33	287,546	1.29	258,785	1.90	496,682	2.38	1,043,013	1.96	10,130,216	11,173,229	9.71	65,775	55,128	83.8%	33,541	17,675	46,313	-4,903	929	1712
34	288,216	1.29	263,818	1.88	1,003,891	2.37	1,555,925	2.09	23,062,087	24,618,012	14.82	104,367	86,779	83.1%	76,923	27,555	72,904	-31,574	1204	1683
35	288,333	1.29	263,859	1.88	1,020,693	2.36	1,572,885	2.09	23,443,657	25,016,542	14.90	105,541	87,742	83.1%	78,245	27,882	73,713	-32,414	1210	1713



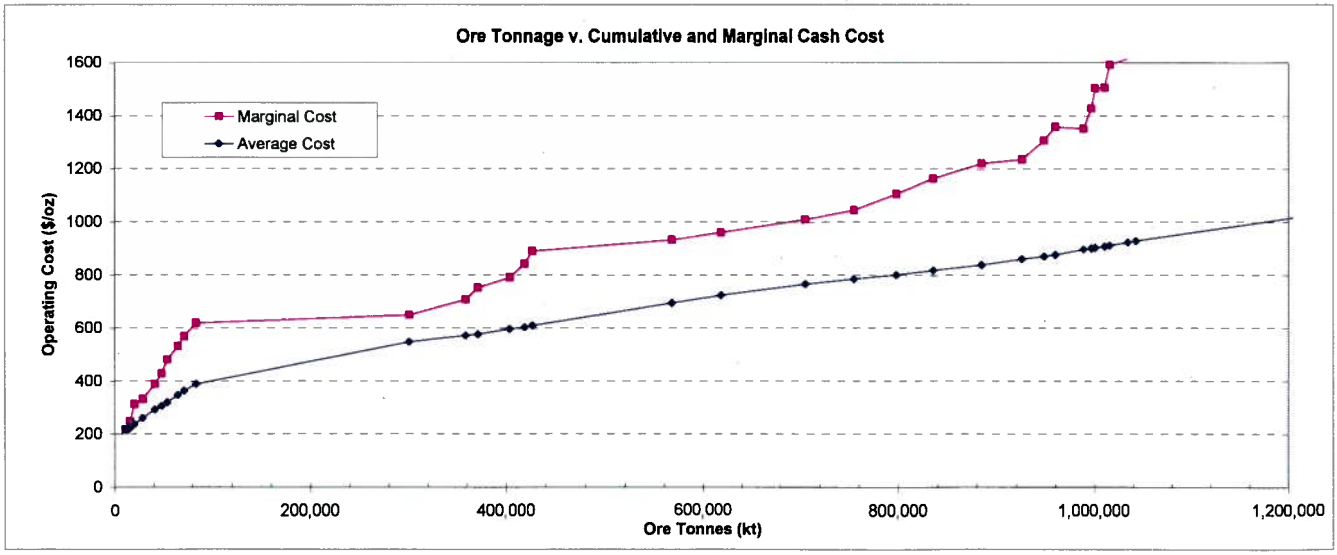


Figure 1
Wonga Pit Optimisation Study
Existing Pit and Dumps Surface

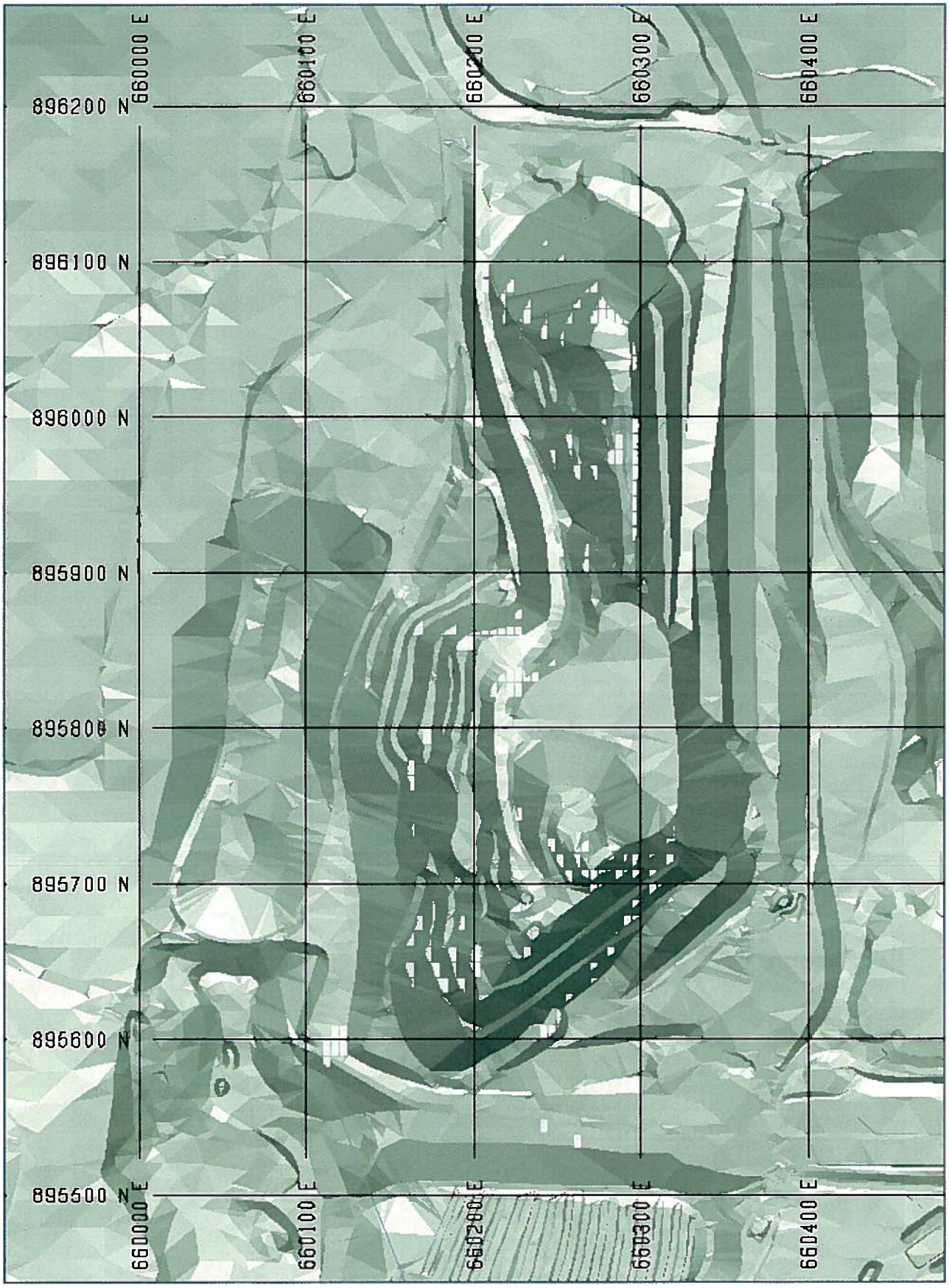


Figure 2
Wonga Pit Optimisation Study
Run01c Pit Shell 15 ($840/oz, Indicated Resource Only)

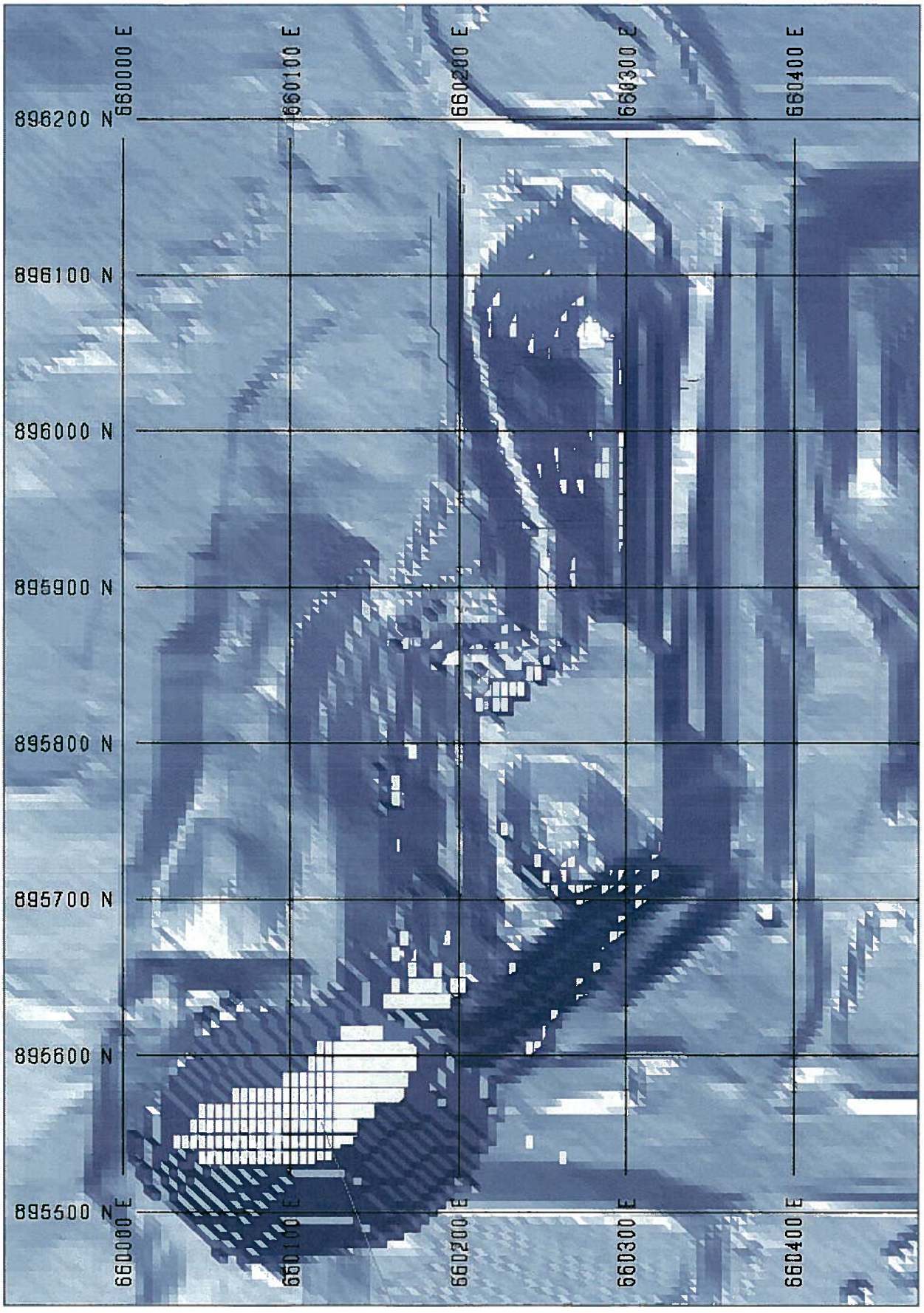


Figure 3
Wonga Pit Optimisation Study
Run02c Pit Shell 15 ($840/oz, Indicated & Inferred Resources)

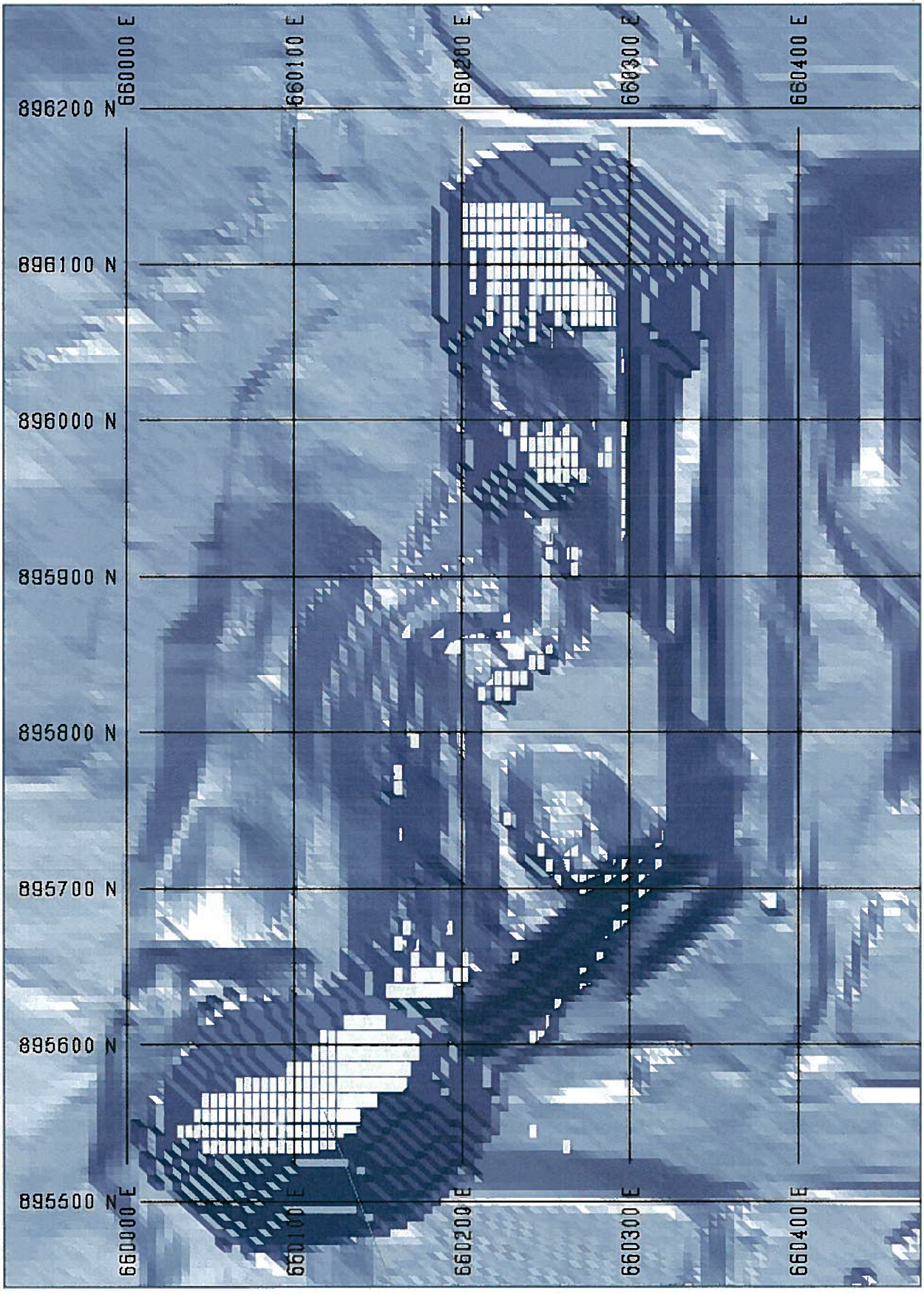


Figure 4
Wonga Pit Optimisation Study
Run02f Pit Shell 22 ($1,200/oz, Indicated & Inferred Resources)

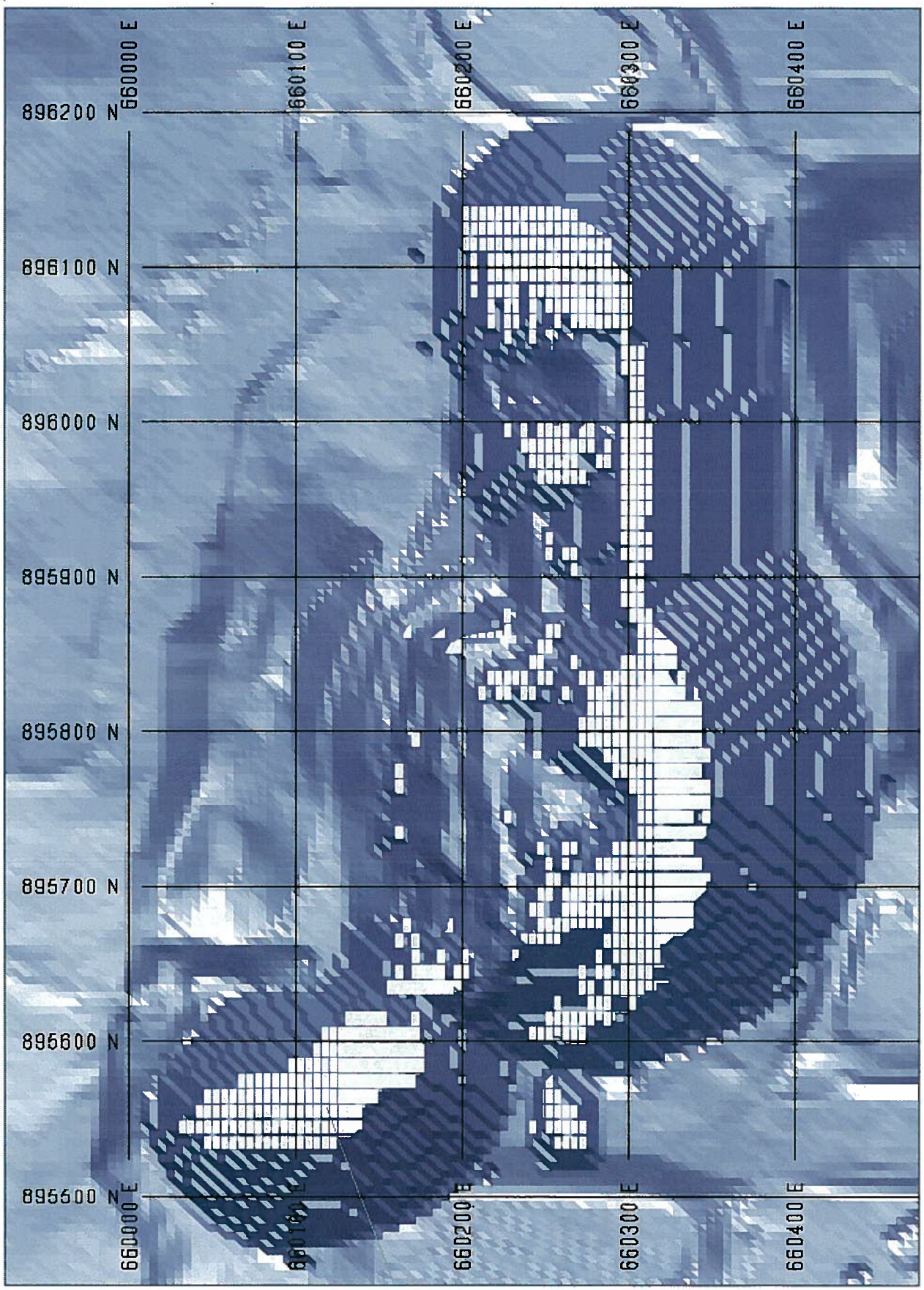
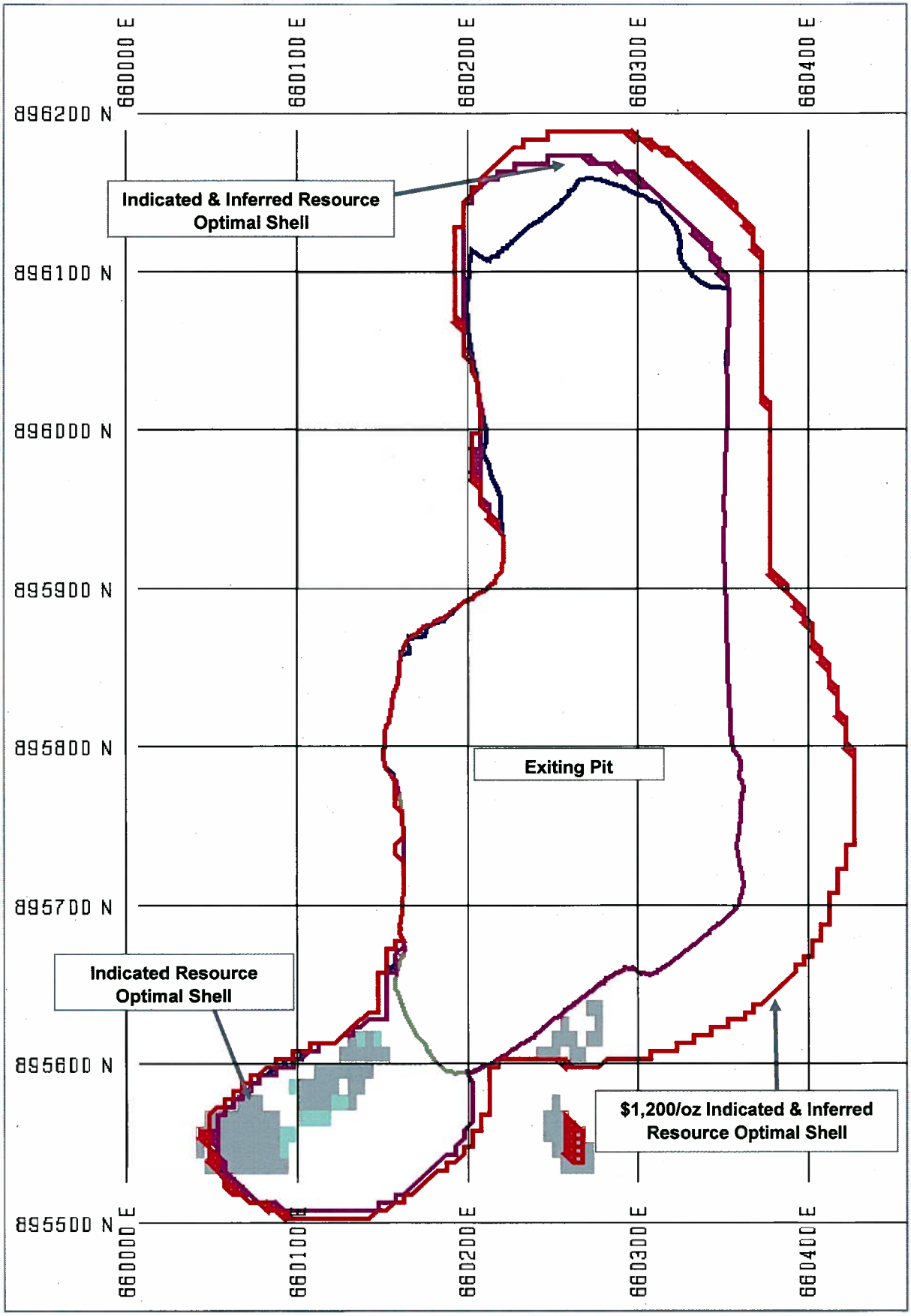


Figure 5
Wonga Pit Optimisation Study
Planer Section Through Pit Shells at -50mRL



Wonga Gold Deposit

Pit Limit Optimisation Study

Appendix 1

Pit Optimisation Sensitivity Result Details

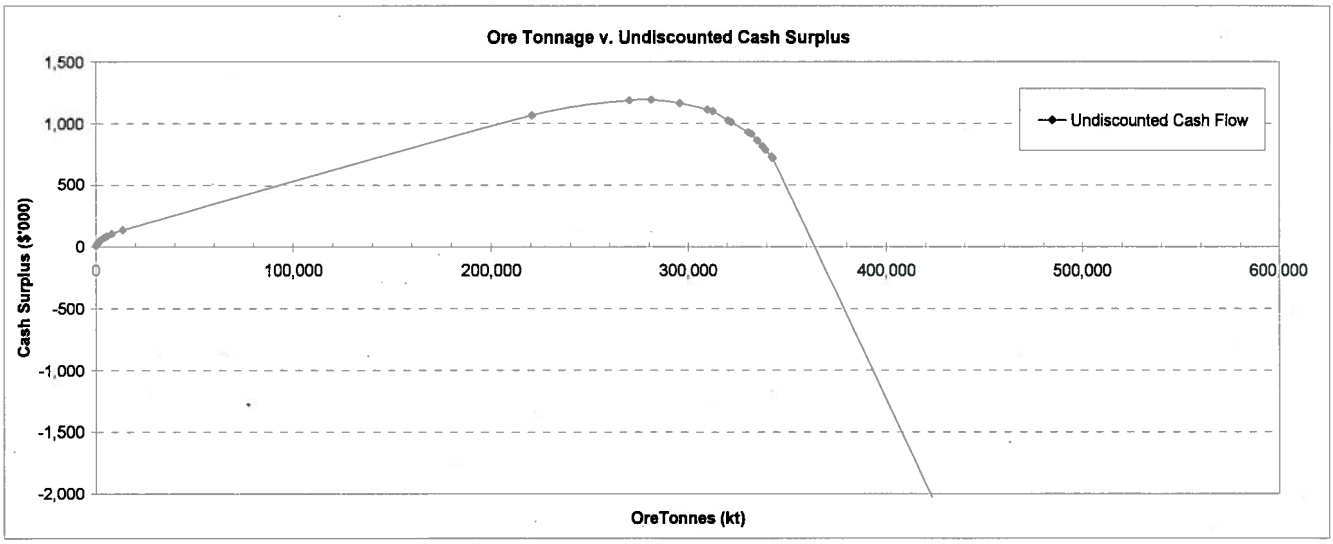
26 June 2007

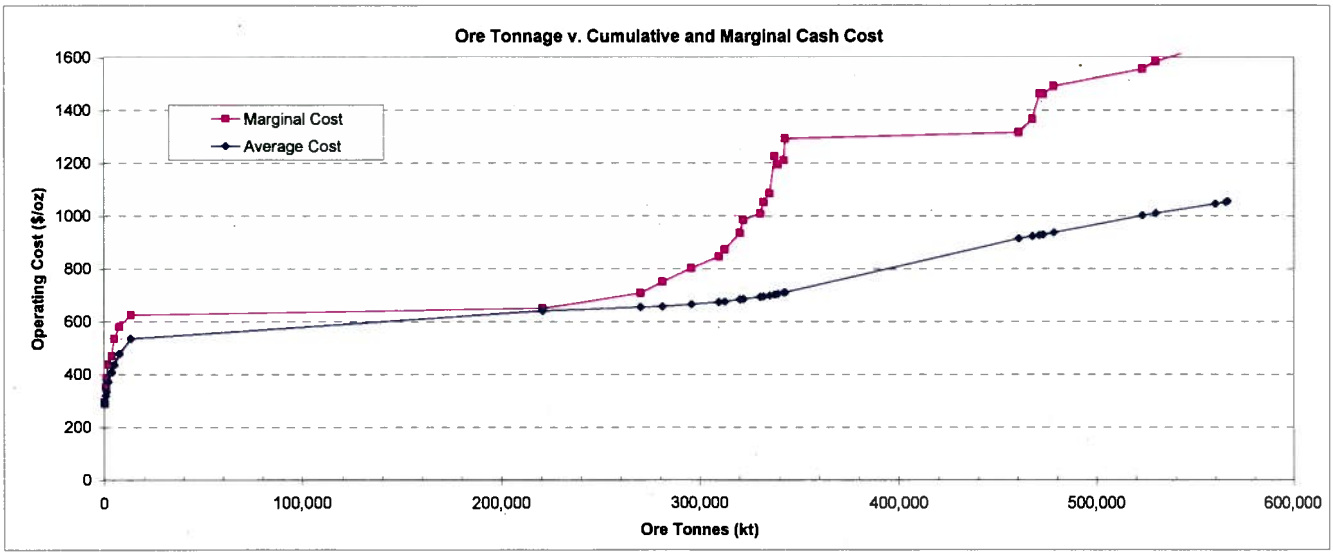
Appendix 1.1

Wonga Pit Optimisation Results - Run01a

$760/oz Price, Indicated Resources only

Pit No	Physicals														Economics					
	Oxide		Transitional		Primary		Total Resource		Waste	Total	W/O Ratio	Contn'd Ounces	Recv'd Ounces	Recv.	Mining Cost	Th'put Cost	Revenue	Operating Surplus	Unit Cost	Marg. Cost
	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Tonnes	t/t	koz	koz	%	$'000	$'000	$'000	$'000	$/oz	$/oz
1	144	2.51					144	2.51	197	341	1.37	12	10	88.0%	1	2	8	5	293	293
2	288	2.51	124	3.14			412	2.70	854	1,266	2.07	36	31	87.0%	3	6	24	15	290	288
3	686	1.98	247	3.14			933	2.28	1,210	2,143	1.30	68	60	86.9%	6	13	45	26	319	352
4	945	1.88	330	3.05			1,275	2.18	1,581	2,856	1.24	89	78	86.9%	8	18	59	33	334	384
5	2,002	1.70	330	3.05			2,332	1.89	2,644	4,976	1.13	142	124	87.3%	14	32	94	48	372	435
6	3,085	1.61	971	2.03			4,056	1.71	4,012	8,068	0.99	223	194	87.1%	22	57	147	68	407	468
7	4,073	1.56	1,187	1.94	107	2.60	5,367	1.67	6,341	11,708	1.18	288	250	87.0%	33	76	190	81	435	534
8	5,986	1.54	1,861	1.69	107	2.60	7,954	1.59	12,369	20,323	1.56	407	354	87.1%	56	113	269	100	477	578
9	9,018	1.47	4,517	1.62	107	2.60	13,642	1.53	26,228	39,870	1.92	670	582	86.9%	114	197	442	131	534	623
10	162,245	1.25	56,097	1.99	2,214	1.98	220,556	1.45	767,412	987,968	3.48	10,251	8,905	86.9%	2,560	3,144	6,766	1,062	641	648
11	178,216	1.24	72,884	1.93	18,904	2.16	270,004	1.49	1,004,987	1,274,991	3.72	12,950	11,181	86.3%	3,358	3,955	8,496	1,183	654	707
12	183,653	1.24	74,749	1.92	22,692	2.07	281,094	1.49	1,041,742	1,322,836	3.71	13,434	11,592	86.3%	3,489	4,132	8,809	1,188	657	750
13	187,097	1.24	78,621	1.91	30,123	2.03	295,841	1.50	1,130,520	1,426,361	3.82	14,235	12,264	86.2%	3,775	4,384	9,319	1,160	665	801
14	194,536	1.22	80,256	1.91	35,112	2.10	309,904	1.50	1,216,698	1,526,602	3.93	14,944	12,861	86.1%	4,055	4,608	9,773	1,110	674	844
15	195,582	1.22	80,974	1.90	36,170	2.09	312,726	1.50	1,233,728	1,546,454	3.95	15,082	12,978	86.0%	4,111	4,654	9,862	1,097	675	870
16	196,510	1.22	83,983	1.89	39,939	2.15	320,432	1.51	1,316,073	1,636,505	4.11	15,596	13,405	86.0%	4,376	4,788	10,186	1,022	684	934
17	197,260	1.22	84,423	1.89	40,365	2.14	322,048	1.51	1,328,185	1,650,233	4.12	15,670	13,469	86.0%	4,413	4,814	10,235	1,008	685	983
18	199,315	1.22	85,767	1.89	45,611	2.06	330,693	1.51	1,386,046	1,716,739	4.19	16,064	13,799	85.9%	4,595	4,964	10,484	925	693	1007
19	200,379	1.22	85,767	1.89	46,141	2.06	332,287	1.51	1,396,097	1,728,384	4.20	16,126	13,851	85.9%	4,625	4,989	10,525	911	694	1050
20	200,714	1.22	86,223	1.89	48,336	2.06	335,273	1.51	1,437,116	1,772,389	4.29	16,326	14,017	85.9%	4,751	5,043	10,651	857	699	1084
21	201,849	1.22	86,337	1.89	49,693	2.06	337,879	1.51	1,462,047	1,799,926	4.33	16,445	14,117	85.8%	4,829	5,087	10,727	811	702	1223
22	201,919	1.22	86,877	1.89	50,586	2.05	339,382	1.51	1,478,720	1,818,102	4.36	16,527	14,185	85.8%	4,883	5,114	10,779	782	705	1193
23	203,400	1.22	87,059	1.89	52,024	2.04	342,483	1.51	1,509,136	1,851,619	4.41	16,665	14,302	85.8%	4,974	5,164	10,866	728	709	1210
25	203,799	1.22	101,316	1.96	155,312	2.23	460,427	1.72	3,834,461	4,294,888	8.33	25,500	21,580	84.6%	12,317	7,395	16,398	-3,314	913	1315
26	203,961	1.22	101,587	1.96	161,787	2.24	467,335	1.73	3,969,853	4,437,188	8.49	26,007	21,997	84.6%	12,753	7,527	16,715	-3,565	922	1365
27	204,073	1.22	102,064	1.96	164,732	2.23	470,869	1.73	4,020,326	4,491,195	8.54	26,194	22,151	84.6%	12,912	7,593	16,832	-3,673	926	1459
28	204,096	1.22	102,428	1.95	166,249	2.23	472,773	1.73	4,057,161	4,529,934	8.58	26,323	22,258	84.6%	13,033	7,628	16,913	-3,748	928	1459
29	204,843	1.22	102,463	1.96	170,791	2.23	478,097	1.74	4,171,156	4,649,253	8.72	26,713	22,579	84.5%	13,412	7,727	17,157	-3,982	936	1489
30	205,075	1.22	102,765	1.95	215,389	2.23	523,229	1.78	5,178,643	5,701,872	9.90	29,955	25,238	84.3%	16,681	8,595	19,177	-6,099	1002	1556
31	205,553	1.22	102,799	1.95	221,376	2.23	529,728	1.78	5,318,213	5,847,941	10.04	30,394	25,599	84.2%	17,130	8,718	19,452	-6,396	1010	1583
32	205,579	1.22	102,799	1.95	251,422	2.19	559,800	1.79	5,868,425	6,428,225	10.48	32,171	27,057	84.1%	18,963	9,298	20,559	-7,702	1045	1656
33	205,579	1.22	103,026	1.95	256,484	2.19	565,089	1.79	6,014,034	6,579,123	10.64	32,585	27,396	84.1%	19,430	9,400	20,818	-8,012	1052	1676
34	205,603	1.22	103,067	1.95	256,573	2.19	565,243	1.79	6,017,049	6,582,292	10.65	32,595	27,404	84.1%	19,441	9,402	20,823	-8,020	1052	1651
35	205,603	1.22	103,084	1.95	257,419	2.19	566,106	1.79	6,042,562	6,608,668	10.67	32,663	27,460	84.1%	19,523	9,419	20,866	-8,076	1054	1771



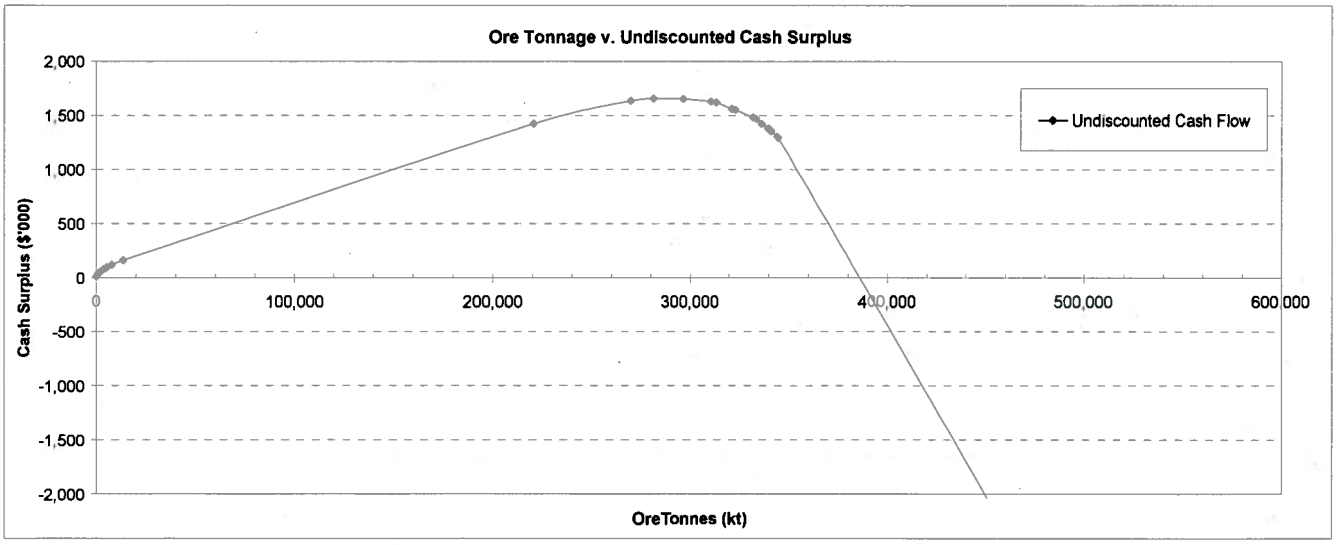


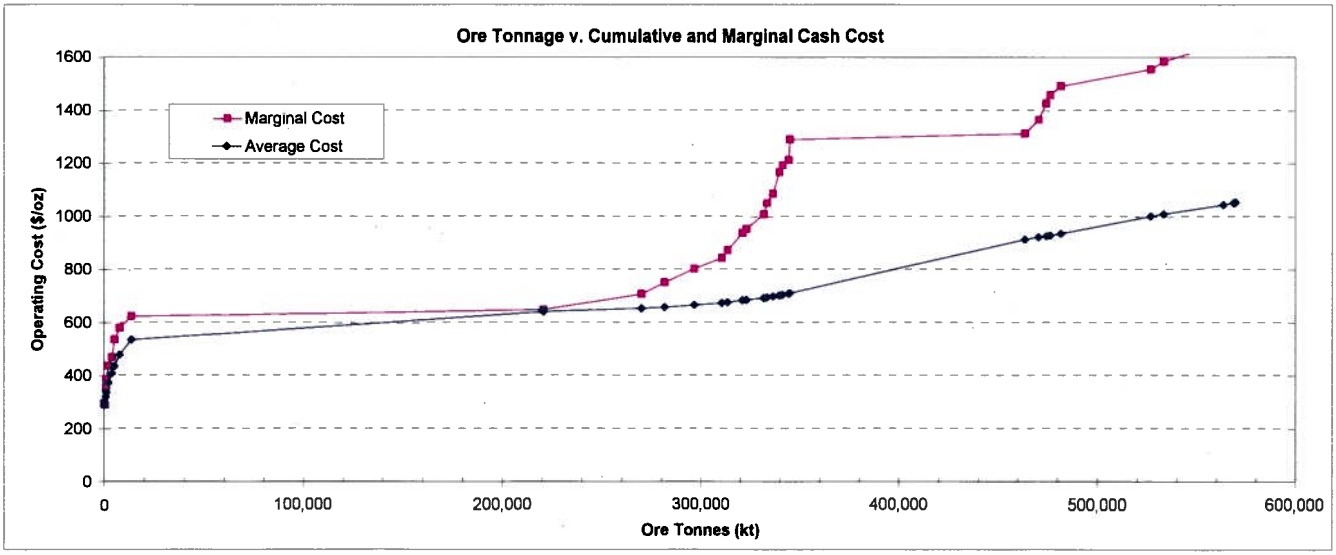
Appendix 1.2

Wonga Pit Optimisation Results - Run01b

$800/oz Price, Indicated Resources only

Pit No	Physicals														Economics					
	Oxide		Transitional		Primary		Total Resource		Waste	Total	W/O Ratio	Contn'd Ounces	Recv'd Ounces	Recv.	Mining Cost	Th'put Cost	Revenue	Operating Surplus	Unit Cost	Marg. Cost
	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Tonnes	t/t	koz	koz	%	$'000	$'000	$'000	$'000	$/oz	$/oz
1	144	2.51					144	2.51	197	341	1.37	12	10	88.0%	1	2	8	5	293	293
2	288	2.51	124	3.14			412	2.70	854	1,266	2.07	36	31	87.0%	3	6	25	16	290	288
3	686	1.98	247	3.14			933	2.28	1,210	2,143	1.30	68	60	86.9%	6	13	47	28	319	352
4	945	1.88	330	3.05			1,275	2.18	1,581	2,856	1.24	89	78	86.9%	8	18	62	36	334	384
5	2,002	1.70	330	3.05			2,332	1.89	2,644	4,976	1.13	142	124	87.3%	14	32	99	53	372	435
6	3,085	1.61	971	2.03			4,056	1.71	4,012	8,068	0.99	223	194	87.1%	22	57	155	76	407	468
7	4,073	1.56	1,187	1.94	107	2.60	5,367	1.67	6,341	11,708	1.18	288	250	87.0%	33	76	200	91	435	534
8	5,986	1.54	1,861	1.69	107	2.60	7,954	1.59	12,369	20,323	1.56	407	354	87.1%	56	113	283	114	477	578
9	9,018	1.47	4,517	1.62	107	2.60	13,642	1.53	26,228	39,870	1.92	670	582	86.9%	114	197	466	155	534	623
10	162,245	1.25	56,097	1.99	2,214	1.98	220,556	1.45	767,412	987,968	3.48	10,251	8,905	86.9%	2,560	3,144	7,123	1,419	641	648
11	178,216	1.24	72,884	1.93	18,904	2.16	270,004	1.49	1,004,987	1,274,991	3.72	12,950	11,181	86.3%	3,358	3,955	8,945	1,632	654	707
12	184,123	1.23	74,749	1.92	22,692	2.07	281,564	1.48	1,041,272	1,322,836	3.70	13,442	11,600	86.3%	3,489	4,138	9,280	1,653	658	750
13	187,567	1.23	78,621	1.91	30,438	2.02	296,626	1.49	1,129,735	1,426,361	3.81	14,253	12,280	86.2%	3,775	4,397	9,824	1,652	665	802
14	195,006	1.22	80,256	1.91	35,428	2.09	310,690	1.50	1,215,912	1,526,602	3.91	14,962	12,877	86.1%	4,055	4,620	10,301	1,626	674	843
15	196,052	1.22	80,974	1.90	36,485	2.08	313,511	1.50	1,232,943	1,546,454	3.93	15,101	12,994	86.0%	4,111	4,666	10,394	1,617	675	871
16	196,981	1.22	83,983	1.89	40,254	2.14	321,218	1.51	1,315,287	1,636,505	4.09	15,614	13,421	86.0%	4,376	4,801	10,737	1,560	684	936
17	197,965	1.22	84,423	1.89	40,681	2.13	323,069	1.51	1,327,164	1,650,233	4.11	15,694	13,489	86.0%	4,413	4,829	10,791	1,549	685	953
18	200,256	1.22	85,767	1.89	45,926	2.05	331,949	1.51	1,384,790	1,716,739	4.17	16,092	13,823	85.9%	4,595	4,983	11,057	1,479	693	1007
19	201,320	1.22	85,767	1.89	46,457	2.05	333,544	1.51	1,394,840	1,728,384	4.18	16,154	13,875	85.9%	4,625	5,008	11,100	1,467	694	1050
20	201,654	1.22	86,223	1.89	48,651	2.06	336,528	1.51	1,435,861	1,772,389	4.27	16,354	14,041	85.9%	4,751	5,062	11,233	1,420	699	1085
21	202,789	1.22	86,337	1.89	50,639	2.04	339,765	1.51	1,460,161	1,799,926	4.30	16,491	14,156	85.8%	4,829	5,118	11,325	1,378	703	1166
22	202,860	1.22	86,877	1.89	51,532	2.03	341,269	1.51	1,476,833	1,818,102	4.33	16,573	14,224	85.8%	4,883	5,145	11,379	1,351	705	1193
23	204,340	1.22	87,059	1.89	52,970	2.02	344,369	1.51	1,507,250	1,851,619	4.38	16,711	14,340	85.8%	4,974	5,195	11,471	1,302	709	1212
24	[illegible]																			
25	204,740	1.22	101,316	1.96	157,653	2.21	463,709	1.72	3,831,179	4,294,888	8.26	25,588	21,654	84.6%	12,317	7,453	17,323	-2,447	913	1313
26	204,902	1.22	101,587	1.96	164,128	2.22	470,617	1.72	3,966,571	4,437,188	8.43	26,095	22,070	84.6%	12,753	7,585	17,656	-2,682	922	1364
27	205,013	1.22	102,064	1.96	167,388	2.20	474,465	1.72	4,016,730	4,491,195	8.47	26,291	22,232	84.6%	12,912	7,657	17,785	-2,784	925	1426
28	205,037	1.22	102,428	1.95	168,906	2.20	476,371	1.73	4,053,563	4,529,934	8.51	26,421	22,339	84.6%	13,033	7,692	17,871	-2,854	928	1457
29	205,783	1.21	102,463	1.96	173,447	2.21	481,693	1.73	4,167,560	4,649,253	8.65	26,810	22,660	84.5%	13,412	7,791	18,128	-3,075	936	1491
30	206,015	1.21	102,765	1.95	218,045	2.22	526,825	1.77	5,175,047	5,701,872	9.82	30,052	25,319	84.3%	16,681	8,659	20,254	-5,086	1001	1556
31	206,493	1.21	102,799	1.95	224,032	2.22	533,324	1.78	5,314,617	5,847,941	9.97	30,491	25,680	84.2%	17,130	8,782	20,544	-5,368	1009	1584
32	206,519	1.21	102,799	1.95	254,078	2.17	563,396	1.78	5,864,829	6,428,225	10.41	32,269	27,138	84.1%	18,963	9,362	21,710	-6,615	1044	1655
33	206,519	1.21	103,026	1.95	259,140	2.18	568,685	1.79	6,010,438	6,579,123	10.57	32,682	27,477	84.1%	19,430	9,464	21,982	-6,912	1052	1679
34	206,543	1.21	103,067	1.95	259,229	2.18	568,839	1.79	6,013,453	6,582,292	10.57	32,692	27,485	84.1%	19,441	9,466	21,988	-6,919	1052	1658
35	206,543	1.21	103,084	1.95	260,075	2.18	569,702	1.79	6,038,966	6,608,668	10.60	32,760	27,541	84.1%	19,523	9,483	22,033	-6,973	1053	1757



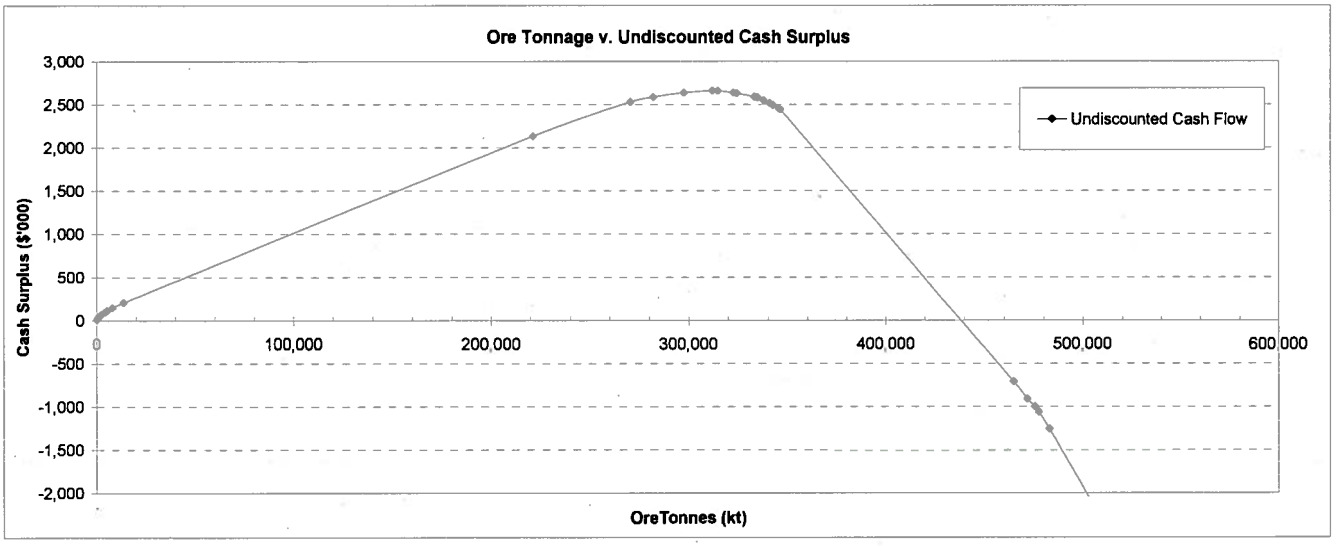


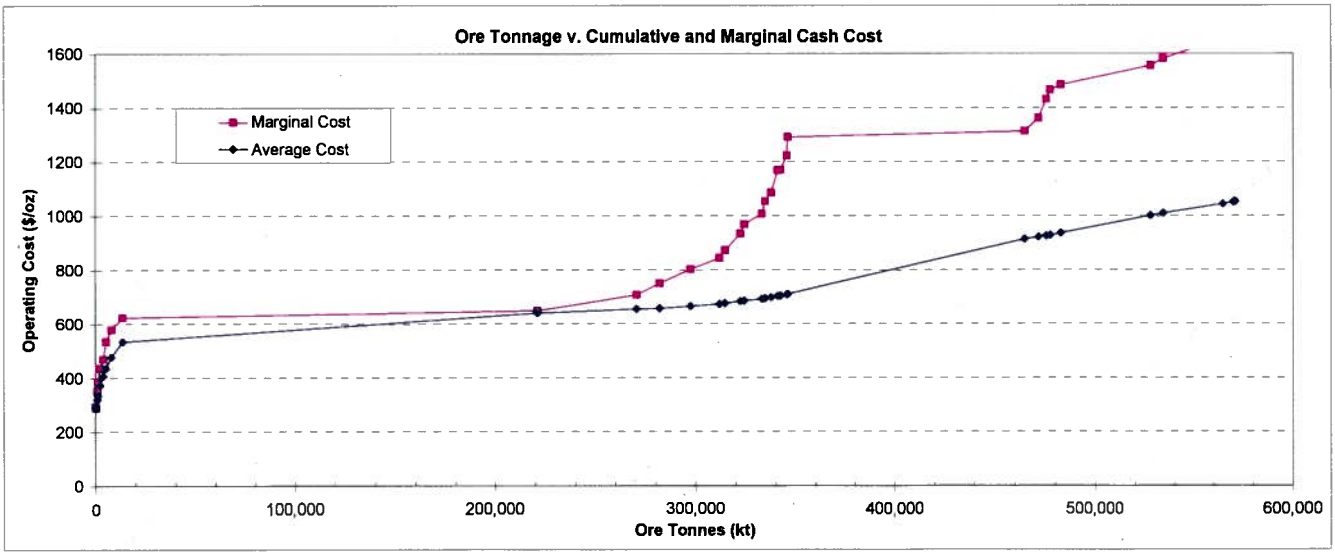
Appendix 1.3

Wonga Pit Optimisation Results - Run01d

$880/oz Price, Indicated Resources only

Pit No	Physicals														Economics					
	Oxide		Transitional		Primary		Total Resource		Waste	Total	W/O Ratio	Contn'd Ounces	Recv'd Ounces	Recv.	Mining Cost	Th'put Cost	Revenue	Operating Surplus	Unit Cost	Marg. Cost
	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Tonnes	t/t	koz	koz	%	$'000	$'000	$'000	$'000	$/oz	$/oz
1	144	2.51					144	2.51	197	341	1.37	12	10	88.0%	1	2	9	6	293	293
2	288	2.51	124	3.14			412	2.70	854	1,266	2.07	36	31	87.0%	3	6	27	18	290	288
3	686	1.98	247	3.14			933	2.28	1,210	2,143	1.30	68	60	86.9%	6	13	52	33	319	352
4	945	1.88	330	3.05			1,275	2.18	1,581	2,856	1.24	89	78	86.9%	8	18	68	42	334	384
5	2,002	1.70	330	3.05			2,332	1.89	2,644	4,976	1.13	142	124	87.3%	14	32	109	63	372	435
6	3,085	1.61	971	2.03			4,056	1.71	4,012	8,068	0.99	223	194	87.1%	22	57	170	91	407	468
7	4,073	1.56	1,187	1.94	107	2.60	5,367	1.67	6,341	11,708	1.18	288	250	87.0%	33	76	220	111	435	534
8	5,986	1.54	1,861	1.69	107	2.60	7,954	1.59	12,369	20,323	1.56	407	354	87.1%	56	113	311	142	477	578
9	9,018	1.47	4,517	1.62	107	2.60	13,642	1.53	26,228	39,870	1.92	670	582	86.9%	114	197	512	201	534	623
10	162,832	1.25	56,097	1.99	2,214	1.98	221,143	1.44	766,825	987,968	3.47	10,262	8,914	86.9%	2,560	3,152	7,843	2,131	641	648
11	178,803	1.24	72,884	1.93	18,904	2.16	270,591	1.49	1,004,400	1,274,991	3.71	12,961	11,191	86.3%	3,358	3,963	9,847	2,526	654	707
12	184,711	1.23	74,749	1.92	22,692	2.07	282,152	1.48	1,040,684	1,322,836	3.69	13,453	11,610	86.3%	3,489	4,146	10,216	2,581	658	750
13	188,449	1.23	78,621	1.91	30,438	2.02	297,508	1.49	1,128,853	1,426,361	3.79	14,270	12,294	86.2%	3,775	4,409	10,818	2,634	666	802
14	196,181	1.22	80,256	1.91	35,428	2.09	311,865	1.49	1,214,737	1,526,602	3.90	14,984	12,896	86.1%	4,055	4,636	11,348	2,657	674	843
15	197,227	1.22	80,974	1.90	36,485	2.08	314,686	1.49	1,231,768	1,546,454	3.91	15,122	13,013	86.1%	4,111	4,682	11,450	2,657	676	872
16	198,156	1.22	83,983	1.89	40,254	2.14	322,393	1.51	1,314,112	1,636,505	4.08	15,636	13,440	86.0%	4,376	4,816	11,826	2,634	684	933
17	199,141	1.22	84,423	1.89	40,681	2.13	324,245	1.51	1,325,988	1,650,233	4.09	15,715	13,508	86.0%	4,413	4,845	11,886	2,628	685	968
18	201,431	1.22	85,767	1.89	45,926	2.05	333,124	1.50	1,383,615	1,716,739	4.15	16,114	13,842	85.9%	4,595	4,999	12,179	2,585	693	1007
19	202,495	1.21	85,767	1.89	46,457	2.05	334,719	1.50	1,393,665	1,728,384	4.16	16,175	13,894	85.9%	4,625	5,024	12,226	2,577	694	1053
20	202,830	1.22	86,223	1.89	48,651	2.06	337,704	1.51	1,434,685	1,772,389	4.25	16,375	14,060	85.9%	4,751	5,078	12,372	2,543	699	1085
21	203,965	1.21	86,337	1.89	50,639	2.04	340,941	1.51	1,458,985	1,799,926	4.28	16,513	14,175	85.8%	4,829	5,134	12,473	2,510	703	1168
22	204,035	1.21	86,877	1.89	51,532	2.03	342,444	1.51	1,475,658	1,818,102	4.31	16,595	14,244	85.8%	4,883	5,160	12,533	2,490	705	1169
23	205,516	1.21	87,059	1.89	52,970	2.02	345,545	1.51	1,506,074	1,851,619	4.36	16,733	14,360	85.8%	4,974	5,211	12,635	2,450	709	1222
24	[illegible]																			
25	205,916	1.21	101,316	1.96	157,653	2.21	464,885	1.71	3,830,003	4,294,888	8.24	25,610	21,673	84.6%	12,317	7,469	19,071	-715	913	1313
26	206,077	1.21	101,587	1.96	164,128	2.22	471,792	1.72	3,965,396	4,437,188	8.40	26,117	22,089	84.6%	12,753	7,600	19,436	-917	921	1362
27	206,189	1.21	102,064	1.96	167,388	2.20	475,641	1.72	4,015,554	4,491,195	8.44	26,313	22,251	84.6%	12,912	7,673	19,580	-1,005	925	1432
28	206,212	1.21	102,428	1.95	168,906	2.20	477,546	1.72	4,052,388	4,529,934	8.49	26,442	22,358	84.6%	13,033	7,708	19,673	-1,068	928	1466
29	206,959	1.21	102,463	1.96	173,447	2.21	482,869	1.73	4,166,384	4,649,253	8.63	26,832	22,679	84.5%	13,412	7,806	19,955	-1,263	936	1485
30	207,191	1.21	102,765	1.95	218,045	2.22	528,001	1.77	5,173,871	5,701,872	9.80	30,074	25,338	84.3%	16,681	8,675	22,296	-3,060	1001	1556
31	207,669	1.21	102,799	1.95	224,032	2.22	534,500	1.78	5,313,441	5,847,941	9.94	30,514	25,700	84.2%	17,130	8,798	22,614	-3,314	1009	1582
32	207,695	1.21	102,799	1.95	254,078	2.17	564,572	1.78	5,863,653	6,428,225	10.39	32,291	27,157	84.1%	18,963	9,378	23,896	-4,445	1044	1656
33	207,695	1.21	103,026	1.95	259,140	2.18	569,861	1.79	6,009,262	6,579,123	10.55	32,704	27,496	84.1%	19,430	9,479	24,194	-4,715	1051	1676
34	207,719	1.21	103,067	1.95	259,229	2.18	570,015	1.79	6,012,277	6,582,292	10.55	32,713	27,504	84.1%	19,441	9,482	24,202	-4,721	1052	1786
35	207,719	1.21	103,084	1.95	260,075	2.18	570,878	1.79	6,037,790	6,608,668	10.58	32,782	27,561	84.1%	19,523	9,499	24,251	-4,771	1053	1757



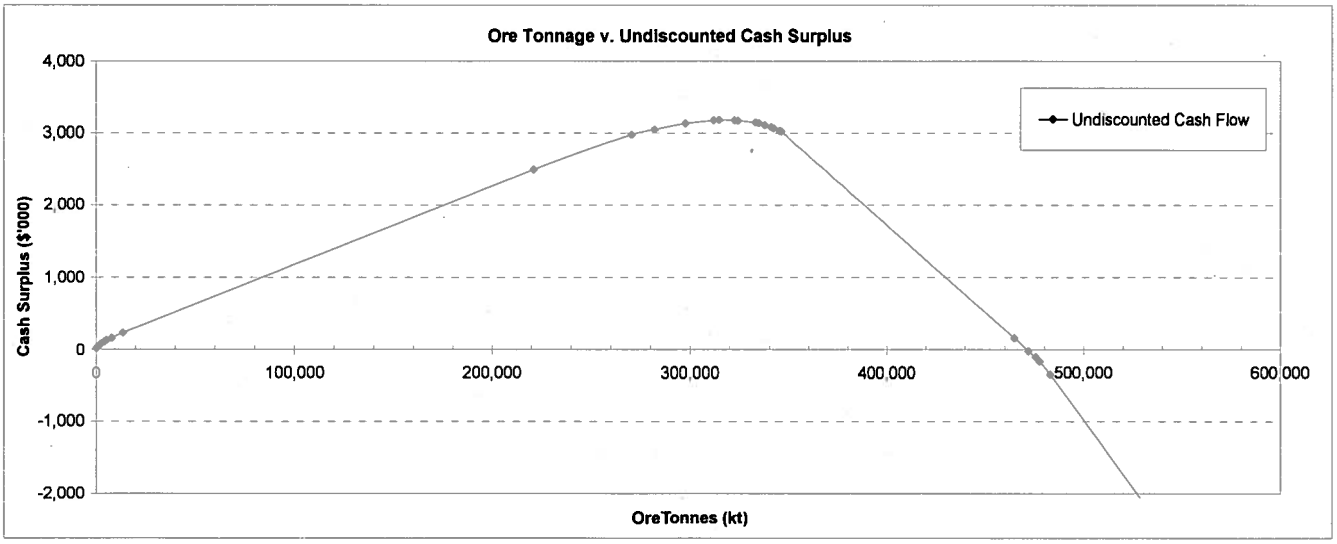


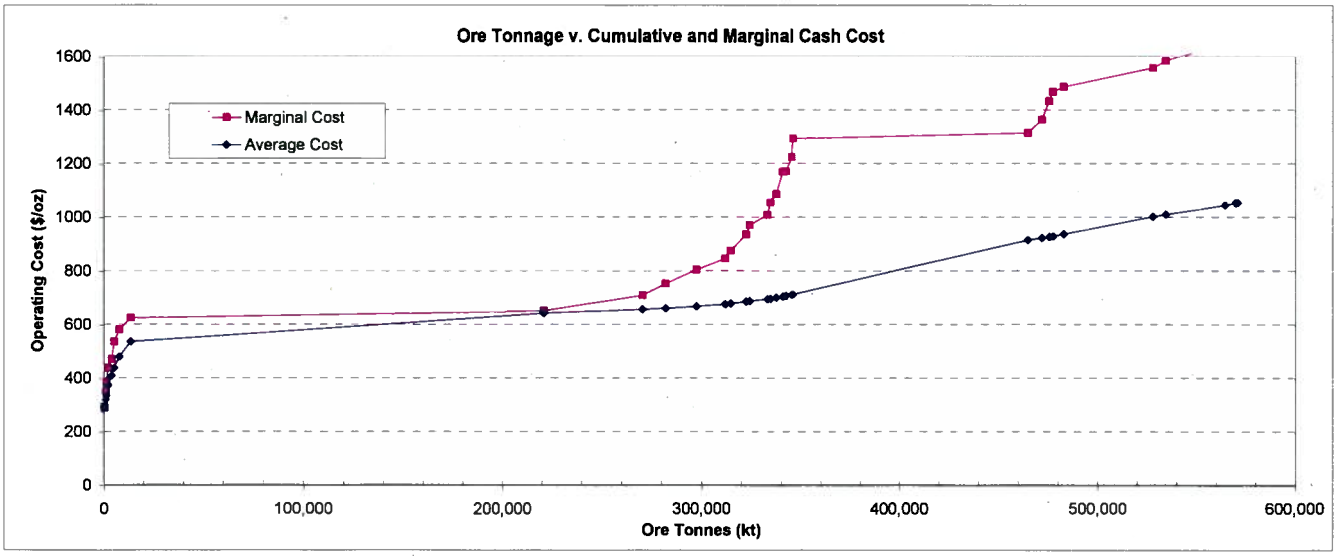
Appendix 1.4

Wonga Pit Optimisation Results - Run01e

$920/oz Price, Indicated Resources only

Pit No	Physicals														Economics					
	Oxide		Transitional		Primary		Total Resource		Waste	Total	W/O Ratio	Contn'd Ounces	Recv'd Ounces	Recv.	Mining Cost	Th'put Cost	Revenue	Operating Surplus	Unit Cost	Marg. Cost
	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Tonnes	t/t	koz	koz	%	$'000	$'000	$'000	$'000	$/oz	$/oz
1	144	2.51					144	2.51	197	341	1.37	12	10	88.0%	1	2	10	7	293	293
2	288	2.51	124	3.14			412	2.70	854	1,266	2.07	36	31	87.0%	3	6	28	19	290	288
3	686	1.98	247	3.14			933	2.28	1,210	2,143	1.30	68	60	86.9%	6	13	55	36	319	352
4	945	1.88	330	3.05			1,275	2.18	1,581	2,856	1.24	89	78	86.9%	8	18	71	45	334	384
5	2,002	1.70	330	3.05			2,332	1.89	2,644	4,976	1.13	142	124	87.3%	14	32	114	68	372	435
6	3,085	1.61	971	2.03			4,056	1.71	4,012	8,068	0.99	223	194	87.1%	22	57	178	99	407	468
7	4,073	1.56	1,187	1.94	107	2.60	5,367	1.67	6,341	11,708	1.18	288	250	87.0%	33	76	230	121	435	534
8	5,986	1.54	1,861	1.69	107	2.60	7,954	1.59	12,369	20,323	1.56	407	354	87.1%	56	113	325	156	477	578
9	9,018	1.47	4,517	1.62	107	2.60	13,642	1.53	26,228	39,870	1.92	670	582	86.9%	114	197	536	225	534	623
10	162,832	1.25	56,097	1.99	2,214	1.98	221,143	1.44	766,825	987,968	3.47	10,262	8,914	86.9%	2,560	3,152	8,201	2,489	641	648
11	178,803	1.24	72,884	1.93	18,904	2.16	270,591	1.49	1,004,400	1,274,991	3.71	12,961	11,191	86.3%	3,358	3,963	10,296	2,975	654	707
12	184,711	1.23	74,749	1.92	22,692	2.07	282,152	1.48	1,040,684	1,322,836	3.69	13,453	11,610	86.3%	3,489	4,146	10,682	3,047	658	750
13	188,449	1.23	78,621	1.91	30,438	2.02	297,508	1.49	1,128,853	1,426,361	3.79	14,270	12,294	86.2%	3,775	4,409	11,312	3,128	666	802
14	196,181	1.22	80,256	1.91	35,428	2.09	311,865	1.49	1,214,737	1,526,602	3.90	14,984	12,896	86.1%	4,055	4,636	11,865	3,174	674	843
15	197,227	1.22	80,974	1.90	36,485	2.08	314,686	1.49	1,231,768	1,546,454	3.91	15,122	13,013	86.1%	4,111	4,682	11,972	3,179	676	872
16	198,156	1.22	83,983	1.89	40,254	2.14	322,393	1.51	1,314,112	1,636,505	4.08	15,636	13,440	86.0%	4,376	4,816	12,366	3,174	684	933
17	199,141	1.22	84,423	1.89	40,681	2.13	324,245	1.51	1,325,988	1,650,233	4.09	15,715	13,508	86.0%	4,413	4,845	12,428	3,170	685	968
18	201,431	1.22	85,767	1.89	45,926	2.05	333,124	1.50	1,383,615	1,716,739	4.15	16,114	13,842	85.9%	4,595	4,999	12,735	3,141	693	1007
19	202,495	1.21	85,767	1.89	46,457	2.05	334,719	1.50	1,393,665	1,728,384	4.16	16,175	13,894	85.9%	4,625	5,024	12,784	3,135	694	1053
20	202,830	1.22	86,223	1.89	48,651	2.06	337,704	1.51	1,434,685	1,772,389	4.25	16,375	14,060	85.9%	4,751	5,078	12,937	3,108	699	1085
21	203,965	1.21	86,337	1.89	50,639	2.04	340,941	1.51	1,458,985	1,799,926	4.28	16,513	14,175	85.8%	4,829	5,134	13,042	3,079	703	1168
22	204,035	1.21	86,877	1.89	51,532	2.03	342,444	1.51	1,475,658	1,818,102	4.31	16,595	14,244	85.8%	4,883	5,160	13,104	3,061	705	1169
23	205,516	1.21	87,059	1.89	52,970	2.02	345,545	1.51	1,506,074	1,851,619	4.36	16,733	14,360	85.8%	4,974	5,211	13,211	3,026	709	1222
25	205,916	1.21	101,316	1.96	157,653	2.21	464,885	1.71	3,830,003	4,294,888	8.24	25,610	21,673	84.6%	12,317	7,469	19,941	155	913	1313
26	206,077	1.21	101,587	1.96	164,128	2.22	471,792	1.72	3,965,396	4,437,188	8.40	26,117	22,089	84.6%	12,753	7,600	20,323	-30	921	1362
27	206,189	1.21	102,064	1.96	167,388	2.20	475,641	1.72	4,015,554	4,491,195	8.44	26,313	22,251	84.6%	12,912	7,673	20,472	-113	925	1432
28	206,212	1.21	102,428	1.95	168,906	2.20	477,546	1.72	4,052,388	4,529,934	8.49	26,442	22,358	84.6%	13,033	7,708	20,570	-171	928	1466
29	206,959	1.21	102,463	1.96	173,447	2.21	482,869	1.73	4,166,384	4,649,253	8.63	26,832	22,679	84.5%	13,412	7,806	20,865	-353	936	1485
30	207,191	1.21	102,765	1.95	218,045	2.22	528,001	1.77	5,173,871	5,701,872	9.80	30,074	25,338	84.3%	16,681	8,675	23,312	-2,044	1001	1556
31	207,669	1.21	102,799	1.95	224,032	2.22	534,500	1.78	5,313,441	5,847,941	9.94	30,514	25,700	84.2%	17,130	8,798	23,645	-2,283	1009	1582
32	207,695	1.21	102,799	1.95	254,078	2.17	564,572	1.78	5,863,653	6,428,225	10.39	32,291	27,157	84.1%	18,963	9,378	24,986	-3,355	1044	1656
33	207,695	1.21	103,026	1.95	259,140	2.18	569,861	1.79	6,009,262	6,579,123	10.55	32,704	27,496	84.1%	19,430	9,479	25,297	-3,612	1051	1676
34	207,719	1.21	103,067	1.95	259,229	2.18	570,015	1.79	6,012,277	6,582,292	10.55	32,713	27,504	84.1%	19,441	9,482	25,305	-3,618	1052	1786
35	207,719	1.21	103,084	1.95	260,075	2.18	570,878	1.79	6,037,790	6,608,668	10.58	32,782	27,561	84.1%	19,523	9,499	25,357	-3,665	1053	1757



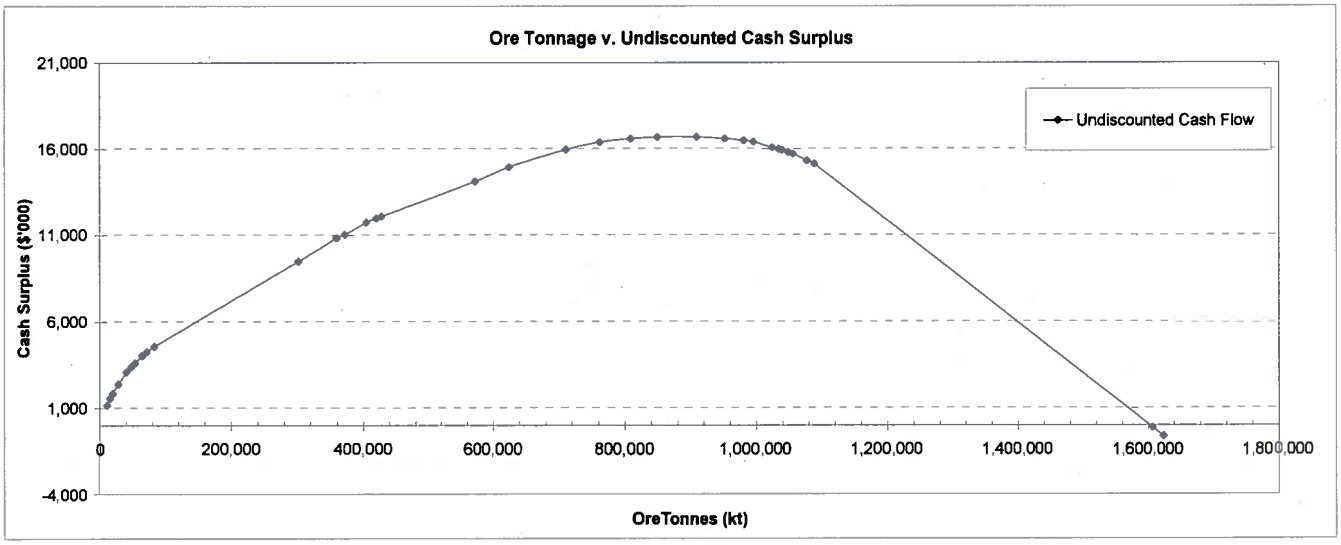


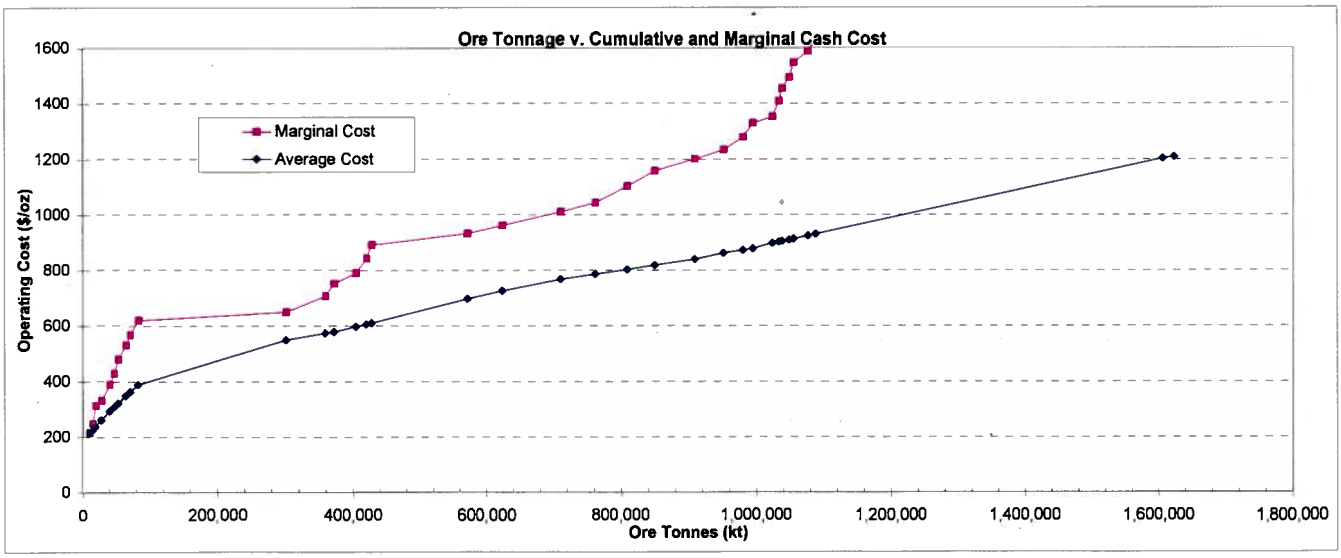
Appendix 1.5

Wonga Pit Optimisation Results - Run02f

$1,200/oz Price, Indicated & Inferred Resources

Pit No	Physicals														Economics					
	Oxide		Transitional		Primary		Total Resource		Waste	Total	W/O Ratio	Contn'd Ounces	Recv'd Ounces	Recv.	Mining Cost	Th'put Cost	Revenue	Operating Surplus	Unit Cost	Marg. Cost
	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Tonnes	t/t	koz	koz	%	$'000	$'000	$'000	$'000	$/oz	$/oz
1	11,011	3.75					11,011	3.75	24,682	35,693	2.24	1,327	1,168	88.0%	102	149	1,402	1,151	215	215
2	15,702	3.58					15,702	3.58	34,532	50,234	2.20	1,810	1,593	88.0%	144	212	1,911	1,555	224	247
3	19,710	3.38	124	3.14			19,834	3.38	43,377	63,211	2.19	2,153	1,895	88.0%	182	268	2,274	1,824	238	311
4	22,047	3.29	6,357	2.66			28,404	3.15	59,321	87,725	2.09	2,879	2,518	87.4%	255	401	3,022	2,366	261	331
5	23,923	3.24	16,866	2.56			40,789	2.96	92,179	132,968	2.26	3,881	3,374	86.9%	391	597	4,049	3,061	293	388
6	25,447	3.16	22,361	2.45			47,808	2.82	104,767	152,575	2.19	4,341	3,767	86.8%	449	707	4,520	3,364	307	427
7	26,530	3.09	26,948	2.37	315	3.55	53,793	2.73	118,526	172,319	2.20	4,718	4,088	86.7%	508	802	4,905	3,595	320	479
8	29,267	2.98	33,807	2.27	1,498	3.52	64,572	2.62	159,609	224,181	2.47	5,441	4,704	86.5%	662	974	5,644	4,008	348	530
9	31,571	2.89	38,122	2.24	1,498	3.52	71,191	2.56	186,320	257,511	2.62	5,849	5,055	86.4%	760	1,075	6,066	4,231	363	567
10	34,738	2.75	46,426	2.13	2,210	3.54	83,374	2.43	224,852	308,226	2.70	6,501	5,611	86.3%	912	1,267	6,733	4,554	388	619
11	189,769	1.51	104,217	2.04	7,826	2.97	301,812	1.73	1,021,692	1,323,504	3.39	16,801	14,535	86.5%	3,563	4,407	17,442	9,472	548	649
12	205,740	1.48	127,189	1.99	26,686	2.48	359,615	1.73	1,309,877	1,669,492	3.64	20,048	17,270	86.1%	4,541	5,360	20,724	10,823	573	706
13	211,647	1.47	129,829	1.98	30,578	2.37	372,054	1.72	1,350,380	1,722,434	3.63	20,588	17,729	86.1%	4,688	5,558	21,276	11,030	578	752
14	215,532	1.46	143,532	1.96	45,740	2.33	404,804	1.74	1,583,300	1,988,104	3.91	22,628	19,436	85.9%	5,468	6,126	23,323	11,729	597	790
15	223,265	1.44	146,273	1.96	50,729	2.35	420,267	1.73	1,675,297	2,095,564	3.99	23,398	20,084	85.8%	5,769	6,371	24,101	11,961	604	842
16	224,382	1.44	149,853	1.96	53,678	2.34	427,913	1.74	1,744,189	2,172,102	4.08	23,878	20,488	85.8%	5,998	6,501	24,585	12,086	610	890
17	244,320	1.38	195,562	1.90	132,066	2.38	571,948	1.79	3,090,285	3,662,233	5.40	32,923	28,006	85.1%	10,480	9,027	33,607	14,100	697	932
18	250,386	1.37	212,541	1.94	161,104	2.47	624,031	1.85	3,820,275	4,444,306	6.12	37,065	31,451	84.9%	12,869	9,945	37,741	14,927	725	960
19	252,977	1.36	216,943	1.94	240,876	2.44	710,796	1.90	4,949,726	5,660,522	6.96	43,522	36,759	84.5%	16,577	11,591	44,110	15,942	766	1009
20	257,673	1.35	221,928	1.93	282,230	2.41	761,831	1.91	5,497,624	6,259,455	7.22	46,794	39,455	84.3%	18,444	12,534	47,345	16,367	785	1042
21	271,430	1.32	232,771	1.93	304,467	2.39	808,668	1.90	5,942,327	6,750,995	7.35	49,299	41,549	84.3%	19,958	13,325	49,857	16,574	801	1101
22	278,937	1.30	237,324	1.92	333,087	2.38	849,348	1.90	6,431,021	7,280,369	7.57	51,771	43,593	84.2%	21,594	14,053	52,310	16,663	818	1157
23	291,570	1.27	250,621	1.88	366,848	2.35	909,039	1.87	6,995,736	7,904,775	7.70	54,766	46,079	84.1%	23,539	15,091	55,293	16,663	838	1200
25	299,792	1.26	260,832	1.86	419,642	2.37	980,266	1.90	8,180,818	9,161,084	8.35	59,747	50,188	84.0%	27,362	16,387	60,224	16,475	872	1278
26	301,570	1.25	264,570	1.85	428,982	2.37	995,122	1.89	8,358,877	9,353,999	8.40	60,532	50,836	84.0%	27,959	16,652	61,002	16,391	878	1329
27	301,885	1.25	264,952	1.85	457,239	2.41	1,024,076	1.92	9,148,302	10,172,378	8.93	63,264	53,078	83.9%	30,432	17,208	63,692	16,052	898	1351
28	301,996	1.25	267,682	1.84	463,824	2.40	1,033,502	1.92	9,272,182	10,305,684	8.97	63,770	53,496	83.9%	30,848	17,381	64,194	15,965	902	1409
29	303,766	1.25	268,519	1.84	466,184	2.40	1,038,469	1.92	9,331,810	10,370,279	8.99	64,004	53,691	83.9%	31,049	17,464	64,429	15,916	904	1454
30	304,518	1.25	270,031	1.84	473,988	2.39	1,048,537	1.91	9,470,258	10,518,795	9.03	64,537	54,131	83.9%	31,521	17,649	64,956	15,786	908	1494
31	306,534	1.24	271,601	1.83	477,349	2.39	1,055,484	1.91	9,574,902	10,630,386	9.07	64,889	54,425	83.9%	31,857	17,767	65,308	15,684	912	1547
32	308,702	1.24	275,531	1.82	492,729	2.38	1,076,962	1.91	9,928,533	11,005,495	9.22	66,097	55,422	83.8%	33,050	18,157	66,504	15,297	924	1587
33	312,921	1.23	275,938	1.82	499,368	2.37	1,088,227	1.90	10,085,002	11,173,229	9.27	66,599	55,839	83.8%	33,541	18,349	67,005	15,115	929	1638
34	314,499	1.23	284,025	1.79	1,007,823	2.36	1,606,347	2.04	23,011,665	24,618,012	14.33	105,292	87,576	83.2%	76,923	28,315	105,088	-150	1202	1681
35	314,616	1.23	284,066	1.79	1,024,624	2.36	1,623,306	2.04	23,393,236	25,016,542	14.41	106,466	88,539	83.2%	78,245	28,641	106,244	-642	1207	1712





TECHNICAL REPORT

on Stawell Gold Mines

Victoria, Australia

28 March 2008

APPENDIX L

Davis Pit South Extension

Reserve Estimate

January 2006

Stawell Gold Mines Pty Ltd

DAVIS PIT SOUTH EXTENSION RESERVE ESTIMATE

Draft

January 2006

mining solutions

Draft Copy Issue Date: *30 January 2006*

Final Copy Issue Date:

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APPENDICES

Appendix A SGM Resource Model Classification Information Memorandum
Appendix B Solid and Surface Data Used to Code the Resource Model
Appendix C SGM Metallurgical Test Results Report
Appendix D Pit Optimisation Result Details

1.0 INTRODUCTION

At the request of the Stawell Gold Mines Pty Ltd (SGM), Mining Solutions Consultancy Pty Ltd (MSC) has undertaken the reserve estimation work for further development of the existing Davis pit. The Davis pit was mined previously to a depth of 50m (-65mRL) and partly backfilled with waste starting from south end where the previous ramp exit located (Figure 1.1). Extensive historic underground workings are present below the existing Davis pit surface.

The reserve estimation work included the following tasks:

- Resource model review for the effect of old underground workings, mining dilution and ore loss estimates
- Review of the mining costs and geotechnical parameters used in the reserve estimate
- Pit limit optimisation using Whittle Four-X to determine the ultimate pit limits and sensitivities
- Pit designs and reporting of the mining inventory by levels
- Project reserves statement based on the calculated mining inventory

The processing recovery, processing and administration costs used in the reserve estimates have been provided by SGM based on the operational experience. A gold price of $700/oz has been used in the pit optimisations an evaluation of the pit designs.

The Davis pit south extension reserve estimate at the end of this report is based on the operating costs only. Any pre-production costs (including contractor mobilisation), finance costs and effect of taxation should be taken in to account in the detailed financial analysis for a decision on the development of the Davis pit. This will be particularly important considering the relatively high operating cost per ounce ($640/oz) for the reserves estimate.

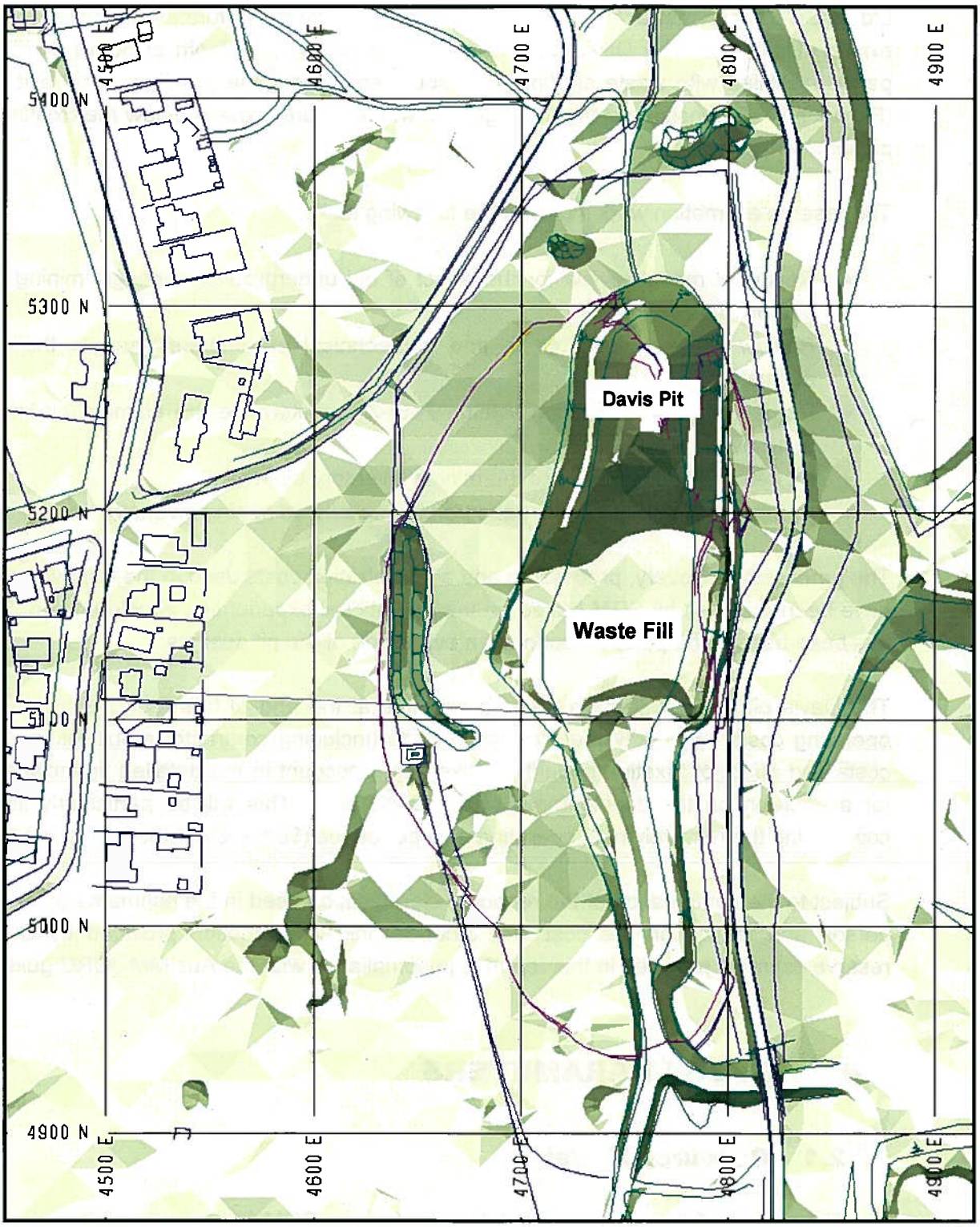
Subject to the confirmation of the resource classification used in the estimates by a qualified person and relying on the cost and other technical information provided by SGM, the reserve estimate provided in this report is in compliance with the AusIMM JORC guidelines.

2.0 STUDY PARAMETERS

2.1 Resource Model

The review of the resource model received from SGM has confirmed that the current model is the same as the resource model used in the 1998-99 Big Hill project feasibility study.

Figure 2.1 Topographic Surface at the Davis Pit Area



SGM confirmed (Appendix A) that the resource estimate in the Davis Pit Extension area is classified as Indicated within the SGM guidelines. Although SGM suggested that a 30% void factor should be applicable to account for the historic underground workings in the estimate, the basis for this suggestion was not clear.

MSC should note that no external factors for voids (other then ORE% item in the resource model) were applied in the 1998-99 Big Hill feasibility study based on the same resource model. The review of the resource model in comparison to the underground solids indicated that some (not all) of the old underground stopes were already incorporated as voids in the resource model.

As a result of the review, MSC decided to deduct the old underground workings (solids for stopes an levels) found in the old Big Hill project directory from the resource model. The ORE% item stored in the resource model was adjusted by coding the model from the available wire-frame data. The mined surface for the existing Davis pit was also applied to eliminate a small number of resource blocks located above the surface. The data used in the manipulation of the model, including underground solids, as mined and backfill pit surfaces, is provided in Appendix B in electronic form.

Table 1.1 summarises the resources reported within the Davis pit area before and after the underground deduction applied. The comparison shows an underground void deduction of 12% on above the deduction already observed in the resource model original ORE% item. MSC tried to quantify the magnitude of the underground voids already incorporated in the resource model but this attempt was not successful since a detailed report was not available.

Table 2.1 Davis Pit Area Resource Report

Cut-off Grade	**g/t Au**	**0.70**	**1.00**
Before U/G Deduction			
Indicated Resource	Tonnes	1,636,000	1,470,000
Gold Grade	g/t Au	1.98	2.10
After U/G Deduction			
Indicated Resource	Tonnes	1,460,000	1,298,000
Gold Grade	g/t Au	1.92	2.05
Underground Deduction			
Indicated Resource	Tonnes	176,000	172,000
Gold Grade	g/t Au	2.45	2.48

Resources were reported below 5,400mN section line and above 130mRL.

The review of the resource model indicated that application of a mining dilution factor of 10% and a resource recovery factor of 95% would be appropriate. These factors may be considered slightly high for the relative continuity of the resource zones but the mining above the old underground workings would result in additional ore loss and dilution.

2.2 Pit Optimisation Parameters

The parameters used in the pit optimisations are summarised in Table 2.2. The SGM metallurgical test result details provided in Appendix C support the 90% recovery used in the pit optimisations and evaluations. The processing and administration costs provided by SGM are based on the operational history at Stawell.

Pit supervision and grade control cost estimates are based on MSC's experience in similar projects/operations. Note that these costs may vary largely depending on the optimal pit resource quantities and stripping ratios.

The pit optimisation cases have included the gold price sensitivity between $550/oz and $700/oz in $50/oz increments and sensitivity for ±20% mining cost variation.

Table 2.2 Pit Optimisation Parameters

Gold Price	$/oz	550	600	650	700
Royalty	$/oz	2.00	2.00	2.00	2.00
Net Gold Price	$/gr	17.62	19.23	20.83	22.44
Cut-off Grade	Au g/t	1.20	1.10	1.01	0.94
Process Recovery	%	90%			
Pit Slopes	deg	43	(Based on old pit design, incl ramps)		
Mining Cost Factors	%	80%	100%	120%	

Thoughput Costs

Processing & Administration	$/t ore	16.00
Pit Supervision	$/t ore	1.50
Grade Control	$/t ore	1.50
Total Throughput Cost	$/t ore	19.00

Model Parameters

Indicated resource category only	
Mining Dilution	10%
Ore Loss	5%

2.3 Pit Optimisation Mining Costs

Table 2.3 below list the mining costs by levels as used in the optimisations. The costs are based on the recent contract mining costs received for similar projects.

The level of mining cost estimates confirm well with the original SGM estimate for Wild Wood project based on a 50t excavator and two 50t articulated trucks. The mining cost estimates in Table 2.3 assume ~100t excavator and ~60t trucks, resulting in an average cost estimate of ~5-10% lower (depending on the mined levels) compared to the original SGM estimate.

The backfill rehandle costs were estimated separately as the backfill blocks in the optimisation model were coded between the backfill and the as-mined surfaces. The densities stored in the block model (2.0t/bcm) were used directly in the calculation of the backfill costs.

Table 2.3 Pit Optimisation Mining Costs

Depth (m)	Density t/bcm	L&H & Ancillary Waste $/bcm	L&H & Ancillary Ore $/bcm	Drill & Blast $/bcm	Other $/bcm	Total Waste $/bcm	Total Ore $/bcm	Total Waste $/t	Total Ore $/t	Backfill Rehandle $/t Mined
-20	2.00	3.37	3.82	1.20	0.44	5.00	5.45	2.50	2.73	1.62
-30	2.00	3.44	3.89	1.20	0.44	5.08	5.53	2.54	2.76	1.65
-40	2.00	3.51	3.96	1.20	0.45	5.16	5.61	2.58	2.80	1.68
-50	2.00	3.58	4.03	1.20	0.45	5.23	5.68	2.62	2.84	1.71
-60	2.00	3.66	4.11	1.20	0.46	5.31	5.76	2.66	2.88	1.75
-70	2.30	4.29	4.74	2.10	0.50	6.89	7.34	2.99	3.19	2.03
-80	2.30	4.37	4.82	2.10	0.51	6.97	7.42	3.03	3.23	2.07
-90	2.30	4.45	4.90	2.10	0.51	7.06	7.51	3.07	3.27	2.11
-100	2.70	5.32	5.77	3.20	0.57	9.10	9.55	3.37	3.54	2.51
-110	2.70	5.42	5.87	3.20	0.58	9.20	9.65	3.41	3.57	2.55
-120	2.70	5.52	5.97	3.20	0.59	9.30	9.75	3.45	3.61	2.59
-130	2.70	5.61	6.06	3.20	0.59	9.41	9.86	3.48	3.65	2.64

Other Cost Details:

Rehabilitation	$/bcm	0.10
Dewatering	$/bcm	0.10
Dayworks	of L&H	7%

3.0 PIT OPTIMISATION RESULTS

The physical and financial quantities for the optimal pit shells obtained from the optimisation cases are summarised in Table 3.1. The optimal pit shells (12 and 14) that have been considered for a detailed pit design at $650/oz and $700/oz gold prices are highlighted in the table. Figure 3.1, 3.2 and 3.3 below show the contours for these pit shells in relation to the town boundary and current site topography.

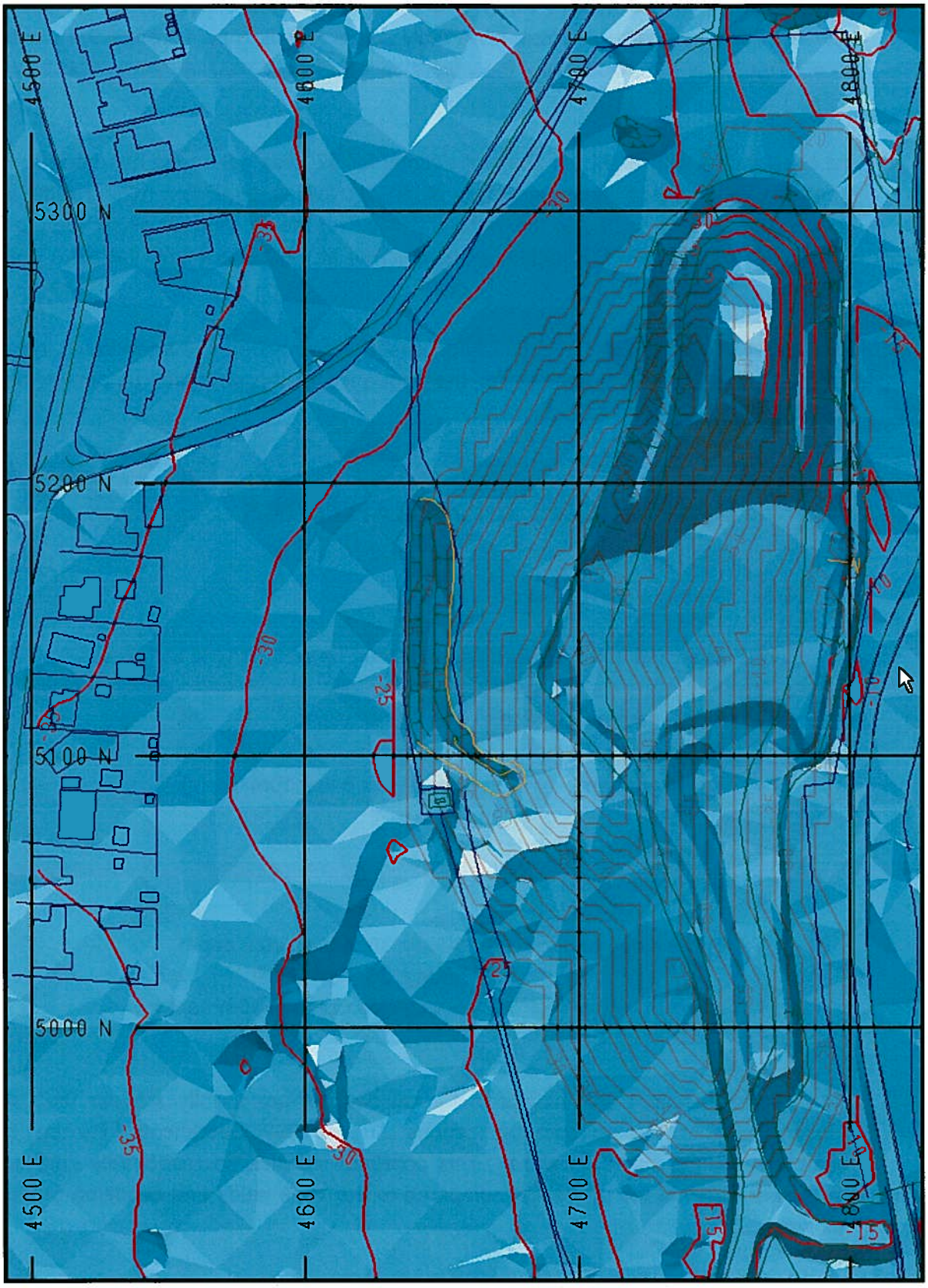
Table 3.1 Pit Optimisation Results Summary

Run ID	Case Description	Pit No	Physicals								Economics						
			Ore		Waste	Total	W/O Ratio	Contn'd Ounces	Recv'd Ounces	Recv.	Mining Cost	Th'put Cost	Reven.	Undisc'd Surplus	Th'put Cost	Min. Cost	Unit Cost
			Total kt	Au g/t	kt	kt	t/t	Au	Au	%	$'000	$'000	$'000	$'000	$/t	$/t	$/oz
Run01a1	$550/oz, Gold Price, 80% Mining Costs	11	83	2.51	639	721	7.7	6,650	5,985	90%	1,359	1,587	3,280	334	19.23	1.88	492
Run01b1	$600/oz, Gold Price, 80% Mining Costs	13	276	2.33	2,286	2,563	8.3	20,728	18,656	90%	5,055	5,310	11,158	793	19.21	1.97	556
Run01c1	$650/oz, Gold Price, 80% Mining Costs	15	314	2.27	2,485	2,798	7.9	22,930	20,637	90%	5,563	6,026	13,370	1,781	19.21	1.99	562
Run01d1	$700/oz, Gold Price, 80% Mining Costs	18	366	2.24	2,926	3,292	8.0	26,306	23,675	90%	6,645	7,029	16,524	2,850	19.21	2.02	578
Run01a2	$550/oz, Gold Price, 100% Mining Costs	8	16	2.19	16	32	1.1	1,100	990	90%	79	301	543	163	19.23	2.45	383
Run01b2	$600/oz, Gold Price, 100% Mining Costs	10	63	2.51	426	490	6.7	5,101	4,591	90%	1,213	1,217	2,746	316	19.23	2.48	529
Run01c2	$650/oz, Gold Price, 100% Mining Costs	12	101	2.40	737	838	7.3	7,796	7,017	90%	1,987	1,943	4,546	616	19.23	2.37	560
Run01d2	$700/oz, Gold Price, 100% Mining Costs	14	312	2.25	2,400	2,712	7.7	22,548	20,293	90%	6,727	5,995	14,164	1,442	19.21	2.48	627
Run01a3	$550/oz, Gold Price, 120% Mining Costs	7	15	2.21	16	32	1.1	1,081	973	90%	93	293	533	148	19.23	2.94	396
Run01b3	$600/oz, Gold Price, 120% Mining Costs	8	16	2.18	16	32	1.0	1,103	993	90%	94	302	594	197	19.23	2.94	399
Run01c3	$650/oz, Gold Price, 120% Mining Costs	9	50	2.36	264	313	5.3	3,777	3,399	90%	924	957	2,202	321	19.23	2.95	553
Run01d3	$700/oz, Gold Price, 120% Mining Costs	11	94	2.33	627	721	6.7	7,047	6,342	90%	2,038	1,808	4,427	581	19.23	2.83	606

Figure 3.1 Optimal Pit Shell 12 Contours ($650/oz Gold Price)

Figure 3.2 Optimal Pit Shell 14 Contours ($700/oz Gold Price)

Figure 3.3 Optimal Pit Shell 14 ($700/oz) Contours and Topographic Surface



The optimal pit shell 12 for $650/oz gold price show two separate pit developments at the south and north. The northern pit development around the existing Davis pit requires the establishment of the access ramp mostly within the backfill material. Accessing the northern and southern pit areas separately would significantly increase the waste mining requirements and consequently the operating costs for a detailed pit design based on pit shell 12.

In consultation with SGM, pit shell 14 optimal for $700/oz gold price has been selected for the detailed pit design process. This pit shell provides significant mining width with 70m and 120m cutback widths to the west and south of the existing Davis pit (before backfill). The distance between the optimal pit shell crest at the surface and the town boundary is 90m.

The detailed results for the optimisation cases are provided in Appendix D, including the physical and financial quantities for the sequence of the pit shells obtained from all the optimisations.

4.0 PIT DESIGNS AND QUANTITIES

4.1 Design Criteria

The criteria used in the Davis pit designs can be summarised as follows:

- Single lane ramp width: 12m
- Passing bay width: 17-18m
- Ramp gradient: 1 in 9 (1 in 8 last 10m depth)

- Berm width: 5m
- Berm intervals: 20m vertical

- Batter slope angles: 50deg, surface to 20m depth
  55deg, 20-40m depth
  60deg, greater than 60m depth

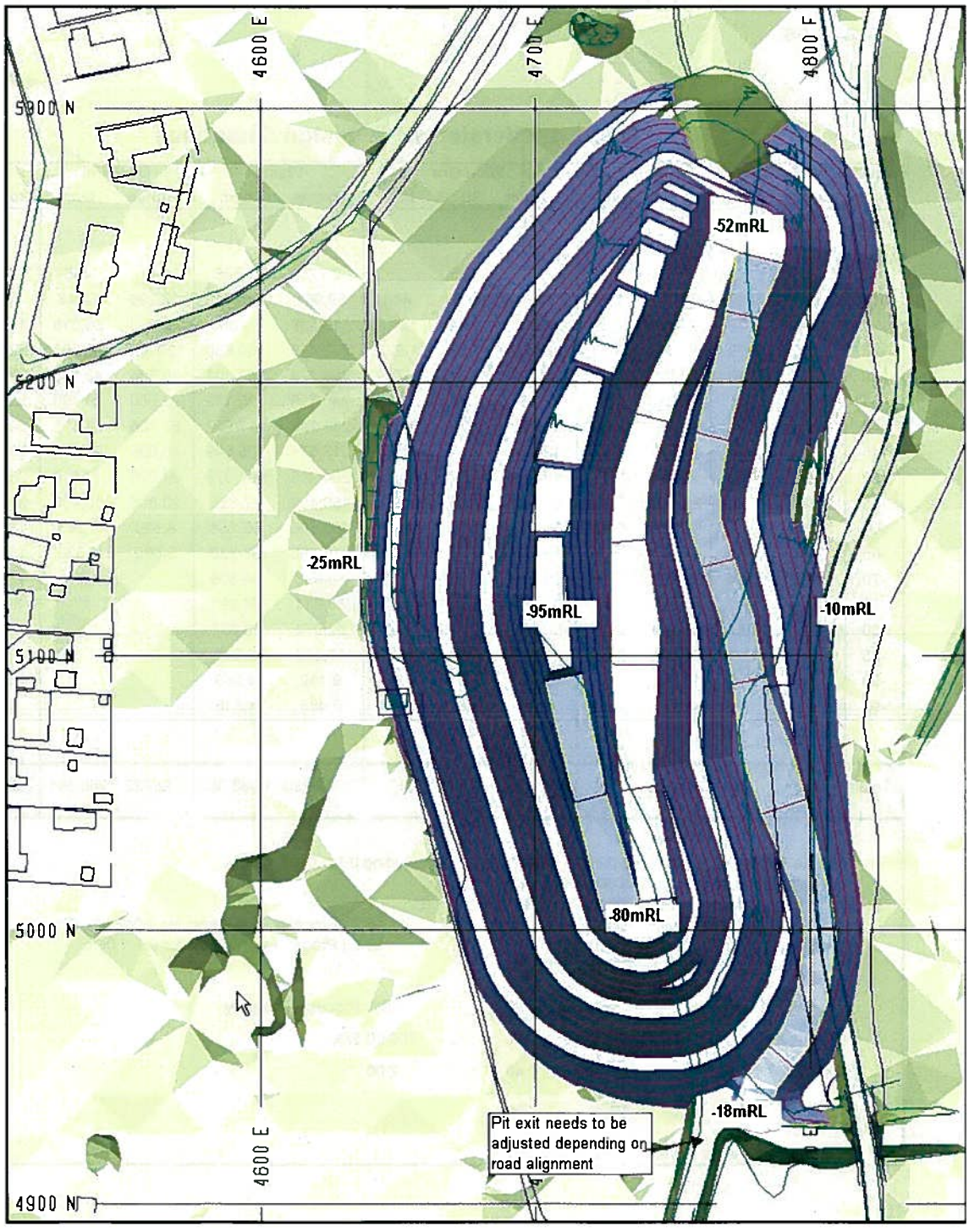
The overall resultant wall slopes in the pit designs are slightly steeper than the Kevin Rosengren & Associates recommendations (August 1999, Geotehcnical Review) but flatter than the existing Davis pit walls. This approach considered reasonable considering the shallower depth of the pit design compared to the 1999 study and condition of the existing pit walls.

The two pit design versions with different ramp systems have been discussed below.

4.2 Version 1 Pit Design and Quantities

The Version 1 pit design (Figure 4.1) is based on a switch back ramp system on the eastern wall located away from the town boundary. The ramp system is not optimal in minimising the waste mining due to (1) the direction of the access (from south) to the final pit floor and (2) the presence of the two switchbacks with flat surfaces.

Figure 4.1 Version 1 Pit Design



The Version 1 design maintains a 90m minimal distance to the town boundary. The pit exit is located at the southeast for minimal ore and waste haul distance at the surface. The operation of the trucks entering and exiting the pit would also be away from the town boundary on the eastern wall in this design version.

The mining quantities reported within the Version 1 pit design are listed in Table 4.1 by levels. The operating financials reported in the table show that the operating margin for the pit design is quite low due to the high operating cost per ounce of $680/oz. This is $50/oz more than the $630/oz cost indicated in the pit optimisation, which is a result of the more waste mined for the access ramp than allowed in the pit optimisation slope angles.

Table 4.1 Version 1 Pit Design Quantities

Level	0.7-1.0g/t		> 1.0g/t Au		Total Ore			Waste		Back Fill		Total	
mRL	tonnes	Au g/t	tonnes	Au g/t	tonnes	Au g/t	bcm	tonnes	bcm	tonnes	bcm	tonnes	bcm
Toe													
-10								2,775	1,388	810	405	3,585	1,793
-15	68	0.87	810	1.19	878	1.17	440	59,961	29,980	13,296	6,648	74,135	37,068
-20	851	0.79	2,665	1.09	3,515	1.02	1,750	118,598	59,090	70,030	35,015	192,143	95,855
-25	1,197	0.79	2,655	1.21	3,852	1.08	1,927	202,459	100,980	120,006	60,003	326,317	162,910
-30	3,375	0.78	8,104	1.20	11,479	1.08	5,739	286,571	143,101	86,366	43,183	384,416	192,023
-35	2,577	0.75	9,589	1.39	12,166	1.26	6,066	240,718	120,179	72,980	36,490	325,864	162,735
-40	2,056	0.77	9,883	2.00	11,938	1.79	5,880	236,223	117,957	57,186	28,593	305,347	152,430
-45	3,928	0.84	14,176	2.09	18,105	1.82	8,903	217,575	108,549	46,036	23,018	281,716	140,470
-50	1,392	0.85	17,714	2.05	19,106	1.97	9,315	204,986	101,973	34,020	17,010	258,112	128,298
-55	2,882	0.84	26,470	2.29	29,352	2.15	14,242	156,460	77,635	20,806	10,403	206,618	102,280
-60	1,809	0.78	36,343	2.12	38,152	2.05	18,483	133,606	66,264	8,526	4,263	180,284	89,010
-65	1,460	0.80	37,301	2.23	38,761	2.17	18,670	113,390	56,380	2,660	1,330	154,811	76,380
-70	989	0.84	38,114	2.25	39,103	2.22	18,857	89,953	44,806			129,056	63,663
-75	1,322	0.82	35,996	2.21	37,318	2.16	18,019	50,014	24,891			87,332	42,910
-80	26	0.87	30,824	2.21	30,850	2.21	14,877	32,974	16,371			63,824	31,248
-85			21,518	2.58	21,518	2.58	10,196	17,360	8,247			38,878	18,443
-90			10,764	3.87	10,764	3.87	4,813	9,112	4,060			19,875	8,873
-95			4,565	2.88	4,565	2.88	1,984	3,485	1,516			8,050	3,500
Total	23,931	0.80	307,490	2.21	331,421	2.11	160,163	2,176,220	1,083,365	532,722	266,361	3,040,363	1,509,889

Waste & LG Mining Cost:	$'000	5,807	Including 0.7-1.0g/t
Backfill Mining Cost:	$'000	889	
Ore (>1.0g/t) Mining Cost:	$'000	1,904	
Ore (>1.0g/t) Processing C	$'000	4,920	16 $/t Process + full admin for 1.0g/t cut-off
Marginal Ore Proces. Cost	$'000	287	12 $/t Proces + part admin for 0.7-1.0g/t
Total Operating Cost	$'000	13,807	
Recovered Ounces	oz	20,250	90% Process Recovery
Revenue	$'000	14,175	700.00 $/oz
Royalty	$'000	40	2.00
Undiscounted Op. Value:	$'000	328	
Operating Cost per Ounce	$/oz	682	

4.3 Version 2 Pit Design and Quantities

The access ramp system is optimal in Version 2 pit design (Figure 4.2) where it is positioned at the lower levels considering the shape of the mineralisation and the optimal pit shell. Development of a temporary ramp switching back to the north and backfilling of the southern end of the pit design will be required for the continuation of the ramp system to the lower levels.

Figure 4.2 Version 2 Pit Design

The distance of the Revision 2 pit design crest is approximately 80m to the town boundary. The operation of the trucks will be relatively close to the town boundary as the ramp exit is located at the northwest. The haul distances to the waste dumps and plant site are also longer compared to the Revision 1 design.

The mining quantities reported within the Version 2 pit design are listed in Table 4.2 by levels. The operating financials calculated in the table shows that the unit cost of $640/oz is in line with the cost of $630/oz reported in the pit optimisations. The pit design is expected to generate $1.1M operating profit based on the cost estimates and $700/oz gold price used in this study.

Table 4.2 Version 2 Pit Design Quantities

Level	0.7-1.0g/t		> 1.0g/t Au		Total Ore			Waste		Back Fill		Total	
mRL	tonnes	Au g/t	tonnes	Au g/t	tonnes	Au g/t	bcm	tonnes	bcm	tonnes	bcm	tonnes	bcm
Toe													
-10								1,940	970	360	180	2,300	1,150
-15			799	1.19	799	1.19	400	27,465	13,732	11,976	5,988	40,240	20,120
-20	798	0.79	3,546	1.35	4,344	1.24	2,141	68,029	33,806	68,746	34,373	141,119	70,320
-25	1,045	0.80	2,640	1.23	3,685	1.11	1,841	152,017	75,764	116,866	58,433	272,568	136,038
-30	1,745	0.79	7,519	1.23	9,264	1.15	4,631	259,741	129,689	83,046	41,523	352,051	175,843
-35	2,054	0.75	9,082	1.41	11,137	1.29	5,552	224,666	112,161	67,200	33,600	303,003	151,313
-40	2,043	0.77	9,839	2.00	11,882	1.79	5,852	219,241	109,483	52,736	26,368	283,859	141,703
-45	3,798	0.84	13,990	2.10	17,788	1.83	8,747	195,955	97,753	45,510	22,755	259,253	129,255
-50	1,392	0.85	17,179	2.07	18,571	1.98	9,052	179,604	89,293	37,740	18,870	235,915	117,215
-55	1,471	0.84	25,386	2.29	26,858	2.21	13,010	140,306	69,597	22,596	11,298	189,760	93,905
-60	1,568	0.78	35,802	2.17	37,369	2.11	18,088	119,102	59,059	9,466	4,733	165,937	81,880
-65	849	0.75	38,974	2.24	39,823	2.20	19,207	99,307	49,351	3,870	1,935	143,000	70,493
-70	1,340	0.85	40,396	2.22	41,735	2.17	20,156	77,691	38,659			119,426	58,815
-75	266	0.90	34,239	2.24	34,506	2.23	16,658	41,697	20,727			76,203	37,385
-80	1,144	0.83	30,077	2.12	31,221	2.07	15,107	24,827	12,318			56,048	27,425
-85	47	0.80	20,256	2.55	20,303	2.55	9,632	14,254	6,833			34,557	16,465
-90			10,642	3.69	10,642	3.69	4,759	11,072	4,971			21,714	9,730
-95			5,456	2.67	5,456	2.67	2,372	7,436	3,233			12,892	5,605
Total	19,560	0.81	305,822	2.20	325,383	2.12	157,206	1,864,350	927,398	520,112	260,056	2,709,845	1,344,660

Waste & LG Mining Cost:	$'000	4,968	Including 0.7-1.0g/t	
Backfill Mining Cost:	$'000	869		
Ore (>1.0g/t) Mining Cost:	$'000	1,849		
Ore (>1.0g/t) Processing Cost	$'000	4,893	16	Process + full admin
Marginal Ore Proces. Cost	$'000	235	12	Proces + part admin for 0.7-1.0g/t
Total Operating Cost	$'000	12,813		
Recovered Ounces	oz	19,967	90%	Process Recovery
Revenue	$'000	13,977	700.00	$/oz
Royalty	$'000	40	2.00	
Undiscounted Op. Value:	$'000	1,124		
Operating Cost per Ounce	$/oz	642		

5.0 RESERVE STATEMENT AND LIMITATIONS

SGM has reviewed the pit design versions in terms of unit costs and operational issues and advised that Revision 2 pit design would be appropriate for a reserves statement. Based on a gold price of $700/oz, the reserve estimate for Davis pit south extension is as follows:

- 330,000 tonnes of Probable Reserve at 2.10g/t Au grade.

This reserve estimate does comply with the JORC guidelines. However, the following limitations will need further confirmation for a development decision:

- The Davis pit south extension reserve estimate in this report is based on the operating costs only. Any pre-production costs (such as mining contractor mobilisation and establishment), finance costs and effect of taxation should be taken in to account in the detailed financial analysis. This will be particularly important considering the relatively high operating cost per ounce ($640/oz) for the reserves estimate.
- The marginal cost analysis can be carried out as part of the financial analyses taking into account the other ore sources at Stawell. Depending on the parameters used, a marginal cost analysis is likely to reduce the costs allocated to Davis pit extension if the mill feed in the site production schedule is in short supply.
- MSC's mining cost estimates in this study are supported with the level of SGM cost estimates for the Wild Wood deposit. The mining costs need to be confirmed with the contractor pricing or detailed owner mining cost estimates specific to Davis pit design.
- Although it is expected that the deductions for old underground workings are sufficient in the reserve estimate, it is recommended to confirm the resource estimates in the Davis pit design with the modelling of the underground workings based on drill hole intercepts. Some in-fill drilling may be required as part of the earlier grade control program to confirm the location of the major stope areas and ore zones.

APPENDIX A
Resource Classification Information Memorandum

STAWELL GOLD MINES PTY LTD
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Leviathan RESOURCES

MEMORANDUM

TO: Troy Cole

CC: Mark Mitchell

FROM: Geoff McDermott

DATE: January 5th 2006

SUBJECT: **Comments on Davis Pit extension resource classification**

During December 2005 I asked our Senior Resource Geologist Chris De-Vitry to review and regenerate resources from the most recent Big Hill resource model in lieu of recent improvement in the gold price and potential for a mining opportunity. He provided me with the following information:

Hi Geoff

I have regenerated resources from the Big Hill Model (s:\Mining-Geology\BigHill\Minesight\BH_amg\bhr15.dat) above an RL of -130. The fields utilised are aubk, ore% and sg. These represent the gold grade, volume and SG above a cutoff of 0.8 g/t. Void is a very rough estimate which was visually estimated from known void and drillhole encountered voids. A default SG of 2.5 was utilised for blocks that did not have an assigned SG.

The resources are:

Mariners
850,500 @ 2.7 (74,000 oz or with 7.5% void 68,500 oz)
Magdala Flanks
2,354,600 @ 2.0 (150,000 oz or with 30% void 105,000 oz)
Iron Duke
295,000 @ 1.7 (16,100 oz or with 2.5% void 15,700 oz)
Allens
276,300 @ 1.8 (16,400 oz no void)

Factored by void this is a total resource of 205,500 oz

The previously reported resource was 215,000 ounces of Indicated.

Regards

Chris De-Vitry

I have subsequently reviewed the "Davis" portion of the resource model and the February 1999 documentation of BH1998 Resource Model of Mike Stewart and have formed the following opinions:

1. Resource for the Southern Pit or "Davis Extension Pit" is currently classified as **Indicated Resource.** I concur with this classification for the portion contained from surface to 130mRL bounded to the south by 4900N or mine section 245N, and to the north by 5380N or mine section 269N. Drill hole spacing in this area is spaced appropriately within SGM guidelines to assume grade and geological continuity. The other resources have not been reviewed by me at this stage.
2. Resources quoted in the model are not diluted or take into account ore loss. The resource estimation block size of 5m (East) X 10m (North) X 5m (Vertical) is relatively small compared to drill hole spacing and therefore should not have much of a diluting effect on resource grade. However, some grade "smoothing" in the estimate would have been created during the interpolation process of ordinary kriging with top cut gold values. In my opinion, for reserve estimation, the resource grades need to be diluted on domain boundaries with the application of a dilution perimeter.
3. Historical void in the resource area is not known with absolute certainty. The percentage of void quoted in Chris De-Vitry's note, above, is thought to be appropriate given the information available to date. Note that for the "Davis" area a factor of 30% should be applied for historic void.

Geoff McDermott BSc (Hons) MAIG
Geology Manager
Leviathan Resources

APPENDIX B
Solid and Surface Data Used to Code the Resource Model
Electronic Copy Only

APPENDIX C
SGM Metallurgical Test Results

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Leviathan RESOURCES

MEMORANDUM

TO: Troy Cole

CC: Les Dunn, Mark Mitchell.

FROM: Senior Metallurgist

DATE: 11th January 2006.

SUBJECT: **DAVIS PIT EXTENSION - METALLURGY.**

INTRODUCTION.

With the current high gold prices, alternative ore sources that will supplement current underground production are being sought. One such source is the Davis Pit Extension or 'Pit 4' from the original Big Hill Project. The following is a summary of test work carried out on samples from within the Pit 4 design from December 2003.

RESULTS/DISCUSSION.

Leaching results for the Davis Extension pit, summarised by section, weathering type and depth (from Big Hill reference) are given below, and illustrated graphically on the following page. Full results are shown in the appendices.

Overall, the expected recovery for the extended pit can be expected to be around 90%. The ore will not be amenable to flotation and regrinding due to its weathered nature, with very little ore originally reported as being in the 'fresh' state. Preg rob is present in most samples tested, however is fairly minor and not likely to greatly affect recovery.

SECTION	METRES TESTED	INTERVALS TESTED	GRADE		PREG ROB	REC % O'ALL
			CALC	COMP		
248	28	4	1.56	1.56	0.3	89.9
250	56	8	3.09	3.10	0.5	95.2
253	36	7	1.72	1.60	0.3	86.6
255-256	47	7	3.09	2.69	0.5	89.1
259	22	3	2.86	1.74	0.5	92.9
262	47	6	1.75	1.80	0.6	90.6
264	22	4	2.08	2.61	0.7	95.5
267	35	6	1.84	1.82	0.3	90.1
OVERALL	293	45	2.32	2.20	0.5	91.7

TABLE 1: *Leaching Recovery Results for Davis Extension Pit by Section.*

Recovery across the pit, from the southern to the northern end is reasonably uniform.

WEATHERING	METRES TESTED	INTERVALS TESTED	GRADE		PREG ROB	REC % O'ALL
			CALC	COMP		
EXTREME - HEAVY	50	8	2.77	2.81	0.3	91.1
MODERATE	179	26	2.13	1.93	0.5	92.7
SLIGHT	43	7	2.41	2.40	0.5	87.1
FRESH	21	4	2.66	2.65	0.7	95.2
OVERALL	293	45	2.32	2.20	0.5	91.7

TABLE 2: *Leaching Recovery Results for Davis Extension Pit by Weathering Classification.*

Recovery by weathering classification is relatively consistent. The small amount of ore reported as being fresh may be amenable to flotation, however its standard recovery is very high as is.

DEPTH	METRES TESTED	INTERVALS TESTED	GRADE		PREG ROB	REC % O'ALL
			CALC	COMP		
0 - 20 METRES	15	3	1.51	1.54	0.4	78.0
20 - 30 METRES	11	2	1.23	1.16	0.3	90.5
30 - 40 METRES	42	5	1.84	1.81	0.2	93.7
40 - 50 METRES	39	7	3.00	3.09	0.5	94.7
50 - 60 METRES	60	9	2.19	1.83	0.6	93.6
60 - 70 METRES	81	12	2.73	2.57	0.4	92.0
70 - 90 METRES	45	7	2.15	2.11	0.6	86.5
OVERALL	293	45	2.32	2.20	0.5	91.7

TABLE 3: *Leaching Recovery Results for Davis Extension Pit by Depth.*

Gold recovery by depth is again reasonably consistent, besides some lower recovery at the very top of the pit. This is mostly affected by one intercept that recovered at 59%.

CONCLUSION.

Gold recovery from ore present in the proposed Davis Extension is likely to be in the order of 90%, with recovery relatively consistent across the pit, at depth and by weathering classification.

Peter Wemyss.
Senior Metallurgist.
11/01/2006.

FIGURE 1a: DAVIS EXTENSION PIT.

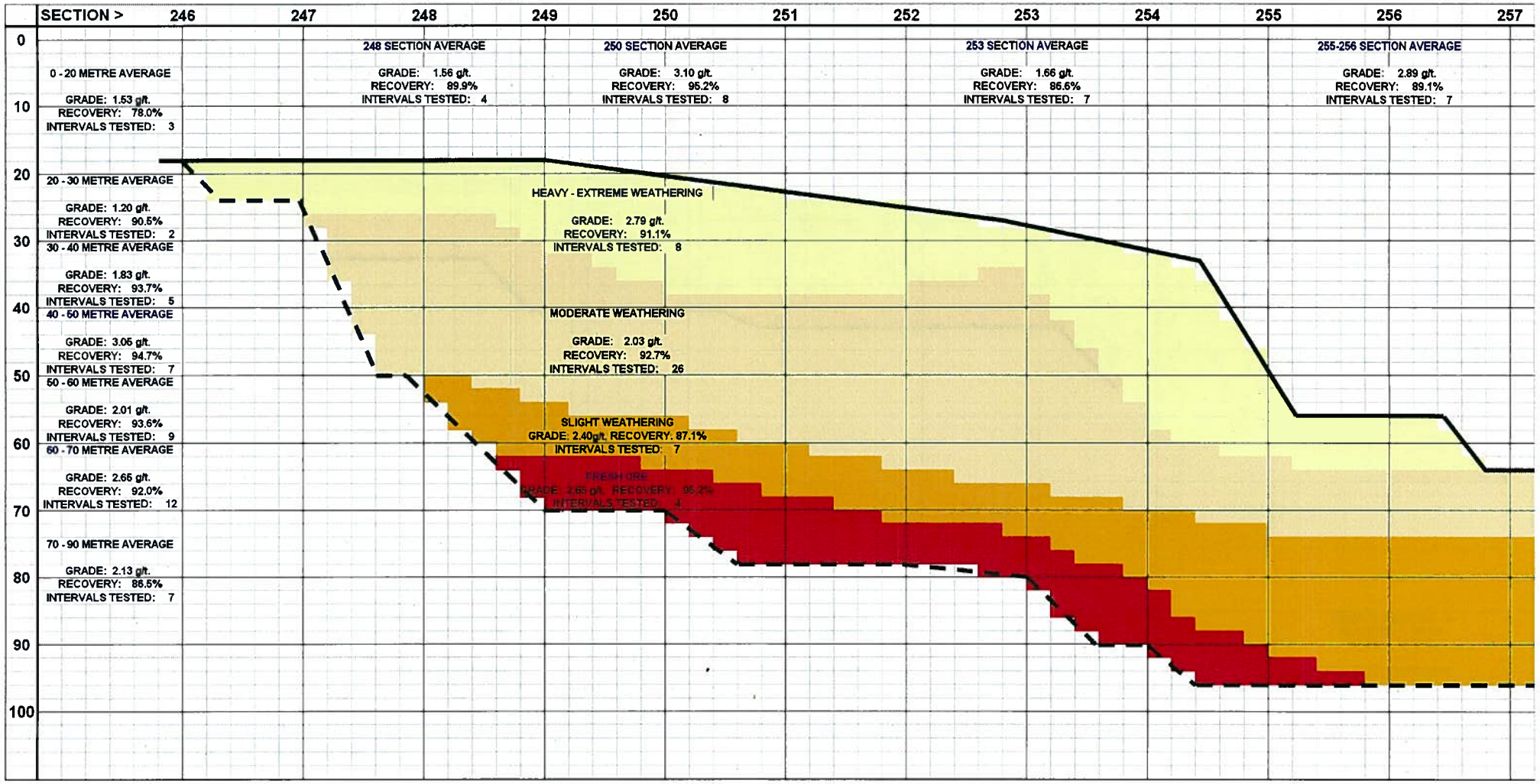
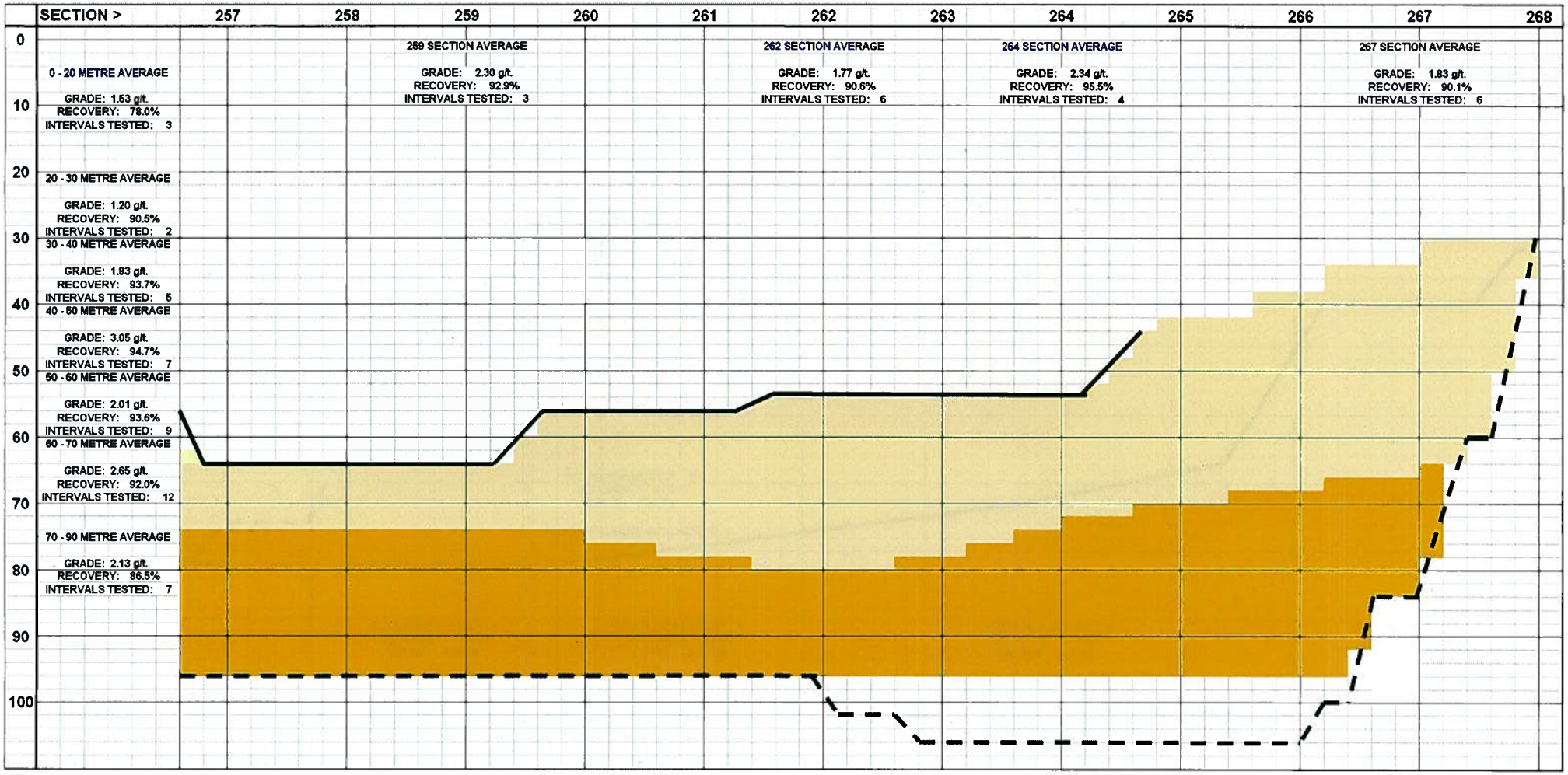


FIGURE 1b: DAVIS EXTENSION PIT.



APPENDIX 1.

Metallurgy results by Section.

		METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
SECTION	HOLE	START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
248	SP311	28	35	7	MW - HW	Volc	32	0.99	1.03	0.3	84.0	
	SP311	35	43	8	MW	Peli	26	1.58	1.57	0.3	94.3	
	SP424	49	54	5	HW	Peli	16	1.79	2.10	0.2	79.0	
	SP424	57	65	8	MW	Quartz/Peli	46	1.89	1.69	0.3	97.3	
	TOTAL			28				1.56	1.56	0.3	89.9	
250	SP210	7	14	7	EW	Peli	31	1.81	1.68	0.1	87.5	
	SP210	30	38	8	MW	Quartz	14	3.06	2.89	0.1	96.7	
	SP417	38	46	8	HW	Peli	54	6.37	6.94	0.8	96.7	
	SP287	42	46	4	MW	Quartz	68	1.83	1.63	0.2	94.8	
	SP417	46	57	11	HW	Quartz	62	2.51	2.50	0.2	94.6	
	SP209	50	59	9	FR	Peli	7	1.88	1.80	0.3	95.8	
	SP208	55	58	3	SW	Peli	3	1.94	1.56	2.9	91.3	48.1
	SP209	64	70	6	FR	Peli	8	4.58	4.73	0.5	95.1	
	TOTAL			56				3.09	3.10	0.5	95.2	
253	SP408	0	3	3	HW	Peli	12	1.54	1.73	1.2	59.4	2.2
	SP413	18	23	5	EW	Peli	27	1.07	1.22	0.2	75.4	
	SP408	34	44	10	MW	Quartz	67	2.15	1.66	0.3	83.1	
	SP413	36	45	9	MW	Quartz	60	1.45	1.40	0.0	96.8	
	SP413	58	62	4	MW	Quartz	74	2.14	1.95	0.6	94.1	
	SP234	61	63	2	SW	Quartz	33	1.67	1.47	0.2	91.2	
	SP234	70	73	3	FR	Peli	6	1.88	2.14	0.0	97.0	
	TOTAL			36				1.72	1.60	0.3	86.6	
255-256	SP309	58	65	7	HW	Quartz	44	3.53	2.99	0.2	93.1	
	SP405	61	66	5	MW	Quartz	57	3.91	3.98	0.4	87.8	
	SP309	65	69	4	MW	Volc	15	8.22	5.03	0.0	95.5	
	SP309	69	77	8	HW - MW	Volc	14	1.16	0.94	0.8	78.7	
	SP328	69	74	5	MW - SW	Clay/Quartz	37	2.37	2.62	0.3	93.1	
	SP328	76	87	11	SW	Volc/Peli	45	2.90	2.66	0.9	82.3	7.7
	SP309	77	84	7	MW	Q/ShZo/CH	43	2.15	2.22	0.4	91.4	
	TOTAL			47				3.09	2.69	0.5	89.1	
259	SP402	47	51	4	MW	Peli/Volc	25	2.69	2.68	0.4	94.0	
	SP402	51	60	9	MW	Quartz	73	3.56	0.76	0.6	74.9	
	SP330	59	68	9	MW	Quartz	73	2.24	2.29	0.5	98.3	
	TOTAL			22				2.86	1.74	0.5	92.9	
262	SP214	33	40	7	MW	Peli	13	0.81	1.59	0.5	99.4	
	SP214	40	45	5	MW	Quartz	67	1.51	1.59	0.4	97.5	
	SP288	49	58	9	MW - SW	Quartz	74	1.62	1.58	0.6	99.1	
	SP288	58	67	9	MW - HW	Quartz	73	2.10	1.78	0.9	97.5	
	SP289	66	78	12	HW	Quartz	60	2.04	1.97	0.6	79.4	
	SP289	80	85	5	SW	Quartz	70	2.18	2.35	0.4	80.6	
	TOTAL			47				1.75	1.80	0.6	90.6	
264	MD1210	45	48	3	MW	ShZo	20	1.70	2.09	2.1	79.2	48.6
	SP378	50	57	7	MW	Quartz	74	1.88	2.40	0.3	95.0	
	SP377	53	59	6	SW	Quartz	66	2.68	3.24	0.6	98.6	
	SP379	64	70	6	MW	Quartz	72	1.91	2.47	0.6	99.0	
	TOTAL			22				2.08	2.61	0.7	95.5	
267	SP394	25	29	4	MW	Peli	32	1.64	1.39	0.4	98.9	
	SP394	41	45	4	MW	Volc	10	1.94	1.48	0.4	91.2	
	SP387	61	70	9	HW	Peli/Quartz	34	0.88	0.93	0.2	82.3	
	SP393	61	66	5	SW	Quartz	91	3.44	3.44	0.4	97.8	
	SP394	63	70	7	MW - SW	Volc	23	1.87	2.15	0.0	83.5	
	SP393	74	80	6	SW	Volc	23	1.99	1.93	0.5	88.1	
	TOTAL			35				1.84	1.82	0.3	90.1	
OVERALL	TOTAL			293				2.32	2.20	0.5	91.7	

APPENDIX 2.

Metallurgy results by Weathering classification.

HEAVY - EXTREME WEATHERING

		METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
SECTION	HOLE	START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
248	SP311	28	35	7	MW - HW	Volc	32	0.99	1.03	0.3	84.0	
248	SP424	49	54	5	HW	Peli	16	1.79	2.10	0.2	79.0	
250	SP210	7	14	7	EW	Peli	31	1.81	1.68	0.1	87.5	
250	SP210	30	38	8	MW	Quartz	14	3.06	2.89	0.1	96.7	
250	SP417	38	46	8	HW	Peli	54	6.37	6.94	0.8	96.7	
253	SP408	0	3	3	HW	Peli	12	1.54	1.73	1.2	59.4	2.2
253	SP413	18	23	5	EW	Peli	27	1.07	1.22	0.2	75.4	
255-256	SP309	58	65	7	HW	Quartz	44	3.53	2.99	0.2	93.1	
	TOTAL			50				2.77	2.81	0.3	91.1	

MODERATE WEATHERING

		METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
SECTION	HOLE	START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
248	SP311	35	43	8	MW	Peli	26	1.58	1.57	0.3	94.3	
248	SP424	57	65	8	MW	Quartz/Peli	46	1.89	1.69	0.3	97.3	
250	SP287	42	46	4	MW	Quartz	68	1.83	1.63	0.2	94.8	
250	SP417	46	57	11	HW	Quartz	62	2.51	2.50	0.2	94.6	
253	SP408	34	44	10	MW	Quartz	67	2.15	1.66	0.3	83.1	
253	SP413	36	45	9	MW	Quartz	60	1.45	1.40	0.0	96.8	
253	SP413	58	62	4	MW	Quartz	74	2.14	1.95	0.6	94.1	
255-256	SP405	61	66	5	MW	Quartz	57	3.91	3.98	0.4	87.8	
255-256	SP309	65	69	4	MW	Volc	15	8.22	5.03	0.0	95.5	
255-256	SP309	69	77	8	HW - MW	Volc	14	1.16	0.94	0.8	78.7	
255-256	SP328	69	74	5	MW - SW	Clay/Quartz	37	2.37	2.62	0.3	93.1	
259	SP402	47	51	4	MW	Peli/Volc	25	2.69	2.68	0.4	94.0	
259	SP402	51	60	9	MW	Quartz	73	3.56	0.76	0.6	74.9	
259	SP330	59	68	9	MW	Quartz	73	2.24	2.29	0.5	98.3	
262	SP214	33	40	7	MW	Peli	13	0.81	1.59	0.5	99.4	
262	SP214	40	45	5	MW	Quartz	67	1.51	1.59	0.4	97.5	
262	SP288	49	58	9	MW - SW	Quartz	74	1.62	1.58	0.6	99.1	
262	SP288	58	67	9	MW - HW	Quartz	73	2.10	1.78	0.9	97.5	
262	SP289	66	78	12	HW	Quartz	60	2.04	1.97	0.6	79.4	
264	MD1210	45	48	3	MW	ShZo	20	1.70	2.09	2.1	79.2	48.6
264	SP378	50	57	7	MW	Quartz	74	1.88	2.40	0.3	95.0	
264	SP377	53	59	6	SW	Quartz	66	2.68	3.24	0.6	98.6	
264	SP379	64	70	6	MW	Quartz	72	1.91	2.47	0.6	99.0	
267	SP394	25	29	4	MW	Peli	32	1.64	1.39	0.4	98.9	
267	SP394	41	45	4	MW	Volc	10	1.94	1.48	0.4	91.2	
267	SP387	61	70	9	HW	Peli/Quartz	34	0.88	0.93	0.2	82.3	
	TOTAL			179				2.13	1.93	0.5	92.7	

SLIGHT WEATHERING

		METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
SECTION	HOLE	START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
253	SP234	61	63	2	SW	Quartz	33	1.67	1.47	0.2	91.2	
255-256	SP328	76	87	11	SW	Volc/Peli	45	2.90	2.66	0.9	82.3	7.7
255-256	SP309	77	84	7	MW	Q/ShZo/CH	43	2.15	2.22	0.4	91.4	
262	SP289	80	85	5	SW	Quartz	70	2.18	2.35	0.4	80.6	
267	SP393	61	66	5	SW	Quartz	91	3.44	3.44	0.4	97.8	
267	SP394	63	70	7	MW - SW	Volc	23	1.87	2.15	0.0	83.5	
267	SP393	74	80	6	SW	Volc	23	1.99	1.93	0.5	88.1	
	TOTAL			43				2.41	2.40	0.5	87.1	

FRESH

		METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
SECTION	HOLE	START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
250	SP209	50	59	9	FR	Peli	7	1.88	1.80	0.3	95.8	
250	SP208	55	58	3	SW	Peli	3	1.94	1.56	2.9	91.3	48.1
250	SP209	64	70	6	FR	Peli	8	4.58	4.73	0.5	95.1	
253	SP234	70	73	3	FR	Peli	6	1.88	2.14	0.0	97.0	
	TOTAL			21				2.66	2.65	0.7	95.2	

APPENDIX 3.

Metallurgy Results by Depth.

0 - 20 Metres

SECTION	HOLE	METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
		START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
253	SP408	0	3	3	HW	Peli	12	1.54	1.73	1.2	59.4	2.2
250	SP210	7	14	7	EW	Peli	31	1.81	1.68	0.1	87.5	
253	SP413	18	23	5	EW	Peli	27	1.07	1.22	0.2	75.4	
	TOTAL			15				1.51	1.54	0.4	78.0	

20 - 30 Metres

SECTION	HOLE	METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
		START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
267	SP394	25	29	4	MW	Peli	32	1.64	1.39	0.4	98.9	
248	SP311	28	35	7	MW - HW	Volc	32	0.99	1.03	0.3	84.0	
	TOTAL			11				1.23	1.16	0.3	90.5	

30 - 40 Metres

SECTION	HOLE	METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
		START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
250	SP210	30	38	8	MW	Quartz	14	3.06	2.89	0.1	96.7	
262	SP214	33	40	7	MW	Peli	13	0.81	1.59	0.5	99.4	
253	SP408	34	44	10	MW	Quartz	67	2.15	1.66	0.3	83.1	
248	SP311	35	43	8	MW	Peli	26	1.58	1.57	0.3	94.3	
253	SP413	36	45	9	MW	Quartz	60	1.45	1.40	0.0	96.8	
	TOTAL			42				1.84	1.81	0.2	93.7	

40 - 50 Metres

SECTION	HOLE	METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
		START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
250	SP417	38	46	8	HW	Peli	54	6.37	6.94	0.8	96.7	
262	SP214	40	45	5	MW	Quartz	67	1.51	1.59	0.4	97.5	
267	SP394	41	45	4	MW	Volc	10	1.94	1.48	0.4	91.2	
250	SP287	42	46	4	MW	Quartz	68	1.83	1.63	0.2	94.8	
264	MD1210	45	48	3	MW	ShZo	20	1.70	2.09	2.1	79.2	48.6
250	SP417	46	57	11	HW	Quartz	62	2.51	2.50	0.2	94.6	
259	SP402	47	51	4	MW	Peli/Volc	25	2.69	2.68	0.4	94.0	
	TOTAL			39				3.00	3.09	0.5	94.7	

50 - 60 Metres

SECTION	HOLE	METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
		START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
248	SP424	49	54	5	HW	Peli	16	1.79	2.10	0.2	79.0	
262	SP288	49	58	9	MW - SW	Quartz	74	1.62	1.58	0.6	99.1	
250	SP209	50	59	9	FR	Peli	7	1.88	1.80	0.3	95.8	
264	SP378	50	57	7	MW	Quartz	74	1.88	2.40	0.3	95.0	
259	SP402	51	60	9	MW	Quartz	73	3.56	0.76	0.6	74.9	
264	SP377	53	59	6	SW	Quartz	66	2.68	3.24	0.6	98.6	
250	SP208	55	58	3	SW	Peli	3	1.94	1.56	2.9	91.3	48.1
248	SP424	57	65	8	MW	Quartz/Peli	46	1.89	1.69	0.3	97.3	
253	SP413	58	62	4	MW	Quartz	74	2.14	1.95	0.6	94.1	
	TOTAL			60				2.19	1.83	0.6	93.6	

60 - 70 Metres

SECTION	HOLE	METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
		START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
255-256	SP309	58	65	7	HW	Quartz	44	3.53	2.99	0.2	93.1	
262	SP288	58	67	9	MW - HW	Quartz	73	2.10	1.78	0.9	97.5	
259	SP330	59	68	9	MW	Quartz	73	2.24	2.29	0.5	98.3	
253	SP234	61	63	2	SW	Quartz	33	1.67	1.47	0.2	91.2	
255-256	SP405	61	66	5	MW	Quartz	57	3.91	3.98	0.4	87.8	
267	SP387	61	70	9	HW	Peli/Quartz	34	0.88	0.93	0.2	82.3	
267	SP393	61	66	5	SW	Quartz	91	3.44	3.44	0.4	97.8	
267	SP394	63	70	7	MW - SW	Volc	23	1.87	2.15	0.0	83.5	
250	SP209	64	70	6	FR	Peli	8	4.58	4.73	0.5	95.1	
264	SP379	64	70	6	MW	Quartz	72	1.91	2.47	0.6	99.0	
255-256	SP309	65	69	4	MW	Volc	15	8.22	5.03	0.0	95.5	
262	SP289	66	78	12	HW	Quartz	60	2.04	1.97	0.6	79.4	
	TOTAL			81				2.73	2.57	0.4	92.0	

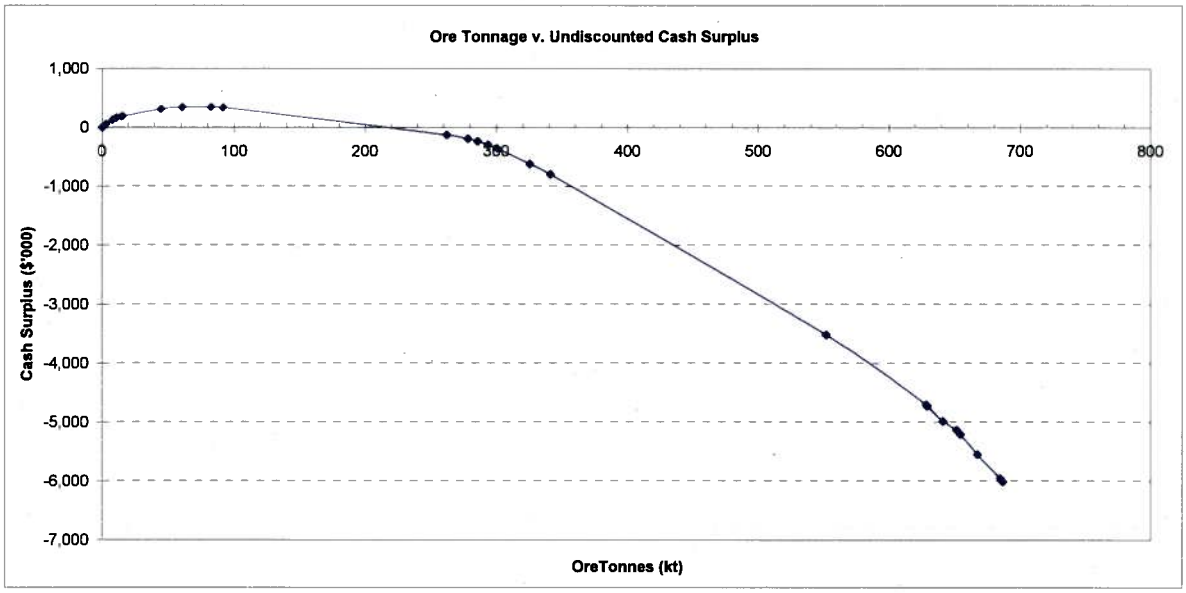
70 - 90 Metres

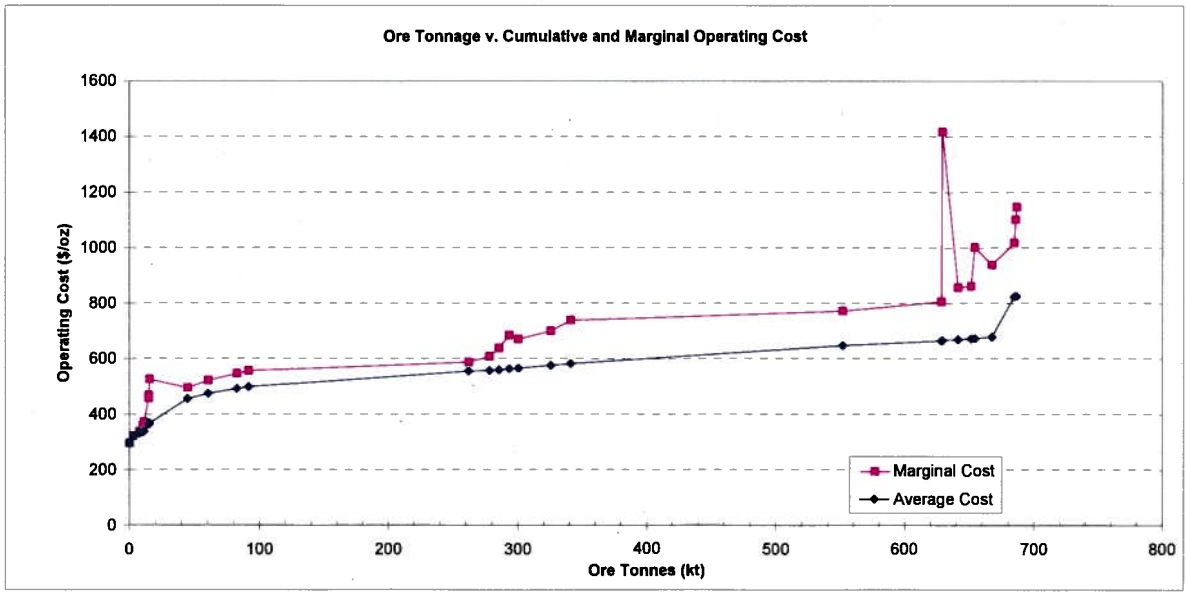
SECTION	HOLE	METRES			PHYSICAL			GRADE		PREG	RECOVERY %	
		START	FINISH	INT	WEATHERING	ROCK	% QUARTZ	CALC	COMP	ROB	O'ALL	AMALG
255-256	SP309	69	77	8	HW - MW	Volc	14	1.16	0.94	0.8	78.7	
255-256	SP328	69	74	5	MW - SW	Clay/Quartz	37	2.37	2.62	0.3	93.1	
253	SP234	70	73	3	FR	Peli	6	1.88	2.14	0.0	97.0	
267	SP393	74	80	6	SW	Volc	23	1.99	1.93	0.5	88.1	
255-256	SP328	76	87	11	SW	Volc/Peli	45	2.90	2.66	0.9	82.3	7.7
255-256	SP309	77	84	7	MW	Q/ShZo/CH	43	2.15	2.22	0.4	91.4	
262	SP289	80	85	5	SW	Quartz	70	2.18	2.35	0.4	80.6	
	TOTAL			45				2.15	2.11	0.6	86.5	

APPENDIX D
Pit Optimisation Result Details

Davis Pit Extension Optimisation Results - Run01a1
$550/oz, Gold Price, 80% Mining Costs

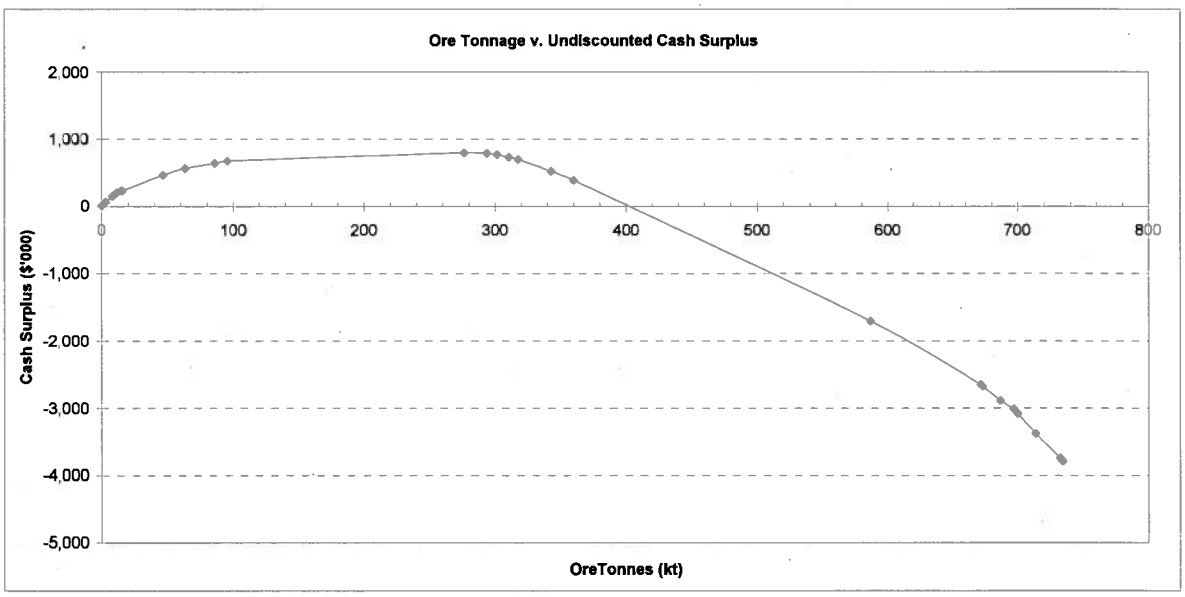
	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	5	3	19.19	2.03	295	295
2	3	2.47	3	6	0.97	246	221	90%	11	59	121	51	19.23	1.83	319	321
3	8	2.35	5	13	0.64	606	545	90%	25	154	299	119	19.23	1.93	330	337
4	11	2.30	6	17	0.61	778	700	90%	33	202	384	149	19.23	1.95	336	358
5	11	2.32	9	20	0.80	846	762	90%	39	219	417	159	19.23	1.93	339	370
6	15	2.22	16	31	1.06	1,064	957	90%	60	287	525	177	19.23	1.96	363	457
7	15	2.21	16	32	1.08	1,081	973	90%	62	293	533	179	19.23	1.96	364	467
8	16	2.19	16	32	1.05	1,100	990	90%	63	301	543	179	19.23	1.96	367	525
9	45	2.50	269	313	6.00	3,600	3,240	90%	616	861	1,775	298	19.23	1.97	456	495
10	61	2.56	429	490	7.07	5,004	4,503	90%	970	1,167	2,468	331	19.23	1.98	475	522
11	83	2.51	639	721	7.74	6,650	5,985	90%	1,359	1,587	3,280	334	19.23	1.88	492	546
12	92	2.53	747	838	8.16	7,454	6,709	90%	1,590	1,759	3,677	328	19.23	1.90	499	557
13	262	2.39	2,300	2,563	8.76	20,209	18,188	90%	5,055	5,043	9,968	-130	19.21	1.97	555	588
14	278	2.39	2,433	2,712	8.74	21,366	19,230	90%	5,382	5,350	10,539	-193	19.21	1.98	558	608
15	286	2.39	2,513	2,798	8.79	21,928	19,735	90%	5,563	5,491	10,816	-239	19.21	1.99	560	639
16	294	2.37	2,575	2,868	8.77	22,418	20,176	90%	5,713	5,643	11,057	-299	19.21	1.99	563	685
17	301	2.38	2,659	2,959	8.84	22,969	20,672	90%	5,913	5,776	11,329	-360	19.21	2.00	565	671
18	326	2.38	2,966	3,292	9.11	24,893	22,404	90%	6,645	6,255	12,278	-621	19.21	2.02	576	699
19	341	2.37	3,131	3,472	9.18	25,941	23,347	90%	7,042	6,553	12,795	-800	19.21	2.03	582	738
20	552	2.22	5,297	5,849	9.60	39,484	35,535	90%	12,401	10,598	19,475	-3,525	19.20	2.12	647	772
22	629	2.21	6,179	6,808	9.83	44,579	40,121	90%	14,625	12,071	21,988	-4,708	19.20	2.15	665	806
23	629	2.20	6,194	6,824	9.84	44,616	40,154	90%	14,659	12,083	22,006	-4,736	19.20	2.15	666	1416
24	642	2.21	6,393	7,034	9.96	45,532	40,979	90%	15,134	12,315	22,458	-4,991	19.20	2.15	670	857
25	652	2.20	6,468	7,120	9.93	46,053	41,447	90%	15,344	12,509	22,715	-5,138	19.20	2.16	672	862
26	655	2.20	6,510	7,164	9.95	46,233	41,610	90%	15,451	12,565	22,804	-5,212	19.20	2.16	673	1001
27	668	2.20	6,737	7,404	10.09	47,215	42,494	90%	16,029	12,815	23,288	-5,556	19.20	2.16	679	938
28	685	2.19	6,950	7,635	10.15	48,186	43,366	90%	16,585	13,147	29,751	-5,965	19.20	2.17	824	1017
29	686	2.19	6,970	7,656	10.16	48,258	43,432	90%	16,635	13,170	29,824	-6,002	19.20	2.17	825	1101
30	687	2.19	6,980	7,666	10.16	48,290	43,461	90%	16,656	13,181	29,856	-6,019	19.20	2.17	825	1147

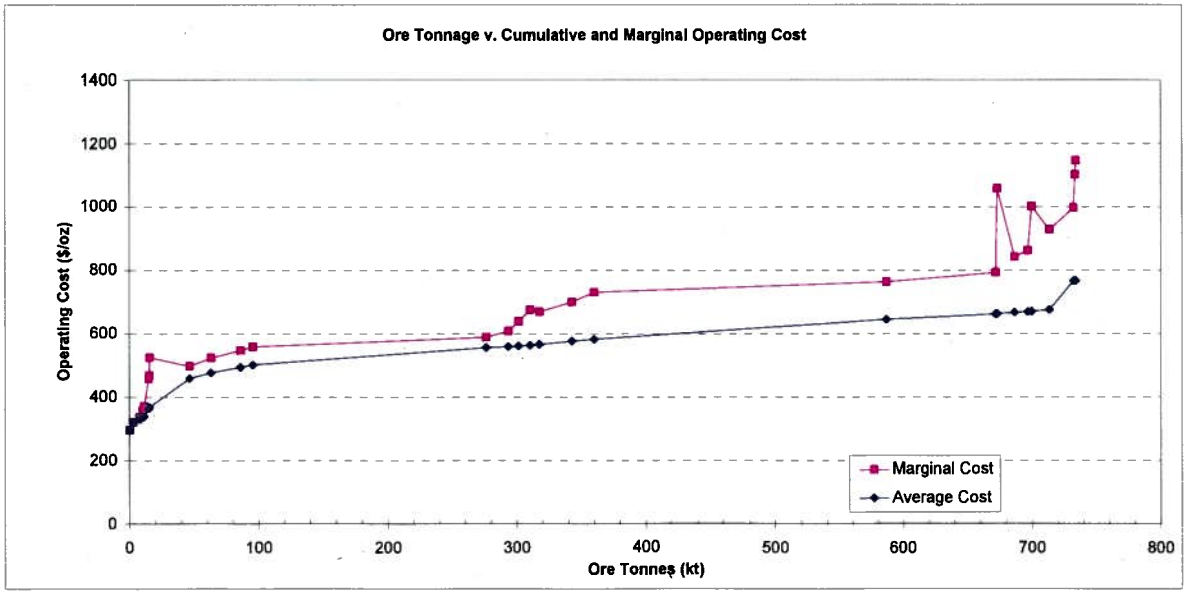




Davis Pit Extension Optimisation Results - Run01b1
$600/oz, Gold Price, 80% Mining Costs

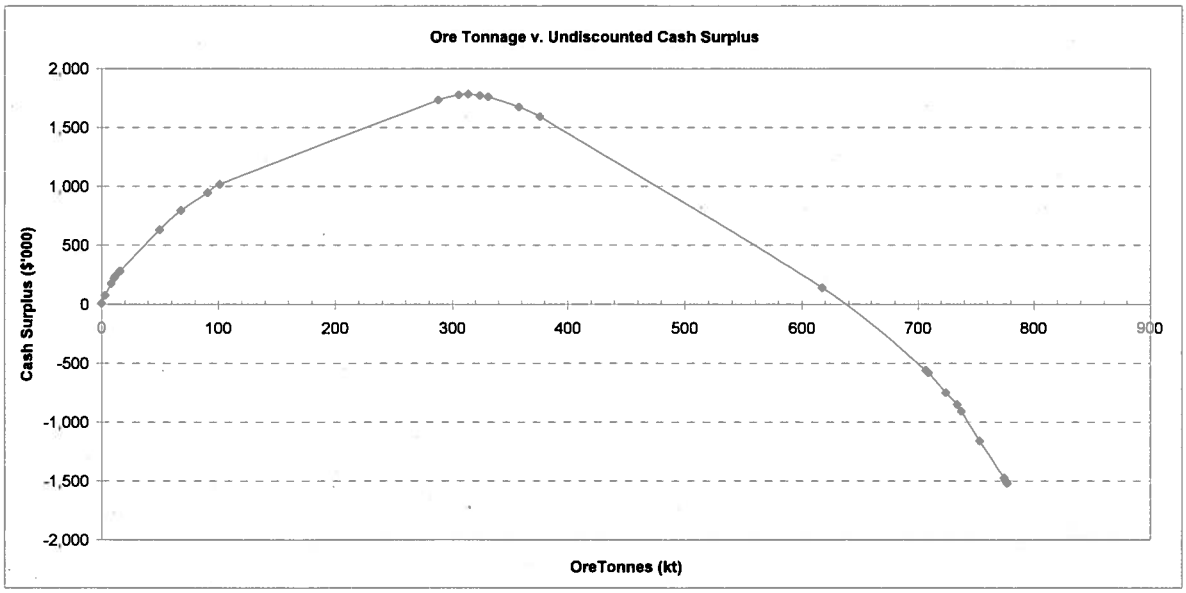
	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	6	3	19.19	2.03	295	295
2	3	2.47	3	6	0.97	246	221	90%	11	59	132	62	19.23	1.83	319	321
3	8	2.35	5	13	0.64	606	545	90%	25	154	326	146	19.23	1.93	330	337
4	11	2.30	6	17	0.61	779	701	90%	33	202	419	184	19.23	1.95	336	358
5	11	2.31	9	20	0.79	847	762	90%	39	219	456	198	19.23	1.93	339	370
6	15	2.21	16	31	1.05	1,067	960	90%	60	289	574	225	19.23	1.96	363	458
7	15	2.20	16	32	1.07	1,084	976	90%	62	294	584	227	19.23	1.96	365	467
8	16	2.18	16	32	1.04	1,103	993	90%	63	302	594	229	19.23	1.96	368	525
9	47	2.44	267	313	5.71	3,672	3,304	90%	616	899	1,976	462	19.23	1.97	458	497
10	63	2.51	426	490	6.73	5,101	4,591	90%	970	1,217	2,746	558	19.23	1.98	477	523
11	86	2.45	635	721	7.38	6,781	6,103	90%	1,359	1,655	3,651	636	19.23	1.88	494	546
12	96	2.47	743	838	7.75	7,613	6,852	90%	1,590	1,841	4,098	667	19.23	1.90	501	557
13	276	2.33	2,286	2,563	8.27	20,728	18,656	90%	5,055	5,310	11,158	793	19.21	1.97	556	587
14	294	2.32	2,418	2,712	8.24	21,933	19,740	90%	5,382	5,641	11,807	784	19.21	1.98	558	607
15	301	2.32	2,497	2,798	8.28	22,513	20,262	90%	5,563	5,792	12,119	764	19.21	1.99	560	637
16	311	2.31	2,558	2,868	8.24	23,046	20,741	90%	5,713	5,965	12,406	727	19.21	1.99	563	675
17	318	2.31	2,642	2,959	8.31	23,609	21,248	90%	5,913	6,104	12,709	691	19.21	2.00	566	669
18	343	2.32	2,949	3,292	8.60	25,538	22,984	90%	6,645	6,586	13,747	516	19.21	2.02	576	699
19	360	2.30	3,112	3,472	8.65	26,643	23,978	90%	7,042	6,913	14,342	386	19.21	2.03	582	729
20	587	2.16	5,262	5,849	8.97	40,784	36,706	90%	12,401	11,265	21,955	-1,711	19.20	2.12	645	763
22	672	2.14	6,136	6,808	9.13	46,200	41,580	90%	14,625	12,902	24,870	-2,657	19.20	2.15	662	792
23	673	2.14	6,150	6,824	9.13	46,261	41,635	90%	14,659	12,926	24,903	-2,683	19.20	2.15	663	1057
24	687	2.14	6,347	7,034	9.24	47,233	42,509	90%	15,134	13,186	25,426	-2,895	19.20	2.15	666	841
25	697	2.13	6,423	7,120	9.21	47,753	42,977	90%	15,344	13,380	25,706	-3,018	19.20	2.16	668	862
26	700	2.13	6,464	7,164	9.24	47,933	43,140	90%	15,451	13,435	25,803	-3,084	19.20	2.16	670	1001
27	714	2.13	6,691	7,404	9.37	48,943	44,049	90%	16,029	13,700	26,347	-3,383	19.20	2.16	675	927
28	732	2.12	6,903	7,635	9.43	49,962	44,965	90%	16,585	14,056	30,661	-3,747	19.20	2.17	765	996
29	733	2.12	6,923	7,656	9.44	50,035	45,031	90%	16,635	14,079	30,733	-3,780	19.20	2.17	766	1101
30	734	2.12	6,932	7,666	9.44	50,066	45,059	90%	16,656	14,091	30,766	-3,796	19.20	2.17	767	1146

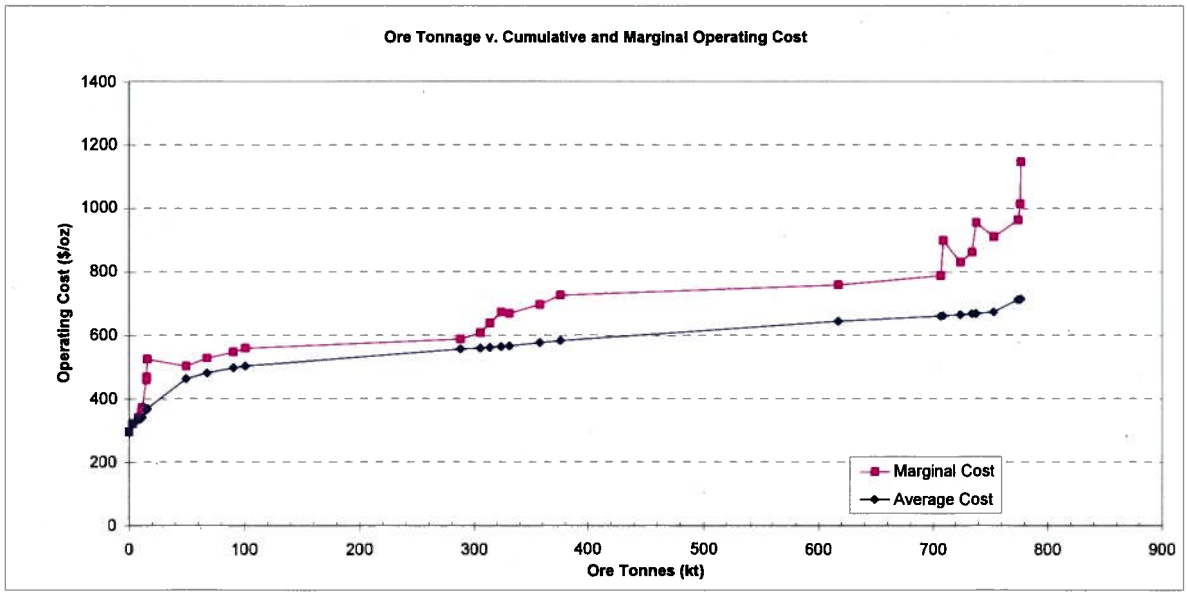




Davis Pit Extension Optimisation Results - Run01c1
$650/oz, Gold Price, 80% Mining Costs

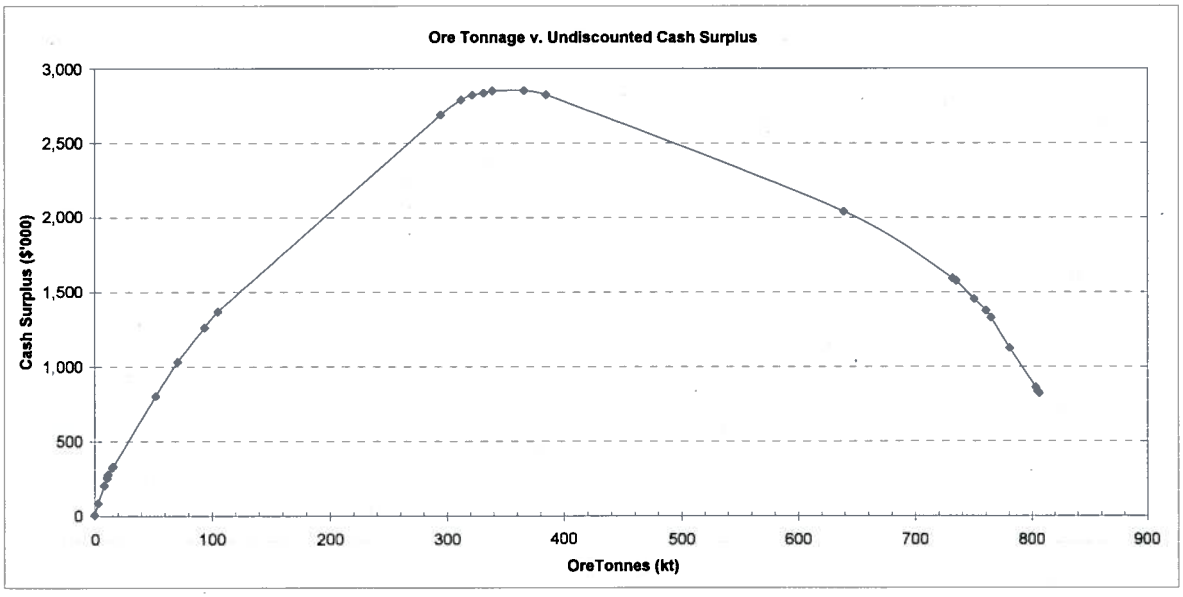
	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	6	4	19.19	2.03	295	295
2	3	2.47	3	6	0.97	246	221	90%	11	59	143	73	19.23	1.83	319	321
3	8	2.33	5	13	0.61	610	549	90%	25	157	356	173	19.23	1.93	332	340
4	11	2.28	6	17	0.59	783	705	90%	33	205	456	219	19.23	1.95	338	358
5	12	2.30	9	20	0.77	851	766	90%	39	221	496	236	19.23	1.93	340	370
6	15	2.20	16	31	1.03	1,072	965	90%	60	292	625	273	19.23	1.96	365	458
7	15	2.19	16	32	1.04	1,089	980	90%	62	297	635	276	19.23	1.96	366	467
8	16	2.17	16	32	1.02	1,108	997	90%	63	305	646	278	19.23	1.96	369	525
9	50	2.36	264	313	5.30	3,777	3,399	90%	616	957	2,202	629	19.23	1.97	463	502
10	68	2.41	422	490	6.20	5,263	4,737	90%	970	1,307	3,069	791	19.23	1.98	481	527
11	91	2.38	630	721	6.94	6,944	6,249	90%	1,359	1,745	4,049	945	19.23	1.88	497	546
12	101	2.40	737	838	7.30	7,796	7,017	90%	1,590	1,943	4,546	1,013	19.23	1.90	503	559
13	288	2.28	2,275	2,563	7.91	21,116	19,004	90%	5,055	5,527	12,313	1,731	19.21	1.97	557	588
14	305	2.28	2,406	2,712	7.88	22,332	20,099	90%	5,382	5,864	13,022	1,776	19.21	1.98	560	607
15	314	2.27	2,485	2,798	7.92	22,930	20,637	90%	5,563	6,026	13,370	1,781	19.21	1.99	562	638
16	324	2.26	2,545	2,868	7.86	23,497	21,148	90%	5,713	6,219	13,701	1,769	19.21	1.99	564	672
17	331	2.26	2,628	2,959	7.94	24,062	21,656	90%	5,913	6,359	14,030	1,758	19.21	2.00	567	669
18	358	2.27	2,934	3,292	8.21	26,043	23,438	90%	6,645	6,870	15,185	1,671	19.21	2.02	577	697
19	376	2.25	3,096	3,472	8.24	27,186	24,468	90%	7,042	7,219	15,852	1,590	19.21	2.03	583	726
20	617	2.11	5,232	5,849	8.48	41,827	37,645	90%	12,401	11,851	24,389	137	19.20	2.12	644	758
22	707	2.09	6,101	6,808	8.63	47,382	42,643	90%	14,625	13,566	27,628	-563	19.20	2.15	661	788
23	709	2.08	6,115	6,824	8.63	47,479	42,730	90%	14,659	13,610	27,684	-585	19.20	2.15	662	899
24	724	2.08	6,310	7,034	8.72	48,497	43,648	90%	15,134	13,898	28,279	-753	19.20	2.15	665	831
25	734	2.08	6,386	7,120	8.70	49,017	44,116	90%	15,344	14,092	28,582	-853	19.20	2.16	667	862
26	738	2.08	6,427	7,164	8.71	49,225	44,302	90%	15,451	14,161	28,703	-910	19.20	2.16	668	956
27	753	2.08	6,651	7,404	8.83	50,291	45,263	90%	16,029	14,458	29,325	-1,162	19.20	2.16	674	910
28	774	2.06	6,860	7,635	8.86	51,404	46,263	90%	16,585	14,867	31,471	-1,479	19.20	2.17	712	964
29	776	2.06	6,880	7,656	8.86	51,494	46,345	90%	16,635	14,900	31,554	-1,509	19.20	2.17	713	1014
30	777	2.06	6,890	7,666	8.87	51,526	46,373	90%	16,656	14,911	31,587	-1,523	19.20	2.17	714	1147

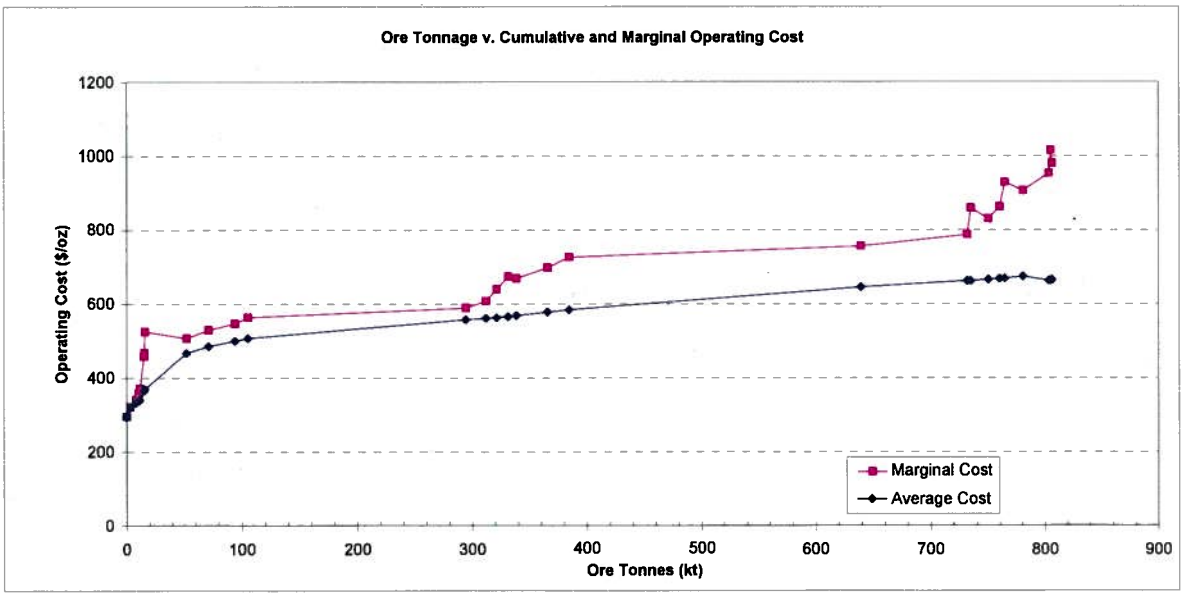




Davis Pit Extension Optimisation Results - Run01d1
$700/oz, Gold Price, 80% Mining Costs

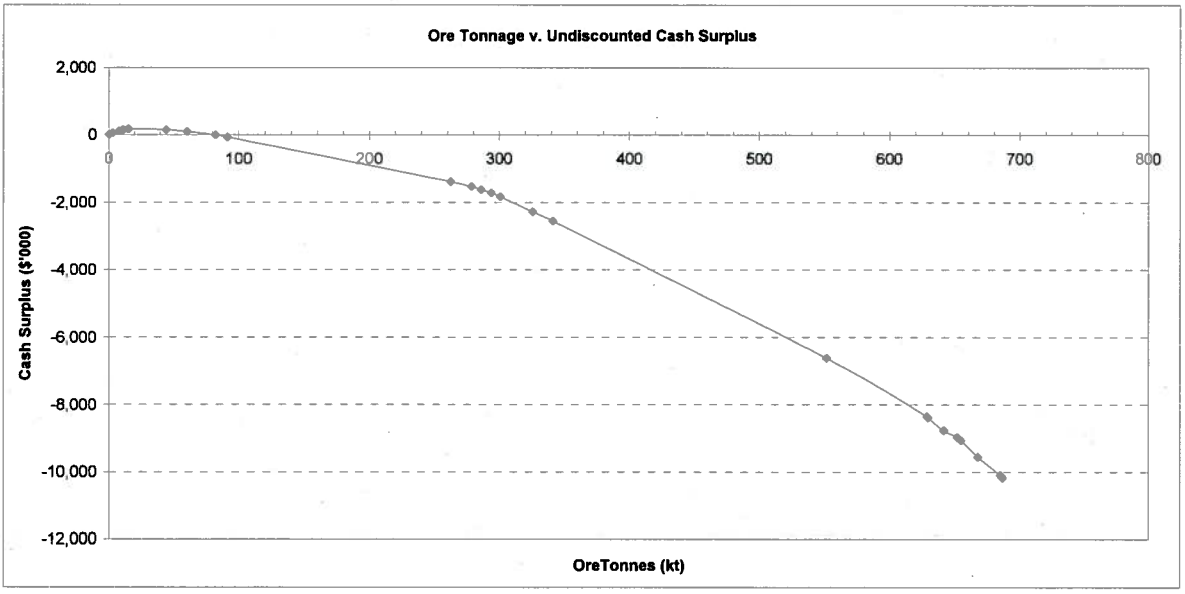
	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	7	4	19.19	2.03	295	295
2	3	2.47	3	6	0.97	246	221	90%	11	59	154	84	19.23	1.83	319	321
3	8	2.33	5	13	0.61	610	549	90%	25	157	383	201	19.23	1.93	332	340
4	11	2.28	6	17	0.59	783	705	90%	33	205	492	254	19.23	1.95	338	358
5	12	2.30	9	20	0.77	851	766	90%	39	221	535	274	19.23	1.93	340	370
6	15	2.20	16	31	1.03	1,072	965	90%	60	292	673	322	19.23	1.96	365	458
7	15	2.19	16	32	1.04	1,089	980	90%	62	297	684	325	19.23	1.96	366	467
8	16	2.17	16	32	1.02	1,108	997	90%	63	305	696	328	19.23	1.96	369	525
9	52	2.30	261	313	5.00	3,855	3,469	90%	616	1,004	2,421	801	19.23	1.97	467	507
10	71	2.34	418	490	5.87	5,366	4,830	90%	970	1,370	3,371	1,031	19.23	1.98	485	529
11	94	2.33	627	721	6.67	7,047	6,342	90%	1,359	1,808	4,427	1,260	19.23	1.88	499	546
12	105	2.34	733	838	6.97	7,928	7,135	90%	1,590	2,023	4,980	1,368	19.23	1.90	506	563
13	294	2.25	2,268	2,563	7.70	21,332	19,198	90%	5,055	5,657	13,400	2,688	19.21	1.97	558	589
14	312	2.25	2,400	2,712	7.69	22,548	20,293	90%	5,382	5,995	14,164	2,787	19.21	1.98	561	607
15	321	2.24	2,477	2,798	7.71	23,179	20,861	90%	5,563	6,177	14,560	2,820	19.21	1.99	563	639
16	332	2.23	2,537	2,868	7.65	23,746	21,372	90%	5,713	6,370	14,917	2,834	19.21	1.99	565	672
17	339	2.23	2,621	2,959	7.73	24,311	21,880	90%	5,913	6,509	15,271	2,848	19.21	2.00	568	669
18	366	2.24	2,926	3,292	8.00	26,306	23,675	90%	6,645	7,029	16,524	2,850	19.21	2.02	578	697
19	385	2.22	3,087	3,472	8.03	27,462	24,716	90%	7,042	7,387	17,251	2,822	19.21	2.03	584	726
20	639	2.07	5,210	5,849	8.15	42,522	38,270	90%	12,401	12,272	26,711	2,038	19.20	2.12	645	756
22	732	2.05	6,076	6,808	8.30	48,176	43,357	90%	14,625	14,047	30,262	1,590	19.20	2.15	661	786
23	735	2.04	6,089	6,824	8.29	48,296	43,466	90%	14,659	14,106	30,338	1,572	19.20	2.15	662	858
24	750	2.05	6,284	7,034	8.38	49,326	44,393	90%	15,134	14,401	30,985	1,450	19.20	2.15	665	829
25	760	2.04	6,359	7,120	8.37	49,846	44,861	90%	15,344	14,594	31,312	1,374	19.20	2.16	667	862
26	765	2.04	6,400	7,164	8.37	50,074	45,068	90%	15,451	14,678	31,456	1,326	19.20	2.16	669	928
27	781	2.04	6,624	7,404	8.48	51,166	46,050	90%	16,029	14,989	32,141	1,122	19.20	2.16	674	905
28	804	2.03	6,831	7,635	8.50	52,325	47,093	90%	16,585	15,426	32,031	857	19.20	2.17	662	952
29	805	2.02	6,851	7,656	8.51	52,415	47,174	90%	16,635	15,460	32,114	832	19.20	2.17	663	1014
30	806	2.02	6,860	7,666	8.51	52,464	47,218	90%	16,656	15,481	32,156	819	19.20	2.17	664	980

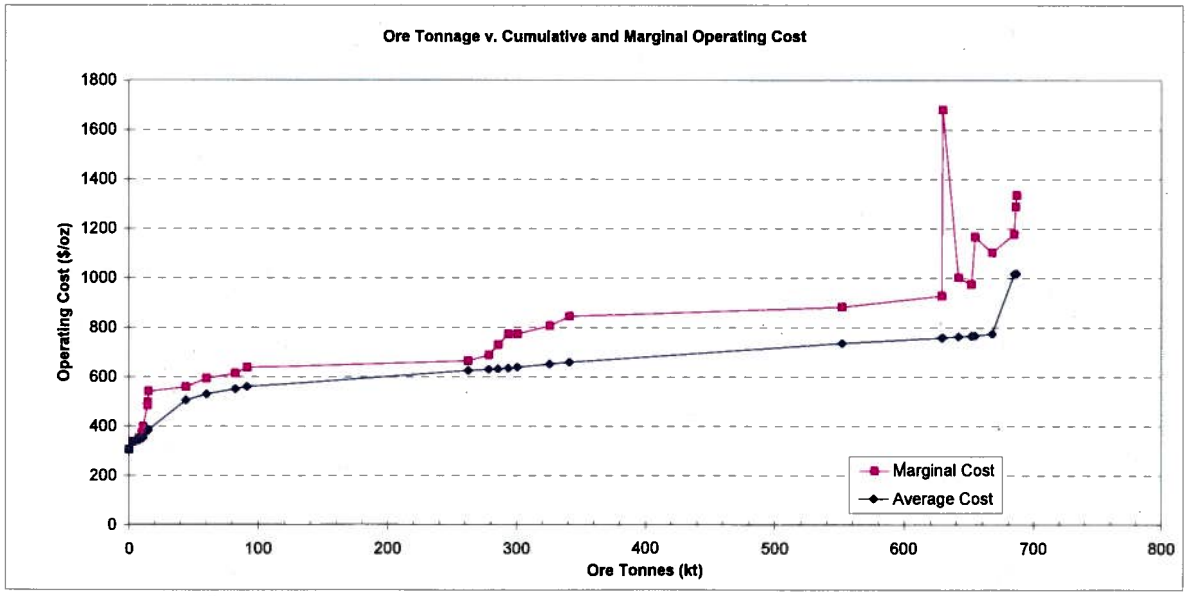




Davis Pit Extension Optimisation Results - Run01a2
$550/oz, Gold Price, 100% Mining Costs

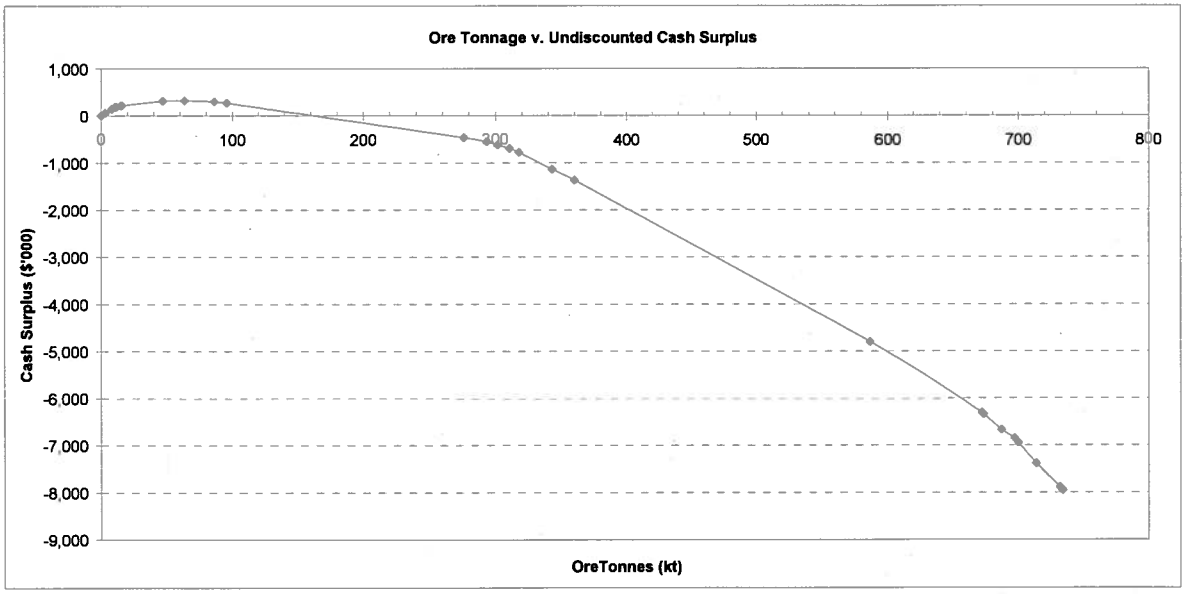
	Physicals								Economics							
	Ore		Waste	Total	W/O	Contn'd	Recv'd	Recv.	Mining	Th'put	Reven.	Undisc'd	Th'put	Min.	Unit	Marg.
Pit No	Total kt	Au g/t	kt	kt	Ratio t/t	Ounces Au	Ounces Au	%	Cost $'000	Cost $'000	$'000	C'Flow $'000	Cost $/t	Cost $/t	Cost $/oz	Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	5	2	19.19	2.54	304	304
2	3	2.47	3	6	0.97	246	221	90%	14	59	121	48	19.23	2.29	332	333
3	8	2.35	5	13	0.64	606	545	90%	32	154	299	113	19.23	2.42	341	348
4	11	2.30	6	17	0.61	778	700	90%	41	202	384	140	19.23	2.44	348	370
5	11	2.32	9	20	0.80	846	762	90%	49	219	417	150	19.23	2.41	352	396
6	15	2.22	16	31	1.06	1,064	957	90%	75	287	525	162	19.23	2.45	379	483
7	15	2.21	16	32	1.08	1,081	973	90%	77	293	533	163	19.23	2.45	380	495
8	16	2.19	16	32	1.05	1,100	990	90%	79	301	543	163	19.23	2.45	383	538
9	45	2.50	269	313	6.00	3,600	3,240	90%	770	861	1,775	144	19.23	2.46	504	557
10	61	2.56	429	490	7.07	5,004	4,503	90%	1,213	1,167	2,468	88	19.23	2.48	528	592
11	83	2.51	639	721	7.74	6,650	5,985	90%	1,698	1,587	3,280	-5	19.23	2.36	549	611
12	92	2.53	747	838	8.16	7,454	6,709	90%	1,987	1,759	3,677	-69	19.23	2.37	558	637
13	262	2.39	2,300	2,563	8.76	20,209	18,188	90%	6,319	5,043	9,968	-1,394	19.21	2.47	625	663
14	278	2.39	2,433	2,712	8.74	21,366	19,230	90%	6,727	5,350	10,539	-1,538	19.21	2.48	628	686
15	286	2.39	2,513	2,798	8.79	21,928	19,735	90%	6,954	5,491	10,816	-1,629	19.21	2.48	631	729
16	294	2.37	2,575	2,868	8.77	22,418	20,176	90%	7,142	5,643	11,057	-1,727	19.21	2.49	634	770
17	301	2.38	2,659	2,959	8.84	22,969	20,672	90%	7,392	5,776	11,329	-1,838	19.21	2.50	637	772
18	326	2.38	2,966	3,292	9.11	24,893	22,404	90%	8,306	6,255	12,278	-2,283	19.21	2.52	650	805
19	341	2.37	3,131	3,472	9.18	25,941	23,347	90%	8,803	6,553	12,795	-2,561	19.21	2.54	658	843
20	552	2.22	5,297	5,849	9.60	39,484	35,535	90%	15,502	10,598	19,475	-6,625	19.20	2.65	734	882
22	629	2.21	6,179	6,808	9.83	44,579	40,121	90%	18,282	12,071	21,988	-8,364	19.20	2.69	757	927
23	629	2.20	6,194	6,824	9.84	44,616	40,154	90%	18,324	12,083	22,006	-8,401	19.20	2.69	757	1681
24	642	2.21	6,393	7,034	9.96	45,532	40,979	90%	18,918	12,315	22,458	-8,775	19.20	2.69	762	1001
25	652	2.20	6,468	7,120	9.93	46,053	41,447	90%	19,180	12,509	22,715	-8,974	19.20	2.69	765	974
26	655	2.20	6,510	7,164	9.95	46,233	41,610	90%	19,314	12,565	22,804	-9,075	19.20	2.70	766	1166
27	668	2.20	6,737	7,404	10.09	47,215	42,494	90%	20,037	12,815	23,288	-9,563	19.20	2.71	773	1101
28	685	2.19	6,950	7,635	10.15	48,186	43,366	90%	20,732	13,147	33,897	-10,112	19.20	2.72	1015	1177
29	686	2.19	6,970	7,656	10.16	48,258	43,432	90%	20,793	13,170	33,982	-10,161	19.20	2.72	1016	1288
30	687	2.19	6,980	7,666	10.16	48,290	43,461	90%	20,820	13,181	34,020	-10,183	19.20	2.72	1017	1334

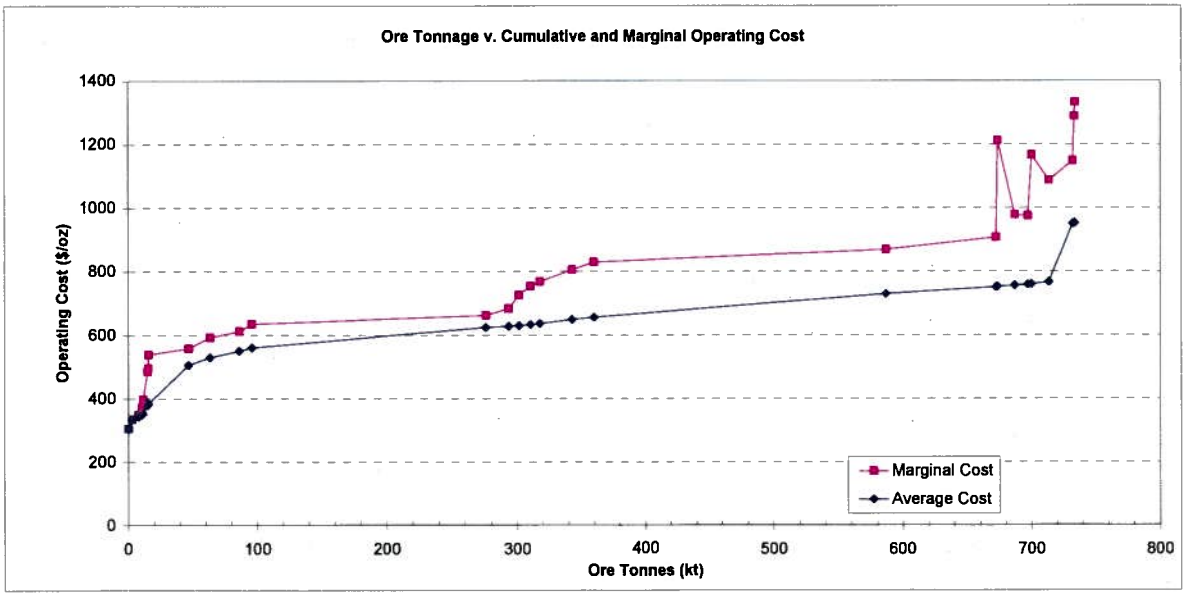




Davis Pit Extension Optimisation Results - Run01b2
$600/oz, Gold Price, 100% Mining Costs

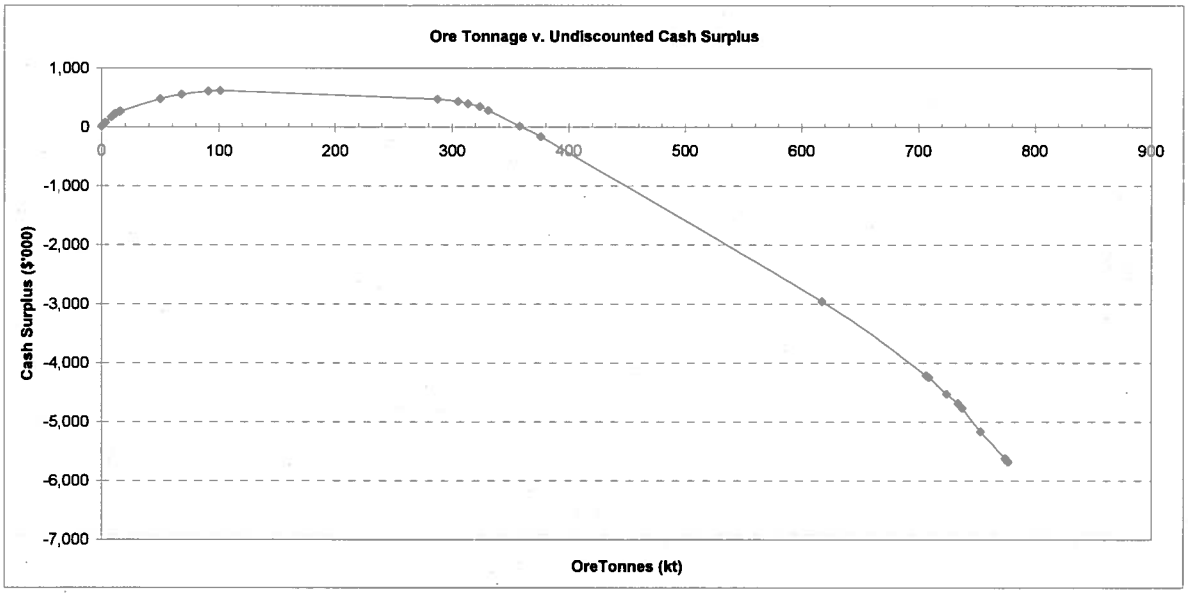
	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	6	3	19.19	2.54	304	304
2	3	2.47	3	6	0.97	246	221	90%	14	59	132	59	19.23	2.29	332	333
3	8	2.35	5	13	0.64	606	545	90%	32	154	326	140	19.23	2.42	341	348
4	11	2.30	6	17	0.61	779	701	90%	41	202	419	175	19.23	2.44	348	370
5	11	2.31	9	20	0.79	847	762	90%	49	219	456	188	19.23	2.41	352	396
6	15	2.21	16	31	1.05	1,067	960	90%	75	289	574	210	19.23	2.45	379	484
7	15	2.20	16	32	1.07	1,084	976	90%	77	294	584	212	19.23	2.45	381	495
8	16	2.18	16	32	1.04	1,103	993	90%	79	302	594	213	19.23	2.45	384	538
9	47	2.44	267	313	5.71	3,672	3,304	90%	770	899	1,976	308	19.23	2.46	505	557
10	63	2.51	426	490	6.73	5,101	4,591	90%	1,213	1,217	2,746	316	19.23	2.48	529	592
11	86	2.45	635	721	7.38	6,781	6,103	90%	1,698	1,655	3,651	297	19.23	2.36	549	611
12	96	2.47	743	838	7.75	7,613	6,852	90%	1,987	1,841	4,098	270	19.23	2.37	559	634
13	276	2.33	2,286	2,563	8.27	20,728	18,656	90%	6,319	5,310	11,158	-470	19.21	2.47	623	661
14	294	2.32	2,418	2,712	8.24	21,933	19,740	90%	6,727	5,641	11,807	-561	19.21	2.48	627	682
15	301	2.32	2,497	2,798	8.28	22,513	20,262	90%	6,954	5,792	12,119	-627	19.21	2.48	629	724
16	311	2.31	2,558	2,868	8.24	23,046	20,741	90%	7,142	5,965	12,406	-701	19.21	2.49	632	753
17	318	2.31	2,642	2,959	8.31	23,609	21,248	90%	7,392	6,104	12,709	-787	19.21	2.50	635	767
18	343	2.32	2,949	3,292	8.60	25,538	22,984	90%	8,306	6,586	13,747	-1,145	19.21	2.52	648	804
19	360	2.30	3,112	3,472	8.65	26,643	23,978	90%	8,803	6,913	14,342	-1,374	19.21	2.54	655	829
20	587	2.16	5,262	5,849	8.97	40,784	36,706	90%	15,502	11,265	21,955	-4,812	19.20	2.65	729	868
22	672	2.14	6,136	6,808	9.13	46,200	41,580	90%	18,282	12,902	24,870	-6,313	19.20	2.69	750	906
23	673	2.14	6,150	6,824	9.13	46,261	41,635	90%	18,324	12,926	24,903	-6,347	19.20	2.69	751	1211
24	687	2.14	6,347	7,034	9.24	47,233	42,509	90%	18,918	13,186	25,426	-6,678	19.20	2.69	755	977
25	697	2.13	6,423	7,120	9.21	47,753	42,977	90%	19,180	13,380	25,706	-6,854	19.20	2.69	758	974
26	700	2.13	6,464	7,164	9.24	47,933	43,140	90%	19,314	13,435	25,803	-6,947	19.20	2.70	759	1166
27	714	2.13	6,691	7,404	9.37	48,943	44,049	90%	20,037	13,700	26,347	-7,390	19.20	2.71	766	1086
28	732	2.12	6,903	7,635	9.43	49,962	44,965	90%	20,732	14,056	34,807	-7,893	19.20	2.72	950	1148
29	733	2.12	6,923	7,656	9.44	50,035	45,031	90%	20,793	14,079	34,892	-7,939	19.20	2.72	951	1288
30	734	2.12	6,932	7,666	9.44	50,066	45,059	90%	20,820	14,091	34,930	-7,960	19.20	2.72	952	1332

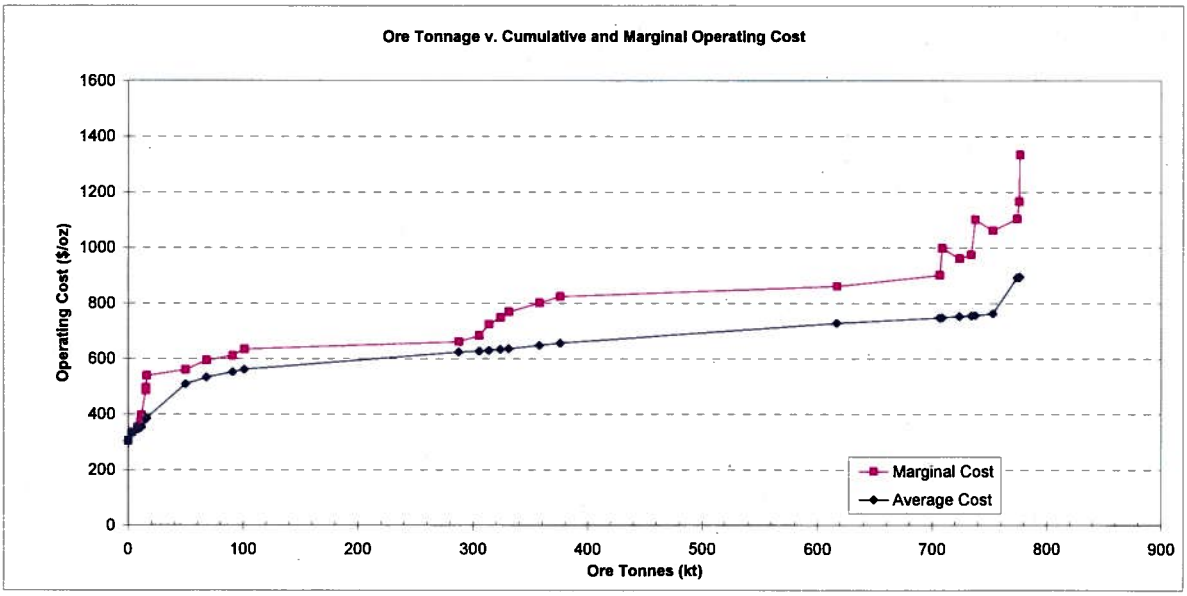




Davis Pit Extension Optimisation Results - Run01c2
$650/oz, Gold Price, 100% Mining Costs

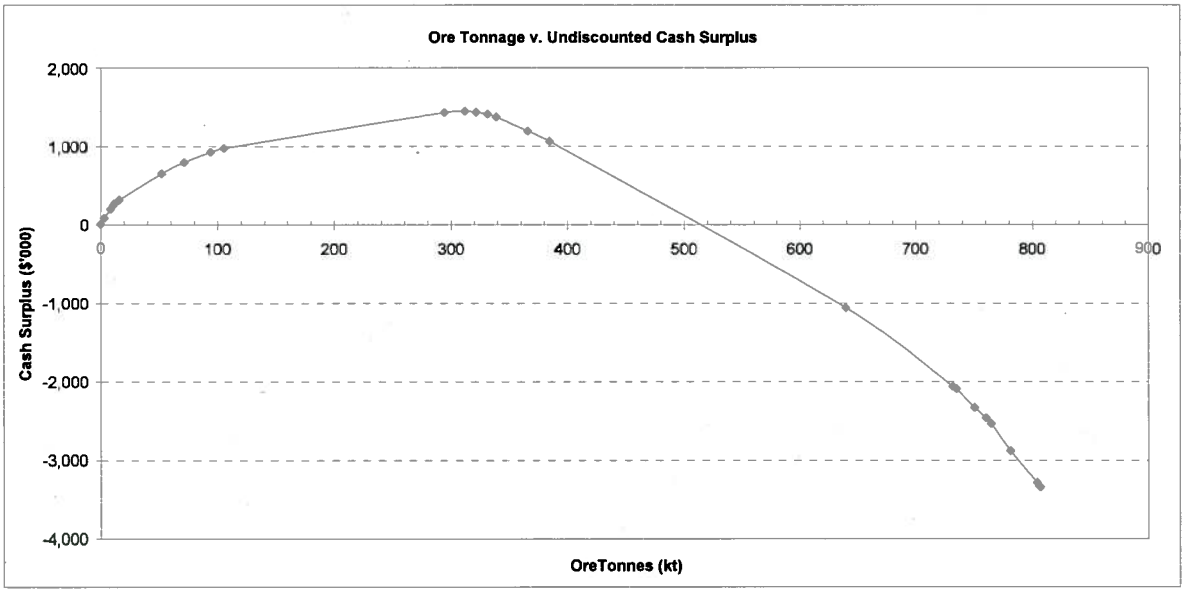
	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	6	3	19.19	2.54	304	304
2	3	2.47	3	6	0.97	246	221	90%	14	59	143	70	19.23	2.29	332	333
3	8	2.33	5	13	0.61	610	549	90%	32	157	356	167	19.23	2.42	344	351
4	11	2.28	6	17	0.59	783	705	90%	41	205	456	210	19.23	2.44	349	370
5	12	2.30	9	20	0.77	851	766	90%	49	221	496	226	19.23	2.41	353	396
6	15	2.20	16	31	1.03	1,072	965	90%	75	292	625	258	19.23	2.45	380	485
7	15	2.19	16	32	1.04	1,089	980	90%	77	297	635	261	19.23	2.45	382	495
8	16	2.17	16	32	1.02	1,108	997	90%	79	305	646	262	19.23	2.45	385	538
9	50	2.36	264	313	5.30	3,777	3,399	90%	770	957	2,202	475	19.23	2.46	508	559
10	68	2.41	422	490	6.20	5,263	4,737	90%	1,213	1,307	3,069	549	19.23	2.48	532	593
11	91	2.38	630	721	6.94	6,944	6,249	90%	1,698	1,745	4,049	605	19.23	2.36	551	611
12	101	2.40	737	838	7.30	7,796	7,017	90%	1,987	1,943	4,546	616	19.23	2.37	560	634
13	288	2.28	2,275	2,563	7.91	21,116	19,004	90%	6,319	5,527	12,313	467	19.21	2.47	623	660
14	305	2.28	2,406	2,712	7.88	22,332	20,099	90%	6,727	5,864	13,022	431	19.21	2.48	626	682
15	314	2.27	2,485	2,798	7.92	22,930	20,637	90%	6,954	6,026	13,370	391	19.21	2.48	629	722
16	324	2.26	2,545	2,868	7.86	23,497	21,148	90%	7,142	6,219	13,701	341	19.21	2.49	632	746
17	331	2.26	2,628	2,959	7.94	24,062	21,656	90%	7,392	6,359	14,030	280	19.21	2.50	635	767
18	358	2.27	2,934	3,292	8.21	26,043	23,438	90%	8,306	6,870	15,185	10	19.21	2.52	647	800
19	376	2.25	3,096	3,472	8.24	27,186	24,468	90%	8,803	7,219	15,852	-170	19.21	2.54	655	823
20	617	2.11	5,232	5,849	8.48	41,827	37,645	90%	15,502	11,851	24,389	-2,963	19.20	2.65	727	860
22	707	2.09	6,101	6,808	8.63	47,382	42,643	90%	18,282	13,566	27,628	-4,220	19.20	2.69	747	899
23	709	2.08	6,115	6,824	8.63	47,479	42,730	90%	18,324	13,610	27,684	-4,250	19.20	2.69	747	997
24	724	2.08	6,310	7,034	8.72	48,497	43,648	90%	18,918	13,898	28,279	-4,537	19.20	2.69	752	960
25	734	2.08	6,386	7,120	8.70	49,017	44,116	90%	19,180	14,092	28,582	-4,689	19.20	2.69	754	974
26	738	2.08	6,427	7,164	8.71	49,225	44,302	90%	19,314	14,161	28,703	-4,773	19.20	2.70	756	1101
27	753	2.08	6,651	7,404	8.83	50,291	45,263	90%	20,037	14,458	29,325	-5,170	19.20	2.71	762	1061
28	774	2.06	6,860	7,635	8.86	51,404	46,263	90%	20,732	14,867	35,618	-5,625	19.20	2.72	891	1103
29	776	2.06	6,880	7,656	8.86	51,494	46,345	90%	20,793	14,900	35,713	-5,667	19.20	2.72	893	1166
30	777	2.06	6,890	7,666	8.87	51,526	46,373	90%	20,820	14,911	35,751	-5,687	19.20	2.72	894	1334

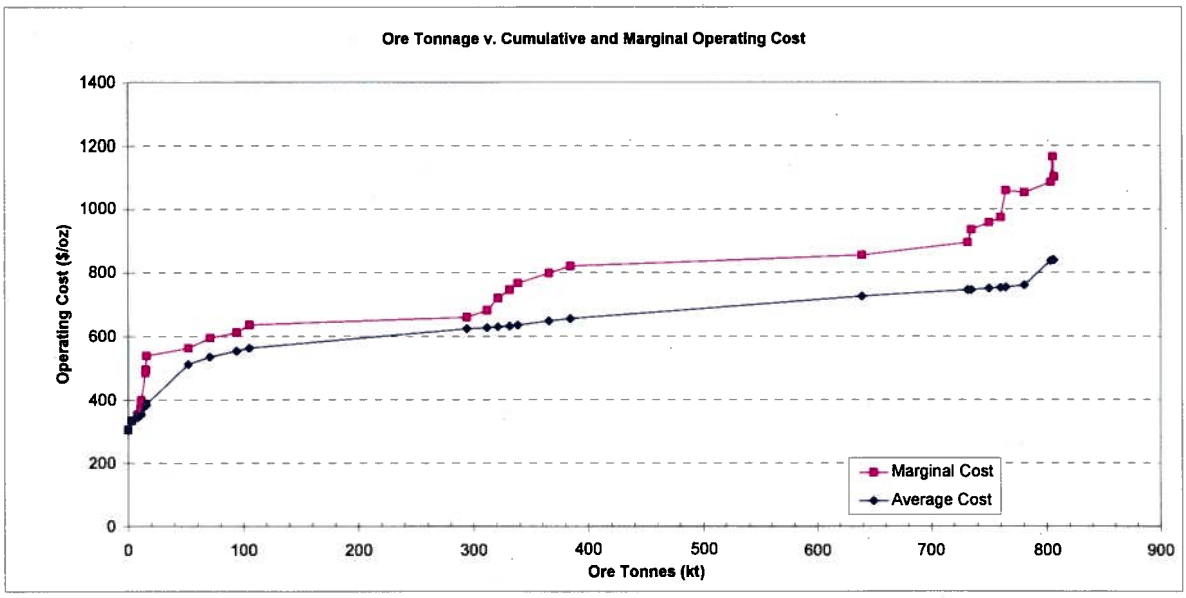




Davis Pit Extension Optimisation Results - Run01d2
$700/oz, Gold Price, 100% Mining Costs

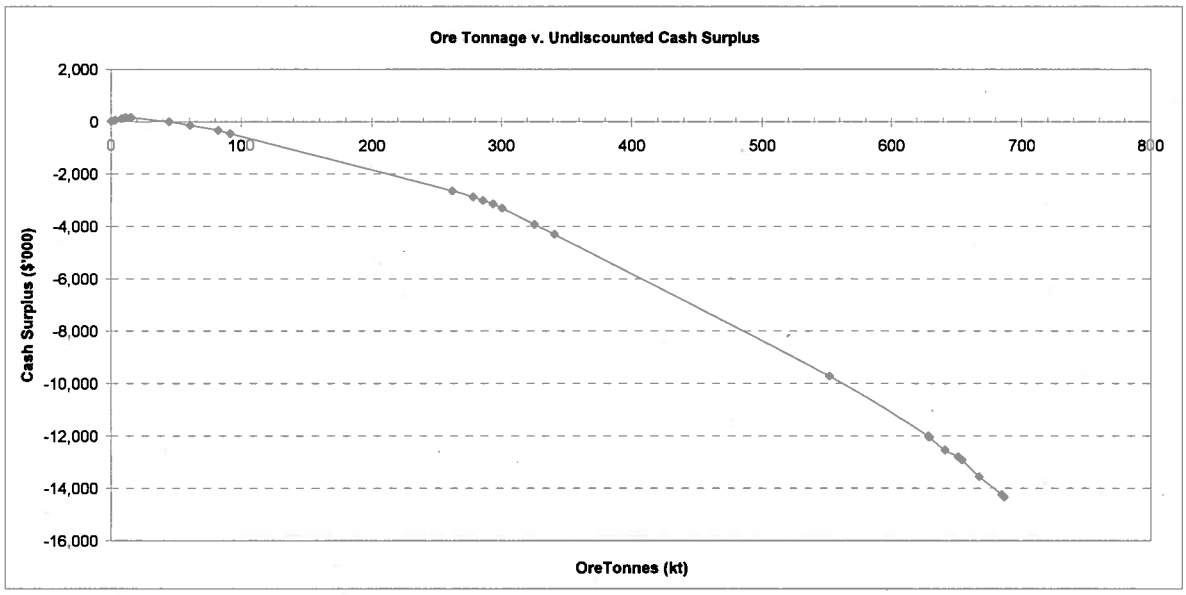
	Physicals								Economics							
Pit No	Ore Total kt	Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	0	3	7	4	19.19	2.54	304	304
2	3	2.47	3	6	0.97	246	221	90%	14	59	154	81	19.23	2.29	332	333
3	8	2.33	5	13	0.61	610	549	90%	32	157	383	195	19.23	2.42	344	351
4	11	2.28	6	17	0.59	783	705	90%	41	205	492	246	19.23	2.44	349	370
5	12	2.30	9	20	0.77	851	766	90%	49	221	535	264	19.23	2.41	353	396
6	15	2.20	16	31	1.03	1,072	965	90%	75	292	673	306	19.23	2.45	380	485
7	15	2.19	16	32	1.04	1,089	980	90%	77	297	684	310	19.23	2.45	382	495
8	16	2.17	16	32	1.02	1,108	997	90%	79	305	696	312	19.23	2.45	385	538
9	52	2.30	261	313	5.00	3,855	3,469	90%	770	1,004	2,421	647	19.23	2.46	511	563
10	71	2.34	418	490	5.87	5,366	4,830	90%	1,213	1,370	3,371	788	19.23	2.48	535	594
11	94	2.33	627	721	6.67	7,047	6,342	90%	1,698	1,808	4,427	921	19.23	2.36	553	611
12	105	2.34	733	838	6.97	7,928	7,135	90%	1,987	2,023	4,980	970	19.23	2.37	562	635
13	294	2.25	2,268	2,563	7.70	21,332	19,198	90%	6,319	5,657	13,400	1,424	19.21	2.47	624	660
14	312	2.25	2,400	2,712	7.69	22,548	20,293	90%	6,727	5,995	14,164	1,442	19.21	2.48	627	682
15	321	2.24	2,477	2,798	7.71	23,179	20,861	90%	6,954	6,177	14,560	1,430	19.21	2.48	629	719
16	332	2.23	2,537	2,868	7.65	23,746	21,372	90%	7,142	6,370	14,917	1,405	19.21	2.49	632	746
17	339	2.23	2,621	2,959	7.73	24,311	21,880	90%	7,392	6,509	15,271	1,370	19.21	2.50	635	767
18	366	2.24	2,926	3,292	8.00	26,306	23,675	90%	8,306	7,029	16,524	1,189	19.21	2.52	648	799
19	385	2.22	3,087	3,472	8.03	27,462	24,716	90%	8,803	7,387	17,251	1,061	19.21	2.54	655	821
20	639	2.07	5,210	5,849	8.15	42,522	38,270	90%	15,502	12,272	26,711	-1,063	19.20	2.65	726	855
22	732	2.05	6,076	6,808	8.30	48,176	43,357	90%	18,282	14,047	30,262	-2,067	19.20	2.69	746	895
23	735	2.04	6,089	6,824	8.29	48,296	43,466	90%	18,324	14,106	30,338	-2,093	19.20	2.69	746	936
24	750	2.05	6,284	7,034	8.38	49,326	44,393	90%	18,918	14,401	30,985	-2,333	19.20	2.69	751	957
25	760	2.04	6,359	7,120	8.37	49,846	44,861	90%	19,180	14,594	31,312	-2,462	19.20	2.69	753	974
26	765	2.04	6,400	7,164	8.37	50,074	45,068	90%	19,314	14,678	31,456	-2,537	19.20	2.70	754	1058
27	781	2.04	6,624	7,404	8.48	51,166	46,050	90%	20,037	14,989	32,141	-2,885	19.20	2.71	761	1053
28	804	2.03	6,831	7,635	8.50	52,325	47,093	90%	20,732	15,426	36,177	-3,289	19.20	2.72	838	1085
29	805	2.02	6,851	7,656	8.51	52,415	47,174	90%	20,793	15,460	36,272	-3,327	19.20	2.72	839	1166
30	806	2.02	6,860	7,666	8.51	52,464	47,218	90%	20,820	15,481	36,320	-3,345	19.20	2.72	840	1102

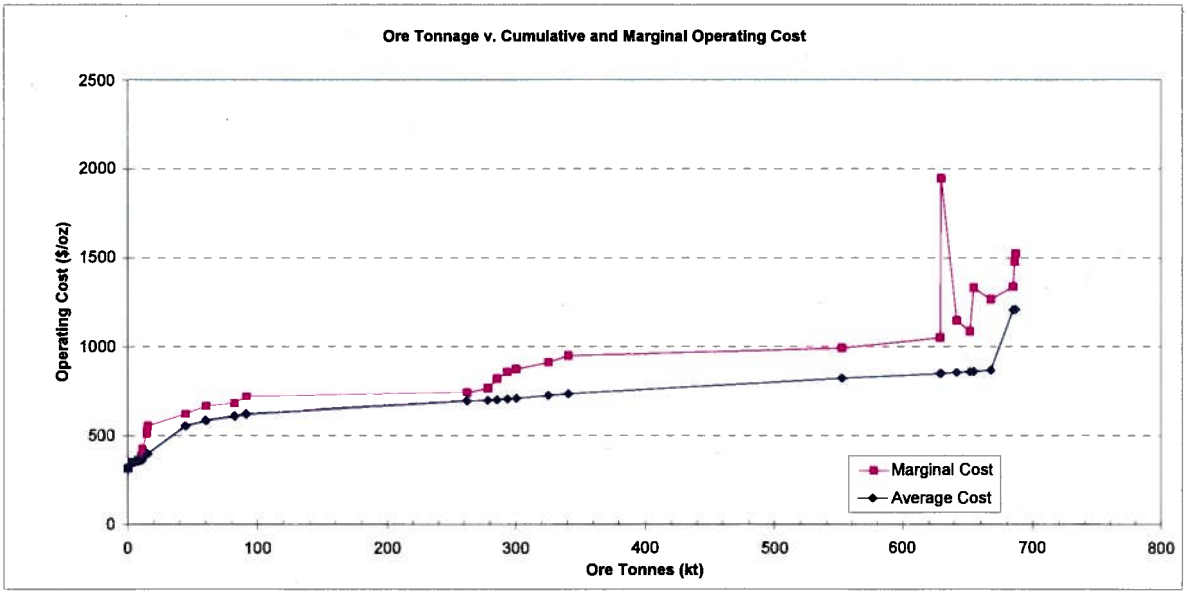




Davis Pit Extension Optimisation Results - Run01a3
$550/oz, Gold Price, 120% Mining Costs

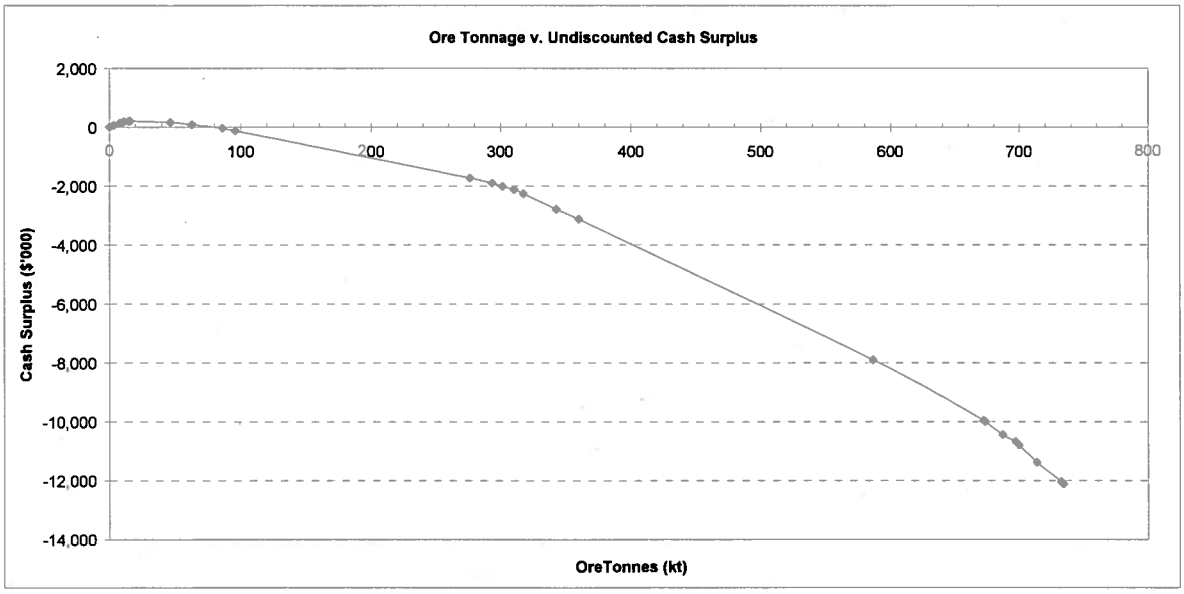
	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	1	3	5	2	19.19	3.05	313	313
2	3	2.47	3	6	0.97	246	221	90%	17	59	121	45	19.23	2.74	345	346
3	8	2.35	5	13	0.64	606	545	90%	38	154	299	106	19.23	2.90	353	359
4	11	2.30	6	17	0.61	778	700	90%	49	202	384	132	19.23	2.93	359	382
5	11	2.32	9	20	0.80	846	762	90%	59	219	417	140	19.23	2.89	365	422
6	15	2.22	16	31	1.06	1,064	957	90%	90	287	525	147	19.23	2.94	394	510
7	15	2.21	16	32	1.08	1,081	973	90%	93	293	533	148	19.23	2.94	396	522
8	16	2.19	16	32	1.05	1,100	990	90%	94	301	543	148	19.23	2.94	399	551
9	45	2.50	269	313	6.00	3,600	3,240	90%	924	861	1,775	-10	19.23	2.95	551	618
10	61	2.56	429	490	7.07	5,004	4,503	90%	1,455	1,167	2,468	-154	19.23	2.97	582	662
11	83	2.51	639	721	7.74	6,650	5,985	90%	2,038	1,587	3,280	-345	19.23	2.83	606	677
12	92	2.53	747	838	8.16	7,454	6,709	90%	2,384	1,759	3,677	-467	19.23	2.84	618	716
13	262	2.39	2,300	2,563	8.76	20,209	18,188	90%	7,582	5,043	9,968	-2,658	19.21	2.96	694	739
14	278	2.39	2,433	2,712	8.74	21,366	19,230	90%	8,073	5,350	10,539	-2,884	19.21	2.98	698	765
15	286	2.39	2,513	2,798	8.79	21,928	19,735	90%	8,345	5,491	10,816	-3,020	19.21	2.98	701	818
16	294	2.37	2,575	2,868	8.77	22,418	20,176	90%	8,570	5,643	11,057	-3,156	19.21	2.99	704	856
17	301	2.38	2,659	2,959	8.84	22,969	20,672	90%	8,870	5,776	11,329	-3,317	19.21	3.00	708	872
18	326	2.38	2,966	3,292	9.11	24,893	22,404	90%	9,967	6,255	12,278	-3,944	19.21	3.03	724	910
19	341	2.37	3,131	3,472	9.18	25,941	23,347	90%	10,564	6,553	12,795	-4,321	19.21	3.04	733	948
20	552	2.22	5,297	5,849	9.60	39,484	35,535	90%	18,602	10,598	19,475	-9,726	19.20	3.18	822	991
22	629	2.21	6,179	6,808	9.83	44,579	40,121	90%	21,938	12,071	21,988	-12,021	19.20	3.22	848	1048
23	629	2.20	6,194	6,824	9.84	44,616	40,154	90%	21,989	12,083	22,006	-12,066	19.20	3.22	849	1945
24	642	2.21	6,393	7,034	9.96	45,532	40,979	90%	22,701	12,315	22,458	-12,558	19.20	3.23	854	1145
25	652	2.20	6,468	7,120	9.93	46,053	41,447	90%	23,016	12,509	22,715	-12,810	19.20	3.23	857	1086
26	655	2.20	6,510	7,164	9.95	46,233	41,610	90%	23,177	12,565	22,804	-12,937	19.20	3.24	859	1331
27	668	2.20	6,737	7,404	10.09	47,215	42,494	90%	24,044	12,815	23,288	-13,571	19.20	3.25	867	1265
28	685	2.19	6,950	7,635	10.15	48,186	43,366	90%	24,878	13,147	38,044	-14,258	19.20	3.26	1206	1336
29	686	2.19	6,970	7,656	10.16	48,258	43,432	90%	24,952	13,170	38,141	-14,319	19.20	3.26	1208	1475
30	687	2.19	6,980	7,666	10.16	48,290	43,461	90%	24,984	13,181	38,184	-14,347	19.20	3.26	1209	1521

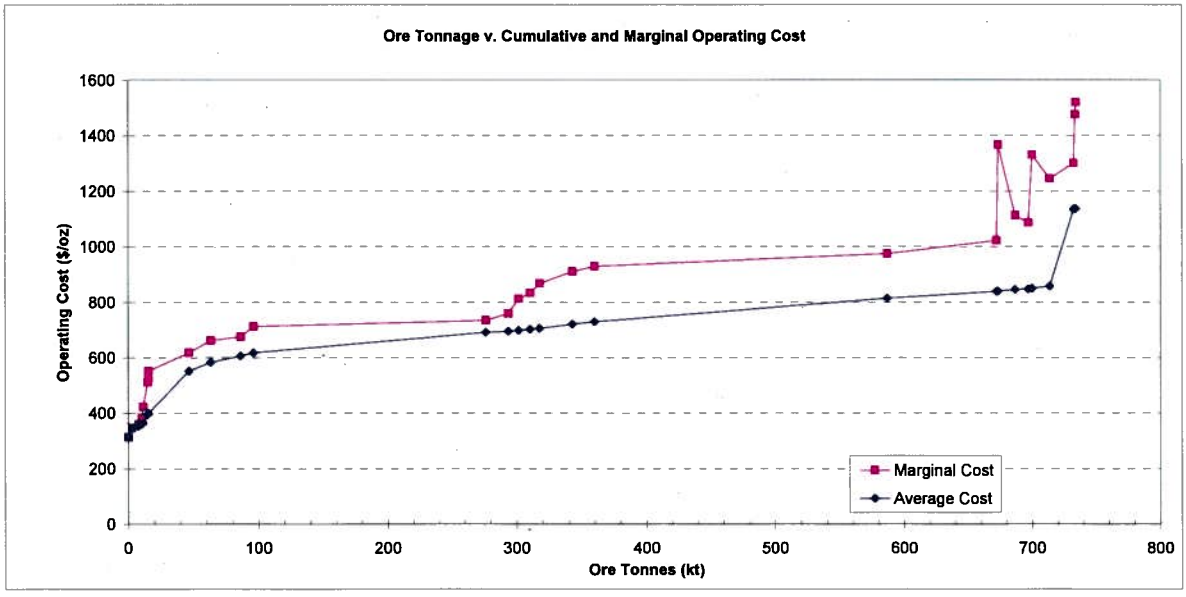




Davis Pit Extension Optimisation Results - Run01b3
$600/oz, Gold Price, 120% Mining Costs

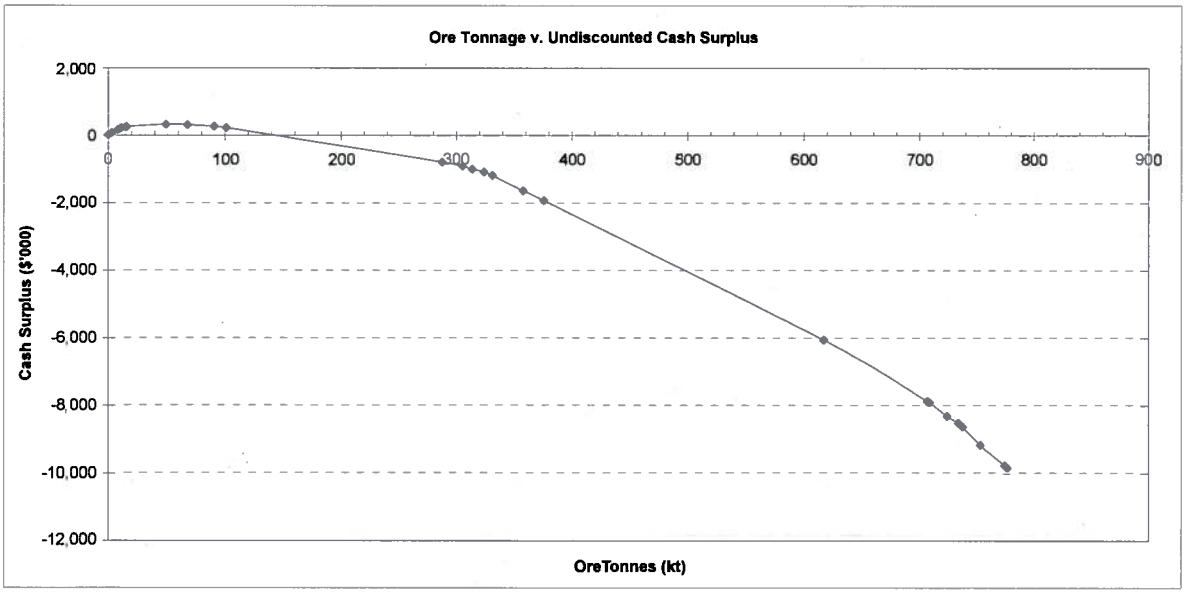
	Physicals								Economics							
	Ore		Waste	Total	W/O	Contn'd	Recv'd	Recv.	Mining	Th'put	Reven.	Undisc'd	Th'put	Min.	Unit	Marg.
Pit No	Total kt	Au g/t	kt	kt	Ratio t/t	Ounces Au	Ounces Au	%	Cost $'000	Cost $'000	$'000	C'Flow $'000	Cost $/t	Cost $/t	Cost $/oz	Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	1	3	6	3	19.19	3.05	313	313
2	3	2.47	3	6	0.97	246	221	90%	17	59	132	56	19.23	2.74	345	346
3	8	2.35	5	13	0.64	606	545	90%	38	154	326	134	19.23	2.90	353	359
4	11	2.30	6	17	0.61	779	701	90%	49	202	419	167	19.23	2.93	360	382
5	11	2.31	9	20	0.79	847	762	90%	59	219	456	178	19.23	2.89	365	422
6	15	2.21	16	31	1.05	1,067	960	90%	90	289	574	195	19.23	2.94	395	511
7	15	2.20	16	32	1.07	1,084	976	90%	93	294	584	196	19.23	2.94	397	522
8	16	2.18	16	32	1.04	1,103	993	90%	94	302	594	197	19.23	2.94	399	551
9	47	2.44	267	313	5.71	3,672	3,304	90%	924	899	1,976	154	19.23	2.95	552	617
10	63	2.51	426	490	6.73	5,101	4,591	90%	1,455	1,217	2,746	73	19.23	2.97	582	661
11	86	2.45	635	721	7.38	6,781	6,103	90%	2,038	1,655	3,651	-43	19.23	2.83	605	675
12	96	2.47	743	838	7.75	7,613	6,852	90%	2,384	1,841	4,098	-128	19.23	2.84	617	711
13	276	2.33	2,286	2,563	8.27	20,728	18,656	90%	7,582	5,310	11,158	-1,734	19.21	2.96	691	734
14	294	2.32	2,418	2,712	8.24	21,933	19,740	90%	8,073	5,641	11,807	-1,907	19.21	2.98	695	758
15	301	2.32	2,497	2,798	8.28	22,513	20,262	90%	8,345	5,792	12,119	-2,018	19.21	2.98	698	811
16	311	2.31	2,558	2,868	8.24	23,046	20,741	90%	8,570	5,965	12,406	-2,130	19.21	2.99	701	831
17	318	2.31	2,642	2,959	8.31	23,609	21,248	90%	8,870	6,104	12,709	-2,266	19.21	3.00	705	866
18	343	2.32	2,949	3,292	8.60	25,538	22,984	90%	9,967	6,586	13,747	-2,806	19.21	3.03	720	909
19	360	2.30	3,112	3,472	8.65	26,643	23,978	90%	10,564	6,913	14,342	-3,135	19.21	3.04	729	929
20	587	2.16	5,262	5,849	8.97	40,784	36,706	90%	18,602	11,265	21,955	-7,912	19.20	3.18	814	973
22	672	2.14	6,136	6,808	9.13	46,200	41,580	90%	21,938	12,902	24,870	-9,970	19.20	3.22	838	1020
23	673	2.14	6,150	6,824	9.13	46,261	41,635	90%	21,989	12,926	24,903	-10,012	19.20	3.22	839	1366
24	687	2.14	6,347	7,034	9.24	47,233	42,509	90%	22,701	13,186	25,426	-10,462	19.20	3.23	844	1112
25	697	2.13	6,423	7,120	9.21	47,753	42,977	90%	23,016	13,380	25,706	-10,690	19.20	3.23	847	1086
26	700	2.13	6,464	7,164	9.24	47,933	43,140	90%	23,177	13,435	25,803	-10,809	19.20	3.24	849	1331
27	714	2.13	6,691	7,404	9.37	48,943	44,049	90%	24,044	13,700	26,347	-11,397	19.20	3.25	857	1245
28	732	2.12	6,903	7,635	9.43	49,962	44,965	90%	24,878	14,056	38,953	-12,040	19.20	3.26	1134	1300
29	733	2.12	6,923	7,656	9.44	50,035	45,031	90%	24,952	14,079	39,051	-12,098	19.20	3.26	1136	1475
30	734	2.12	6,932	7,666	9.44	50,066	45,059	90%	24,984	14,091	39,094	-12,124	19.20	3.26	1137	1519

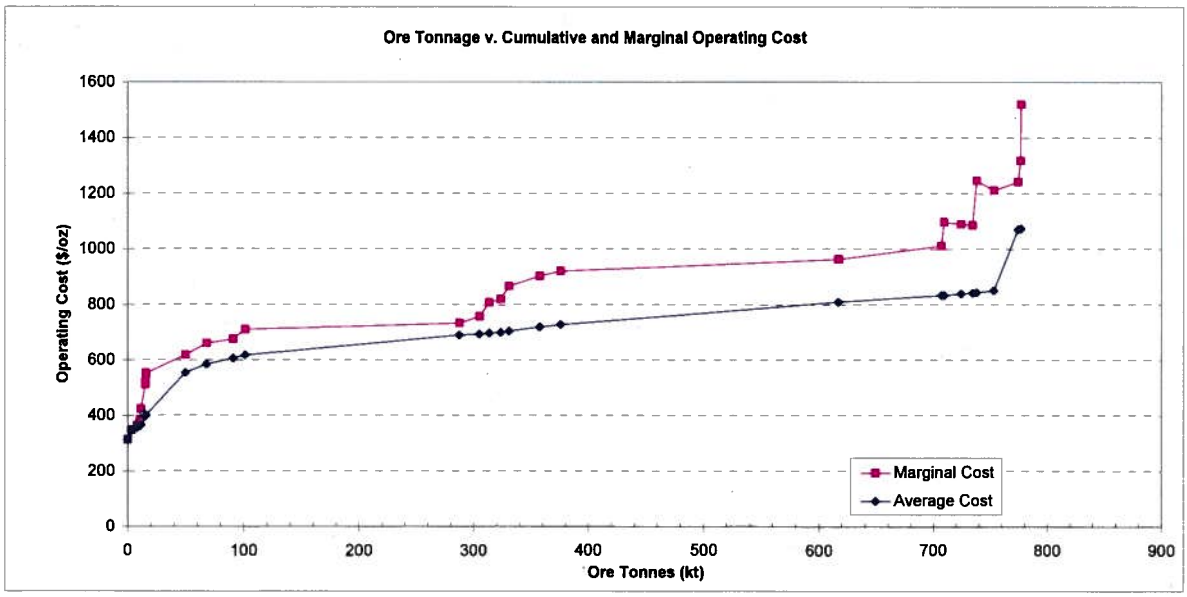




Davis Pit Extension Optimisation Results - Run01c3
$650/oz, Gold Price, 120% Mining Costs

	Physicals								Economics							
Pit No	Ore Total kt	Ore Au g/t	Waste kt	Total kt	W/O Ratio t/t	Contn'd Ounces Au	Recv'd Ounces Au	Recv. %	Mining Cost $'000	Th'put Cost $'000	Reven. $'000	Undisc'd C'Flow $'000	Th'put Cost $/t	Min. Cost $/t	Unit Cost $/oz	Marg. Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	1	3	6	3	19.19	3.05	313	313
2	3	2.47	3	6	0.97	246	221	90%	17	59	143	67	19.23	2.74	345	346
3	8	2.33	5	13	0.61	610	549	90%	38	157	356	161	19.23	2.90	355	362
4	11	2.28	6	17	0.59	783	705	90%	49	205	456	202	19.23	2.93	361	383
5	12	2.30	9	20	0.77	851	766	90%	59	221	496	216	19.23	2.89	366	422
6	15	2.20	16	31	1.03	1,072	965	90%	90	292	625	243	19.23	2.94	396	511
7	15	2.19	16	32	1.04	1,089	980	90%	93	297	635	245	19.23	2.94	398	522
8	16	2.17	16	32	1.02	1,108	997	90%	94	305	646	247	19.23	2.94	401	551
9	50	2.36	264	313	5.30	3,777	3,399	90%	924	957	2,202	321	19.23	2.95	553	617
10	68	2.41	422	490	6.20	5,263	4,737	90%	1,455	1,307	3,069	306	19.23	2.97	583	659
11	91	2.38	630	721	6.94	6,944	6,249	90%	2,038	1,745	4,049	265	19.23	2.83	605	675
12	101	2.40	737	838	7.30	7,796	7,017	90%	2,384	1,943	4,546	218	19.23	2.84	617	709
13	288	2.28	2,275	2,563	7.91	21,116	19,004	90%	7,582	5,527	12,313	-796	19.21	2.96	690	733
14	305	2.28	2,406	2,712	7.88	22,332	20,099	90%	8,073	5,864	13,022	-915	19.21	2.98	693	756
15	314	2.27	2,485	2,798	7.92	22,930	20,637	90%	8,345	6,026	13,370	-1,000	19.21	2.98	696	806
16	324	2.26	2,545	2,868	7.86	23,497	21,148	90%	8,570	6,219	13,701	-1,088	19.21	2.99	699	819
17	331	2.26	2,628	2,959	7.94	24,062	21,656	90%	8,870	6,359	14,030	-1,198	19.21	3.00	703	865
18	358	2.27	2,934	3,292	8.21	26,043	23,438	90%	9,967	6,870	15,185	-1,652	19.21	3.03	718	902
19	376	2.25	3,096	3,472	8.24	27,186	24,468	90%	10,564	7,219	15,852	-1,931	19.21	3.04	727	919
20	617	2.11	5,232	5,849	8.48	41,827	37,645	90%	18,602	11,851	24,389	-6,063	19.20	3.18	809	962
22	707	2.09	6,101	6,808	8.63	47,382	42,643	90%	21,938	13,566	27,628	-7,876	19.20	3.22	833	1010
23	709	2.08	6,115	6,824	8.63	47,479	42,730	90%	21,989	13,610	27,684	-7,915	19.20	3.22	833	1095
24	724	2.08	6,310	7,034	8.72	48,497	43,648	90%	22,701	13,898	28,279	-8,320	19.20	3.23	839	1089
25	734	2.08	6,386	7,120	8.70	49,017	44,116	90%	23,016	14,092	28,582	-8,525	19.20	3.23	841	1086
26	738	2.08	6,427	7,164	8.71	49,225	44,302	90%	23,177	14,161	28,703	-8,636	19.20	3.24	843	1246
27	753	2.08	6,651	7,404	8.83	50,291	45,263	90%	24,044	14,458	29,325	-9,177	19.20	3.25	851	1211
28	774	2.06	6,860	7,635	8.86	51,404	46,263	90%	24,878	14,867	39,764	-9,771	19.20	3.26	1071	1242
29	776	2.06	6,880	7,656	8.86	51,494	46,345	90%	24,952	14,900	39,871	-9,826	19.20	3.26	1072	1317
30	777	2.06	6,890	7,666	8.87	51,526	46,373	90%	24,984	14,911	39,915	-9,851	19.20	3.26	1073	1521





Davis Pit Extension Optimisation Results - Run01d3
$700/oz, Gold Price, 120% Mining Costs

	Physicals								Economics							
	Ore		Waste	Total	W/O	Contn'd	Recv'd	Recv.	Mining	Th'put	Reven.	Undisc'd	Th'put	Min.	Unit	Marg.
Pit No	Total kt	Au g/t	kt	kt	Ratio t/t	Ounces Au	Ounces Au	%	Cost $'000	Cost $'000	$'000	C'Flow $'000	Cost $/t	Cost $/t	Cost $/oz	Cost $/oz
1	0	2.57	0	0	0.34	11	10	90%	1	3	7	4	19.19	3.05	313	313
2	3	2.47	3	6	0.97	246	221	90%	17	59	154	78	19.23	2.74	345	346
3	8	2.33	5	13	0.61	610	549	90%	38	157	383	188	19.23	2.90	355	362
4	11	2.28	6	17	0.59	783	705	90%	49	205	492	237	19.23	2.93	361	383
5	12	2.30	9	20	0.77	851	766	90%	59	221	535	254	19.23	2.89	366	422
6	15	2.20	16	31	1.03	1,072	965	90%	90	292	673	291	19.23	2.94	396	511
7	15	2.19	16	32	1.04	1,089	980	90%	93	297	684	294	19.23	2.94	398	522
8	16	2.17	16	32	1.02	1,108	997	90%	94	305	696	297	19.23	2.94	401	551
9	52	2.30	261	313	5.00	3,855	3,469	90%	924	1,004	2,421	493	19.23	2.95	556	619
10	71	2.34	418	490	5.87	5,366	4,830	90%	1,455	1,370	3,371	546	19.23	2.97	585	659
11	94	2.33	627	721	6.67	7,047	6,342	90%	2,038	1,808	4,427	581	19.23	2.83	606	675
12	105	2.34	733	838	6.97	7,928	7,135	90%	2,384	2,023	4,980	573	19.23	2.84	618	708
13	294	2.25	2,268	2,563	7.70	21,332	19,198	90%	7,582	5,657	13,400	160	19.21	2.96	690	732
14	312	2.25	2,400	2,712	7.69	22,548	20,293	90%	8,073	5,995	14,164	96	19.21	2.98	693	756
15	321	2.24	2,477	2,798	7.71	23,179	20,861	90%	8,345	6,177	14,560	39	19.21	2.98	696	799
16	332	2.23	2,537	2,868	7.65	23,746	21,372	90%	8,570	6,370	14,917	-23	19.21	2.99	699	819
17	339	2.23	2,621	2,959	7.73	24,311	21,880	90%	8,870	6,509	15,271	-108	19.21	3.00	703	865
18	366	2.24	2,926	3,292	8.00	26,306	23,675	90%	9,967	7,029	16,524	-472	19.21	3.03	718	901
19	385	2.22	3,087	3,472	8.03	27,462	24,716	90%	10,564	7,387	17,251	-700	19.21	3.04	726	917
20	639	2.07	5,210	5,849	8.15	42,522	38,270	90%	18,602	12,272	26,711	-4,163	19.20	3.18	807	953
22	732	2.05	6,076	6,808	8.30	48,176	43,357	90%	21,938	14,047	30,262	-5,723	19.20	3.22	830	1005
23	735	2.04	6,089	6,824	8.29	48,296	43,466	90%	21,989	14,106	30,338	-5,758	19.20	3.22	830	1015
24	750	2.05	6,284	7,034	8.38	49,326	44,393	90%	22,701	14,401	30,985	-6,117	19.20	3.23	836	1085
25	760	2.04	6,359	7,120	8.37	49,846	44,861	90%	23,016	14,594	31,312	-6,298	19.20	3.23	838	1086
26	765	2.04	6,400	7,164	8.37	50,074	45,068	90%	23,177	14,678	31,456	-6,400	19.20	3.24	840	1189
27	781	2.04	6,624	7,404	8.48	51,166	46,050	90%	24,044	14,989	32,141	-6,892	19.20	3.25	848	1200
28	804	2.03	6,831	7,635	8.50	52,325	47,093	90%	24,878	15,426	40,324	-7,435	19.20	3.26	1014	1218
29	805	2.02	6,851	7,656	8.51	52,415	47,174	90%	24,952	15,460	40,431	-7,486	19.20	3.26	1016	1318
30	806	2.02	6,860	7,666	8.51	52,464	47,218	90%	24,984	15,481	40,484	-7,509	19.20	3.26	1016	1224