PERSEUS MINING LIMITED

SISSINGUÉ GOLD PROJECT
CÔTE D'IVOIRE

NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

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SISSINGUÉ GOLD PROJECT

FEASIBILITY STUDY

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SISSINGUÉ GOLD PROJECT

FEASIBILITY STUDY

NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

1893-000-GEREP-0002

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1.0 SUMMARY

1.1 Introduction

In October 2014 Perseus Mining Limited (Perseus) engaged Lycopodium Minerals Pty Ltd (Lycopodium) to coordinate a Definitive Feasibility Study (DFS) for their Sissingué Gold Project (the Project), located in northern Côte d'Ivoire.

The DFS drew on previously published reports and included new contributions from resources within Perseus and external consulting groups. This Form 43-101 Technical report reflects the outcomes of that study and provides a summary of the content of the Sissingué Gold Project Definitive Feasibility Study report 1893-000-GEREP-0001 of May 2015.

1.2 Project Location and Tenure

The Sissingué Gold Project, formerly known as the Tengrela Gold Project, is located in northern Côte d'Ivoire, adjacent to the Mali border and approximately 620 km north of the commercial capital, Abidjan (Figure 1.2.1). The nearest town is Tengrela, located approximately 15 km to the west of the concession boundaries. Korhogo, the principal regional centre, is located 150 km to the southeast. The Project area is centred at National Grid reference 1,141,666N and 804,166E.

Perseus acquired its interest in the Project by agreements dated March 2004.

The Tengrela East Permit was last renewed on April 21, 2009 and expired on November 19, 2011. In replacement of the Tengrela East exploration permit, Perseus's Ivorian subsidiary Occidental Gold Sarl (‘OGIC’) was granted exploitation permit PE 39 (the 'Exploitation Permit') for the development of the Sissingué gold deposit in August 2012. The Exploitation Permit was subsequently transferred to another Ivorian subsidiary company, Perseus Mining Côte d'Ivoire SA ('PMCI') in July 2013.

As a result of the grant of the Exploitation Permit for the Project, the Government of Côte d'Ivoire has a statutory 10% free carried interest in the Project and OGIC's interest has reduced to 85%. This is reflected in the shareholding of PMCI as follows:

- Perseus (through OGPL): 85%
- The State of Côte d'Ivoire (held by SOMICI and to be transferred to the State): 10%
- SOMICI: 5%

SOMICI is Société Minière De Côte d'Ivoire SARL

Occidental Gold Pty Ltd (‘OGPL’ is an Australian registered company and a wholly owned subsidiary of Perseus Mining Limited.

Under the conditions of the Exploitation Permit, PMCI had to commence production from the Permit by no later than 8 August 2014. By order dated 17 March 2014, the Ivorian Government has granted an extension to commence production until 16 March 2016.

Figure 1.2.1 General Location of the Sissingué Gold Project in Côte d'Ivoire



Sissingué is currently the main focus of development at the Project, however several other mineral occurrences have been identified in the Project area. All known zones of mineralisation and Mineral Resources have been located accurately and there is no risk of any of the known Mineral Resources being outside the permit boundaries due to survey errors.

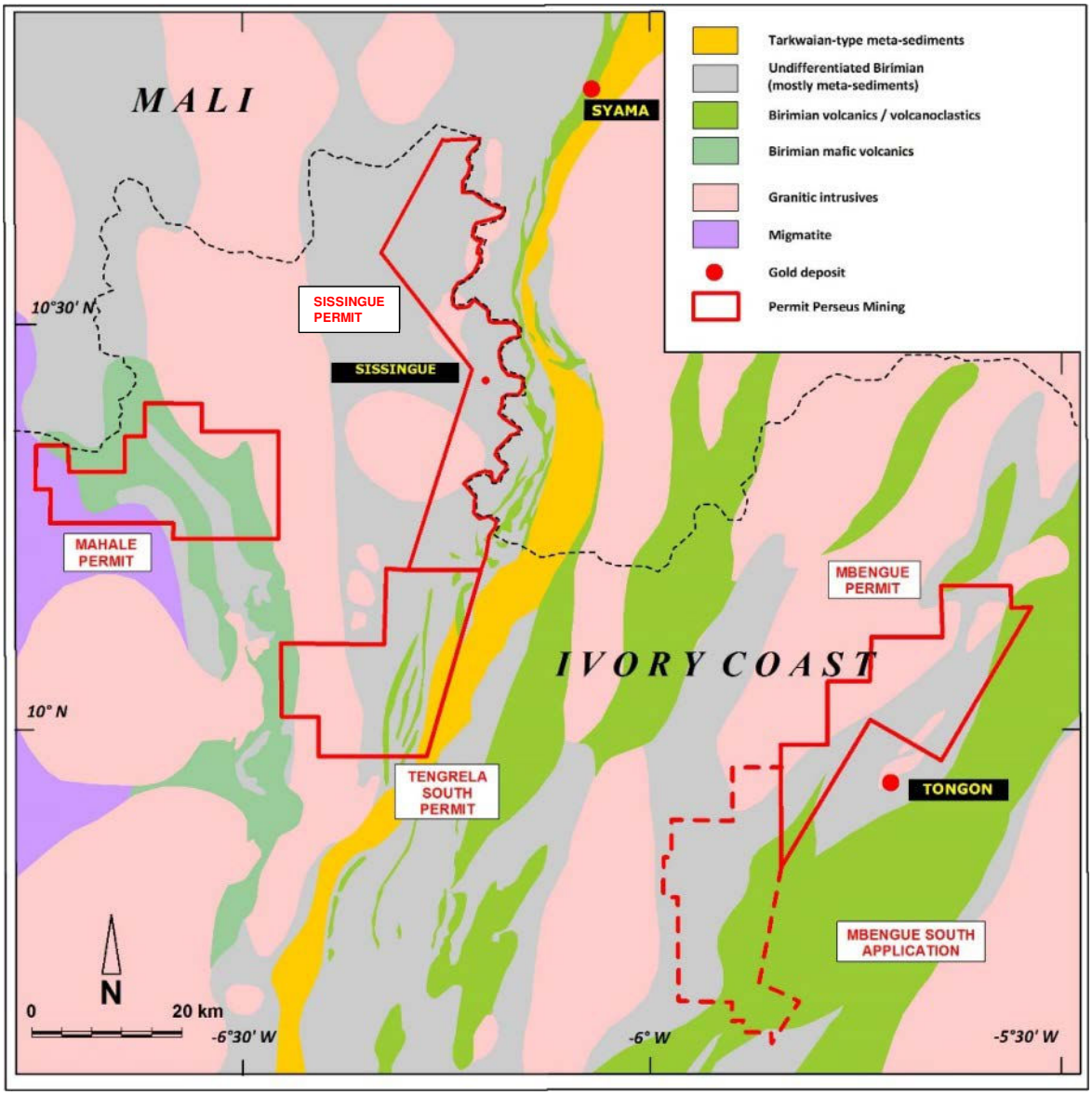
1.3 Geology and Resources

1.3.1 Regional and Local Geology and Mineralisation

The Sissingué deposit is situated in the Syama-Boundiali Greenstone Belt. The geology of the belt has similarities to the better explored Ashanti Belt in neighbouring Ghana, where Birimian volcanoclastics dominate over competent volcanics, with the development of inner belt basins filled with Tarkwaian epiclastics. In the Bagoe River region near Tengrela, the terrain is comprised of undifferentiated granitoids, flysch sediments, intermediate volcanic and small occurrences of mafic intrusive and molasse sediments.

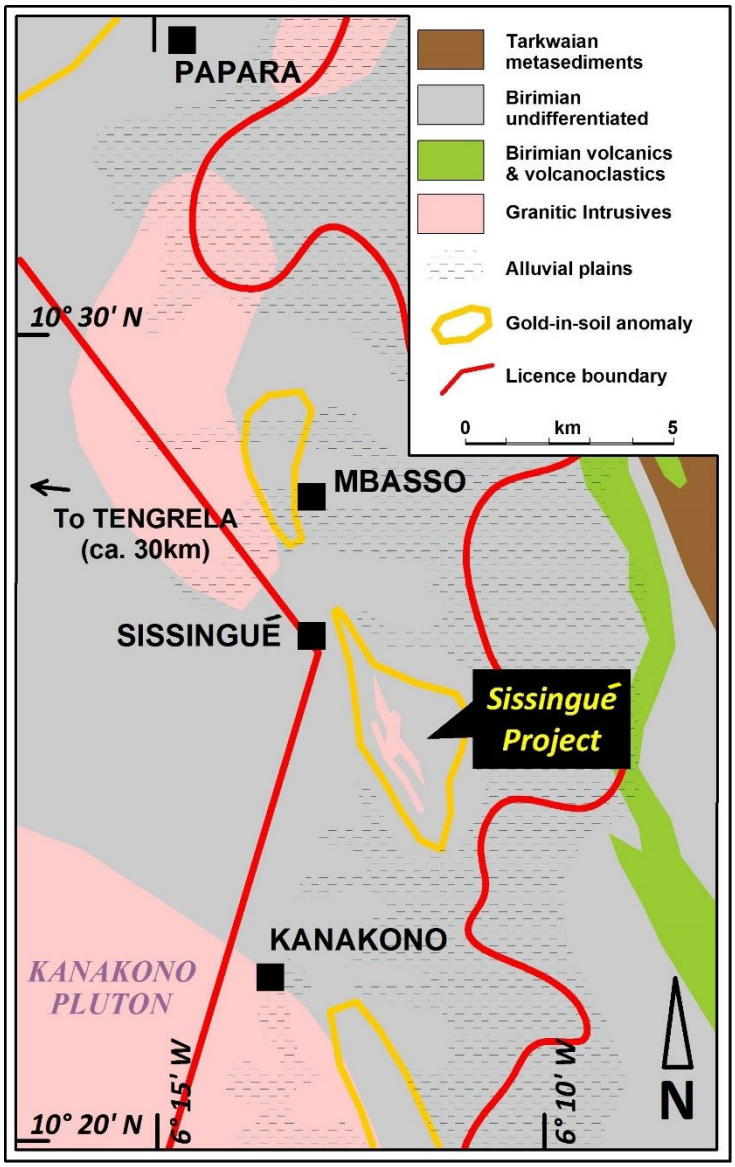
The Sissingué permit area covers basin sediments west of the strongly deformed Birimian Greenstone Belt, intruded by granitoid bodies (Figure 1.3.1). Interpretation of both aeromagnetics and radiometric data indicates the inclusion of an extensive sequence of north trending Birimian volcanoclastics, intruded by a large felsic body (Kanakono Pluton) as well as several small uranium-rich felsic intrusions.

Figure 1.3.1 Geology of Northern Côte d'Ivoire (Image Supplied by Perseus)



The Sissingué deposit is defined by a 4 km long and up to 1.5 km wide gold-in-soil anomaly situated 5 – 6 km west of the Syama-Boundiali Greenstone Belt (Figure 1.3.2). Rocks encountered in outcrops and drilling comprise predominantly north-north east striking, steeply west dipping and isoclinally folded sediments (sandstones, mudstones and subordinate conglomerates) of the Birimian Supergroup, interpreted as units of turbiditic flows. These sediments are cross cut by a swarm of narrow porphyritic dykes (sub-metre to several metres thick) which trend obliquely to the sedimentary package north-west with sub-vertical or steep to moderate dips towards the north-east. Some minor dykes appear to have exploited weak zones parallel to sedimentary bedding.

Figure 1.3.2 Sissingué Property Geology (Image Supplied by Perseus)



Gold mineralisation is found as three distinct styles as summarised below:

- Disseminated or fracture related mineralisation in the felsic intrusive closely associated sericite-carbonate alteration, silification and a pyrite and arsenopyrite mineralisation. The average gold tenor of this type of mineralisation is usually relatively high (2 g/t Au to 5 g/t Au).

- Significant low grade gold mineralisation (commonly below 2 g/t Au) is encountered within sediments immediately adjacent to the intrusive bodies within their respective alteration halos. The gold tenor are perceived to be closely correlated to alteration intensity and hence with primary porosity of the rocks. Sandstones and conglomerate are usually better mineralised than fine grain mud and siltstones.

- High grade mineralisation (with grades exceeding 1,000 g/t Au) is associated with quartz-veins which are encountered in the upper portion of the intrusive bodies and in the altered and brittle portion of sediments directly above. Visible gold is common.

It appears that weathering has an effect on the gold distribution. Boundary plots and sample histograms for the various weathering domains suggest that gold has been depleted in weathered (oxidised) material. Descending waters resulted in gold enrichment in the transition zone as well as in nugget growth where existing nuggets have been freed from country rock during weathering processes.

1.3.2 Exploration Overview

Previous extensive systematic exploration on the Project, including airborne geophysics and extensive surface geochemical sampling, identified a series of soil anomalies over a strike length of 75 km, mostly along the Syama-Shear. The Morila-Tongon region has a historically high conversion rate of significant soil anomalies to gold discoveries.

Past exploration activities have been carried out by Randgold Resources Ltd, and OGIC. A summary of this work is included in Section 6 of this report. Since 2004 all the exploration activities have been carried out by OGIC, as a wholly-owned subsidiary of Perseus.

An airborne geophysical survey was carried out by HI Sense in December 1998, and flown at 75 m above ground level on east-west oriented lines. The digital airborne magnetics, radiometric and VLF database were re-interpreted by Southern Geoscience Consultants in Perth, Australia. From this, detailed structural and geological interpretations of the area were prepared and used to assist with exploration planning.

RAB drilling commenced in September 2002, however, the programme was disrupted by the outbreak of the civil war. Exploration activities and RAB drilling resumed in 2005. Since 2007, extensive RC and diamond core drilling has been carried out over the Sissingue prospect. Contract drillers Ivorian-based Foramin SARL (formerly Drillex Ltd) and Boart Longyear were used for the programs.

Between March and May 2008, an airborne VTEM/Magnetics survey was flown by Geotech Airborne Ltd, in a joint venture with Resolute Mining Ltd and Etruscan Resources Inc, both operating on the other side of the Bagoe River in Mali.

In 2008, Sagax Afrique was contracted for a ground IP survey over the Sissingue deposit, including processing and imaging of the data. Coffey Mining Pty Ltd in Notting Hill, Australia, was contracted to carry out a structural study during a site visit that resulted in the current geological model of the Sissingué ore body. Resource definition drilling at Sissingué was completed in mid-2012.

Since 2010, the focus of exploration has gradually shifted towards other prospects close to Sissingue (e.g. Papara, Kanakono, and Mbasso) in an effort to add resources to a potential mine.

In 2012, consulting geophysicist Paolo Constantini was contacted to reprocess the existing ground IP data and to reinterpret the available geophysical data set for regional target identification.

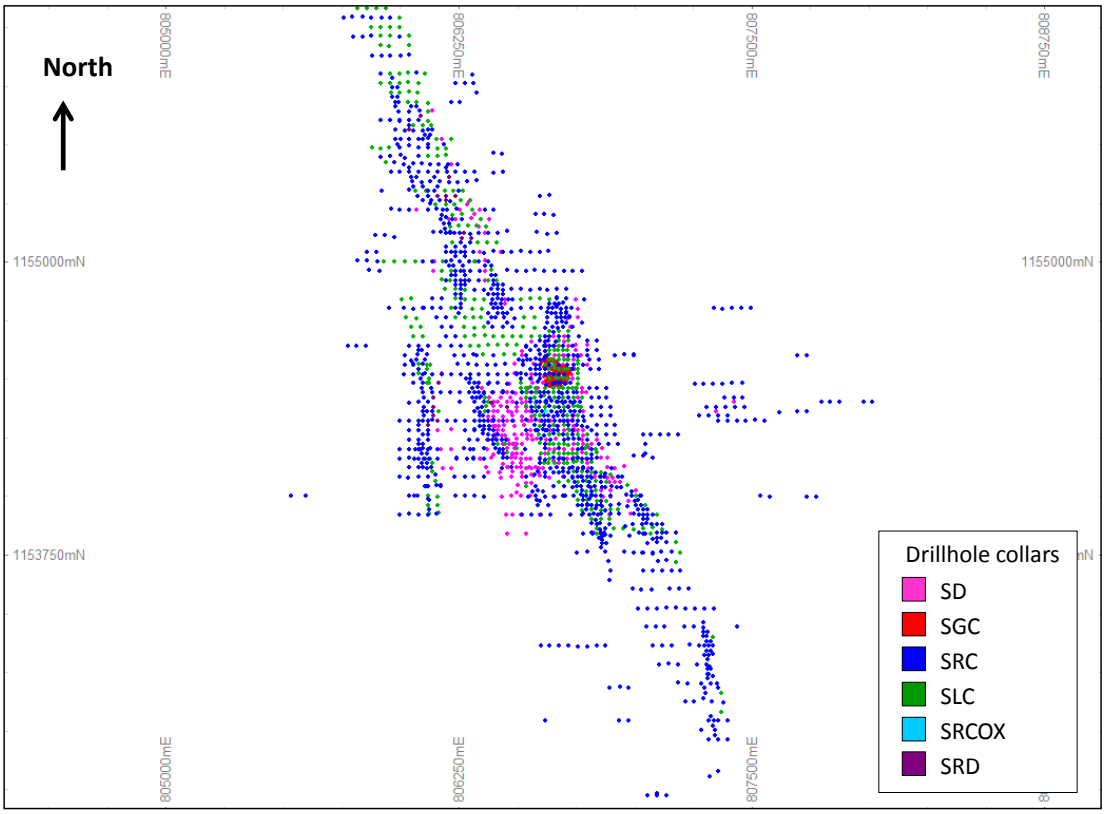
For the Mineral Resource estimate only samples from RC and diamond core drilling, carried out by Perseus since 2005, have been used. Exploration results by previous owners are not included.

1.3.3 Drillhole Data

The input dataset used for the Sissingué resource estimate contained 69,728 m RC and 62,019 m DD samples for 201,551 m of which 36,113 m lie within the mineralised domains.

The total dataset within the Sissingué Mineral Resource area is illustrated in plan view Figure 1.3.3.

Figure 1.3.3 Plan View of Sissingué Drillhole Data

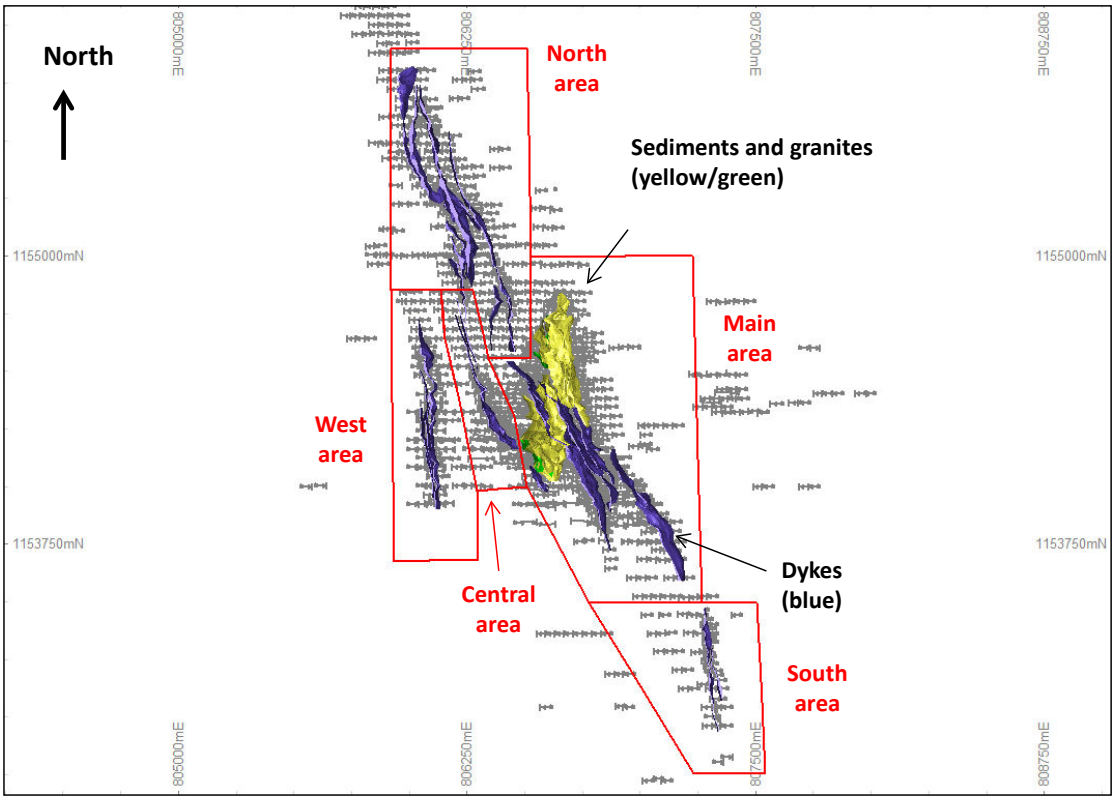


1.3.4 Geology and Mineralisation Domains

Mineralisation was domained by its host lithology for modelling (granite, porphyritic dykes and sediments). Within the dyke domain mineralised sediments in the alteration halo of the dykes were included in the dyke domain to maintain a minimum width of the wireframes and to maintain continuity along strike. Where geological contacts were not clearly controlling the distribution of mineralisation, a grade cut-off of 0.3 g/t Au was used to construct Mineral Resource boundaries and to provide overall geometry to mineralised zones. A minimum of 4 m width was used for the wireframes and samples of grades below the nominal cut-off of 0.3 g/t Au were included where the wireframe would otherwise be less than 4 m wide. Analysis of the global grade distribution shows that there is a natural change in grade population at around 0.3 g/t Au.

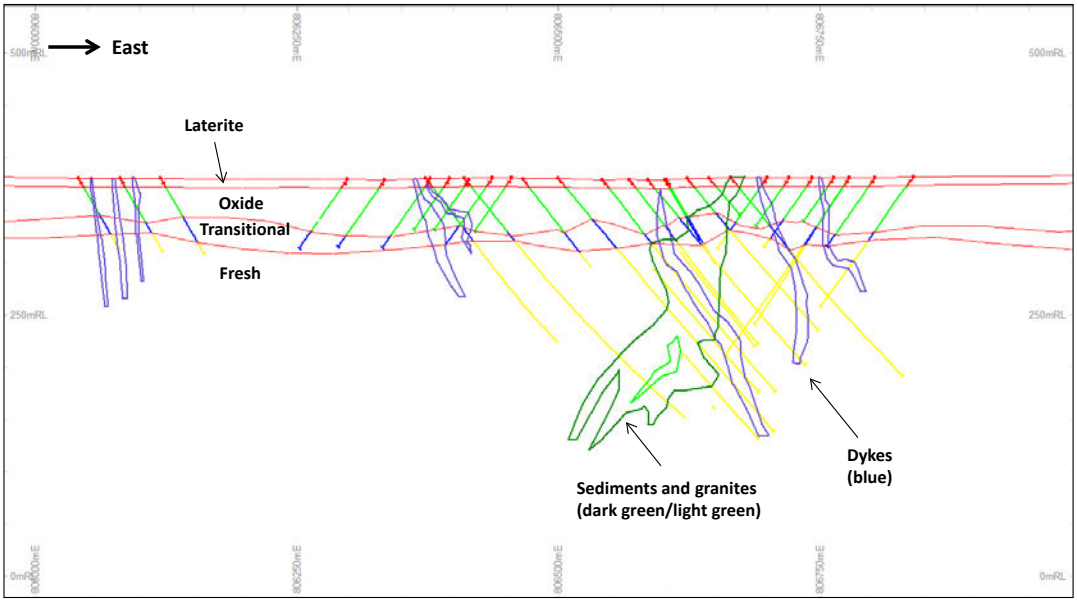
A plan view of the major modelling domains is shown in Figure 1.3.4.

Figure 1.3.4 Plan View of Mineralisation Domains with Drillholes and Project Areas



A topographic surface and three surfaces of the weathering profile were created by Perseus who supplied the wireframes to Snowden Mining Industry Consultants. A cross section illustrating the topography and the four weathering surfaces is shown in Figure 1.3.5. Review of the oxide, transitional and fresh weathering domains within the granites, sediments and dykes indicates that the grade distributions have similar shapes, with typically higher grades in the transitional domain.

Figure 1.3.5 Cross Section Showing Topography and Weathering Surfaces



1.3.5 Mineral Resource Classification and Reporting

The Sissingue Mineral Resource has been classified in the Measured, Indicated and Inferred categories. A range of criteria has been considered in determining this classification including:

- Geological continuity
- Data quality
- Drill hole spacing
- Modeling technique
- Estimation properties including search strategy, number of informing data and average distance of data from blocks

Trial optimisation has been run at a USD2400 gold price to define the base of potentially economic material and the vertical limit of the Mineral Resource.

Table 1.3.1 summarises the resources by category and area reported above a 0.6 g/t Au cut-off grade. The classification categories of Inferred, Indicated and Measured are in accordance with the JORC Code (2012). They are equivalent to the CIM categories of the same name (CIM, 2010).

Table 1.3.1 October 2014 Sissingué Mineral Resource Reported at a 0.6 g/t Au Cut-off

Category	Area	Tonnage (kt)	Grade (g/t Au)	Contained Gold (oz)
Measured	Oxide	1,000	1.8	59,000
	Transitional	650	2.3	49,000
	Fresh	3,200	2.5	260,000
Total Measured		**4,800**	**2.4**	**370,000**
Indicated	Oxide	3,100	1.3	130,000
	Transitional	800	1.5	38,000
	Fresh	7,100	1.5	350,000
Total Indicated		**11,000**	**1.4**	**510,000**
Measured + Indicated	Oxide	4,100	1.4	190,000
	Transitional	1,400	1.9	87,000
	Fresh	10,000	1.8	600,000
Total Measured + Indicated		**16,000**	**1.7**	**880,000**
Inferred	Oxide	310	1.2	12,000
	Transitional	54	1.2	2,100
	Fresh	760	2.0	49,000
Total Inferred		**1,100**	**1.7**	**63,000**

Notes: Mineral Resources are inclusive of Mineral Reserves. Mineral Resources are reported to two significant figures. Rounding may cause minor discrepancies in the table. Oxide includes small portions of laterite (571 kt total).

1.4 Mining and Mine Reserves

A mining study was completed by RungePincockMinaro (RPM) in close liaison with Perseus personnel.

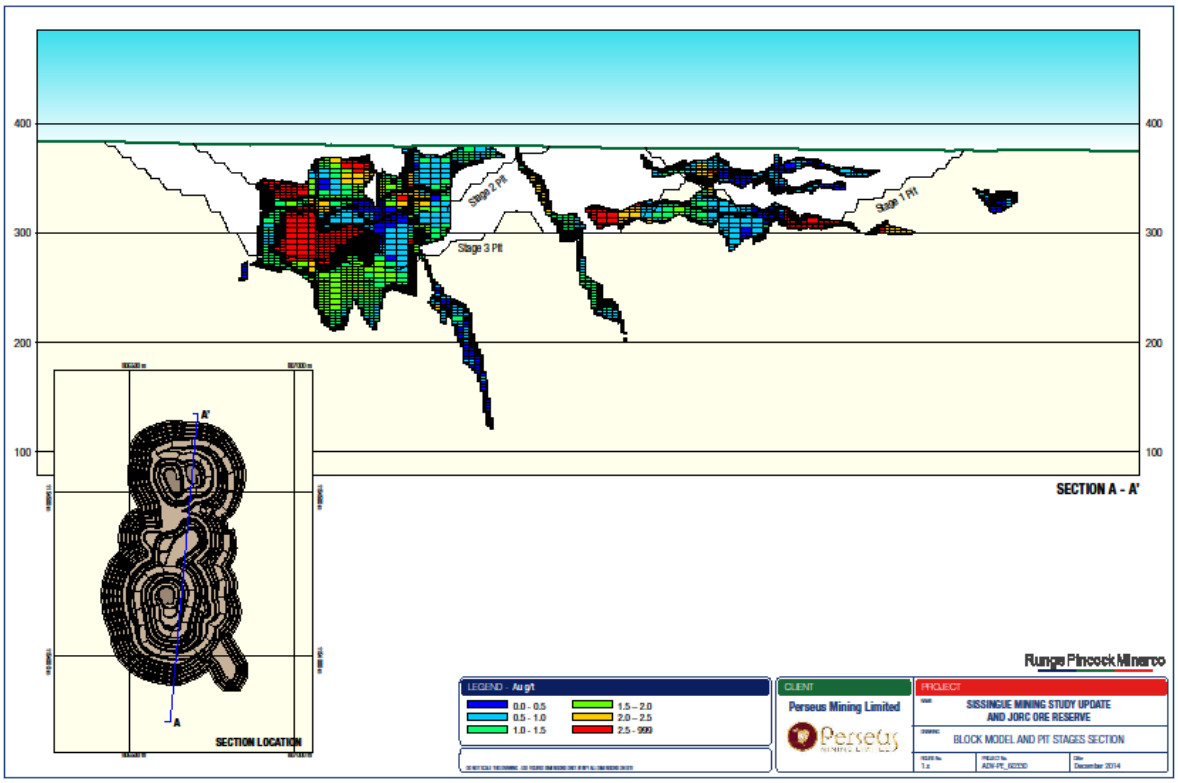
The economic pit limits were determined by using the Whittle 4X pit limit optimisation software ('Whittle 4X Optimiser'). The Whittle 4X optimisation was based on the Surpac format block model prepared by Snowden Mining Industry Consultants, geotechnical parameters estimated by Coffey Mining and metallurgical recovery, operating costs and metal price supplied by Perseus. The Base Case metal price was US$1,200/tr oz.

1.4.1 Minable Quantities

The mining method planned for the operation is conventional open pit excavation with ex-pit waste dumping. The pit will be accessed by a 20 m wide haul road incorporating a 10 percent gradient and two lane ramps. At the pit floor the final benches will be accessed using single lane ramps.

Based on the established pit design parameters and pit optimisation three detailed phase designs were prepared. The three mining phases are shown in section view in Figure 1.4.1.

Figure 1.4.1 Cross Section Through Pit Phases



The ramp exit points for each design were placed on the east to provide the shortest hauls to the ROM pad and waste dump. The waste rock dump was located to the east of the pit and adjacent to the processing plant.

The resulting ore and waste mineable quantities for the three phase designs are presented in Table 1.4.1.

Table 1.4.1 Mineable Quantities

Pit Phase	Weathering Type	Waste Mt	Ore Mt	Strip Ratio t/t	Ore Grade Au g/t
1	Oxide	3.3	0.5	6.2	2.1
	Transition	0.4	0.2	1.8	2.7
	Fresh	0.3	0.4	0.6	2.1
Phase 1 Total		**3.9**	**1.2**	**3.3**	**2.2**
2	Oxide	3.2	0.8	4.2	1.4
	Transition	0.3	0.3	1.1	1.7
	Fresh	0.1	0.5	0.3	3.6
Phase 2 Total		**3.6**	**1.5**	**2.4**	**2.1**
3	Oxide	6.1	0.7	8.3	1.4
	Transition	1.6	0.3	5.2	2.4
	Fresh	2.5	1.8	1.4	3.1
Phase 3 Total		**10.2**	**2.8**	**3.6**	**2.6**
Total of All Phases		**17.7**	**5.5**	**3.2**	**2.4**

1.4.2 Mining Costs

RPM prepared a Mining Specific Cost Model within its XERAS financial modelling software for a Contractor Mining scenario.

Mining costs were estimated based on a quotation from DTP Terrassement ('DTP') (the locally experienced earthworks division of a French contractor Bouygues Construction Group) and by RPM for the Owner's component, including site based management, supervision and technical support, with input from Perseus.

LOM mining cost, following adjustment of the DTP estimate based on the final mining schedule, and after addition of fuel costs, are shown in Table 1.4.2 below.

Table 1.4.2 Adjusted Contractor Estimate

RPM Contractor Estimate						
Item	**Item Description**	**Units**	**Driver**	**Rate**	**Units**	**Total US$M**
1	Mobilisation, Site Est & Demob	No.	1	3,609	US$k	3.6
2	Fixed Monthly Fee	Months	65	512	US$k	33.3
3	Area Preparation	km^2	796	0.88	US$	0.7
4	Drilling	kdrm	172	22.40	US$	3.8
5	Blasting	kholes	21.8	173	US$	3.8
6	Excavate, Load, Haul and Dump	kbcm	10,872	3.63	US$	39.5
7	Stockpile Rehandle	kt	5,525	0.60	US$	3.3
8	Dewatering	Months	65	32.19	US$k	2.1
9	Grade Control	ore kt	5,525	0.24	US$	1.3
Total						**91.4**

Over the life of the Project, this equates to an average cost of $16.55 per ore tonne or $8.42/bcm or $3.95/tonne rock. The 'fixed monthly fee' and the 'excavate, load and haul' activity account for 80% of the total cost.

1.4.3 Mineral Reserves

Mineral Reserves were estimated by RungePincockMinarco Ltd (RPM). As shown in Table 1.4.3, a total of 5.5 Mt of Open Cut Mineral Reserves at 2.4 g/t gold were estimated as at 1 February 2015.

Table 1.4.3 February 2015 Open Pit Ore Reserve

Classification	Tonnes dry Mt	Au Grade g/t	Gold koz
Proved	3.4	2.8	312
Probable	2.1	1.7	115
Proved + Probable	**5.5**	**2.4**	**429**

The estimates have been rounded to two significant figures to reflect accuracy.

The previously published Mineral Reserve as at 22 December 2010 was 9.7 Mt at 2.1 g/t for 657 koz. The major changes since the previous Ore Reserve are the updates to the processing recoveries, cost and to a reduction in the annual throughput rate of ore.

1.4.4 Production Schedule

A detailed life of mine production schedule was generated for the Project. The entire life of mine was scheduled in monthly periods. Mill production begins in period six and ramps up from 60% capacity to 100% capacity over a period of three months.

Figure 1.4.2 shows the total monthly material movement from the pit by phase while Figure 1.4.3 shows the monthly processing plant ore tonnage by ore type.

Figure 1.4.2 Total Monthly Material ex Pit by Phase

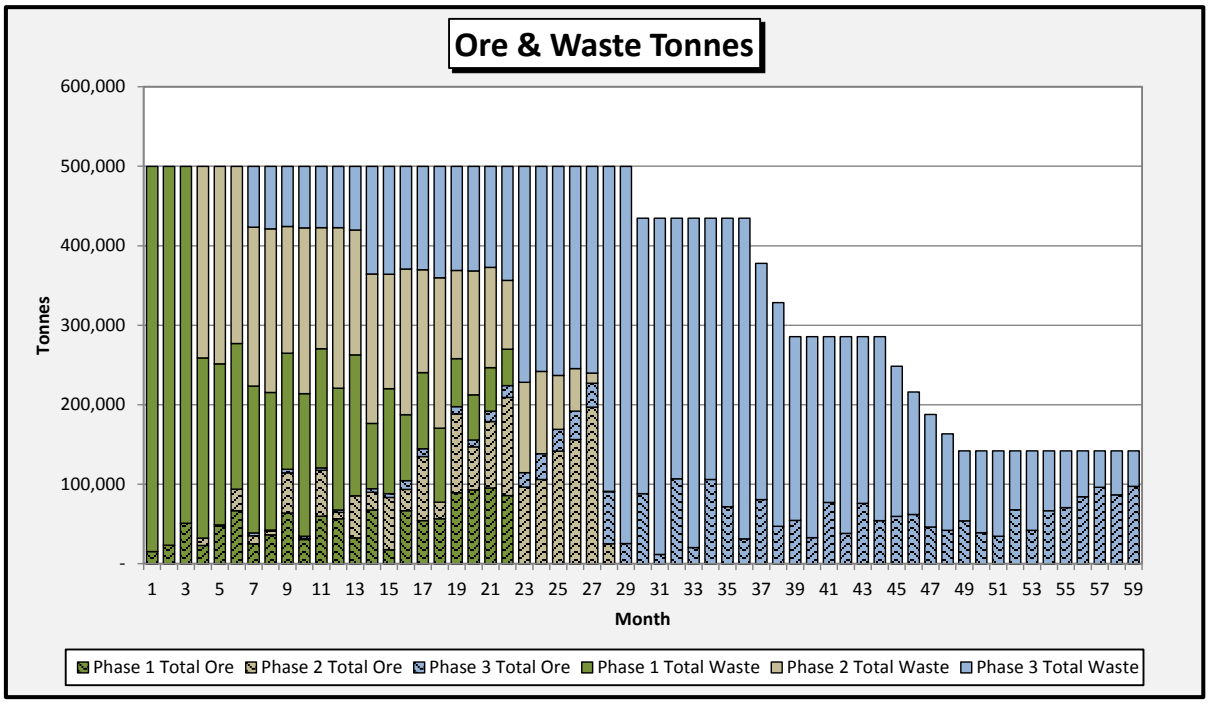


Figure 1.4.3 Monthly Processing Plant Ore Tonnage by Ore Type

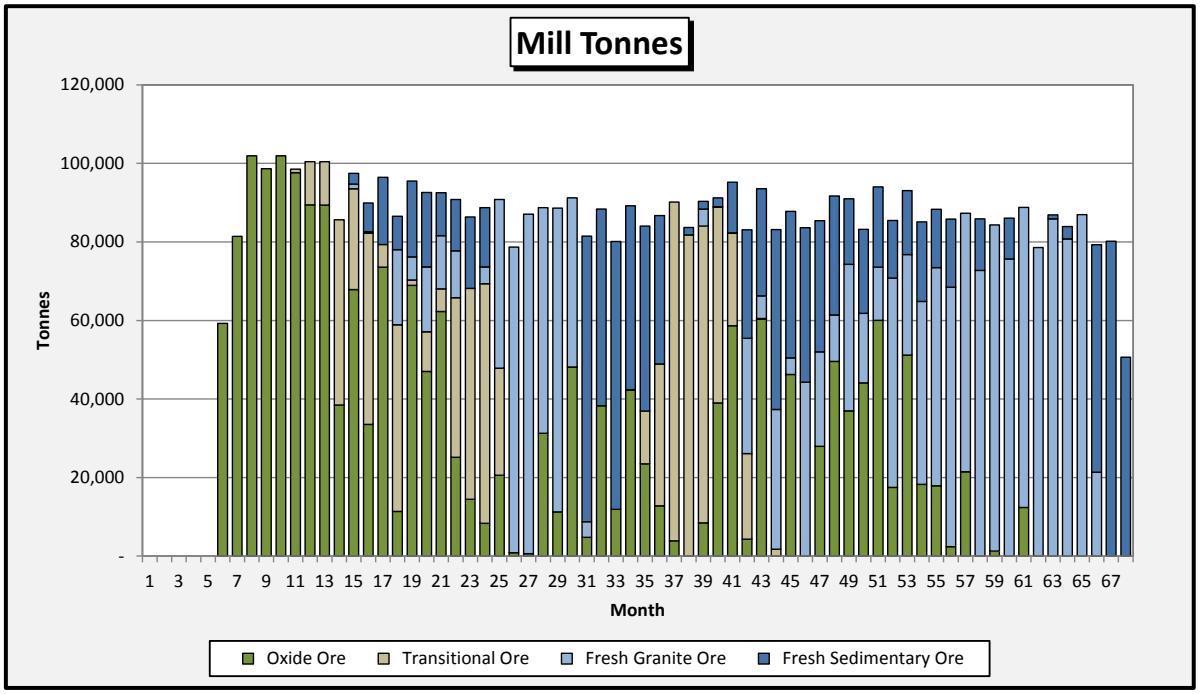
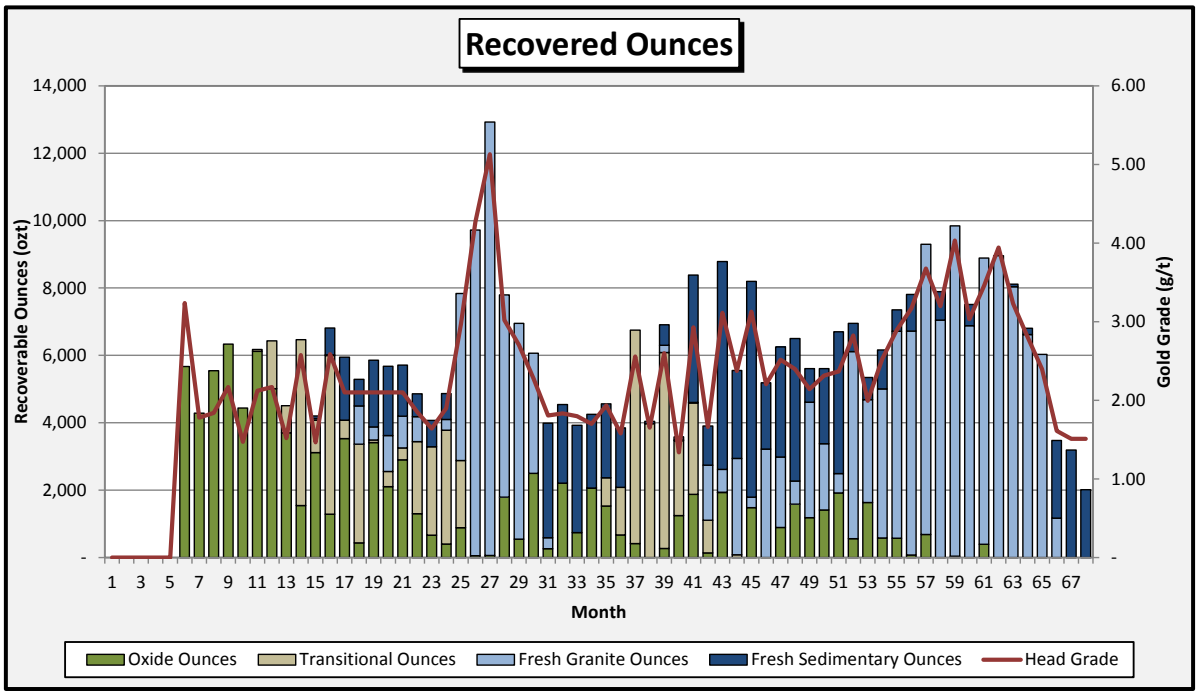


Figure 1.4.4 shows the recovered gold by month.

Figure 1.4.4 Recovered Gold by Month



1.5 Metallurgy and Process Plant

1.5.1 Metallurgy

Between 2010 and 2012 a series of three metallurgical testwork programmes have been completed to support the feasibility study of the Sissingué Gold Project.

Conclusions from the test work programs on the Sissingué samples are as follows:

- Sissingué samples are predominantly 'free-milling', non–'preg-robbing' and are amenable to gold extraction by conventional cyanidation. Some of the samples tested, exhibited a refractory nature, however it is noted that only two of these are located inside the planned pit shell.
- The oxide samples typically yield high recoveries (90% to 98%) at a grind size P_{80} of 106 µm. Leach kinetics are variable with approximately 40% of the samples tested completing leaching with 24 hours, while the remainder exhibit a slow leaching component to 48 hours.
- The granite hosted sulphide samples typically; have high recoveries (87% to 94%) and exhibit a slow leaching component with leach time of 48 hours being required.

- The porphyry hosted sulphide samples responded similarly to granite samples.
- The sediment hosted sulphide samples had variable recoveries ranging from 39.9% to 95.6%. A slow leaching component is present and a leach time of 48 hours is required.
- The oxide and sulphide samples have dissimilar physical and comminution characteristics. The oxide material is soft and non-abrasive, while the sulphide materials are abrasive, hard and competent to very competent. Oxide material will be processed by a two stage crushing circuit, with a third stage of crushing added for processing of the sulphide material.
- Gravity test-work showed variable recovery to (0.7% to 87.4%) of gold by centrifugal gravity concentration and intensive cyanidation of gravity concentrate. A gravity circuit has been included in the flowsheet as visible gold is present in drill core.
- Pre-oxidation is found to be beneficial in leach kinetics and overall recovery, predominantly for sulphide samples.

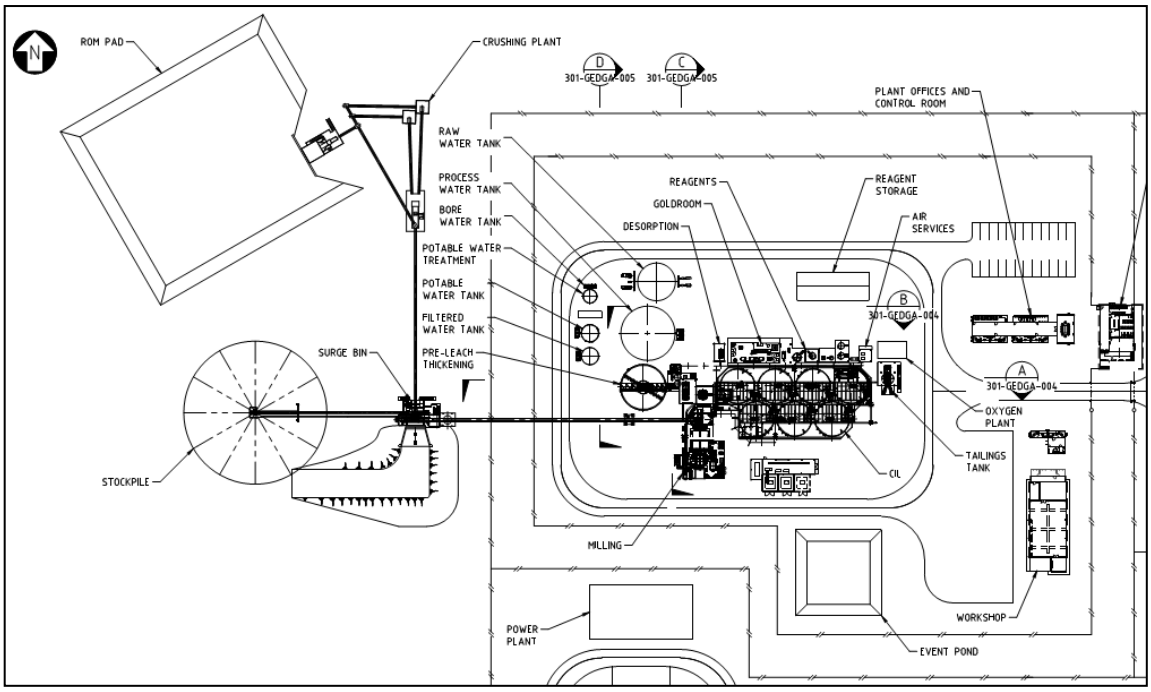
1.5.2 Process Plant

The proposed process plant design for the Sissingué Gold Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is constructed from unit operations that are well proven in industry.

The Sissingué plant will process a range of ore types (oxide, transition and primary porphyry, granite and sediment ores) with variable ore characteristics, gold grades and metallurgical treatment requirements. The primary ores are significantly more competent than the oxide ores and require a longer leach time in order to optimise gold recovery.

The layout of the plant is shown in Figure 1.5.1.

Figure 1.5.1 Plant Layout



Ores will be mined so that predominantly oxide/transitional ore will be processed in two phases; during the first year and then again for a five month period in Year 3. Some blending with primary ores will be required in the final two years. Oxide ore will be processed at a rate of up to 1.2 Mt/y using a two stage crushing and single stage SAG mill circuit. Primary ore will be processed at a rate of up to 1.0 Mt/y using a three stage crushing and ball mill circuit, where the SAG mill has been re-configured to operate as a ball mill.

The key process design criteria used as the basis of the process plant design and equipment selection are shown in Table 1.5.1.

Table 1.5.1 Summary of Key Process Design Criteria

		Oxide	Primary Blend	Source
Crushing Circuit Capacity	t/y	1,200,000	1,000,000	Perseus
Plant Throughput - Design	t/y	1,200,000	800,000	Perseus
Plant Throughput – Nominal Max.	t/y		1,000,000	Perseus
Gold Head Grade	g Au/t	2.50	3.75	Perseus
Design Gold Recovery	%	92	90	Perseus
Crushing Plant Utilisation	%	70.0		Lycopodium
Plant Availability	%	91.3		Lycopodium
Comminution Circuit		2° Crush & SAG	3° Crush & Ball	Perseus / OMC
Crush Size, P_{80}	mm	32	9.5	OMC
Grind Size, P_{80}	µm	106	106	Perseus
Mill Pinion Power	kW	620	1,750	Perseus
Leach /CIL Residence Time	hrs	31	48	Perseus
Leach Slurry Density	% w/w	45	52	Testwork
Number of Pre-Oxidation Tanks		0	1	Perseus
Number of Leach Tanks		1	0	Perseus
Number of Adsorption Tanks		6	6	Perseus
Cyanide Consumption[5]	kg/t	0.51	0.59	Perseus
Quicklime Consumption[6]	kg/t	0.76	0.32	Perseus
Elution Circuit Type		Zadra		Perseus
Elution Circuit Size	t	4		Lycopodium
Frequency of Elution	strips / week	7	4	Lycopodium

It is planned to process primary ore at rates faster than the design rate 0.8 Mt/y selected at the start of the Project. The leach feed density will be increased to compensate for the higher throughput rates and cyanide will be added to the first leach tank if required. The increased mill throughput is achievable because the ball mill was sized to accommodate the hardest ore type, providing additional throughput capacity on the softer blend of ores in the mine plan.

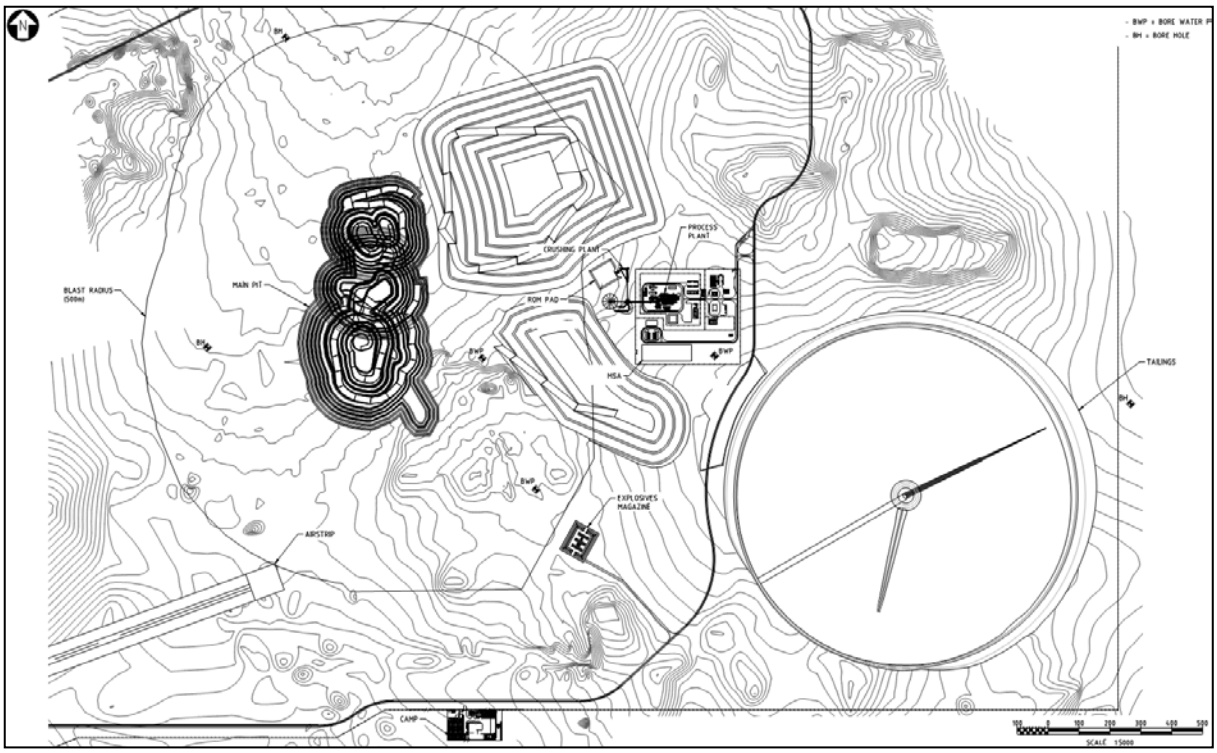
The general control philosophy for the plant will be one with a low level of automation and remote control facilities. Instrumentation will be provided within the plant to measure and control key process parameters. The main control room, which will be located adjacent to the plant offices, will house two PC based operator interface terminals (OIT). Both of the OITs will act as the control system supervisory control and data acquisition (SCADA) servers as well as configuration / operator stations. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system.

1.6 Infrastructure

1.6.1 Site Layout

The overall site development plan is shown in Figure 1.6.1. The drawing shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, airstrip, mine services area, mine open pit and mine waste dump.

Figure 1.6.1 Site Layout



The process plant and tailings storage facility are located on the eastern side of the Sissingué open pit, just outside of the 500 m blast zone. The accommodation camp and airstrip are located south / southwest of the process plant.

The main access road approaches the site from the west and the layout provides easy access for personnel and material movements.

1.6.2 Buildings and Infrastructure

The site as a whole will be fenced to clearly delineate the mine area, prevent animal access and deter access by unauthorised persons. Road access into the fenced area will be through a manned checkpoint. Site security is based on concentric lines of fencing / control. Security fencing will surround the accommodation camp and general site infrastructure. Monitored high security fencing will surround the process plant.

The existing road, approximately 23 km long, between Tengrela and Sissingué will be upgraded to a six metre wide gravel, all weather, and free draining carriageway to provide access for the delivery of equipment, materials and services to the site.

A 1,200 m long x 20 m wide all weather gravel airstrip will be provided for secure transport of bullion to Abidjan for on-shipment to a refinery. The airstrip can also be used for emergency medivac.

The closest connection point to grid power is the 225 kV substation at Boudiali. As a connection is dependent on providing additional infrastructure to connect Tengrela to the national grid this is not an attractive option. Power will therefore be generated on site by a diesel fuelled power station located adjacent to the fuel storage facility. The power station will supply the main HV switchroom inside the processing plant from which power will be distributed.

An accommodation camp will be located approximately 2.2 km south west of the process plant and will provide accommodation for 130 salaried and security staff not originating from the local area.

The existing exploration camp supplemented with the available two bedroom units will be used for pioneer accommodation pending early completion of elements of the permanent camp.

The permanent camp will then be used for Perseus and EPCM contractor staff with any surplus capacity made available to senior contractor personnel if available.

During construction contractors will be required to provide their own accommodation for their construction workforce. An area will be set aside for temporary contractor camps adjacent to the construction site. Alternatively some contractors may choose to source temporary accommodation in nearby towns and villages.

A vendor package modular potable water treatment plant including filtration, ultra-violet sterilisation and chlorination will be installed. Potable water will be stored in the plant potable water tank and will be reticulated to the plant buildings, site ablutions, safety showers and other potable water outlets.

Effluent from all water fixtures in the process plant, mine services area and accommodation camp will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewer pump station from where it will discharge via a pressure main to a vendor package sewage treatment plant system located at the process plant.

Site buildings will be 'fit for purpose' industrial type structures. The workshop and warehouse will be constructed of a concrete slab on ground with structural steel frame and metal cladding. Offices and amenity buildings will be prefabricated structures.

The following facilities will be located in a fenced area adjacent to the process plant:

- Gatehouse with turnstile and entry boom gate control.
- Weighbridge for control of fuel and consumable deliveries.
- Main administration building.
- First aid / clinic.
- Administration junior staff mess.
- Warehouse and stores.

- Contract laboratory.
- Power station.
- Fuel storage facility.
- Mine services area (facilities to be provided by the mining contractor).

The following buildings will be located inside the process plant high security area:

- Security access building and change room including laundry.
- Plant offices, training room, junior staff mess and ablutions.
- Plant workshop including small store, welding bay and overhead crane.
- Engineering offices and ablutions.
- Reagent storage area.
- Goldroom.

Internal communications and IT services shall be via a site wide fibre optic network.

One of the local mobile phone providers will be contracted to install facilities on site and provide a link into the local, national and international telecommunication network.

A radio network will be established with dedicated operational, security and emergency channels.

A local ground station will be installed to provide global satellite voice and data connection.

1.7 Tailings Storage and Site Water Balance

1.7.1 Tailings Storage Facility

The Tailings Storage Facility (TSF), located about 300 metres south east of the plant, will receive the tailings slurry at about 43% solids (oxide) and 48% solids (primary). Return water from the TSF will be used to meet plant process water requirements.

The TSF has a nominal diameter of 1,100 m occupying a storage surface area of approximately 100 ha. The facility and will be surrounded by mine waste to form an Integrated Waste Landform (IWL). The facility is a geotechnically engineered clay structure designed to contain tailings and minimise seepage losses. The TSF has been designed to have a maximum storage capacity for 8.7 Mt. However, at this stage only 5.5 Mt be deposited over a life of five years. The facility will be constructed in two stages. Stage 1 providing storage for 4.5 Mt will be constructed using mine pre-strip materials during initial construction. Stage 2 will occur in Year 4 and will utilise stockpiled oxide waste.

1.7.2 Site Water Balance

The site water balance indicates that sufficient water will be available life of mine even if no water is taken from the near-by Bagoe River. It has been assumed, however, that an abstraction licence to top up the water level in the TSF should it fall below 400,000 m³ will be acquired.

The water balance model indicated that if production commences during the dry season there should still be sufficient water to run the process plant provided that a suitable starting volume exists in the TSF pond, either from water collected or abstracted from the river during the previous wet season, or from pit dewatering.

For the base case average rainfall scenario the Project is in water surplus so care must be taken during operations to optimise the use of decant water from the TSF to avoid accumulating so much water on the TSF that its tailings storage capacity is compromised.

The water balance is based on pit dewatering volumes of 80 m³/h, with no allowance for water used in dust suppression. Pump testing of the proposed dewatering bores have yet to be conducted to confirm the predicted flow rates.

1.8 Environment and Social Impact

Studies, investigations and research, conducted by consultants from CECAF International commenced in 2010 and culminated in the publishing of the Environmental and Social Impact Assessment Final Report in December 2011. Although this report was based on the earlier concept of a 1.6 Mtpa operation it remains relevant for the scope of the proposed 1.0 Mtpa operation.

The Environmental Permit was granted in February 2012.

An early activity of the Project will be the development of comprehensive Environmental and Social Management Plans setting out the standards and outlining procedures and practices that shall be adopted during development and operation of the Project.

A preliminary Closure Plan outlining the closure and rehabilitation of the site has been prepared, a more detailed Closure Plan will be developed during the early stages of operations, following consultation with relevant stakeholders.

Although the positive benefits of the Project should significantly outweigh the negative, ongoing monitoring will be required to ensure unforseen adverse impacts are identified and addressed. This may relate to issues such as the blocking of access or rights of way, ensuring employment opportunities are offered to local people, and minimising any negative impact of an influx of migrants to the area looking for employment opportunities. As a social partner, Perseus through its subsidiary PMCI, commits to maintaining an ongoing dialogue with stakeholders in the area to identify and address concerns and issues as they arise.

1.9 Project Implementation

The proposed approach to project implementation is to engage a suitable Engineering, Procurement and Construction Management (EPCM) contactor for design and construction management of the process plant and infrastructure, which will then be handed over to an Owner's operating team, and to engage an experienced mining contractor for the development of mine infrastructure and provision of ongoing drill and blast and mine operating services under an Owner's mine technical team.

The Owner's team will be progressively expanded to widen its skills and knowledge base to meet the needs of the Project. The team of full and part time personnel will manage both the onshore and offshore activities of the principal EPCM contractor and specialist subcontractors as well as providing specialist technical input into the Project design.

Personnel required for key onshore mine operations roles will be filled early to contribute to mine design and manage the mine contractor undertaking development of the mining infrastructure and open pit.

Perseus' in-country management, administration and services department will manage environment and community issues and prepare the site for the coming influx of operating personnel.

A preliminary schedule shows the Project can be executed within 16 months from EPCM award. Key project milestone dates are provided in Table 1.9.1.

Table 1.9.1 Project Schedule

Milestone Description	Planned Completion
EPCM Award Date	Week 0
Ball Mill Contract Award	Week 14
Earthworks and Civil Contractor Mobilisation	Week 19
Building Contractor Mobilisation	Week 35
SMP Contractor Mobilisation	Week 38
E&I Contractor Mobilisation	Week 60
Mills Mechanical Installation Commencement	Week 63
Power On	Week 60
First Gold Pour	Week 76

Several items and activities are potentially on or close to the schedule critical path for design and construction including procurement, manufacturing, transport and installation of long lead mechanical and electrical equipment items for the process plant. Their criticality will be investigated further during early engineering.

1.10 Operations

1.10.1 Transition to Operations

Drawing on Perseus' recent experience in starting up a new mine in Ghana, comprehensive policies will be developed and induction, training and operating procedures put in place as part of an overall Operational Readiness Plan that will be prepared to ensure the transition from exploration to project development to operations is managed in a safe and effective manner.

Perseus is committed to providing employment opportunities to Ivorian nationals, based on identifying individuals with the necessary qualifications, experience and skills. It is anticipated, however, that expatriate staff will fill a number of senior management, key supervisory and training roles. A succession plan will be developed to transition Ivorian nationals into roles initially filled by expatriates where possible.

Expatriate positions will be advertised internationally and/or regionally as appropriate with Perseus leveraging off its position as an established produced, with a trained and experienced workforce, in adjacent Ghana.

It is recognised that the site is remote from the main population areas. Allowance has, therefore, been made to employ senior Ivorian nationals on a single status residential 'fly in, fly out' basis. Employment opportunities will be offered to local residents where feasible and appropriate.

1.10.2 Ramp Up and Production

Mining and processing plant ramp up and production estimates have been prepared to facilitate the development of a financial cash flow model and thus more accurately determine the timing of likely expenditure and revenue streams.

The mining contractor, or contractors, will be mobilised during the construction phase to allow establishment of the mine services area, development of haul roads and other mining infrastructure. Completion of the mine pre-strip, construction of the TSF and accumulation of an initial ROM stockpile.

The plant ramp up schedule has been developed to reflect the simple, robust flowsheet of the processing facility, using ramp up data from similar regional operations as a basis. It is estimated that nameplate plant ore throughput will be achieved in the third month after ore is first introduced to the circuit.

Life of mine (LOM) ore production will be 5.5 Mt at a grade of 2.4 g/tAu. Mine pre-strip will commence five months in advance of plant commissioning.

Ore processed during the first year of mill operations and again for a five month period in Year 3 will be predominantly oxide and transitional material. During this period the plant will be operated in two stage crush / primary ball mill configuration. In this configuration, treating soft material, the plant has a nominal annualised capacity of 1.2 Mtpa providing 'catch up' capacity for unexpected operational delays due to wet oxide ore handling problems and similar issues.

As the proportion of fresh ore in plant feed increases the comminution circuit will be reconfigured to three stage crushing and conventional ball milling with the harder material requiring a finer crush size to be fed to the ball mill. The reconfiguration of the crushing plant is a relatively simple process and during the period that mine production consists of a mix of softer materials and fresh ore the crushing plant can be switched back and forth to batch treat material with differing properties if necessary.

1.10.3 Gold Production

Over the life of mine gold production is estimated to be 385 koz. The annual gold production schedule from commencement of plant commissioning is shown in Table 1.10.1.

Table 1.10.1 LOM Gold Production

Mill Months	1-12	13-24	25-36	37-48	49-60	61-64	LOM
'000 Ounces	67	82	61	74	93	8	385

1.11 Operating Costs

1.11.1 Mining Costs

Mine Contractor

Contract mining costs for a conventional drill and blast, shovel and truck operation were developed with key underlying data for the mining cost model coming from a mining contractor quote developed by DTP Mining. Labour for mine management and supervision by Perseus personnel has been developed from an in-house organisation structure. The mining costs were developed to a ±25% level of accuracy.

The overall mining cost for the life of the Project, of contractor expenditure including establishment, mobilisation and demobilisation, and adjusted to include fuel costs at US$1.01/L is shown in Table 1.11.1.

Table 1.11.1 Mine Contractors Costs Life of Mine

RPM Contractor Estimate						
Item	**Item Description**	**Units**	**Driver**	**Rate**	**Units**	**Total US$M**
1	Mobilisation, Site Est & Demob	No.	1	3,609	US$k	3.6
2	Fixed Monthly Fee	Months	65	512	US$k	33.3
3	Area Preparation	km2	796	0.88	US$	0.7
4	Drilling	kdrm	172	22.40	US$	3.8
5	Blasting	kholes	21.8	173	US$	3.8
6	Excavate, Load, Haul and Dump	kbcm	10,872	3.63	US$	39.5
7	Stockpile Rehandle	kt	5,525	0.60	US$	3.3
8	Dewatering	Months	65	32.19	US$k	2.1
9	Grade Control	ore kt	5,525	0.24	US$	1.3
Total						**91.4**

Contractor costs incurred in the five months prior to ore processing ($11.05 million) have been capitalised as mine development costs.

Over the life of the Project, this equates to an average cost of $16.55 per ore tonne or $8.42/bcm or $3.95/tonne rock.

Owner Management and Supervision

Owner mine management and supervision costs total $5.99 million life of mine of which $0.53 million are incurred during pre-stripping and are capitalised.

1.11.2 Processing and Administration Costs

The LOM processing and general administration operating cost estimates are presented at summary level in Table 1.11.2. The estimate is considered to have an accuracy of ±15%, is presented in US dollars (US$) and is based on prices obtained during the fourth quarter of 2014 (4Q14).

General administration operating costs include site administration functions and costs for maintaining an office in Abidjan. Corporate costs outside of Cote d'Ivoire are excluded.

Table 1.11.2 Life of Mine Process Plant and Administration Operating Costs

Cost Centre	LOM	
	US$	US$/t
Power	39,867,729	7.22
Operating Consumables	25,389,577	4.60
Maintenance	6,236,076	1.13
Laboratory	4,240,832	0.77
Process & Maintenance Labour	14,180,385	2.57
Plant Feed Rehandle	1,546,694	0.28
Total Processing Cost	**91,461,293**	**16.56**
Abidjan Office (DGA) Labour	1,273,507	0.23
Administration Labour	13,539,305	2.45
General & Administration Costs	27,738,812	5.02
Total General & Administration	**42,551,624**	**7.70**
Total Cost including G&A	**134,012,917**	**24.26**

1.12 Capital Cost Estimate

The capital estimate is summarised in Table 1.12.1. The initial project capital cost was estimated at US$105.98 million.

Table 1.12.1 Initial Capital Cost Estimate Summary (US$, 1Q2015, ±15%)

Initial Capital		
Item	**Main Area**	**Total US$M**
1	Mining	11.89
2	Construction Indirects	8.01
3	Treatment Plant	21.75
4	Reagents & Plant Services	8.43
5	Infrastructure & Tailings	22.86
6	Owners Costs	14.46
7	EPCM Costs	9.18
8	Contingency	9.40
Subtotal		**105.98**

Exclusions to the capital estimate include the following:

- Project sunk costs.
- Import duties and taxes on the assumption that the Project will be exempt.
- Land compensation and community costs.

- Financing costs.
- Escalation.

1.13 Economic Analysis

The Sissingué Gold Project is an economically viable Project at current gold prices. Applying a long term gold price of US$1,200/oz on a flat line basis from the commencement of production, estimated pre-tax cashflows generated are US$112.4 million and the payback of the Project is estimated to occur after 2.7 years. The mine life is estimated at 5.3 years.

An internal rate of return of 27.0% and net present value ('NPV') of US$52.5 million, based upon real, post-tax cash flows, calculated using a 10% discount rate are estimated.

Life of mine average C1 cash costs of production are US$570/oz, while all-in sustaining costs are US$632/oz (net of silver credits, US$3/oz).

Project returns are mostly sensitive to gold prices and head grade/recovery. For a US$100/oz increase in gold prices, a further US$24.3 million NPV is added whilst for a US$100/oz decrease in gold prices, there is a US$26.1 million reduction in NPV.

Table 1.13.1 Financial Analysis

Gold Price	Unit	US$1,100/oz	US$1,200/oz	U$S1,300/oz
Waste + Ore Mined	'000 tonnes	23,183	23,183	23,183
Ore processed	'000 ozs	5,524	5,524	5,524
Head Grade	g/t gold	2.42	2.42	2.42
Weighted Average Recovery	%	89.72	89.72	89.72
Gold Produced	ozs	385,211	385,211	385,211
Development Capital	$'000	105,982	105,982	105,982
Sustaining Capital	$'000	5,249	5,249	5,249
Mining Costs	$'000	85,835	85,835	85,835
Processing Costs	$'000	91,461	91,461	91,461
Administration Costs	$'000	42,358	42,550	42,743
LOM Cash Operating Cost (C1)	$/oz	570	570	571
Year 1 & 2 Cash Operating Cost (C1)	$/oz	596	596	597
Total Site Cost	$/oz	628	632	643
Free cash	$'000	75,645	112,414	146,680
IRR	%	18.8	27.0	34.2
NPV 10%	$'000	26,382	52,515	76,794
Payback period	months	38	32	26
Tax paid	$'000	-	2	140
Royalties paid (State)	$'000	17,258	18,798	22,842

1.14 Risks and Opportunities

1.14.1 Risks

Sovereign Risk

KPMG published a snapshot report covering sovereign risk in the first quarter of 2013. At that time none of the three major rating agencies rated sovereign risk in the country.

KPMG's report states that 'Foreign interest in Ivory Coast has recovered since the stabilisation of the political environment, albeit that political risk remains relatively high'.

Sovereign risk is mitigated by the short payback period anticipated for the Project.

Cross Border Incursions

With the Project site adjacent to the Côte d'Ivoire / Mali border there exists the possibility of security threats from people / groups crossing from Mali and retreating back across the border before Ivorian authorities can respond.

Non-violent attempts by individuals / groups targeting the theft of portable assets such as vehicles, tools, copper wire, etc. will be discouraged by concentric lines of fixed security installations. Patrolled fencing around the entire site will deter casual access and opportunistic theft. A second fence line with more rigorous access control will ensure only authorised employees and visitors have access to core facilities and storage areas. A third, double fence, with camera monitoring and a further level of access control will surround the process plant. Finally, gold bullion will be processed and stored in a secure building with strict access control, intrusion alarms and camera monitoring.

The fixed security assets will be patrolled and monitored by a site security team that will develop a close working relationship with local Ivorian police and military personnel for the purposes of intelligence gathering and emergency response.

Gold bullion will be recovered on a weekly basis and promptly flown off site. If feasible, smelting days and the bullion flight schedule will be randomly varied to discourage attempts at theft.

1.14.2 Opportunities

Project Timing

Prompt development of the Project at a time when construction activity in West Africa and elsewhere is well below recent peaks offers the opportunity to negotiate competitive prices for equipment supply and construction contracts. At the very least project timing reduces the risk of cost escalation and overruns, reducing the risk of exceeding project contingency allowances. It is likely, with the Project underway and 'money on the table' for equipment purchase and contract award, more favourable supplier prices and contractor rates could be negotiated.

Discussion with DTP has indicated there is potential to reduce the mining cost of the Project by 6-7% below their initial quote. Experience from the recent tendering process at Edikan indicates that the reduction could ultimately be larger due to the very competitive nature of the mining contractor sector at this point in time.

Extension to Mine Life

The wider Tengrela leases are prospective with additional zones of mineralisation identified. Although this project is based on exploitation of the Sissingué deposit there is the opportunity that the operating life of the process plant will be extended.

Drilling to date at the Mahale / Bélé deposits, 40 km SW of Sissingué, shows potential for additional mill feed material. The deposit is within trucking distance of the Sissingué plant site. Additional work is required to develop resource and reserve estimates.

Grid Power Supply

Grid power supply as a standalone proposition at Sissingué is not the preferred option. However, there is a risk associated with the continuity of power supply to Perseus' Edikan operation in Ghana, and a potential mitigation method is to access power from Côte d'Ivoire through CI Energy. As part of a potential deal, Perseus could provide the additional capital to fund the construction of the power line from Boudiali to site and the overall outcome may be economically attractive to Perseus as a whole. The supply of grid power to site would potentially lower Sissingué operating costs.

1.15 Interpretation and Conclusions

The Sissingué Gold Project Feasibility Study has identified, defined and costed a low technical risk conventional open cut mine and carbon in leach processing facility.

Commercial gold mining has been established in West Africa for over 100 years and Cote d'Ivoire and surrounding countries have varying levels of established mining infrastructure.

Sovereign risk is understood and mitigated by the relatively short payback of the Project. Residual environmental and social issues are low and manageable.

The Sissingué Gold Project is an economically viable Project at current gold prices. Applying a long term gold price of US$1,200/oz on a flat line basis from the commencement of production, estimated pre-tax cashflows generated are US$112.4M and the payback of the Project is estimated to occur after 2.7 years. The mine life is estimated at 5.3 years.

1.15.1 Mineral Resource

The Mineral Resource model on which the Project is based is considered to be robust, with the key outcomes being:

- Resource classification is based on information and data provided from the Perseus database. The drilling techniques, survey, sampling, sample preparation, analytical techniques and database management/validation are well within industry standard. The database represents an accurate record of the drilling undertaken in the Project;
- Geological continuity is understood with reasonable confidence. The classification of the Mineral Resource reflects the level of confidence; and
- The estimation technique and parameters are appropriate for the style of mineralisation.

1.15.2 Mining

The mine planning study has demonstrated the Project to be a viable and attractive prospect for development. The key outcomes of the Study are:

- 3 phase mine development strategy from south to north.
- Total material movement and vertical rates of advance are readily achievable.
- Areas have further been identified that would likely improve the Project value.
- Re-run the Whittle optimisation to reflect owner-operator mining operating costs.
- Re-run the schedule optimising the output based on an owner-operator model.
- Further studies into the potential for backfilling of the Phase 1 pit.

1.15.3 Mineral Reserve

The Mineral Reserve Estimate on which the Project is based is considered to be robust, with the key outcomes being:

- Total mineable ore of 5.5 Mt at 2.2 g/t Au and strip ratio of 3.2:1.
- Annual gold production is variable due to grade and the transition from oxide to sulphide, ranging from 61/koz to 94/koz per year.
- 5.5 year mining life.

1.15.4 Processing

The metallurgical and ore processing studies have demonstrated the Project to be viable and attractive for development. The key outcomes of the Project are:

- A simple crushing and grinding circuit with recovery via gravity and CIL is the preferred processing method.
- High recoveries are achievable from all major ore types that are planned to be processed.

Areas have further been identified that would likely improve the Project value, namely:

- Investigate methods for improving the recovery from sediment type ores.
- Investigate the availability of immediately available new or second-hand items of long lead equipment such as the crushing circuit and/or mill.

1.16 Recommendation

It is recommended that Perseus Mining Limited commence implementation of the Project with a view to demonstrating commitment to commercial production before the expiry of the 8 August 2016 deadline currently in place as a condition of their Exploitation Permit. The project is expected to be developed for an initial capital cost of US$106.0M and a life of mine average operating cost (C1) of $39.75/t ore, or US$570/oz gold produced.

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2.0 INTRODUCTION

2.1 Introduction

Perseus Mining Limited (Perseus), a Canadian reporting issuer, has requested Lycopodium Minerals Pty Ltd (Lycopodium) coordinate and prepare an independent Technical Report on their Sissingué Gold Project (Sissingué or The Project) in accordance with Canadian National Instrument 43-101 requirements. This Technical Report is based on the outcomes of the Sissingué Gold Project Definitive Feasibility Study (the DFS) completed in May 2015.

2.2 Contributing Consultants

Lycopodium Minerals Pty Ltd is a subsidiary of Lycopodium Limited, an Australian listed public corporation, and has provided engineering and project management services to the international mining industry for over 20 years. The Perth, Western Australia, office of Lycopodium Minerals undertook the compilation of this report.

Sections of this report were authored or co-authored by Qualified Persons from Perseus, RungePincockMinarco (Perth) and Worley Parsons (Perth).

RungePincockMinarco (RPM) is the world's largest group of mining technical experts with history dating back to 1968. The Perth (Western Australia) office of RPM contributed to this report.

Worley Parsons provides expertise in engineering, procurement and construction and offers consulting and advisory services across a wide range of industries and geographical locations. Their geotechnical office located in Perth (Western Australia) contributed to this report.

2.3 Site Visits and Inspections

QP Steffen Brammer (Perseus) has visited the site since 2005 on a regular basis for several months per year to inspect drill location, drill progress, sampling practices, sampling quality and logging practices. Site visits also included laboratory inspections, his most recent visit was from 19 October to 13 November 2014.

QP Christopher Waller (Lycopodium) has not visited the site but is familiar with the region having been resident in West Africa for over 10 years at various times during his career. He has filled operating roles to the level of General Manger and has managed multiple feasibility studies in respect of projects in the region.

QP Paul Thompson (Perseus) has not visited the site.

QP Joe McDiarmid (RPM) has not visited the site.

QP Chris Lane (Worley Parsons) has not visited the site.

2.4 Information and Data Sources

This report is based in part on information provided by Perseus and others including documents, data and reports compiled by Perseus management and technical staff and previous reports by other independent experts (refer Sections 3 and 27).

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3.0 RELIANCE ON OTHER EXPERTS

3.1 Reports and Contributions from Other Experts

In preparing the feasibility study report on which this Technical Report is based the Qualified Persons have relied on previous work, reports and opinions from a number of sources. These include the following:

- Theodore Hoegah & Michel Ette, Avocats associés, Abidjan; Legal opinion on Exploitation Licence in Côte d'Ivoire and on Perseus Mining Côte d'Ivoire; 13 February 2015 for confirmation of the incorporation and standing of PMCI and status and ownership of the Exploitation Licence and the absence of encumbrances as addressed in Section 4 and the approval of the environmental impact study and issue of an approval certificate by the National Agency for the Environment as addressed in Section 20.

- CECAF International (authors M. Gilbert Guey et al); Occidental Gold (Côte d'Ivoire) SARL Sissingué Gold Project Environmental and Social Impact Assessment Final Report; December 2011 for details of baseline studies contributing to the ESIA final report on which was based the granting of an Environmental permit in February 2012 (refer Section 20).

- Snowden Mining Industry Consultants Pty Ltd (Perth) (author Lynn Olssen); 'Sissingué Resource Estimate' prepared for Perseus Mining Ltd; October 2014 for the JORC Resource Estimate quoted in Section 14.

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4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Sissingué Gold Project, formerly known as the Tengrela Gold Project, is located in northern Côte d'Ivoire, adjacent to the Mali border and approximately 620 km north west of the commercial capital, Abidjan (Figure 4.1.1) (approximately 720 km by road). The nearest town is Tengrela, located approximately 15 km to the west of the concession boundaries and 25 km from the project site by existing roads.. Korhogo, the principal regional centre, is located 150 km to the southeast. The village of Sissingué is approximately 2.2 km by road from the Project site. The Project area is centred at National Grid reference 1,141,666N and 804,166E.

Principal access is gained by sealed road through Korhogo and Boundiali to Bolona and thereafter by all-weather formed gravel road to Tengrela. From Mali, access to Tengrela is also possible by the Bougouni–Kolondieba-Kadiana secondary road. This road is currently poorly maintained and this route would be impractical during the wet season.

The major towns in the area are Tengrela and Kanakono and these towns offer very little in terms of major infrastructure. Tengrela has two hotels and a few markets, and fuel appears to be readily available at the petrol stations. The town of Tengrela is connected to grid power, and running water is readily available. Tengrela also has land and mobile telephone coverage and internet services on mobile phones and satellite dishes; a cable system is not yet available.

Figure 4.1.1 General Location of the Sissingué Gold Project in Côte d`Ivoire



The Sissingué Gold Project area is situated in an area of low to moderate relief, lying to the west of the Bagoé River. The relief is generally gently rolling, relatively flat peneplain with areas of lateritic plateaux. The highest point of the concession reaches approximately 370 m with the Bagoé at around 320 m above mean sea level (amsl). The vegetation is mainly savannah with long grass and moderately sparse trees and bushes. Land use is dominated by cash cropping and subsistence cultivation, mainly cashews, cotton, corn or mangoes.

Côte d'Ivoire has an emerging mining sector with several significant projects being commissioned in recent years. Experience and support in personnel and industry is on a steady incline.

4.2 Mineral Tenure

Initially, the Project area consisted of the exploration permit, RP145 ('Tengrela East') covering an aggregate area of approximately 446 km^2. Exploration permits confer on holders, within boundaries, surface and depth, the exclusive right to explore for mineral substances as well as the free disposal of the products extracted during exploration. Tengrela East was registered to Perseus's Ivorian subsidiary Occidental Gold Sarl ('OGIC'). The Tengrela East Permit was last renewed on 21 April, 2009 and expired on 19 November 2011. In replacement of the Tengrela East exploration permit, OGIC was granted exploitation permit PE 39 (the 'Exploitation Permit') for the development of the Sissingué gold deposit in August 2012. The Exploitation Permit was subsequently transferred to another Ivorian subsidiary company, Perseus Mining Côte d'Ivoire SA ('PMCI') in July 2013.

Under the conditions of the Exploitation Permit, PMCI had to commence production from the Permit by no later than 8 August 2014. By order dated 17 March 2014, the Ivorian Government has granted an extension to commence production until 16 March 2016.

The area surrounding the open pit, waste dumps, tailings storage facility, process plant, airstrip, accommodation and other infrastructure will be fenced to discourage unauthorised access and covers an area of approximately 1,050 hectares. Of this some 590 hectares will be directly impacted / disturbed by the Project.

The position of the Sissingué Project permit area and other gold deposits in the area are shown in Figure 4.2.1. The Sissingué deposit is currently the main focus of development at the Project, however several other mineral occurrences have been identified in the Project area. Figure 4.2.2 shows the location of all known mineralised zones, mineral resources, mineral reserves and mine workings relative to the outside property boundaries. All known zones of mineralisation and Mineral Resources have been located accurately and there is no risk of any of the known Mineral Resources being outside the permit boundaries due to survey errors.

Figure 4.2.1 Sissingué Project Area

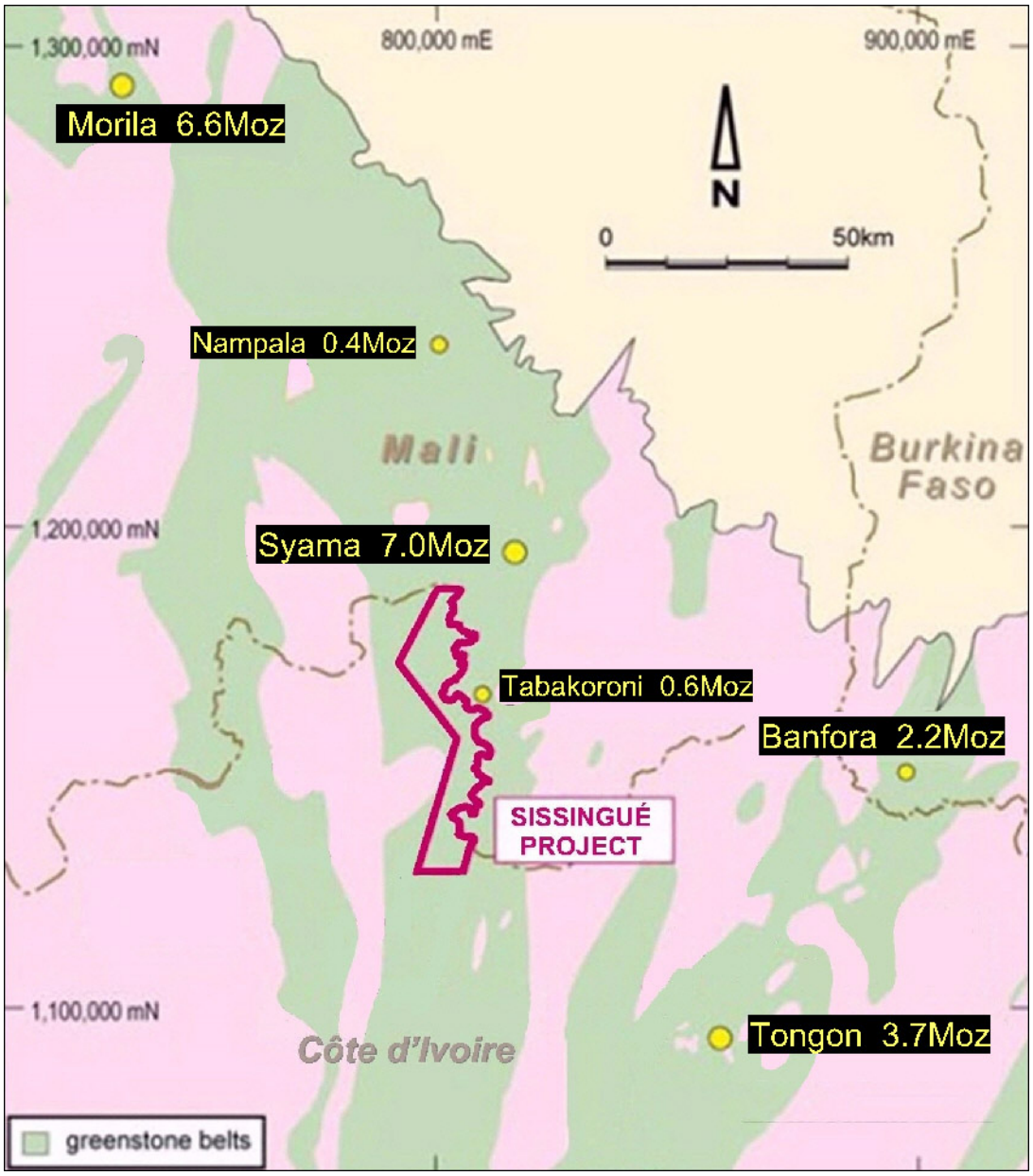
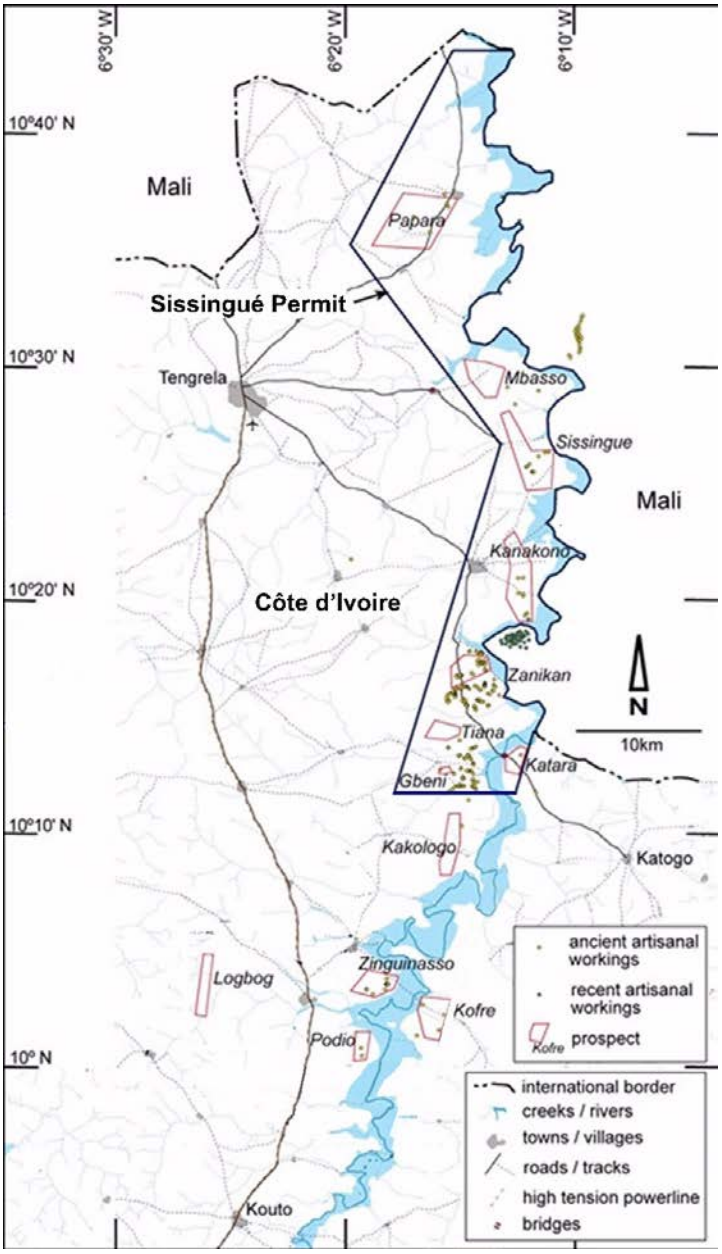


Figure 4.2.2 Location of Known Mineralised Zones



4.3 Ownership

As a result of the grant of the Exploitation Permit for the Project, the Government of Côte d'Ivoire has a statutory 10% free carried interest in the Project and OGIC's interest has reduced to 85%. This is reflected in the shareholding of PMCI as follows:

- Perseus (through OGPL): 85%
- The State of Côte d'Ivoire (held by SOMICI and to be transferred to the State): 10%
- SOMICI: 5%

OGPL is an Australian registered company and a wholly owned subsidiary of Perseus Mining Limited.

Confirmation of the current legal status of the Exploitation Licence has been provided by Theodore Hoegah & Michel Ette, Avocats associés, Abidjan (refer Section 3).

4.4 Mining Convention and Mining Code

In March 2014, Côte d'Ivoire adopted a new Mining Code. The new Mining Code formalises the requirement of the holder of an exploitation permit to enter into a mining convention with the State and sets out what should be included in a convention. A mining convention may not deviate from the Mining Code. PMCI is currently negotiating a mining convention with the State. The convention is expected to have a term of 12 years from signing, renewable for successive periods of no more than 10 years. The draft convention received from the State includes, without limitation, the following:

- Fiscal stability for the duration of the convention.
- Legal stability for the duration of the convention with the exception of social, health, safety and environmental regulation.
- An annual contribution to a social and economic development fund of 0.5% of the gross revenue from gold produced, after deduction of transportation and refining costs during the previous year.
- An annual contribution of FCFA 25,000,000 (approx US$ 45,000) to the Ivorian Mining Training Fund, a fund dedicated to the training of mining professionals.
- An annual payment of 4% of the estimated rehabilitation cost into an escrow account and annual provision of a bank guarantee for 16% of the estimated rehabilitation cost.
- Free choice of contractors and suppliers with priority to be given to Ivorian providers to the extent the relevant services or products are available at least equivalent quality, price, skills, warranties and delivery times.

- The obligation for at least 80% of the workforce to be Ivorian subject to availability of required qualifications.

Under the new Mining Code, the State may apply for an additional interest in PMCI of up to 15%. Such interest will be contributory and at market conditions.

The main features of the fiscal and customs regime laid down in the General Taxation Code, the new Mining Code and to be included in the Mining Convention are the following:

- A five year tax holiday on revenue from the date of first production. After that period the corporate tax rate is 25%.
- An exemption from VAT on the acquisition of foreign and Ivorian services and goods and sales in connection with the operations. This exemption extends to contractors.
- An exemption from export duties and taxes on product.
- An exemption from various other taxes, including, without limitation, real estate tax and employer's contribution tax on expat salaries.
- A reduction by 50% of withholding tax to 9% on interest paid on loans with a term longer than three years by the Exploitation Company.
- An exemption from import duties on imported plant and parts during construction and any expansion phase.
- An exemption from import duties on fuel, lubricants, chemical products, etc. during the operations phase.

4.5 Environmental Liabilities

There are currently no known environmental liabilities or significant risks that could affect access, title or the right or ability to undertake project activities. An Environmental Permit was issued by the Agence Nationale De l'Environment (National Environment Agency) in February 2012 and the Company will endeavor to comply with all applicable laws, regulations, decrees and guidelines. Other permits and approvals required for the project activities will be obtained prior to the commencement of the relevant works.

Site plans indicating the proposed locations of various waste facilities were submitted to the National Environment Agency as part of the Environmental Impact Statement (EIS). The ensuing Environmental Permit issued allows the Company to mine and process ore as well as dispose of waste materials including waste rock, tailings and other general wastes.

4.6 Rights Royalties and Encumbrances

Franco-Nevada Royalty

Equus Mining Limited retains a royalty of 0.5% of the value of all minerals recovered from the Tengrela Exploration Permits less refining and smelting costs. In June 2013 the right to receive the royalty was assigned to Franco-Nevada Corporation and the obligation to pay the royalty in respect of minerals recovered from the Exploitation Permit, has since been assumed by PMCI.

Ivorian Partners Royalty Agreement

The Ivorian Partners (refer Section 6.1) retain a royalty of US$0.80 per ounce of refined gold produced from the Tengrela East Research Permit 145 (the whole of the current area of the Exploitation Permit is part of the area covered by Research Permit 145), payable monthly.

State of Côte d'Ivoire Interest

In addition to its 10% free carried interest in the Project, the State of Côte d'Ivoire is entitled to production royalties as follows:

Table 4.6.1 Government Royalty Rates

Royalty	Gold Price
3%	Up to US$1,000
3.5%	US$1,000 – US$1,300
4%	US$1,300 – US$1,600
5%	US$1,600 – US$2,000
6%	Over US$2,000

The royalty is based on the gross revenue from gold produced, after deduction of transportation and refining costs.

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5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography

The Sissingué Project area is situated in an area of low to moderate relief, lying to the west of the Bagoé River. The relief is generally gently rolling, relatively flat peneplain with areas of lateritic plateaux. The highest point of the concession reaches approximately 370 m with the Bagoé at around 320 m above mean sea level (amsl). The vegetation is mainly savannah with long grass and moderately sparse trees and bushes. Land use is dominated by cash cropping and subsistence cultivation, mainly cashews, cotton, corn or mangoes.

5.2 Access

International airlines including Emirates, Air France and South African Airways fly regularly to Abidjan. Internal commercial flights are then available to Korhogo 150 km from the Project site.

The port of Abidjan is on the regular cargo route around West Africa and has been used as a gateway port for imports to Francophone West Africa for decades.

Principal access to site is gained by sealed road through Korhogo, located 150 km to the southeast, and Boundiali to Bolona and thereafter by all weather formed gravel road to Tengrela. From Mali, access to Tengrela is also possible by the Bougouni–Kolondieba-Kadiana secondary road. This road is currently poorly maintained and this route would be impractical during the wet season.

Sissingue is the closest village to the project site, at approximately 3 km to the north. The major towns in the area are Tengrela, located approximately 15 km to the west of the concession boundaries, and Kanakono, approximately 10km to the south of the project site, however these towns offer very little in terms of major infrastructure. Tengrela has two hotels and a few markets, and fuel appears to be readily available at the petrol stations. The town of Tengrela is connected to grid power, and running water is readily available. Tengrela also has land and mobile telephone coverage and internet services on mobile phones and satellite dishes; a cable system is not yet available.

5.3 Climate

The Project area has a warm semi-arid climate where there is only one wet and one dry season, with rainfall heaviest in summer, culminating in September, and lightest in January.

The average annual rainfall is 1,160 mm. Approximately 75% of the rainfall occurs between the months of June to September with the highest average monthly rainfall of 309 mm occurring during August.

Temperatures are comparatively consistent year round with mean daily minimum average between 15.2°C (December) and 24.6°C (April) and mean daily maximum between about 30°C (August) and 37°C (April).

During the wet season, access and transport can become difficult unless well-formed and drained roads are constructed. As a result, exploration activities are often curtailed for several months during the wet season. Mining operations will be set up to operate continuously throughout the year.

The climatic data for the Project has been based on national records held for Tengrela from 1960 - 2002. Table 5.3.1 shows the average rainfall data used for the Project design basis:

Table 5.3.1 Average Rainfall

Month	Rainfall in mm
January	3.1
February	6.6
March	17.1
April	49.1
May	94.6
June	140.4
July	246.3
August	309.0
September	194.3
October	79.2
November	17.6
December	3.5
Total	**1,160.9**

The 1 in 50 year monthly rainfall intensity has been estimated at 600 mm.

5.4 Local Infrastructure and Resources

Local infrastructure is limited and the Project will have to be self sufficient in most areas.

Connection to the power grid is not a cost effective option at this stage and site generated power is proposed.

Indications are that the project will be able to source sufficient water from pit dewatering and rainwater but allowance has been made to source supplementary water from the Bagoe River if necessary. The tailings storage facility has been designed as a water retaining structure and will be used to store make-up process water during the wet season for use during the dry.

A VSAT land station is proposed for voice and data connection. It is anticipated that a local mobile phone provider will erect a tower to connect the site to the local network.

Specialist professional and trade skills are limited in the near vicinity but it is expected that an adequate workforce will be available from resources elsewhere in the country. This will be supplemented by regional and international expatriate workers in key management, engineering, operating and training roles. Up to 20% non-Ivorian staff are permitted and this number can be exceeded should local skills not be available.

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6.0 HISTORY

6.1 Historical Background

The Sissingué Gold Project is governed by a joint venture agreement dated September 29, 1997 between Occidental Gold Sarl (OGIC), then a wholly-owned subsidiary of Afminex Limited (Afminex), with a 90% interest and Société Minière De Côte d'Ivoire SARL (SOMICI) with a 10% interest, and a royalty agreement dated June 22, 2002 among OGIC, Leo Resources SARL (Leo IC) (at the time, a wholly owned subsidiary of OGIC) and Le Groupement des Hommes d'Affaires Ivoriens (the Ivorian Partners), pursuant to which OGIC effectively granted the Ivorian Partners a royalty of US$0.80 per ounce of refined gold produced from Tengrela East.

Randgold Resources Limited (Randgold) negotiated a joint venture agreement with OGIC in June 1998, but subsequently withdrew.

Perseus acquired its interest in the Project by agreements dated March 2004 pursuant to which Afminex sold all of the shares of Occidental Gold Pty Ltd (OGPL) (which holds all of the shares of OGIC) and assigned loans to Perseus for consideration comprising shares in Perseus and a royalty interest.

An agreement dated May 20, 2009 was made between OGIC and SOMICI pursuant to which an option was granted to OGIC to acquire a 5% interest in the Project from SOMICI. In December 2010, OGIC exercised this option thereby increasing its interest to 95% and reducing SOMICI's interest to 5%.

As a result of the grant of the Exploitation Permit for the Project, the Government of Côte d'Ivoire has a statutory 10% free carried interest in the Project and OGIC's interest has reduced to 85%. This is reflected in the shareholding of the operating company PMCI as follows:

Perseus (through OGPL):	85%
The State of Côte d'Ivoire (held by SOMICI and to be transferred to the State):	10%
SOMICI:	5%

OGPL is an Australian registered company and a wholly owned subsidiary of Perseus Mining Limited.

6.2 Past Exploration Work

6.2.1 Pre-2003

The Tengrela Project area is covered by a lateritic terrain with multiple levels of ferricrete, sand and silt of alluvial origin. Outcrop exposure is rare. The exploration activities conducted between 1998 and 2004, when Perseus acquired its interest in the Project are summarised in Table 6.2.1.

Table 6.2.1 Summary of Historical Exploration Work

Period	Explorer	Surveys	Geochemical Samples	Pitting	Drilling m
1998- 1999	Randgold	Magnetics, Radiometrics VLF	12,254	370	-
2001 - 2002	Occidental-Afminex		5,737	119	2,211
2003 - 2004	No exploration conducted due to civil unrest.				

6.2.2 Exploration by Randgold

Lag sampling (1,841 sites) at a density of 1.1 samples per km^2 was undertaken over both permits outside the regional soil grids. Significant lag gold anomalies were identified in the following locations:

- Podio E (14 km^2 up to 2,400 ppb)
- Zinguinasso S (10.8 km^2 up to 208 ppb)
- Zinguinasso SE (2.8 km^2 up to 1,670 ppb)
- Zinguinasso NE (10 km^2 up to 920 ppb)
- Zinguinasso E (4.7 km^2 up to 240 ppb)
- Ouomon E (4.4 km^2 up to 370 ppb)
- Ouomon SE (1.5 km^2 up to 300 ppb)
- Gbeni S (21.7 km^2 up to 450 ppb)
- Gbeni E (1.7 km^2 up to 4,600 ppb)
- Gbeni (20.8 km^2 up to 780 ppb)
- Zanikan (12.6 km^2 up to 22,400 ppb)
- Kanakono (16.7 km^2 up to 4,090 ppb)
- Kanakono N (16.4 km^2 up to 1,000 ppb)
- Papara (36 km^2 up to 4,090 ppb)
- Dounaso (4.8 km^2 up to 1,430 ppb)

These coarse fraction anomalies appear to closely coincide with regional structural trends, confirmed by broader targets generated in finer fraction sampling. The aggregate area of anomalous results approximates 180 km².

Regional sampling on a 500 m by 100 m pattern was completed over a 288 km² area covering the structurally complex western portion of the Tengrela South permit. This soil sampling defined a single discrete anomaly (the Logbog anomaly) from 200 m to 400 m wide and extending for 3.5 km, incorporating values from 20 ppb to 200 ppb gold. However no further exploration was completed and Randgold withdrew from the joint venture having spent US$500,000.

6.2.3 Exploration by Occidental (2002)

Occidental explored in its own right from April 2002 with a program of soil sampling and pitting designed to refine the extensive lag anomalies defined by Randgold along the southern extensions of the Syama tectono-stratigraphic corridor. In addition to mapping and limited rock chip sampling, initial soil sampling was undertaken at 50 m intervals along 800 m to 1,600 m spaced lines. Samples were screened to a nominal -2 mm, with four consecutive 50 m sub-samples (200 m) composited for analysis. In all, 974 composites were collected, representing 3,894 individual sample sites from 8 separate targets. The composite samples were submitted to SGS laboratories in Ghana for 2 kg bottle roll gold analysis and AAS arsenic analysis following an aqua regia digest.

The initial results from a number of areas were encouraging and 624 of the 50 m sub-sample residues were analysed for gold individually. Infill soil sampling was variously completed on an 800 m by 50 m, 400 m by 50 m or 200 m by 50 m grid spacing in five separate areas for gold analysis.

Regolith cover prevented effective soil sampling at the Papara and Kanakono anomalies and 52 pits were excavated on a 200 m by 50 m pattern to a depth of approximately 3 m prior to sampling. The resulting samples were assayed by fire assay at the SGS laboratory in Ghana.

Sixteen potentially significant lag anomalies and two soil anomalies had been defined at the completion of the program in July 2002. Prior to commencement of the 2002 program, 13 of the lag anomalies had been tested and three remained untested. Of the 13 tested anomalies, 10 were confirmed and refined by soil sampling, while three were rejected as being of little significance. Seven anomalies were considered to be of major regional significance and a further three targets required additional assessment before their significance could be accurately determined. The majority of the more significant targets were associated with elevated arsenic values. Weakly to moderately anomalous gold and arsenic values were also identified from pitting at the Papara and Kanakono Prospects.

Exploration work was suspended at the Project in September 2002 until end of 2004 due to civil unrest in the country.

6.3 Historical Production

No industrial scale mining or production has been carried out at the Project although small scale artisanal mining has been carried out in a number of locations.

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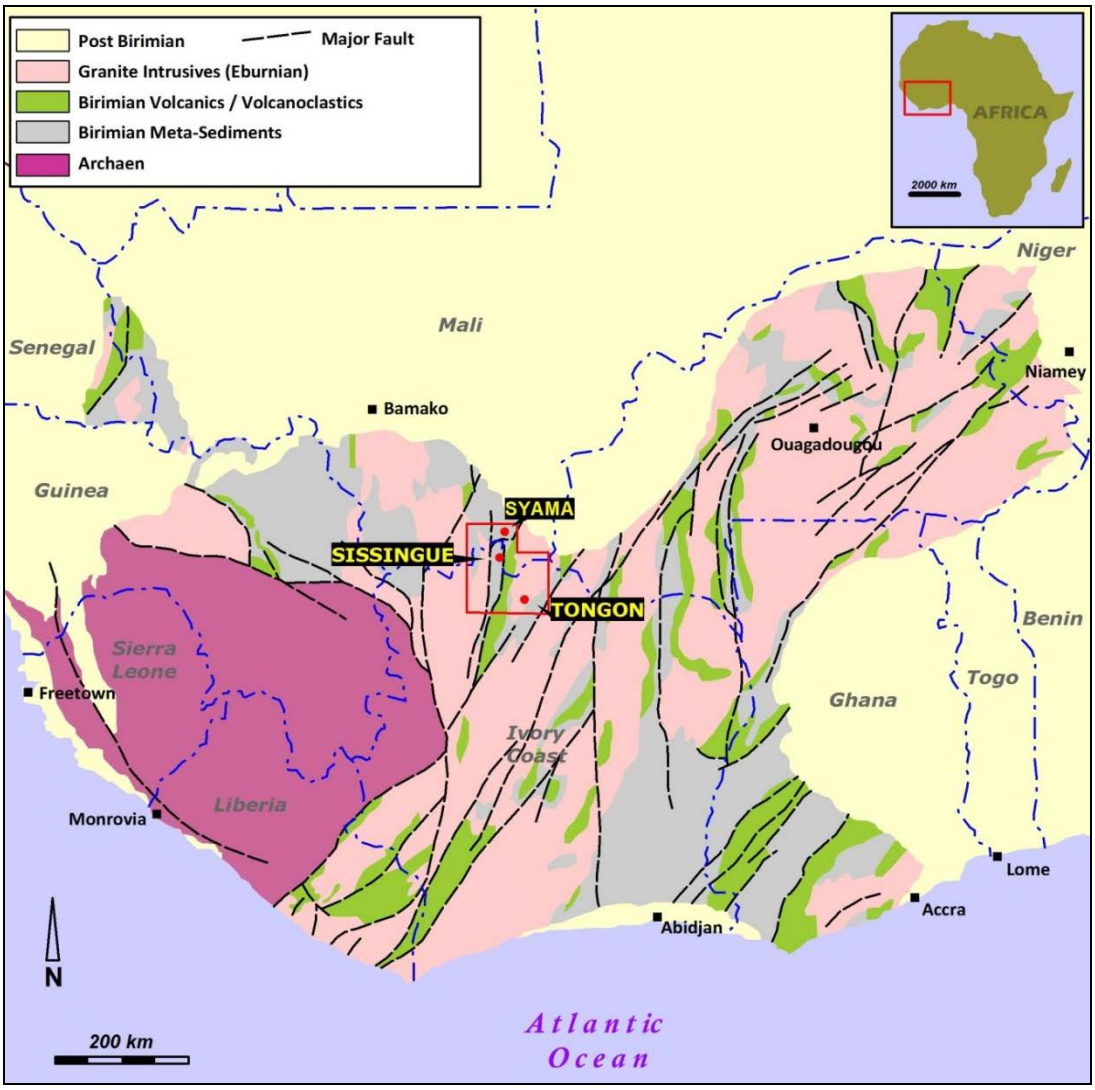
7.0 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

Most gold deposits in the Proterozoic Birimian Shield of West Africa occur in volcano-sedimentary greenstone belts. These belts generally consist of fine-grained sedimentary rocks and tholeiitic to calc-alkaline volcanic rocks. Granitic and gneissic rocks surround the greenstone belts. The metamorphic grade of the belts is usually lower greenschist but may increase to amphibolite grade adjacent to major intrusive bodies. The volcano-sedimentary sequences are in most places exposed in arcuate monoclonal sequences.

A map of the regional geology is shown in Figure 7.1.1.

Figure 7.1.1 Regional Geology of West Africa (Image Supplied by Perseus)



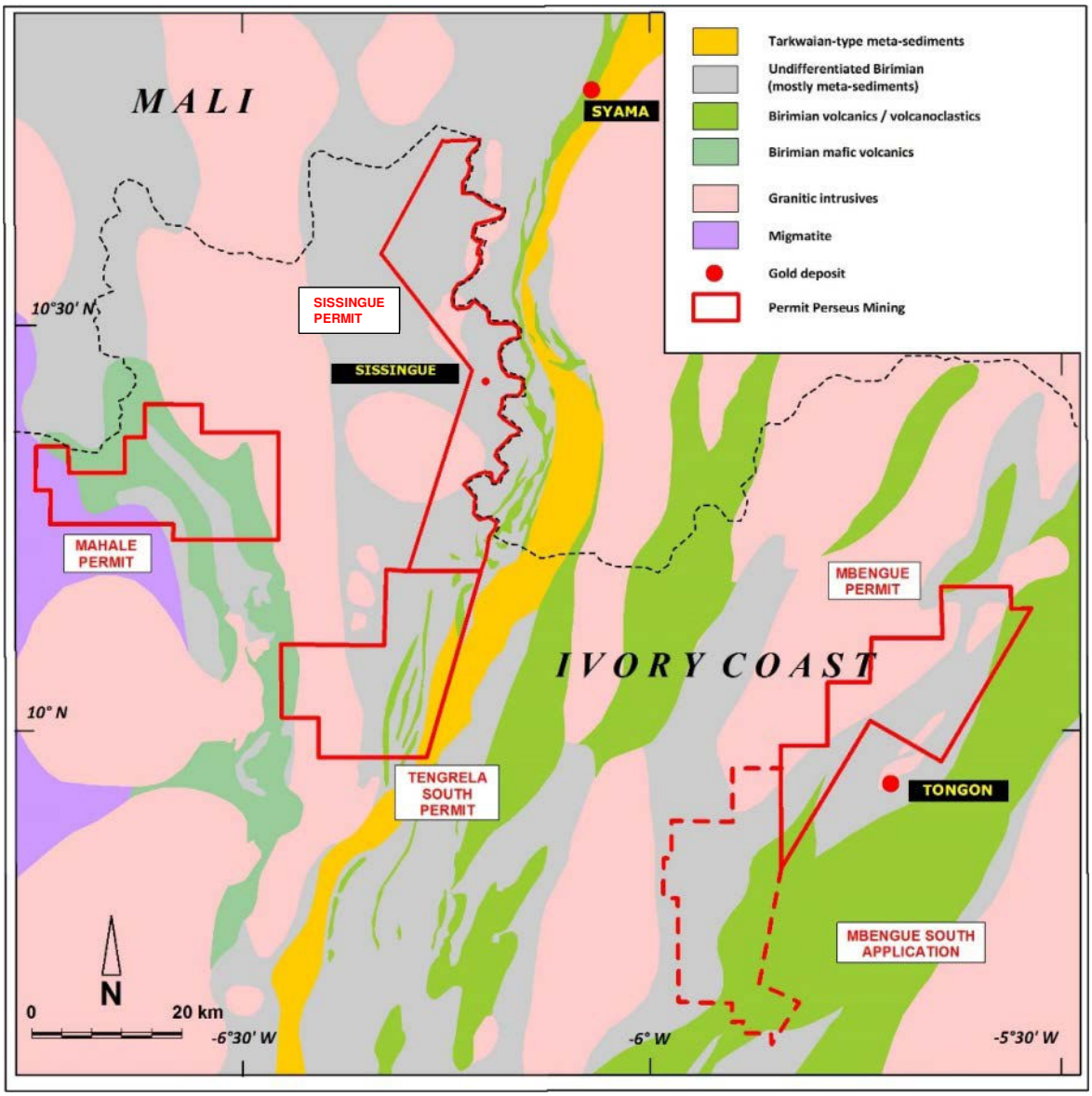
The southern part of the West African Craton (the Man Shield) comprises an Archean core in the south and west and the Proterozoic Baoulé-Mossi Domain, containing the Birimian Formations to the east. Côte d'Ivoire is situated almost entirely over the later domain which has been involved in the Eburnean orogenic event (2,200 Ma to 2,000 Ma). It consists mainly of polyphase granite-gneiss complexes, sedimentary flysch-type deposits and volcanic and volcano-sedimentary complexes. Remnant Archean greenstone belts exist in Côte d'Ivoire, mainly as relict narrow northeast-southwest striking belts that are situated in the gneissic complexes. They contain an assemblage of tholeiitic, volcanic, plutonic and volcano-sedimentary mafic rocks.

7.2 Local Geology

The Sissingué deposit is situated in the Syama-Boundiali Greenstone Belt. The geology of the belt has similarities to the better explored Ashanti Belt in neighbouring Ghana, where Birimian volcanoclastics dominate over competent volcanics, with the development of inner belt basins filled with Tarkwaian epiclastics. In the Bagoe River region near Tengrela, the terrain is comprised of undifferentiated granitoids, flysch sediments, intermediate volcanic and small occurrences of mafic intrusive and molasse sediments.

The Sissingué permit area covers basin sediments west of the strongly deformed Birimian Greenstone Belt, intruded by granitoid bodies (Figure 7.2.1). Interpretation of both aeromagnetics and radiometric data indicates the inclusion of an extensive sequence of north trending Birimian volcanoclastics, intruded by a large felsic body (Kanakono Pluton) as well as several small uranium-rich felsic intrusions.

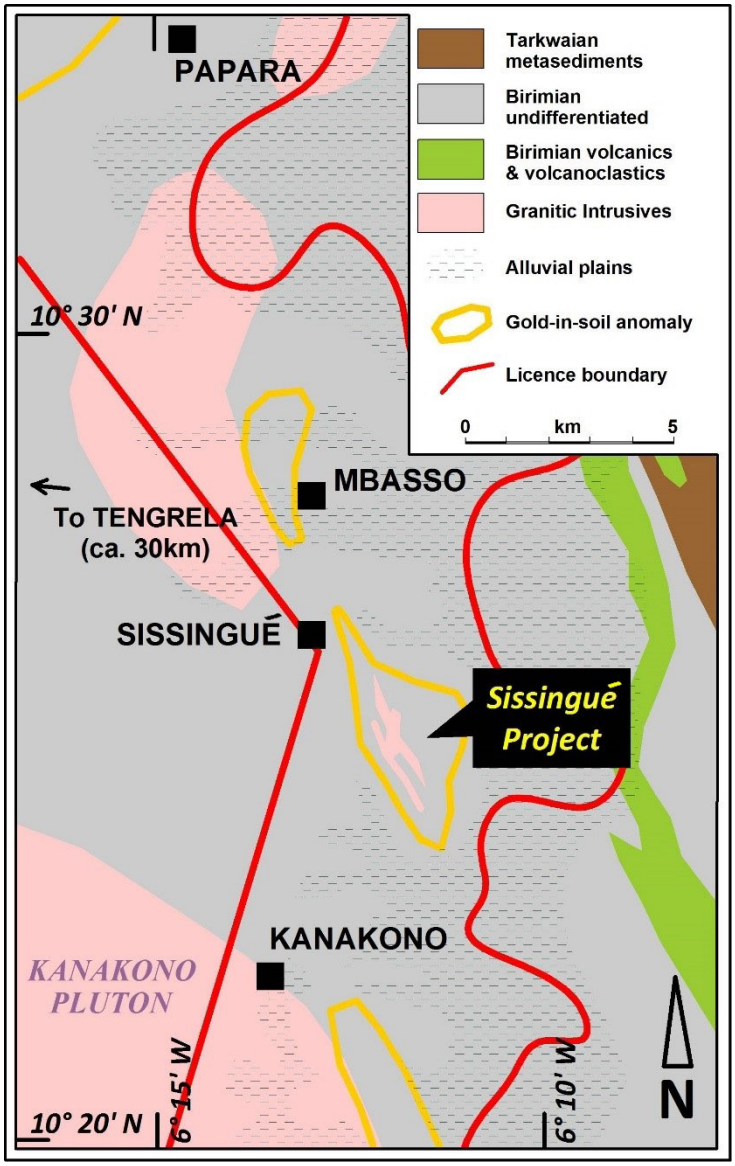
Figure 7.2.1 Geology of Northern Côte d'Ivoire (Image Supplied by Perseus)



7.3 Property Geology

The Sissingué deposit is defined by a 4 km long and up to 1.5 km wide gold-in-soil anomaly situated 5 – 6 km west of the Syama-Boundiali Greenstone Belt (Figure 7.3.1). Rocks encountered in outcrops and drilling comprise predominantly north-north east striking, steeply west dipping and isoclinally folded sediments (sandstones, mudstones and subordinate conglomerates) of the Birimian Supergroup, interpreted as units of turbiditic flows. These sediments are cross cut by a swarm of narrow porphyritic dykes (sub-metre to several metres thick) which trend obliquely to the sedimentary package north-west with sub-vertical or steep to moderate dips towards the north-east. Some minor dykes appear to have exploited weak zones parallel to sedimentary bedding.

Figure 7.3.1 Sissingué Property Geology (Image Supplied by Perseus)



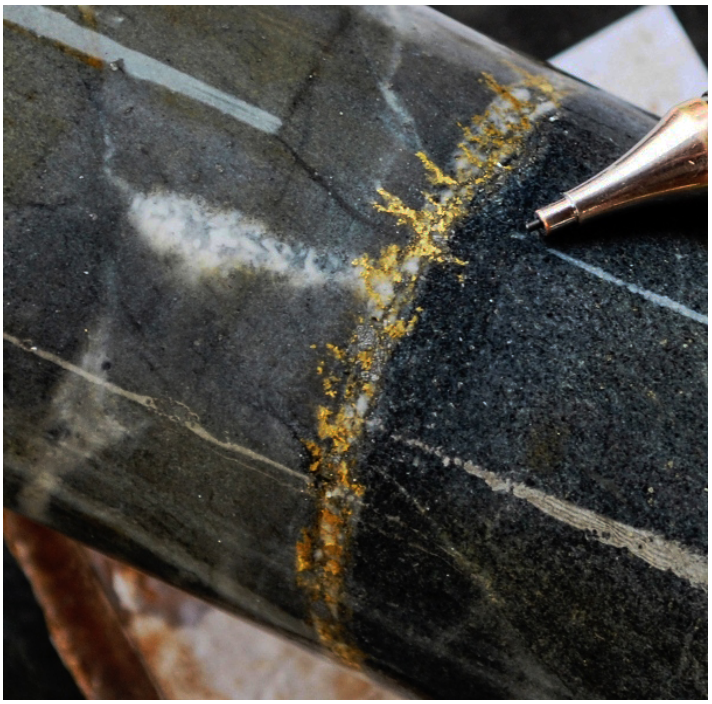
7.4 Mineralisation

Gold mineralisation at Sissingué is associated with the porphyritic dykes of tonalitic composition that cross cut the flysch sediments (turbidites). Subsequent metasomatism of the tonalite has led to:

- a sericite-carbonate alteration within the intrusives and the more permeable horizons (sandstones and conglomerates) of the turbidites
- low to moderate grade disseminated gold mineralisation.

Late stage high grade Au-As-quartz-carbonate veins exploited the altered and brittle portions of the intrusives and sediments with common occurrences of visible gold (Figure 7.4.1).

Figure 7.4.1 Quartz-Carbonate Vein with Visible Gold from Drill Core (SD223) (Image Supplied by Perseus)



Gold mineralisation is found as three distinct styles as summarised below:

- Disseminated or fracture related mineralisation in the felsic intrusive closely associated sericite-carbonate alteration, silification and a pyrite and arsenopyrite mineralisation. The average gold tenor of this type of mineralisation is usually relatively high (2 g/t Au to 5 g/t Au).

- Significant low grade gold mineralisation (commonly below 2 g/t Au) is encountered within sediments immediately adjacent to the intrusive bodies within their respective alteration halos. The gold tenor are perceived to be closely correlated to alteration intensity and hence with primary porosity of the rocks. Sandstones and conglomerate are usually better mineralised than fine grain mud- and siltstones.

- High grade mineralisation (with grades exceeding 1,000 g/t Au) is associated with quartz-veins which are encountered in the upper portion of the intrusive bodies and in the altered and brittle portion of sediments directly above. Visible gold is common.

Figure 7.4.2 shows a typical mineralised granite sediment contact with cross cutting quartz veins.

Figure 7.4.2 Mineralised Granite in Contact with Sediments (after Runge, 2010)



It appears that weathering has an effect on the gold distribution. Boundary plots and sample histograms for the various weathering domains suggest that gold has been depleted in weathered (oxidised) material. Descending waters resulted in gold enrichment in the transition zone as well as in nugget growth where existing nuggets have been freed from country rock during weathering processes.

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8.0 DEPOSIT TYPES

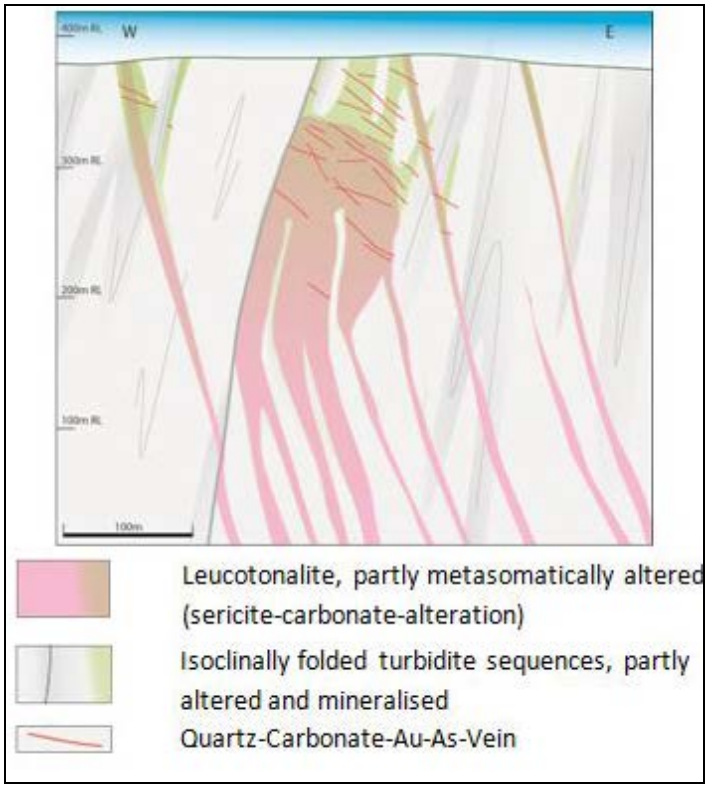
8.1 Deposit Types

Perseus carried out the geological interpretation for Sissingué. Widenbar and Snowden reviewed the interpretation prior to estimation and considers it to be an accurate reflection of the geology of the orebody. The following section is taken from Widenbar (2013) and summarises the geological interpretation provided by Perseus.

In the central part of the Sissingué prospect ('Sissingué Main'), the major host rock is a medium to coarse grained, veined, altered and sulphidised granite. Chemically identical to the porphyritic dykes, it is formed by several of these dykes that have been attenuated by a structural feature, possibly a fold plane, to flow into each other and to form a pear-shaped, approximately 50 m thick body (Figure 8.1.1).

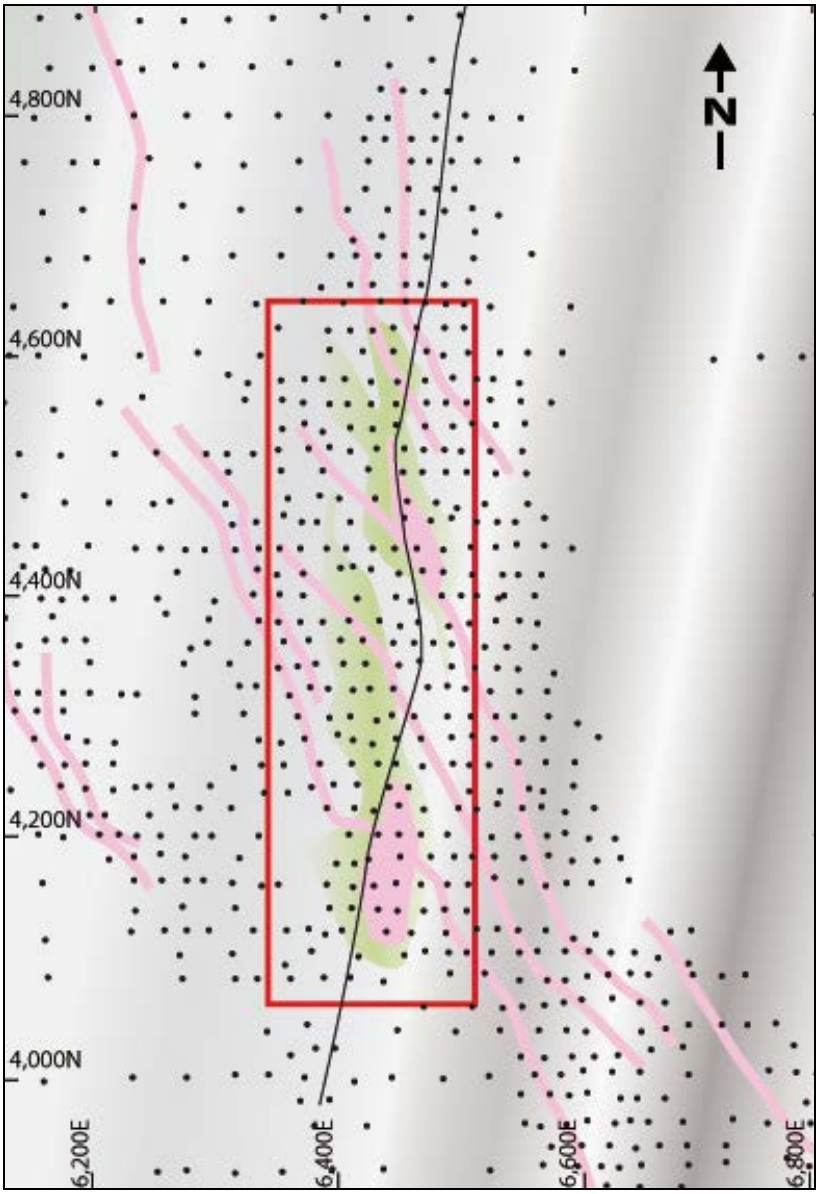
Alteration is most intense and gold grades, on average, are highest in the roof of this intrusive body and in the coarse grained sediments above. Emplacement of the dykes is believed to have occurred in several stages or over an extended period as several dykes cross cut the granitic body, some with fuzzy contacts (interpreted as emplaced during cooling period of the granitic body), others with sharp contacts (emplaced after cooling of granite). Gold mineralisation is relatively homogeneous in the upper portion the granite. At depth, with decreasing intensity of alteration, the tenor of the mineralisation decreases.

Figure 8.1.1 Schematic Cross Section (around, 1,154,200 mN) (after Widenbar, 2013)



Elsewhere, mineralisation is interpreted to be associated with isolated dykes and altered sediments in their respective vicinities. This distribution leads to a bi-directional ore body: Granite and associated altered sediments in north-north-eastern extent and with sub-vertical or steep westerly dips, and north-west striking dykes with associated altered sediments, obliquely cross cutting and steep to moderate easterly dips. The most significant zone of gold mineralisation is the Main Zone with its granitic bodies, various dykes and a pervasive alteration halo due to the larger mass of the intrusives (Figure 8.1.2).

Figure 8.1.2 Surface Geology of Sissingué Main Zone (Red) With Drill Collar Layout (after Widenbar, 2013)



8.2 Geology and Mineralisation Domains

A plan view of the major modelling domains and a list of the domains and files is shown in Figure 8.2.1 and Table 8.2.1, respectively.

Figure 8.2.1 Plan View of Mineralisation Domains with Drillholes and Project Areas

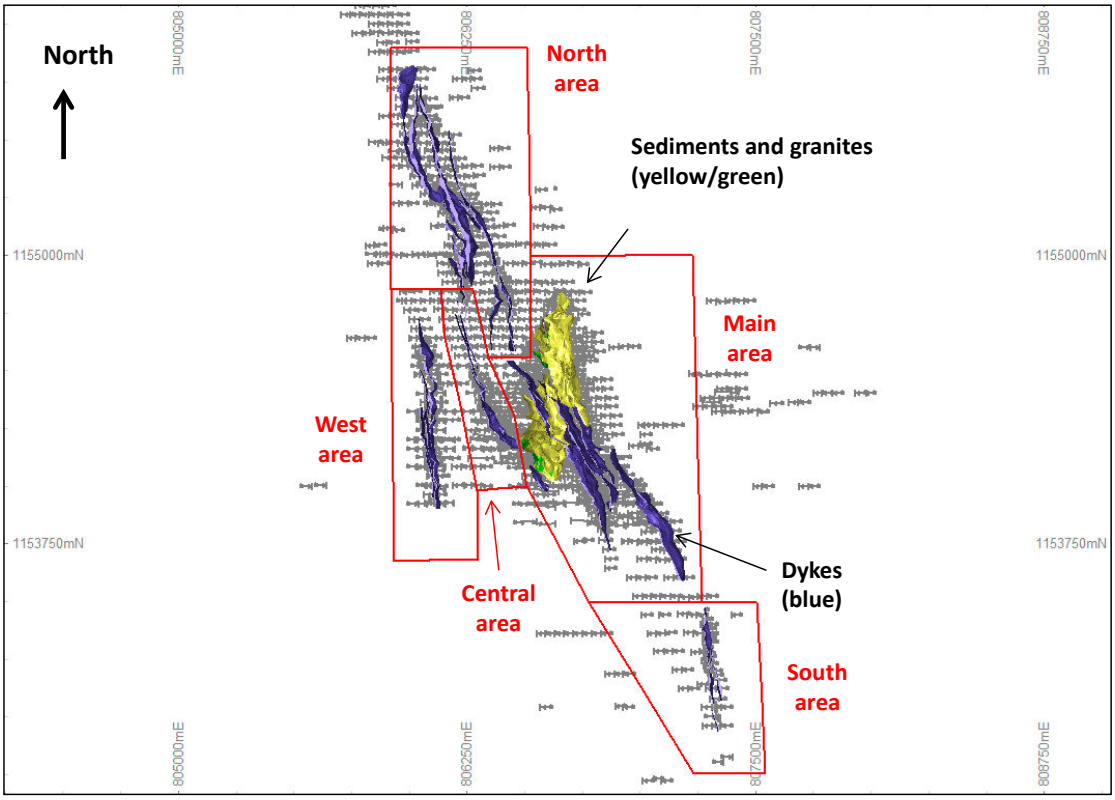


Table 8.2.1 Summary of Mineralisation Domains

Prospect	Domain	Wireframe
Main	Granite South	granite1.dxf
	Granite North	granite2.dxf
	Sediments South	sed_env1.dxf
	Sediments North	sed_snv2.dxf
	Dyke 1	dyke_1.dxf
	Dyke 2	dyke_2.dxf
	Dyke 3	dyke_3.dxf
	Dyke 4	dyke_4.dxf
	Dyke 5	dyke_5.dxf
	Dyke 6	dyke_6.dxf
	Dyke 7	dyke_7.dxf
	Dyke 8	dyke_8.dxf
Central	Dyke 11	dyke_c1.dxf
	Dyke 12	dyke_c2.dxf
	Dyke 13	dyke_c3.dxf
	Dyke 14	dyke_4c.dxf
North	Dyke21	dyke_n1.dxf
	Dyke22	dyke_n2.dxf
	Dyke23	dyke_n3.dxf
West	Dyke31	dyke_w1.dxf
	Dyke32	dyke_w2.dxf
	Dyke33	dyke_w3.dxf
South	Dyke41	dyke_s1.dxf
	Dyke42	dyke_s2.dxf

Mineralisation was domained by its host lithology for modelling (granite, porphyritic dykes and sediments). Within the dyke domain mineralised sediments in the alteration halo of the dykes were included in the dyke domain to maintain a minimum width of the wireframes (see below) and to maintain continuity along strike. Where geological contacts were not clearly controlling the distribution of mineralisation, a grade cut-off of 0.3 g/t Au was used to construct Mineral Resource boundaries and to provide overall geometry to mineralised zones. A minimum of 4 m width was used for the wireframes and samples of grades below the nominal cut-off of 0.3 g/t Au were included where the wireframe would otherwise be less than 4 m wide. Analysis of the global grade distribution shows that there is a natural change in grade population at around 0.3 g/t Au.

The interpreted geology and grade boundaries were digitised on 10 m and 20 m spaced, west-east orientated vertical sections and manually triangulated to form wireframe solids. To form ends to the wireframes along strike, either:

- the end sections strings were copied to a position midway to the next section and adjusted to match dip, strike and plunge of the zone

- a single point was digitised midway to the next section. Vertically, strings were closed at a depth that matched that of the adjacent sections.

The wireframe objects were validated prior to compiling the volume model. In total, 23 solids were created.

8.3 Topography and Weathering Domains

A topographic surface and three surfaces of the weathering profile were created by Perseus who supplied the wireframes to Snowden as dxf files as detailed in the following table (Table 8.3.1).

Table 8.3.1 Summary of Topography and Weathering Wireframes

Description	Wireframe
Topography	topo1.dxf
Base of laterite	bolat1.dxf
Base of total oxidation	botox.dxf
Base of transitional	bottr1.dxf

The topography covering the extent of the Sissingué Mineral Resource model was created as a digital terrain model (DTM) in Surpac using surveyed drillhole collars and an additional 639 survey points across the prospect.

Weathering interfaces were digitised on 20 m spaced, west-east orientated drill sections and automatically triangulated to form DTMs. A cross section illustrating the topography and the four weathering surfaces is shown in Figure 8.3.1.

Figure 8.3.1 Topography Cross Section

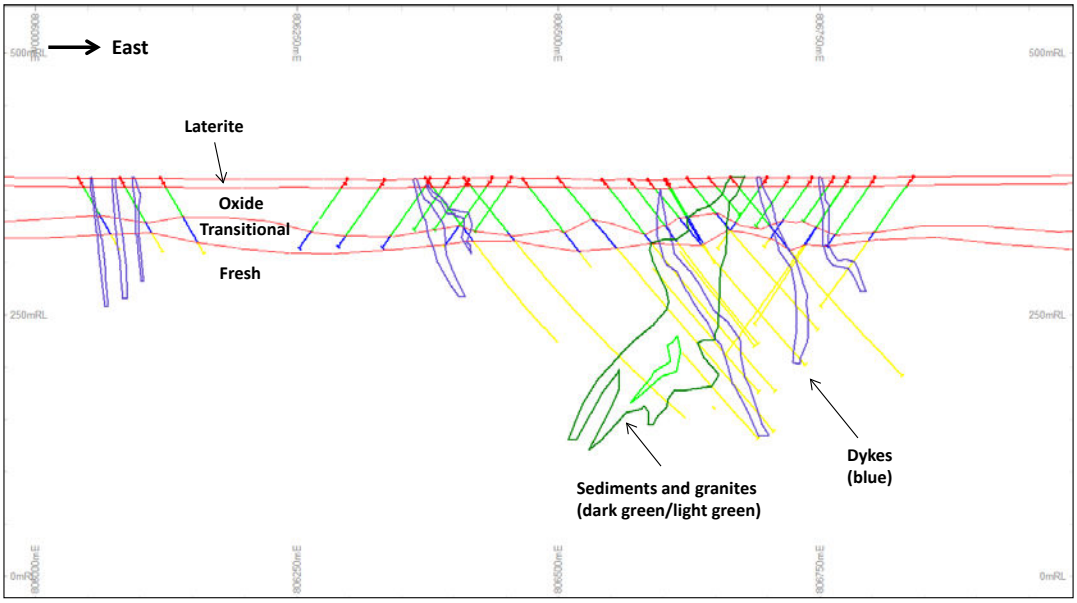


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9.0 EXPLORATION

9.1 Exploration Overview

Previous extensive systematic exploration on the Project, including airborne geophysics and extensive surface geochemical sampling, identified a series of soil anomalies over a strike length of 75 km, mostly along the Syama-Shear. The Morila-Tongon region has a historically high conversion rate of significant soil anomalies to gold discoveries.

Past exploration activities have been carried out by Randgold Resources Ltd, and Occidental Gold Sarl. A summary of this work is included in Section 6 of this report. Since 2004 all the exploration activities have been carried out by Occidental Gold Sarl, as a wholly-owned subsidiary of Perseus.

An airborne geophysical survey was carried out by HI Sense in December 1998, and flown at 75 m above ground level on east-west oriented lines. The digital airborne magnetics, radiometric and VLF database were re-interpreted by Southern Geoscience Consultants in Perth, Australia. From this, detailed structural and geological interpretations of the area were prepared and used to assist with exploration planning.

RAB drilling commenced in September 2002, however, the programme was disrupted by the outbreak of the civil war. Exploration activities and RAB drilling resumed in 2005. Since 2007, extensive RC and diamond core drilling has been carried out over the Sissingue prospect. Contract drillers Ivorian-based Foramin SARL (formerly Drillex Ltd) and Boart Longyear were used for the programs.

Between March and May 2008, an airborne VTEM/Magnetics survey was flown by Geotech Airborne Ltd, in a joint venture with Resolute Mining Ltd and Etruscan Resources Inc, both operating on the other side of the Bagoe River in Mali.

In 2008, Sagax Afrique was contracted for a ground IP survey over the Sissingue deposit, including processing and imaging of the data. Coffey Mining Pty Ltd in Notting Hill, Australia, was contracted to carry out a structural study during a site visit that resulted in the current geological model of the Sissingué ore body. Resource definition drilling at Sissingué was completed in mid-2012.

Since 2010, the focus of exploration has gradually shifted towards other prospects close to Sissingue (e.g. Papara, Kanakono, and Mbasso) in an effort to add resources to a potential mine.

In 2012, consulting geophysicist Paolo Constantini was contacted to reprocess the existing ground IP data and to reinterpret the available geophysical data set for regional target identification.

A summary of the exploration completed is given in Table 9.1.1. Drilling at Sissingué and in other adjacent areas is presented in Table 9.1.2.

For the Mineral Resource estimate only samples from RC and diamond core drilling, carried out by Perseus since 2005, have been used. Exploration results by previous owners are not included.

Table 9.1.1 Summary of Exploration Activities

Period	Surveys	Geochemical Samples	Pitting	Auger	Drilling			
					RAB / AirCore		RC / DD	
		#	#	#	#	[m]	#	[m]
1998 - 1999	Airborne Magnetics, Radiomagnetics, VLF	12,254	370					
2001 - 2002		5,737	119		1	22		
2002 - 2004	No exploration activities due to civil unrest							
2005			55		104	4,608	0	0
2006					346	14,271	0	0
2007					388	13,932	138	11,236
2008	VTEM, IP				450	14,869	332	37,888
2009		2,954			738	28,629	549	43,908
2010		1,868			573	26,444	687	66,394
2011		560			737	35,068	206	22,028
2012					697	35,333	460	51,240
2013					0	0	0	0
2014				252	136	5,592	0	0

Table 9.1.2 Summary of Drilling at Sissingue and at Other Prospects on the Permit

Year	Drilling							
	Sissingué				Other			
	RAB / AirCore		RC / DD		RAB / AirCore		RC / DD	
	#	[m]	#	[m]	#	[m]	#	[m]
2005	104	4,608	0					
2006	271	11,346	0		75	2,925		
2007	114	4,887	138	11,236	274	9,045		
2008	0		332	37,888	450	14,869		
2009	539*	21,531	549	43,908	199	7,098		
2010	261*	14,219	507	50,347	312	12,225	180	16,047
2011	470	25,434	169	18,757	267	9,634	37	3,271
2012	227	14,035	327	39,415	470	21,298	133	11,825
2013	0		0		0		0	
2014	0		0		136	5,592	0	

** Including sterilisation drilling for mine planning*

9.2 Exploration Potential

All existing geochem anomalies, as identified through soil sampling, have been thoroughly drill tested, without significant discovery apart from Sissingué itself. 'Near-mine' exploration potential exists underneath the large alluvial plains of the Bagoe River to the east and north-east of Sissingué that have not been soil sampled, or where soil sampling was ineffective. Auger drilling with RAB follow-up has been carried out since 2014 and is currently on-going. The structural corridors between Sissingué and Papara to the north, as well as between Sissingué and Kanakono to the south, as identified by the airborne geophysics, represent another near-mine target, though soil sampling (2009-2011) did not return anomalous results.

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10.0 DRILLING

10.1 Introduction

The input dataset used for the Sissingué resource estimate contained 69,728 m RC and 62,019 m DD samples for 201,551 m of which 36,113 m lie within the mineralised domains.

The total dataset within the Sissingué Mineral Resource area is summarised in Table 10.1.1 and illustrated in plan view Figure 10.1.1. Figure 10.1.2 to Figure 10.1.5 show representative drill sections through the mineral deposit.

Table 10.1.1 Sissingué Drillhole Data Summary

Hole Prefix	Drill Type	Number of Holes	Number of Metres
SD	DD	315	67,862.75
SGC	RC	58	3,230
SLC	RC	356	33,246
SRC	RC	1,271	94,551
SRCOX	DD	4	285,6
SRD	RC/DD	18	2,375.7
Total		**2,02**2	**201,**551.0**5**

The disseminated nature of the mineralisation and lack of discrete structures controlling grade distribution meant that selective sampling was not warranted. Nominal sample length was 2 m for RC drilling and 1 m for DD drilling throughout the Project. 58 RC holes (SGC prefix) were drilled at 10 x 10 m over one of the high grade area with 1m samples assayed.

Drill holes have been oriented and inclined in order to intercept the targeted mineralisation perpendicularly i.e. towards the east to test the west-dipping granite, and towards the west for the east-dipping porphyritic dykes. True thickness of intersections is typically close to the downhole thickness. The steep nature of mineralisation within the sediments, however, meant that holes were at a moderate angle to mineralisation and true thickness of intersections is typically about 50-70% of the downhole thickness.

Perseus observed that core and RC samples showed very good recoveries with overall average of 99.5% recovery in fresh rock for core drilling. Perseus considers the samples to be representative of the mineralisation defined by the drilling.

Figure 10.1.1 Plan View of Sissingué Drillhole Data

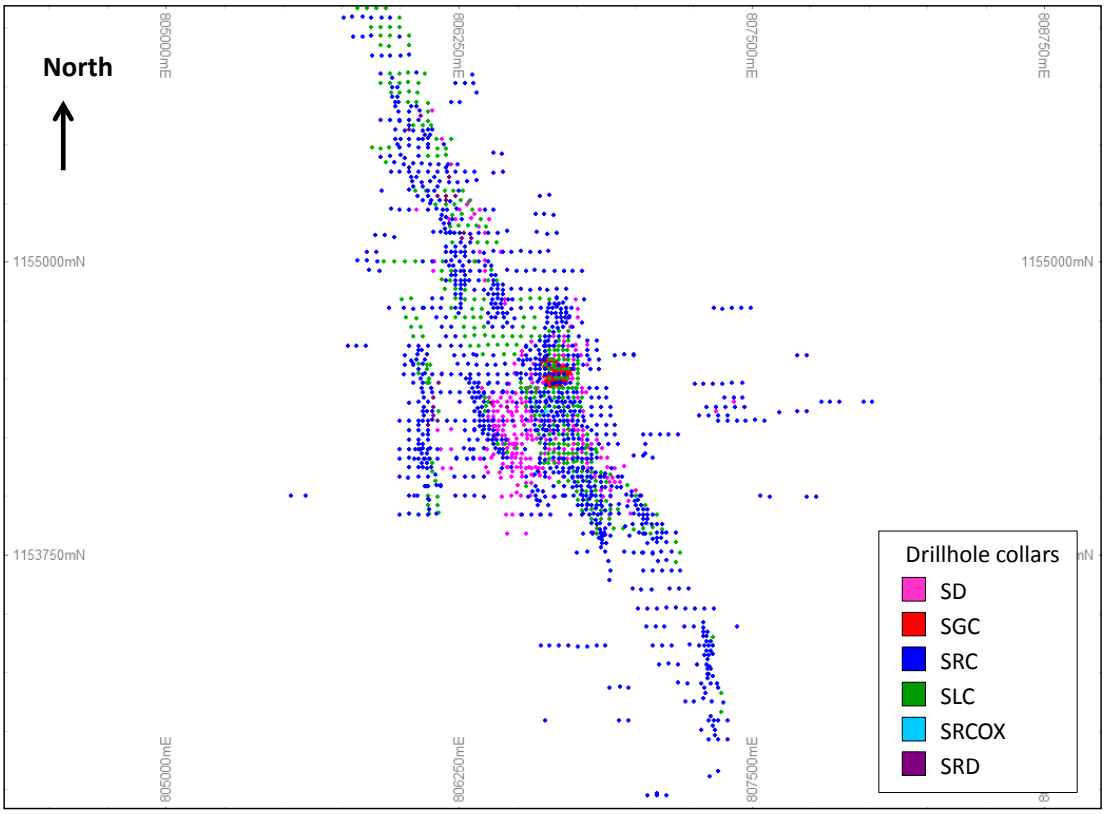


Figure 10.1.2 Sissingué Drill Section 1,154,080N

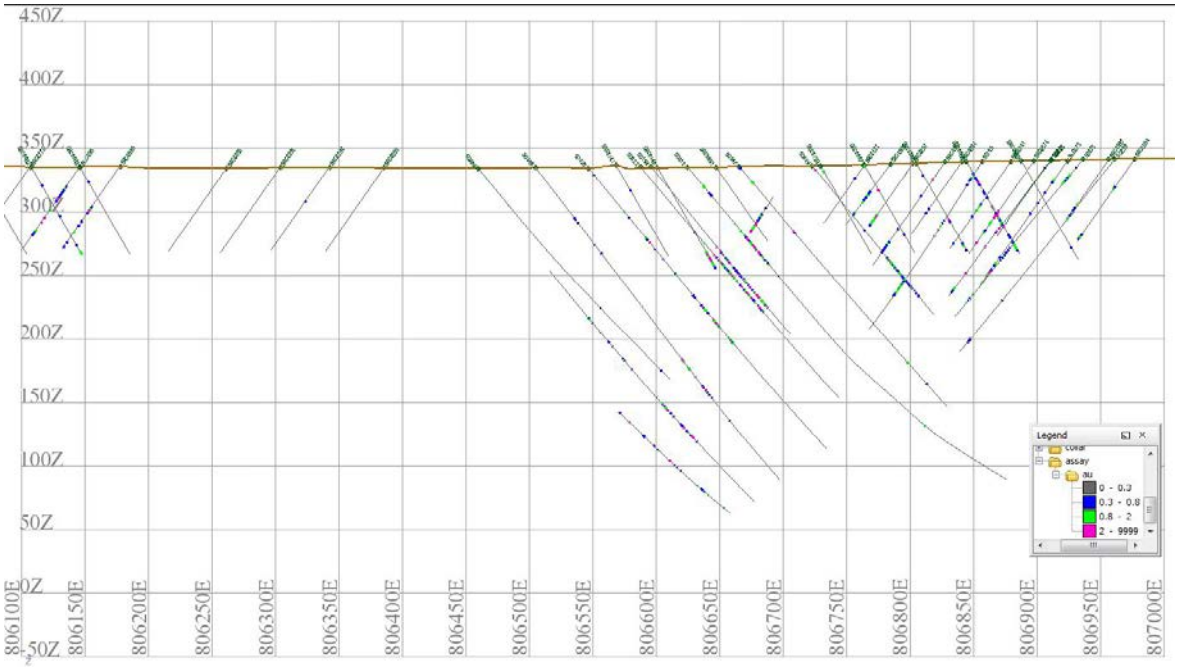


Figure 10.1.3 Sissingué Drill Section 1,154,200N

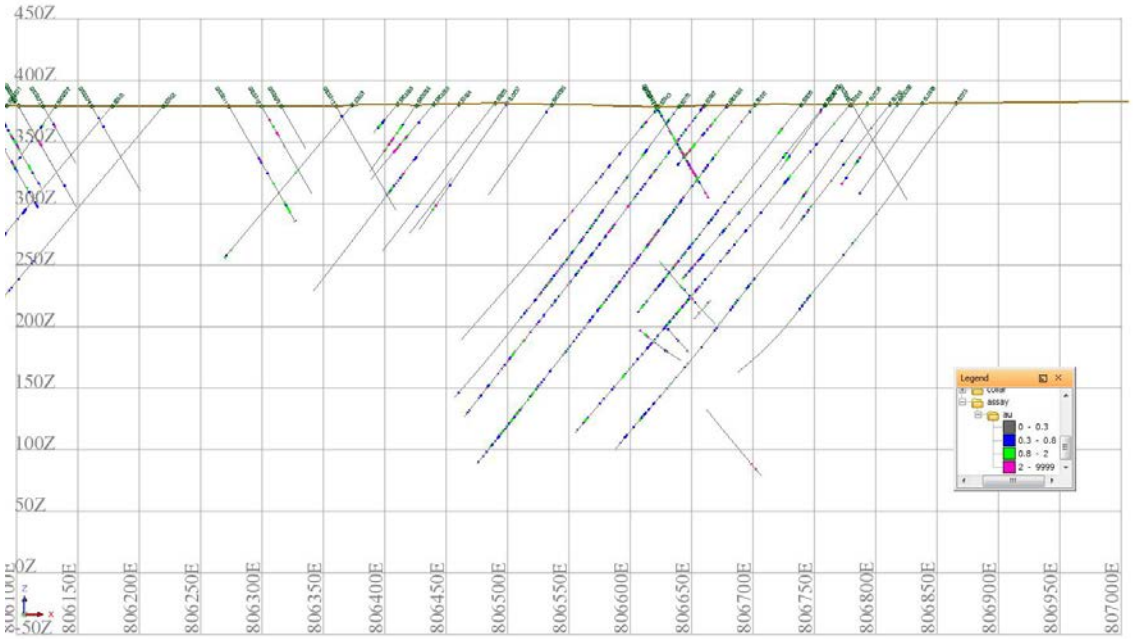


Figure 10.1.4 Sissingué Drill Section 1,154,320N

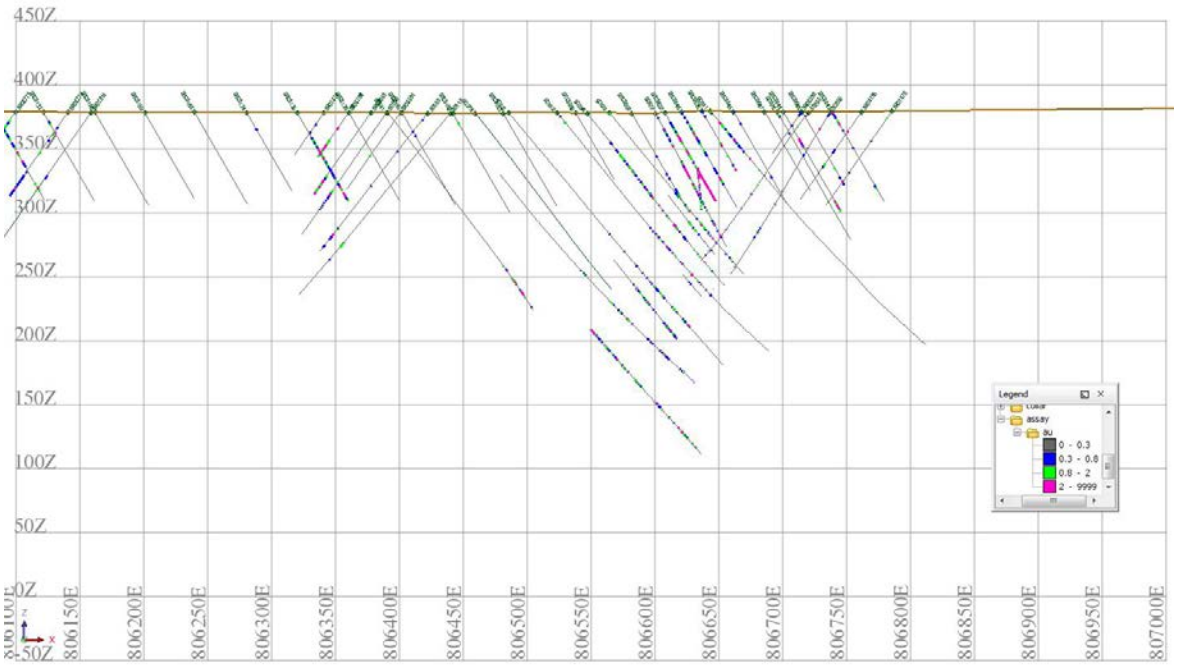
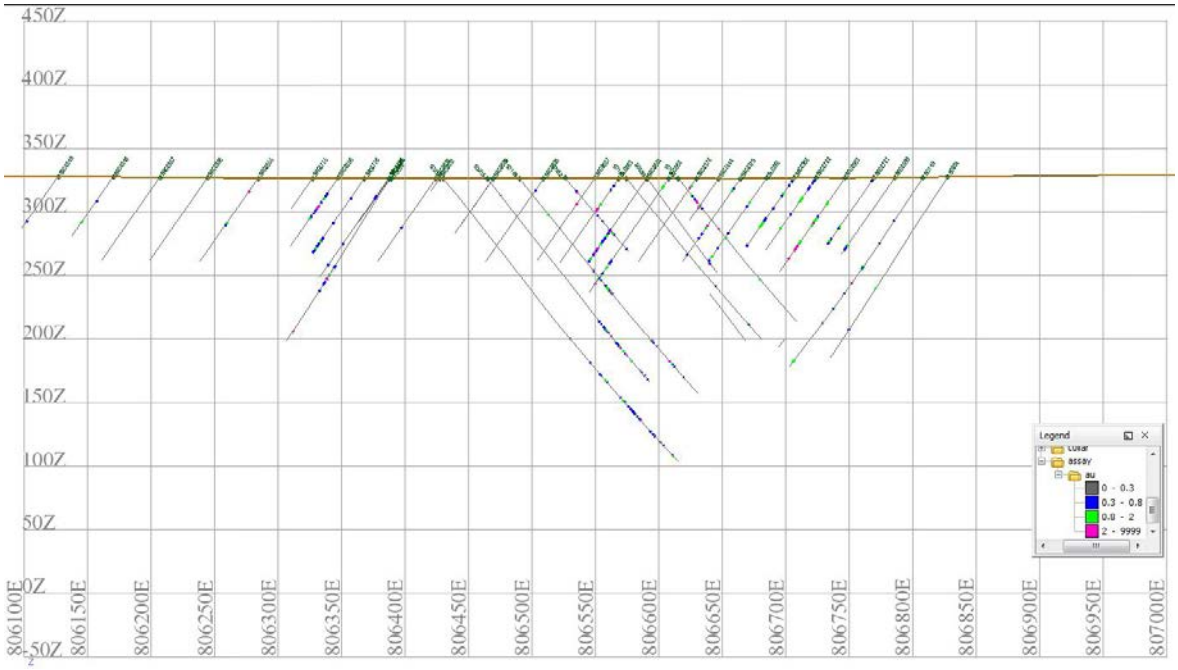


Figure 10.1.5 Sissingué Drill Section 1,154,440N



10.2 Reverse Circulation Drilling

Since March 2007, a total of 1,685 RC holes totalling 131,027 m have been drilled at Sissingué within an area of 5.7 km strike and 2.5 km width, of which 931 holes have been used for the Mineral Resource estimate. Drilling was carried out on west-east aligned lines at nominal 80 m by 80 m spacing followed by 40 m by 40 m and 20 m by 20 m infill. A 120 m long zone between 1,154,460 mN and 1,154,580 mN has been drilled on a 10 m by 10 m grid (59 holes). All RC holes were aligned either 90° east or 270° west with inclinations between -50° and -60°.

1,326 RC holes were completed by contract drilling company Foramin Limited (formerly Drillex Limited), using four different drill rigs as follows:

- Two GK850 multi-purpose rigs, mounted on tracks with auxiliary Sullair compressors (900 cfm/350psi).
- DB750 multi-purpose rig, mounted on tracks and equipped with an Elgi compressor (900 cfm/350psi) and a separate Hydco V8 booster for additional 1,200 cfm/650psi.
- DB900 track mounted RC rig with an Elgi compressor (900 cfm/350psi) and a separate Hydco V8 booster for additional 2,300 cfm/750psi.

The other 359 RC holes have been drilled by contract drilling company Boart Longyear using a CATMAX RC rig, equipped with a Sullair compressor (1,350 cfm/500psi).

All RC rigs used at Sissingué were equipped with 5¼ inch diameter hammer. Most RC holes (1,072) were designed to stop at a nominal depth of 80 m or less. The deepest RC hole was 150 m.

RC sampling was generally conducted for the full length of the drillholes at regular 1 m intervals with 2 m composites submitted for analysis, with exception of the 58 RC holes drilled on the 10 m by 10 m grid where 1 m samples were submitted for analysis. Compositing of two consecutive samples was done using a scale to ensure equal representation of the individual samples. Nominally, composite samples consist of two 2.5 kg samples collected from the RC drilling are considered to be representative of the mineralisation.

RC drilling intersected sediments, granite and feldspar porphyry lithologies. The holes were sampled in their entirety. All RC samples were collected at the drill site at 1 m interval and split using a multi-stage riffle splitter. Each two consecutive samples were composited (where applicable) in one bag and given a pre-printed sample ticket.

The natural water table varies by season, but is typically at around 30 m to 40 m vertical in depth. With the use of a booster compressor during drilling, samples generally stay dry until approximately 80 m depth. Wet samples were taken from the drill bag at the cyclone using a spear. Approximately 4% of all RC drill samples have been reported to be wet. There is no evidence of downhole smearing, depletion or enrichment during wet RC drilling.

A total of 69,728 RC samples were submitted to analytical laboratories for gold-only analysis.

10.3 Diamond Core Drilling

Diamond (DD) drilling commenced at Sissingué in January 2008. A total of 319 diamond drillholes for 68,148.35 m have been completed over the Sissingué prospect and an additional 18 RC holes that have been continued with diamond tails (for 898 m). All but one diamond drillhole were drilled either 90° east or 270° west with inclination between -50° and -60°. Drillhole SD80 was drilled at -50° towards 0° north.

260 diamond holes, and all 18 combined RC and diamond holes, have been completed by Foramin Limited with a track mounted YDXL / Coretec core rig, and the two multi-purpose rigs (see above). The other 59 diamond holes have been drilled by Boart Longyear with a platform-mounted Boart Longyear LY44 core rig. Drilling was typically completed at HQ size in weathered material and NQ size in fresh rock. All core was orientated by the 'AceTool' orientation tool. The first 43 holes drilled by Boart Longyear were orientated using a spear.

Diamond core was geologically and structurally logged, then sawn in half using a motorised diamond blade saw. Half core samples of oxide and transitional material were taken accordingly to core run (1.5 m) and every meter in fresh rock where core recovery usually was close to 100%. The material of the right side of the core was submitted for analysis, the left side stored in trays for reference.

Core sampling was conducted for the full length of the drillholes in regular intervals. No attempt was made to selectively sample portion of the holes or according to lithology. A total of 62,019 samples were dispatched to analytical laboratories for gold-only assaying. Samples collected from diamond drilling are considered to be representative of the mineralisation.

10.4 Collar Location and Survey

Drill holes were planned, pegged and surveyed using the WGS84 (Zone 29 North) metric grid system.

The collar locations have been surveyed from 1,225 of the RC and diamond core drill holes within the Mineral Resource area. Only four RC drill hole collars from drill holes used for the Mineral Resource estimate have not been surveyed, but estimated to be within 2 m to 3 m accuracy of the position established by handheld GPS.

All RC and diamond holes until September 2009 were surveyed using differential GPS, by a certified contract surveyor (SEMS Exploration Services Ltd, Ghana). Drillholes between September 2009 and October 2010 were surveyed by contract surveyor CBM Surveys Ltd of Ghana. All subsequent drillholes were surveyed by the Company's surveyor.

All diamond core holes were downhole surveyed nominally every 30 m with a 'Flexit SmartTool' digital downhole survey instrument. Most RC holes have not been surveyed downhole. Measurements of inclination have been taken at collar with a goniometer, azimuths taken with a precision compass.

All compass readings and magnetic azimuths recorded by the Flexit tool were adjusted in the data base by -6° to allow for the prevalent magnetic declination at the Project area and to convert the data from magnetic north to true north.

10.5 Geological Logging

RC drillholes have been logged from chip boards. Logged items include lithology, weathering, colour, grain size, vein type and estimated percentage, sulphide mineralisation and their estimated percentage, alteration and alteration intensity. While logging of fresh RC chip generally poses little challenges, strongly or entirely oxidised RC chips are often difficult to log. After commencement of diamond core drilling and a better understanding of the encountered rock types and structures, all chip boards have been re-logged.

Diamond core has been logged at the core yard on full core prior to cutting. Logged items include lithology, weathering, colour, grain size, vein type and vein volume percentage, sulphide mineralisation and their estimated percentage, alteration and alteration intensity. Structural measurements were taken regularly on all orientated core including fault, fold, cleavage and joint orientation, lithological contacts, vein orientation and bedding.

Geotechnical logging was carried out on all core and includes core recovery, RQD, rock strength and number and type of defects.

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11.0 SAMPLING PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation and Assaying

11.1.1 Analytical Laboratories Used

Three analytical laboratories have been used to assay samples from the Sissingué project:

- ALS Chemex Laboratories (ALS), in Bamako, Mali with head office in Vancouver, Canada. This laboratory was formerly operated by Abilabs Ltd (Abilabs) until taken over by ALS Global in March 2007.

- Intertek Minerals Ltd (Intertek) in Tarkwa, Ghana under the umbrella of Intertek Minerals / Genalysis Laboratory Services Pty Ltd with head office in Maddington, Australia. This facility was until August 2010 operated by Transworld Laboratories Ltd (TWL).

- Bureau Veritas Minerals Laboratory (BVML) in Abidjan, Côte d'Ivoire with head office in Paris, France.

All three laboratories are independent of Perseus.

A summary of the sample dispatch and assay methods are given in Table 11.1.1. All assaying has been carried out via fire assay (FA) or BLEG bottle roll (BR).

ALS Global Canada are covered by ISO 9001:2008 certification for the provision of assay and geochemical analytical services by QMI Quality Registrars (1 February 2011) and accredited under ISO 17025 for its analytical capabilities by SCC (31 August 2008).

ALS Global South Africa is accredited under ISO 17025 by SANAS (6 February 2013). The laboratory in Bamako is not accredited, but operates under the same quality protocol and the same work instructions as the accredited ones in Canada and South Africa.

Intertek / Genalysis Laboratory Services are accredited under ISO 17025 by NATA (No 3244 of 25 May 2011).

BVML Abidjan, Côte d'Ivoire has been accredited by Ultra Trace Pty Ltd in September 2010 but do not have a formal international accreditation. Ultra Trace accreditation covers technical competence in sorting and drying, weighing, crushing, splitting, pulverisation, sieving and gold analysis by fire assay with a detection limit of 0.01 ppm.

Table 11.1.1 Summary of Assay Methods

Laboratory	Drill Type	Number of Samples			Period
		Dispatched	Method	Analysed	
ALS	DD	15,737	FA BR	15,583 154	since March 2007
	RC	18,831	FA	18,831	
Abilabs	DD	0			until March 2007 (then acquired by ALS)
	RC	3,475	FA BR	307 3,168	
Intertek	DD	23,600	FA	23,600	since August 2010
	RC	6,020	FA	6,020	
TWL	DD	21,370	FA	22,373	until August 2010 (then acquired by Intertek)
	RC	36,726	FA	36,726	
BVML	DD	1,312	FA	1,312	since December 2012
	RC	4,676	FA	4,676	
Total	**DD**	**6**2,019	**FA** **BR**	**6**1,865 **154**	
	RC	**69,728**	**FA** **BR**	**66,560** **3,168**	

Perseus staffs have carried out regular unannounced laboratory visits to BVML, Côte d'Ivoire; the latest during April 2013. No material issues were observed.

11.1.2 Sample Preparation

RC samples are collected at the drill site and split through a multistage riffle splitter and each two consecutive samples are bagged into a 2 m composite and sent to the laboratory for preparation and analysis. Perseus employees carry out the sampling at the drill site.

Diamond drilling, core cutting and bagging of samples is conducted by Perseus employees and then sent to the laboratory for preparation and analysis. The right hand side of the core looking downhole was always submitted for analysis with the left side being stored in trays on site. After dispatch from the Project, Perseus staff had no involvement in the sample preparation or assay procedures of either RC or diamond samples.

Both core and RC samples followed a standard path of drying at 105° to 110°C, crushing and grinding. Samples were pulverised with a ring mill and thoroughly mixed on a rolling mat ('carpet roll'), and then a 200 g of sub-sample was split off. Internal laboratory checks required at least 90% of the pulp passing -75 microns.

11.1.3 Sample Analysis

Two types of analysis for gold were performed, a standard fire assay and BLEG bottle roll. Similar fire assay procedures were followed by ALS, Intertek and BVML. 50 g sub-samples (and at BVML 40 g sub-samples) were taken from the pulverised material and mixed with flux, then fired at 1,100°C. The prill is then dissolved in Aqua Regia solution and the resultant liquor read by AAS with a detection limit of 0.01 g/t Au.

For the bottle roll analysis, a 1 kg portion of the pulverised material is placed into the BLEG bottles with cyanide solution and water. The mix is then rolled for 24 hours with an additional two hours for settling. A 50 mL sub-sample of the clear liquor is filtered into a flask and extracted into 5 mL of DIBK which is then read by AAS with a detection limit of 0.01 g/t Au.

The first 26 RC holes were analysed by 1 kg bottle roll, however, analysis of the tails showed that, on average, 20% of the gold was not recovered with this method. Subsequently, almost all samples were analysed by fire assay. In total, 3,168 RC samples and 154 diamond core samples (from two drillholes) were analysed by bottle roll.

11.2 Quality Assurance and Quality Control

Perseus retains overall responsibility for the database quality including quality assurance (QA) and quality control (QC). QAQC analysis shown in the following sections is sourced from Widenbar, 2013 (Sissingué Resource Estimate).

Extensive quality control (QC) programs have been used throughout the Perseus drilling programs to check the precision and accuracy of the laboratory analysis.

Initially, every 25th RC sample was duplicated and a blank inserted at a ratio of 1 in 25. Since April 2007, certified standard material was submitted in a rate of 1 in 50 samples and then at a rate of 1 in 20 since early 2008 for both RC and diamond core samples. Certified material was supplied by Gannet and Rocklabs until November 2008 and Geostats Pty Ltd (Geostats) since November 2008. QC samples were inserted by Perseus employees prior to sample dispatch.

In early 2012, Perseus standardized its QAQC protocols for drilling and sampling across its various projects by implementation of formal, company-wide standard operating procedures (SOPs).

The results of the QC analysis indicate that reasonable precision and accuracy has been achieved. Previous consultants including RPM, Widenbar and Snowden consider that the sample preparation, security and analytical procedures are adequate for the project. This conclusion reflects accurately the opinion of the author.

11.2.1 Blanks

Blank reference material samples were inserted during RC and diamond sampling as part of the QC routine. Table 11.2.1 lists the blanks used since 2011. The analysis of the blank material indicates that generally there is good sample preparation with little evidence of contamination. Two isolated occurrences of failed blanks at ALS (Job No BK11116000 in 2011 and BK11224176 in 2012) remain unexplained. In both cases, the preceding samples were not or only very weakly mineralised and contamination is not likely.

Blank analysis prior to 2011 has been discussed in Runge's Technical Report (Runge, 2010) and indicates no material issues.

Table 11.2.1 Blanks Used Since 2011

CRM	Au Value	Standard	Source
	ppb	Deviation	
GLG302-4	3.23	1.27	Geostats
GLG307-1	2.86	1.70	Geostats
GLG307-3	2.83	1.80	Geostats
GLG907-1	3.58	2.61	Geostats
GLG910-5	2.78	1.58	Geostats
GLG911-3	3.74	2.84	Geostats

11.2.2 Standards

Certified reference material (CRM) sourced from Gannet and Rocklabs (until November 2008) and Geostats (since November 2008) has been used to test the accuracy of the sample analysis for both RC and diamond core samples. Since beginning of 2011, 29 different CRM have been used covering a grade range between 0.154 g/t Au and 7.24 g/t Au (Table 11.2.2).

Since early 2011, QAQC data has been reviewed for each of the assay batches upon return from the laboratory. QAQC samples which fall outside specified tolerance ranges are flagged and investigated.

Results of CRM assays were plotted over time for each sample batch and over grade range for each laboratory. An analysis of performance of CRMs by type and grade range shows that all three laboratories operate satisfactorily. 1% of all standards submitted since 2011 reported values three standard deviations below or above the expected value, in all three laboratories (Table 11.2.3). This is considered by Snowden and in the opinion of the author to be acceptable and in line with industry standards.

Detailed quality control plots for all batches since 2011 were developed; earlier batches have been discussed in Runge's Technical Report (Runge, 2010) and indicate no material issues.

Table 11.2.2 Certified Reference Materials Used

CRM	Au Value ppm	Standard Deviation	Source
GLG304-1	0.154	0.017	Geostats
G902-1	0.390	0.040	Geostats
G909-6	0.570	0.030	Geostats
G999-2	0.630	0.060	Geostats
G398-4	0.660	0.050	Geostats
G901-9	0.690	0.040	Geostats
G998-6	0.800	0.060	Geostats
G907-2	0.890	0.060	Geostats
G907-2	0.890	0.060	Geostats
G910-10	0.970	0.040	Geostats
G302-6	0.990	0.050	Geostats
G306-2	1.049	0.072	Geostats
G307-2	1.080	0.050	Geostats
G02	1.200	0.080	Geostats
G307-8	1.990	0.080	Geostats
G302-3	2.330	0.120	Geostats
G308-3	2.500	0.110	Geostats
G901-1	2.580	0.130	Geostats
G909-5	2.630	0.100	Geostats
G398-6	2.940	0.160	Geostats
G998-1	2.950	0.120	Geostats
G907-4	3.840	0.150	Geostats
G397-2	4.490	0.180	Geostats
G306-7	4.500	0.200	Geostats
G307-5	4.870	0.172	Geostats
G904-8	5.530	0.180	Geostats
G996-7	5.990	0.390	Geostats
G905-10	6.750	0.250	Geostats
G906-8	7.240	0.270	Geostats

Table 11.2.3 CRM Results Since 2011

CRM				ALS / Mali *			Intertek / Ghana **			BVML / Mali ***			All CRM		
ID	Source	Exp Value	Std Dev	#	> +-3 Std Dev	% Failed	#	> +-3 Std Dev	% Failed	#	> +-3 Std Dev	% Failed	#	> +-3 Std Dev	% Failed
ST16/5357	Gannet	0.52	0.03	7	0	0.0							7	0	0.0
GLG304-1	Geostats	0.154	0.017	70	0	0.0							70	0	0.0
G902-1	Geostats	0.39	0.04							77	0	0.0	77	0	0.0
G909-6	Geostats	0.57	0.03	129	2	1.6	48	5	10.4				177	7	4.0
G999-2	Geostats	0.63	0.06	190	0	0.0	136	0	0.0				326	0	0.0
G398-4	Geostats	0.66	0.05				136	0	0.0	11	0	0.0	147	0	0.0
G901-9	Geostats	0.69	0.04	30	1	3.3							30	1	3.3
G998-6	Geostats	0.8	0.06				19	0	0.0	28	1	3.6	47	1	2.1
G907-2	Geostats	0.89	0.06	155	0	0.0	154	1	0.6	6	0	0.0	315	1	0.3
G910-10	Geostats	0.97	0.04							33	1	3.0	33	1	3.0
G302-6	Geostats	0.99	0.05	24	2	8.3							24	2	8.3
G306-2	Geostats	1.049	0.072	18	0	0.0	116	0	0.0	19	0	0.0	153	0	0.0
G307-2	Geostats	1.08	0.05	26	2	7.7							26	2	7.7
G02	Geostats	1.2	0.08	14	0	0.0							14	0	0.0
G307-8	Geostats	1.99	0.08							7	1	14.3	7	1	14.3
G308-3	Geostats	2.5	0.11							23	0	0.0	23	0	0.0
G901-1	Geostats	2.58	0.13				78	1	1.3				78	1	1.3
G909-5	Geostats	2.63	0.1	21	0	0.0	19	0	0.0				40	0	0.0
G398-6	Geostats	2.94	0.16	8	1	12.5							8	1	12.5
G907-4	Geostats	3.84	0.15	170	0	0.0	63	1	1.6				233	1	0.4
G397-2	Geostats	4.49	0.018				4	1	25.0	48	0	0.0	52	1	1.9
G307-7	Geostats	4.5	0.2	13	0	0.0							13	0	0.0
G904-8	Geostats	5.53	0.18	6	0	0.0							6	0	0.0
G996-7	Geostats	5.99	0.39				126	0	0.0	6	0	0.0	132	0	0.0
G905-10	Geostats	6.75	0.25				10	0	0.0				10	0	0.0
G906-8	Geostats	7.24	0.27	148	1	0.7				189	2	1.1	337	3	0.9
Summary				**1,029**	**9**	**0.9**	**909**	**9**	**1.0**	**447**	**5**	**1.1**	**2,385**	**23**	**1.0**

* ALS/Mali June 2011 to June 2012, ** Intertek/Ghana July 2012 to January 2013, *** BVML/Mail December 2012 to January 2013.

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12.0 DATA VERIFICATION

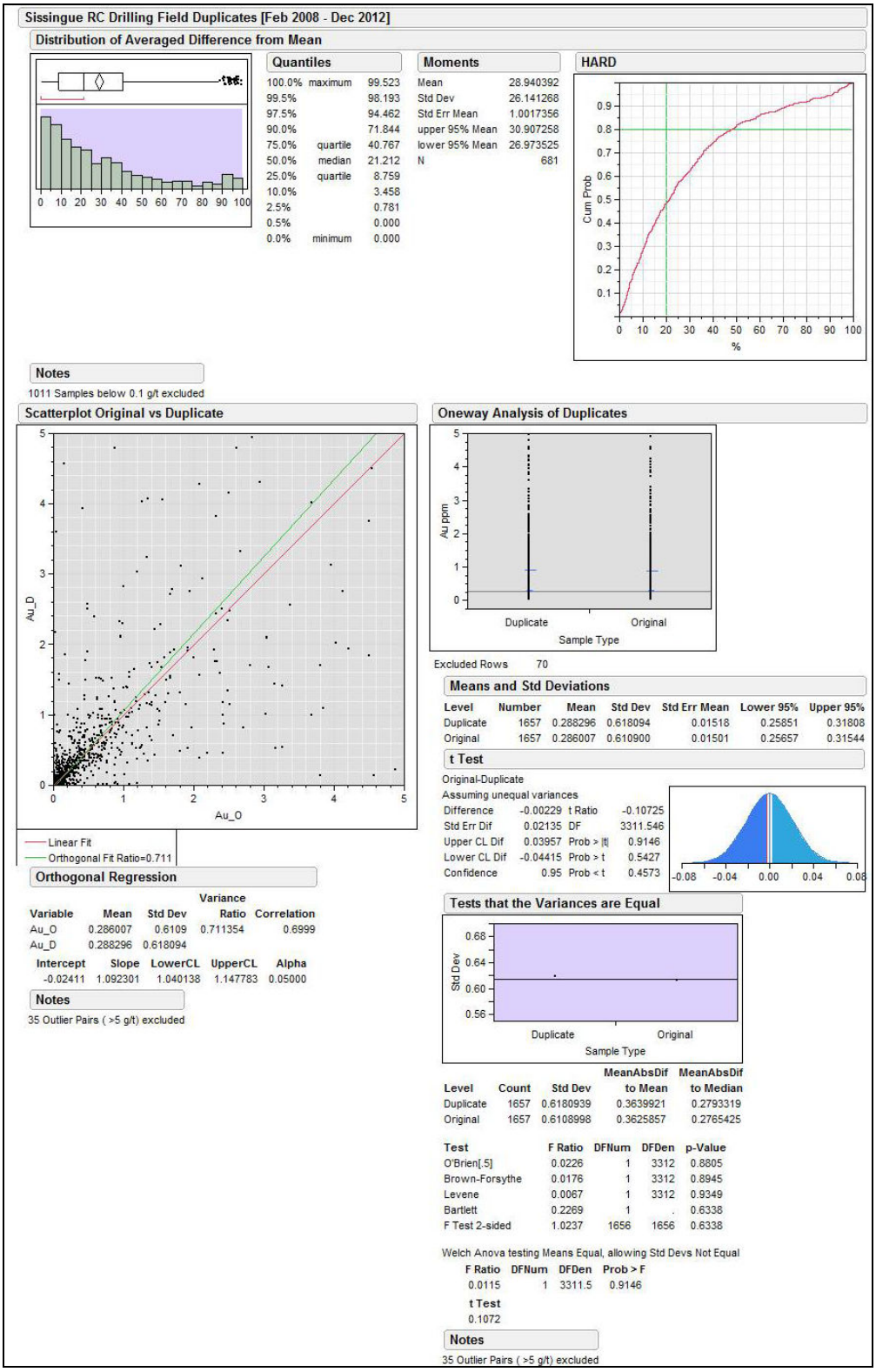
12.1 Field Duplicates

A total of 1,696 field duplicates were taken during RC drilling between February 2008 and the December 2012. Initially, nominally every 50th RC sample was duplicated. However, with increasing knowledge of the mineralisation, selective intervals were duplicated, usually one 2 m composite per drillhole, to achieve a meaningful data set and to avoid duplicating barren intervals.

No field duplicates have been submitted from diamond core drilling as Perseus considered the use of ¼ core as a field duplicate to be inadequate given the style of mineralisation, and the submission of the full remaining ½ core was considered to be undesirable. Snowden recommends that some of the remaining 50% splits should be submitted to assess the precision of the sampling process, even though it will mean there will be some intervals with no residual core.

When outliers (both original and duplicate sample >5 g/t Au) and near detection limit samples (below 0.1 g/t Au = 10x detection limit) are removed, the Analysis of Variance (ANOVA) of the field duplicates shows that no systematic sample bias has been introduced during routine RC sampling: means and standard deviation of both populations are quasi-identical (Figure 12.1.1). However, the HARD plot (without outliers removed) indicates that only 50% of duplicates are within 20% of the value of the original samples. This suggests a nuggety mineralisation and supports the high nugget effects, particularly for the sediment domains.

Figure 12.1.1 Analysis of Sissingué RC Drilling Field Duplicates

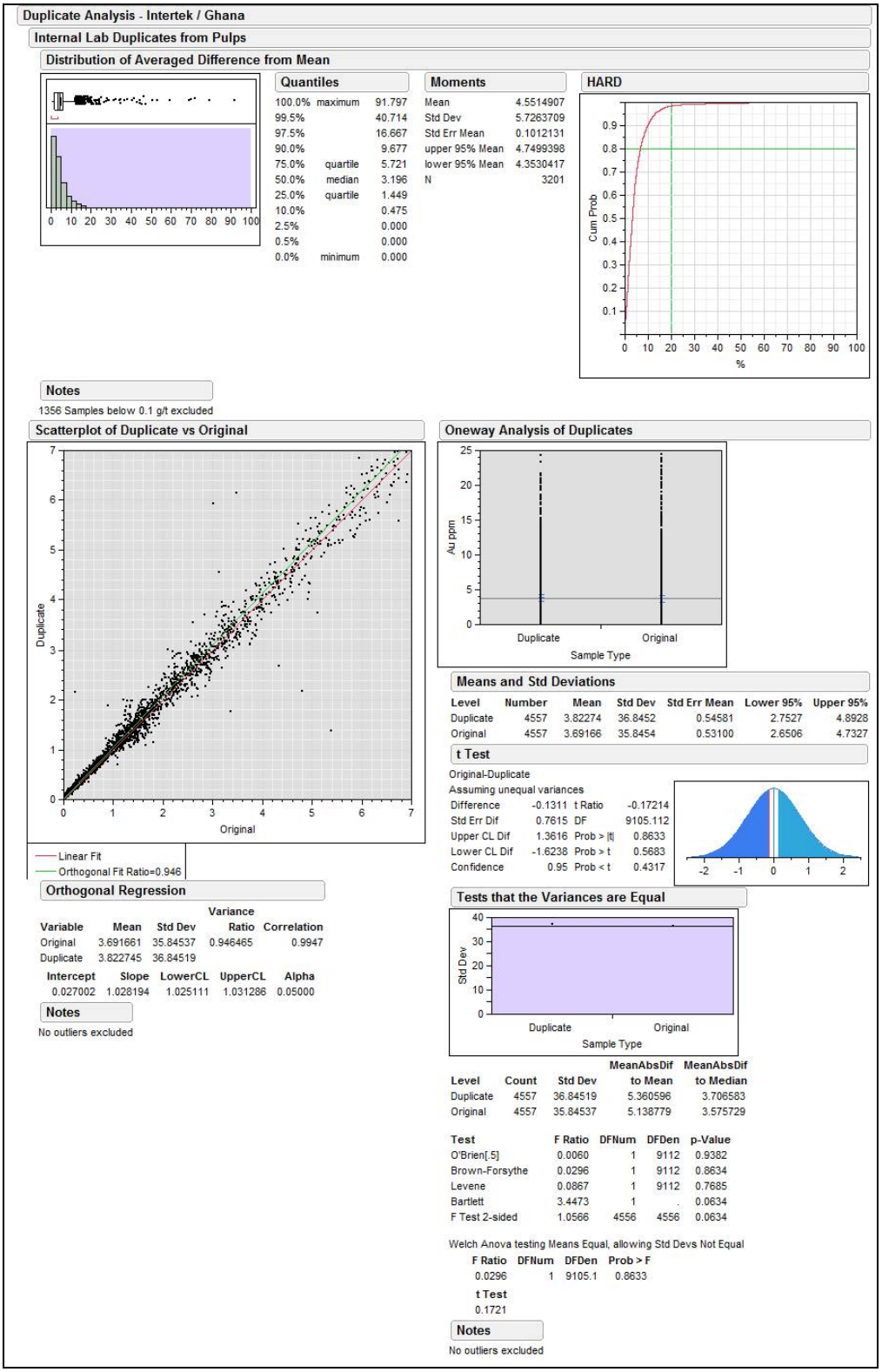


12.2 Pulp Duplicates

A total of 4,557 pulp duplicates were analysed at Intertek in Ghana for diamond core samples. These pulp duplicates are a mixture of random and preferential re-assays (internal laboratory duplicates) on selected mineralised samples.

The HARD plot shows that 90% of the duplicates are within 10% of the original samples which is acceptable precision for pulp duplicates (Figure 12.2.1).

Figure 12.2.1 Analysis of Sissingué Diamond Pulp Duplicates



12.3 Internal Laboratory Quality Control

The following outlines the internal laboratory quality control procedures being carried out. Perseus considers these procedures to be adequate and in line with industry standard.

12.3.1 ALS Internal Quality Control

The following procedure was forwarded by ALS for the quality assurance and quality control (QAQC) protocols:

- Preparation:
  - samples are crushed to >70% passing 2 mm and pulverised to better than 85% passing 75 µm
  - screen tests are done on 5% of all samples to ensure that this is achieved
  - a duplicate prep sample is taken after every 50th sample, prepared and subject to analyses.
- Analytical:
  - work in batches of 24, consisting of 20 client samples, a CRM, a blank and two repeat samples
  - participate in round robins tests organised from ALS Vancouver on a monthly basis
  - participate in independent round robin analyses from Geostats, Australia on a six monthly basis
  - participate in six monthly tests from Rocklabs New Zealand in proficiency testing.

12.3.2 Intertek Internal Quality Control

Intertek has several control procedures to monitor precision and accuracy of results. These are as follows:

- Internal standards (1 in 20).
- Internal duplicates (1 in 20).
- Internal repeats (1 in 20).
- Reagent and sample blanks (1 in 20).
- Inter-laboratory cross checks.

- Sizing tests on pulverised material.
- Blank tests on jaw crusher, roll mills and pulverising mills.
- Fire assay copper maps and loss-lead maps.
- Repeat assays on unusual outliers (upon visual check of results).

12.3.3 BVML Internal Quality Control

The following procedure was forwarded by BVML for the QAQC protocols:

- Internal standards (2 in 25 – one blank, one CRM).
- Internal duplicates (3 in 50).
- Sizing test on jaw crusher (2% for 90% passing 2 mm).
- Sizing tests on pulverised material (5% of sample passing 85% 75 µm).

12.4 Umpire Assaying

A total of 673 samples including 73 quality control samples (CRMs and blanks) where submitted in 10 batches for umpire assaying between January 2010 and June 2011. Pulps of samples previously assayed by Transworld / Intertek were randomly selected from mineralised intervals and submitted to ALS Minerals in Kumasi, Ghana and Genalysis in Perth, Australia.

An overview of comparison between principal and umpire laboratory is given in Table 12.4.1. Globally, the umpire assays were lower by about 8%, with outliers >10 g/t Au removed.

A likely explanation of this discrepancy is that preferential re-sampling for umpire assaying introduced a bias into the procedure. This is supported when comparing only those samples where umpire assay results lie within ±30% of the original value (Table 12.4.2), under the assumption that wider margins would not result from inaccurate analysis, but from inconsistency of the sample material and poor repeatability. Gold grains physically settling down in the sample bag during storage could also play a role when the re-assaying laboratory has scooped pulp material from the top of the bag: an issue that has been observed elsewhere during re-assaying exercises. The results in Table 12.4.2 suggest that Intertek, as principal analytical laboratory, performed satisfactorily over the time period observed.

Figure 12.4.1 shows quantile to quantile (QQ) plots comparing the primary and umpire laboratory results for all assays (left) and assays within ±30% (right).

Umpire assaying has been discontinued after June 2011 mainly due to continuous logistic difficulties (pulps from Intertek laboratory stored in Ghana) and political instability in the wake of general elections in Côte d'Ivoire in late 2011.

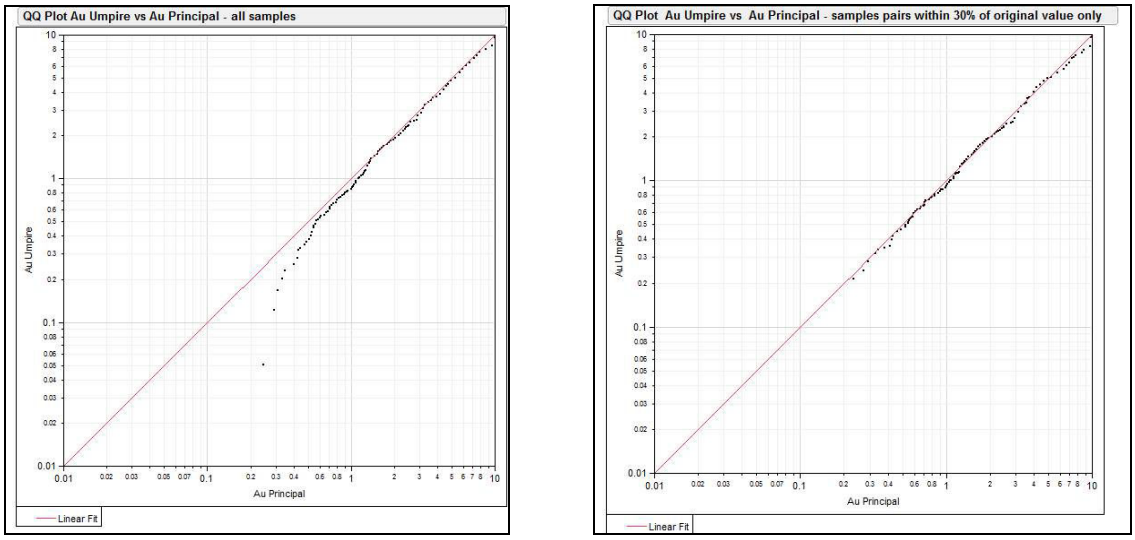
Table 12.4.1 Umpire Laboratory Results

Date	Umpire Lab	No. Samples	Original Lab	All Samples				Outliers (>10 g/t Au) Excluded			
				Mean Au g/t Umpire	Mean Au g/t Original	Difference Au g/t	Difference %	Mean Au g/t Umpire	Mean ug/t Original	Difference Au g/t	Difference %
January 2010	ALS Ghana	51	TWL	2.98	2.84	0.1	4.8	2.18	2.11	0.1	3.6
February 2010	ALS Ghana	50	TWL	4.67	5.08	-0.4	-8.1	2.34	2.66	-0.3	-12.1
March 2010	ALS Ghana	45	TWL	4.68	3.92	0.8	19.4	1.52	1.46	0.1	3.8
June 2010	Genalysis Australia	139	TWL	4.52	4.98	-0.5	-9.1	2.78	3.14	-0.4	-11.5
July 2010	ALS Ghana	93	TWL	11.06	36.92	-25.9	-70.0	2.13	2.31	-0.2	-7.9
August 2010	ALS Ghana	73	TWL	2.57	3.09	-0.5	-17.0	1.65	1.76	-0.1	-6.2
September 2010	ALS Ghana	46	TWL	1.61	1.76	-0.2	-8.6	1.49	1.51	0.0	-1.3
October 2010	ALS Ghana	46	Intertek	0.89	1.37	-0.5	-35.3	0.89	1.37	-0.5	-35.3
November 2010	ALS Ghana	41	Intertek	2.19	1.55	0.6	41.4	1.59	1.44	0.2	10.8
June 2011	ALS Ghana	16	Intertek	0.92	1.07	-0.1	-13.8	0.92	1.07	-0.1	-13.8
Total		**600**		**4.43**	**8.59**	**-4.16**	**-48.4**	**1.96**	**2.12**	**-0.16**	**-7.5**

Table 12.4.2 Umpire Laboratory Results – Only Samples Pairs Within ±30%

Date	Umpire Lab	No. Samples	Original Lab	Mean Au g/t Umpire	Mean Au g/t Original
January 2010	ALS Ghana	35	TWL	3.23	3.22
February 2010	ALS Ghana	35	TWL	5.86	6.18
March 2010	ALS Ghana	30	TWL	6.25	5.17
June 2010	Genalysis Australia	72	TWL	3.80	4.02
July 2010	ALS Ghana	58	TWL	2.97	3.01
August 2010	ALS Ghana	46	TWL	1.80	1.88
September 2010	ALS Ghana	31	TWL	1.39	1.51
October 2010	ALS Ghana	27	Intertek	1.11	1.14
November 2010	ALS Ghana	29	Intertek	1.65	1.61
June 2011	ALS Ghana	9	Intertek	0.74	0.74
Total		**372**		**3.12**	**3.13**

Figure 12.4.1 Umpire Laboratory versus Primary Laboratory QQ Plots – All Assays (left) and Assays within ±30% (right)



12.5 Sample Security

Samples from RC drilling are collected and bagged at drill site during the drilling operation. Core samples are cut in a central facility in Tengrela and samples placed into sample bags as they are cut.

All samples are then catalogued and placed in large woven bags and sealed (Figure 12.5.1) prior to dispatch to ALS, Intertek or BVML for preparation and analysis. Dispatch from site to Korhogo (Intertek) is by Perseus staff and vehicles. Samples dispatched to ALS and BVML are collected from Tengrela by staff and vehicles of the respective laboratories.

Sample numbering is limited to non-descriptive number. All aspects of the process are supervised by Perseus personnel and limited opportunity exists for tampering with samples.

Figure 12.5.1 Samples at Tengrela Office Facility Ready for Dispatch (after Runge, 2010)



12.6 Conclusions

Perseus retains overall responsibility for the database quality including quality assurance (QA) and quality control (QC).

Extensive quality control (QC) programs have been used throughout the Perseus drilling programs to check the precision and accuracy of the laboratory analysis. The results of the QC analysis suggest reasonable precision and accuracy has been achieved.

Overall the QC data provided for the project does not indicate any bias and supports the suitability of the assay data for resource estimation purposes. The author is of the opinion that the sample preparation, security and analytical procedures are adequate for the project. QAQC procedures are considered to be to industry standard.

12.7 Data Validation

Perseus carried out detailed validation of the dataset and retains overall responsibility for the database quality. All drillhole data was validated during data entry by or under the supervision of the author including the following:

- Checks for duplicate collars (LogChief, Datashed).
- Checks for missing samples (Datashed).
- Checks for downhole from-to interval consistency (LogChief, Datashed).
- Checks for overlapping samples (LogChief, Datashed).
- Checks for samples beyond hole depth (LogChief, Datashed).

- Checks for inexistent or misspelt log items (LogChief).
- Check for missing assays (Datashed).
- Check for downhole information beyond hole depth (Datashed).

Snowden carried out a basic statistical and visual validation prior to estimation including the following:

- Drillholes with overlapping sample intervals.
- Sample intervals with no assay data.
- Duplicate records.
- Assay grade ranges.
- Collar coordinate ranges.
- Valid hole orientation data.

No issues were found with the data and Snowden considers the data to be appropriate for estimation. This conclusion reflects accurately the opinion of the author.

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13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A series of three metallurgical testwork programmes have been completed to support the feasibility study of the Sissingué Gold Project. The programs were conducted by AMMTEC Ltd ('AMMTEC') in Perth Western Australia, between 2010 and 2012:

- 2010: Phase 1[1] - composite and limited variability testwork for 2010 Definitive Feasibility Study.
- 2011: Phase 2[2] - variability testwork for the 2010 Feasibility Study.
- 2012: Phase 3[3] - additional variability testwork.

The programmes included testwork to establish:

- comminution characteristics of fresh ores
- mineralogy of the main fresh ore composite
- diagnostic leach analysis of main fresh ore composite
- flotation recovery of main fresh composite at various grind sizes
- gravity recoverable gold content of main fresh composite
- optimum gravity / CIL grind size for main fresh composite
- optimum cyanidation conditions for the main fresh composite
- gravity / CIL leach performance of two oxide / transition composites
- gravity / CIL leach performance of five depth composites, six grade composites and five lithology composites
- gravity / CIL leach performance of 42 variability samples
- engineering data, including testing of oxygen uptake, slurry rheology, carbon kinetics and thickener settling rate[4].

The DFS and Phase 2 test results have been reported in Section 6 of the 2010 Definitive Feasibility Study report that was updated on 15 August 2011.

[1] Refer AMMTEC Report A12440
[2] Refer AMMTEC Report A13207
[3] Refer AMMTEC Report A14120
[4] Thickener tests were completed by Outotec

This document summarises the previous work and draws conclusions relevant for process plant design criteria.

An additional phase of testwork was completed in 2014 to assess the amenability of heap leaching and pre-concentration (gravity and flotation) followed by ultra-fine grinding and cyanidation. The results are not discussed here.

13.2 Summary

Conclusions from the test work programs on the Sissingué ore samples are:

- Sissingué ores are predominantly 'free-milling', non–'preg–robbing' and are amenable to gold extraction by conventional cyanidation. Some of the samples tested, exhibited a refractory nature, however only two of these are located inside the planned pit shell.
- The oxide ores typically yield high recoveries (90% to 98%) at a grind size P_{80} of 106 µm. Leach kinetics are variable with approximately 40% of the samples tested completing leaching with 24 hours, while the remainder exhibit a slow leaching component to 48 hours. Four lower recovery (85.1% to 88.2%) oxide samples were observed clustered in a small zone of about 10,000 tonne in the centre of the pit. A fifth sample in the same zone returned a recovery in excess of 95%.
- The sulphide ores have moderate to high recoveries and vary between lithology.
- The dominant sulphide minerals present are pyrite and arsenopyrite. Variability sample assays show little base metal sulphides present.
- The granite hosted sulphide ores typically; have high recoveries (87% to 94%) at a grind size P_{80} of 106 µm, and exhibit a slow leaching component with leach time of 48 hours being required. The samples contained between 1.6% to 3.1% pyrite and arsenopyrite. No significant relationship between head grade and sulphide mineral abundance was observed, however leach residue grades increased linearly with the combined amount of pyrite and arsenopyrite. One granite sample exhibited poor recovery (75%), with residue gold grade showing a similar relationship to sulphide mineral abundance as noted in the Type 2 sediment hosted ores.
- The porphyry hosted sulphide ore responded similarly to granite ore.

- The sediment hosted sulphide ores had variable recoveries ranging from 39.9% to 95.6% at a grind size P_{80} of 106 µm. A slow leaching component is present and a leach time of 48 hours is required. Detailed examination showed there are two types of behaviours noted:

  - Type 1 sediments: these have, higher recovery (average 85.4%, range 76.6% to 94.3%), exhibit no relationship between gold head grade and sulphide mineral content, while gold in residue increases linearly with pyrite and arsenopyrite content[1]. The residue grades are marginally higher than the granite ores for the same sulphide mineral content.

  - Type 2 sediments: these lower recovery (average 63.7% range 39.9% to 92.1%), exhibit no relationship between gold grade and sulphide mineral content, but gold in residue increases exponentially with pyrite and arsenopyrite content[2].

- The oxide and sulphide ores have dissimilar physical and comminution characteristics. The oxide ore is soft and non-abrasive, while the sulphide ores are abrasive, hard and competent to very competent. Oxide ores will be processed by a two stage crushing circuit, with a third stage of crushing added for processing of the sulphide ores.

- Gravity test-work showed variable recovery to (0.7% to 87.4%) of gold by centrifugal gravity concentration and intensive cyanidation of gravity concentrate. In general terms oxide ores had the highest gravity gold recovery, followed by Type 1 sediment, granite then Type 2 sediment ores. Overall, gold recovery including leaching of gravity tails showed no improvement compared to the whole-of ore leach, however a gravity circuit has been included in the flowsheet as visible gold is present in drill core.

- Diagnostic leach analysis showed that where recoveries were low, gold deportment in tails was variable, ranging from:

  - predominantly in arsenopyrite, with lesser pyrite and silicate

  - equally in pyrite and arsenopyrite

  - equally in pyrite, arsenopyrite and silicates

  - equally pyrite and silicates.

- The variable deportment suggests the gold is not in solid solution, but occurs as fine discrete gold particles locked in arsenopyrite, pyrite and/or silicate.

- Pre-oxidation is found to be beneficial in leach kinetics and overall recovery, predominantly for sulphide ores.

[1] Type 1 sediment pyrite content ranges from 0.17% to 3.07%, arsenopyrite content from 0.05% to 1.88% and a combined content of 0.22% to 4.88%
[2] Type 2 sediment pyrite content ranges from 0.58% to 2.63%, arsenopyrite content from 0.15% to 2.86% and a combined content of 0.73% to 4.80%

- The oxide and sulphide ores both have thickener specific settling rates of 1.0 t/m^2h. Oxide ores require 30 g/t to 50 g/t of flocculant to produce an underflow with a density of 49% to 55% solids w/w, whereas the sulphide ores require 20 g/t to 30 g/t of flocculant for an underflow density of 63% to 65% solids w/w. Viscosity related materials handling issues will prevent oxide slurries from being thickened beyond 45% solids w/w.

- The selected design criteria from the various metallurgical testwork programs are summarised in Table 13.2.1.

Table 13.2.1 Selected Design Criteria

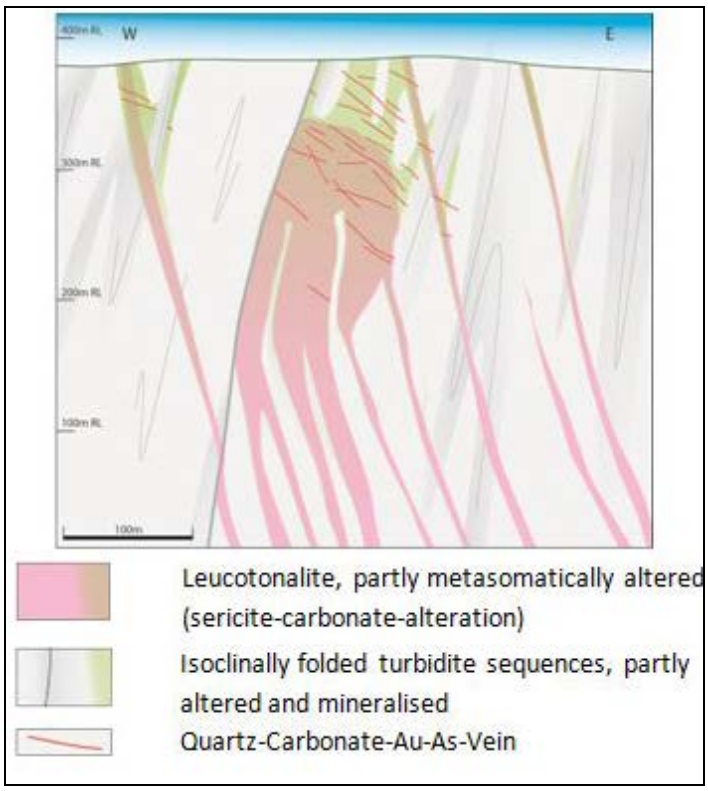
Parameter	Unit	Oxide	Granite / Porphyry	Sediment	Reference
Comminution					
Ore SG	t/m^3	2.78	2.78	2.77	Section 13.4.8
A x b		100	32.4	24.2	Section 13.4.8
CWI	kWh/t	5	13.4	10	Section 13.4.8
RWI	kWh/t	3.7	18.3	23.4	Section 13.4.8
BWI	kWh/t	5.4	17.0	17.9	Section 13.4.8
Ai		0.0026	0.48	0.39	Section 13.4.8
Milling product P_{80} size	µm	106	106	106	Section 13.6.1
Pre-Leach Thickening					
Specific settling rate	t/m^2h	1.0	1.0	1.0	Section 13.8.5
Underflow density	%w/w	45	52	52	Section 13.8.5
Pre-oxidation					
Residence time	h		4	4	Section 13.6.7
CIL					
Residence time	h		48	48	
Carbon Kinetic Parameter k	h^{-1}		296	296	Section 13.8.3
Carbon Kinetic Parameter n			0.63	0.63	Section 13.8.3

13.3 Samples

13.3.1 Geology

There are three main lithological domains at Sissingué; granite, porphyry and sediments as illustrated in Figure 13.3.1.

Figure 13.3.1 Sissingué Geological Model



- Granite: there is at least one granitic intrusive body known in the area and it is interpreted to be emplaced into structurally favourable locations, possibly a fold hinge. The granite dips 60-70° to the west and plunges moderately to the north.

- Porphyry: there are porphyritic dykes that range from less 1 m to 10 m thick and are interpreted to cover the entire project area with dykes being traceable over several hundred metres. They are late stage intrusions that cut across all lithologies and dip at 70-80° to the east.

- Sediments: weakly mineralised altered sediments surround the granite and porphyry intrusions. The alteration is typically weak and associated with moderate silica, carbonate and chlorite overprints.

Gold mineralisation is characterised by silica-feldspar alteration and sulphide mineralisation consisting of arsenopyrite, pyrrhotite, pyrite and trace chalcopyrite (Morila), or characterised by variable combinations of fine-grained albite, calcite and silica with deposition of pyrite, arsenopyrite, chalcopyrite plus chlorite and haematite, and gold (Lero). Visible gold is generally a common occurrence.

Gold mineralisation at Sissingué has a close association with veined, altered and sulphidised felsic intrusive bodies. The highest grades are located within the felsic bodies themselves, whilst the meta-greywackes adjacent to the felsic mineralised bodies typically host lower grades. The gold mineralisation is found as three distinct styles and are summarised below:

- Disseminated or fracture-related mineralisation in the felsics associated with silicification, chlorite alteration and pyrite and arsenopyrite.
- Coarse gold has been observed in milky quartz veins cutting the intrusive or adjacent sediments.
- Significant low grade gold mineralisation is also found within the meta-greywackes immediately adjacent to the intrusive bodies, either in veins or in a more dispersed form through micro-fractures and sulphidisation.

It appears that in the central part of the Sissingué prospect (around 1,154,160 m), the major host rock to some of the highest grade mineralisation is a medium-to-coarse grained, veined, altered and sulphidised granite. A range of veining styles is observed within the granite and is summarised below:

- Milky quartz veins, sometimes observed to host coarse gold.
- Finer quartz-carbonate-chlorite veins that appear to follow brittle fracture networks in the granite.
- Minor ankerite veining was also observed.

The granite is pervasively silicified and carbonate and sericite altered (possibly after biotite). There is minor, disseminated sulphide mineralisation, dominated by pyrite but also with bladed arsenopyrite.

Some of the highest grade mineralised sections occur between 1,154,500 mN and 1,154,560 mN in a 45° east dipping sediment domain where individual samples have been assayed at over 1,200 g/t Au.

Resource grade mineralisation has been defined over a total strike length of 3.1 km. Individual lodes and zones have lengths of up to 840 m, widths of up to 100 m and down dip extents of up to 250 m.

The current Sissingué (measured and indicated) resource, based on a 0.65 g/t gold cut-off grade, contains 16.1 Mt of ore @ 1.75 g/t, as summarised in Table 13.3.1.

Table 13.3.1 Sissingué Resource (M&I) @ 0.65 g/t Au cut–off grade

Ore Type	Tonnes	Grade (g/t)	Contained Gold (ozs)
Oxide	3,298,700	1.73	183,729
Granite	3,100,449	2.50	248,887
Porphyry	1,654,870	1.25	66,398
Sediment	8,069,199	1.57	406,965
Subtotal	**16,123,218**	**1.75**	**906,046**

The estimated distribution by tonnage of the 5.5 Mt[1] to be processed is shown in Table 13.3.2. The average grade for each ore type is shown in Table 13.3.3.

Table 13.3.2 Sissingué Ore – Type Distribution

Ore Type	Tonnes	Distribution, %
Oxide/transitional	2,786,679	50.5
Granite/porphyry	1,664,764	30.1
Sediment	1,072,466	19.4
Subtotal	**5,523,909**	**100.0**

Table 13.3.3 Sissingué Grade Distribution

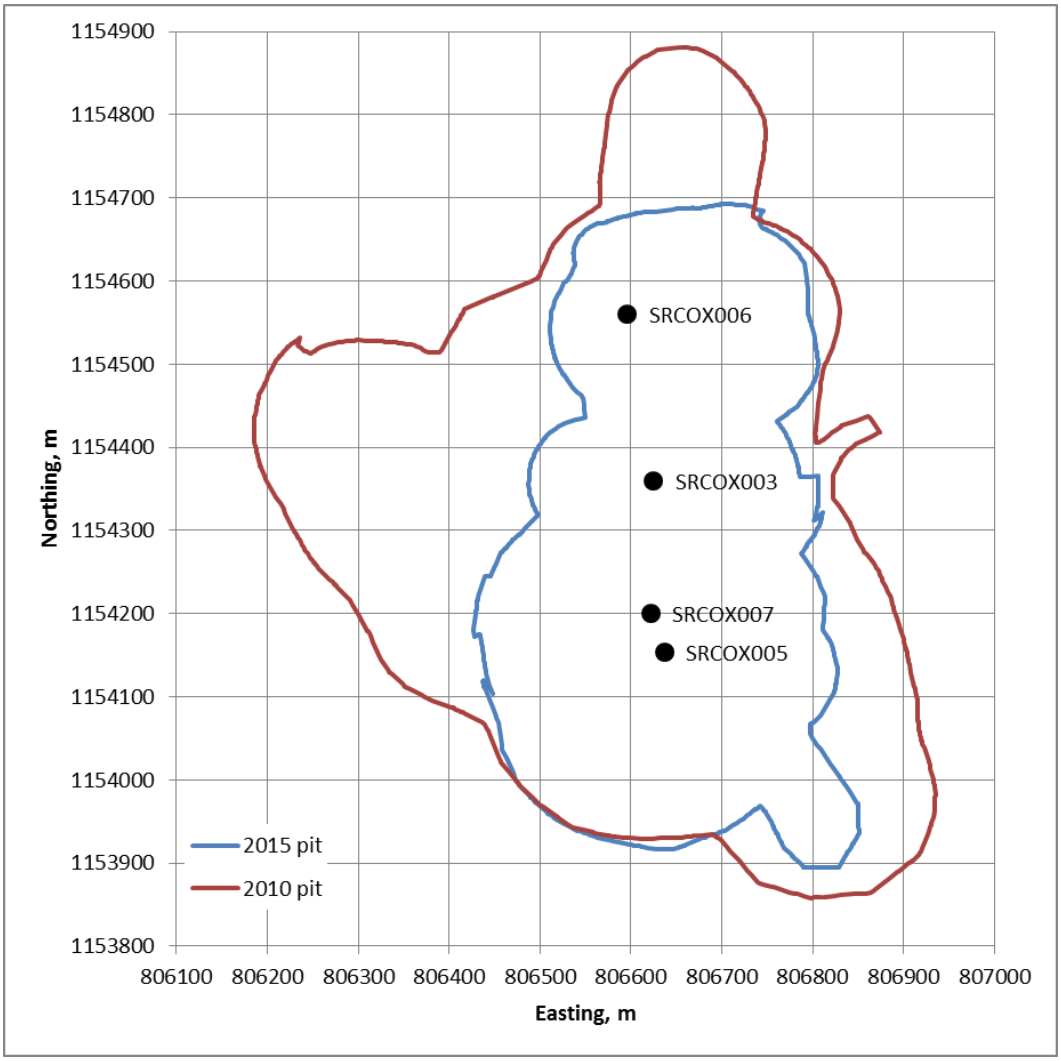
Ore Type	Unit	Grade
Oxide/transitional	g/t	1.77
Granite/porphyry	g/t	3.40
Sediment	g/t	2.57
Subtotal		**2.42**

13.3.2 Comminution Samples

Twelve comminution samples were selected from core retrieved from four large core drill holes drilled at a vertical dip. The drill holes extend across the pit along the pit centre-line as shown in Figure 13.3.2. No comminution testwork has been completed since the 2010 DFS.

[1] Based on mine production schedule '5.5Mt Scedule_March_2015.xls'

Figure 13.3.2 Location of Comminution Sample Drill Holes



13.3.3 Metallurgical Composites

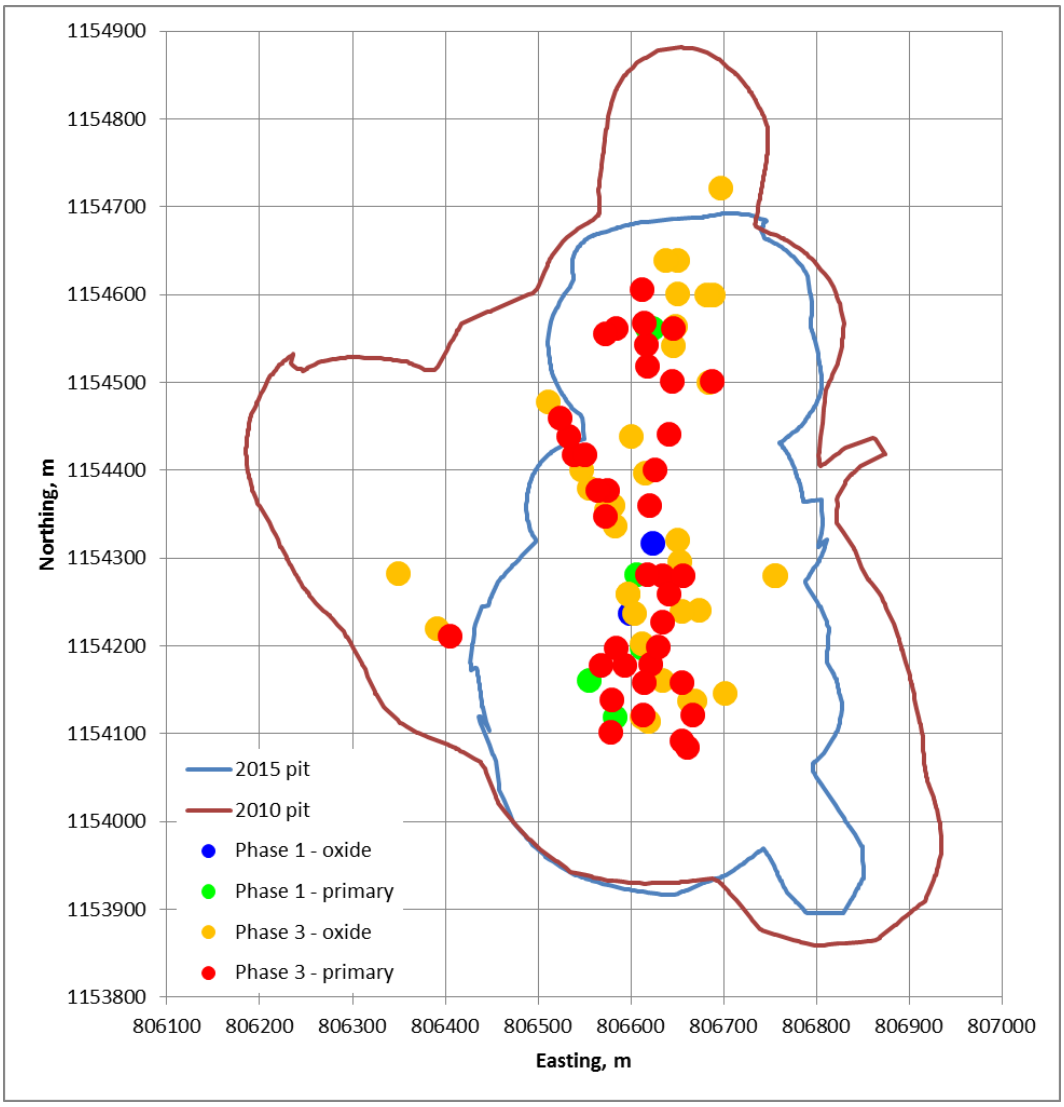
A number of ore type composite samples were generated from Sissingué core samples over the various phases of the Project:

- Primary ore (Phase 1).
- Oxide ore (Phase 1).
- Pit North Oxide ore (Phase 3).
- Pit North Sulphide (Phase 3).
- Pit South Oxide (Phase 3).
- Pit South Sulphide (Phase 3).

- Oxide Master[1] (Phase 3).
- Sulphide Master[2] (Phase 3).

The locations of each sample used for composite testwork are shown in Figure 13.3.3.

Figure 13.3.3 Metallurgical Composite Sample Location



[1] A composite of the Pit North Oxide and the Pit South Oxide samples
[2] A composite of the Pit North Sulphide and the Pit South Sulphide samples

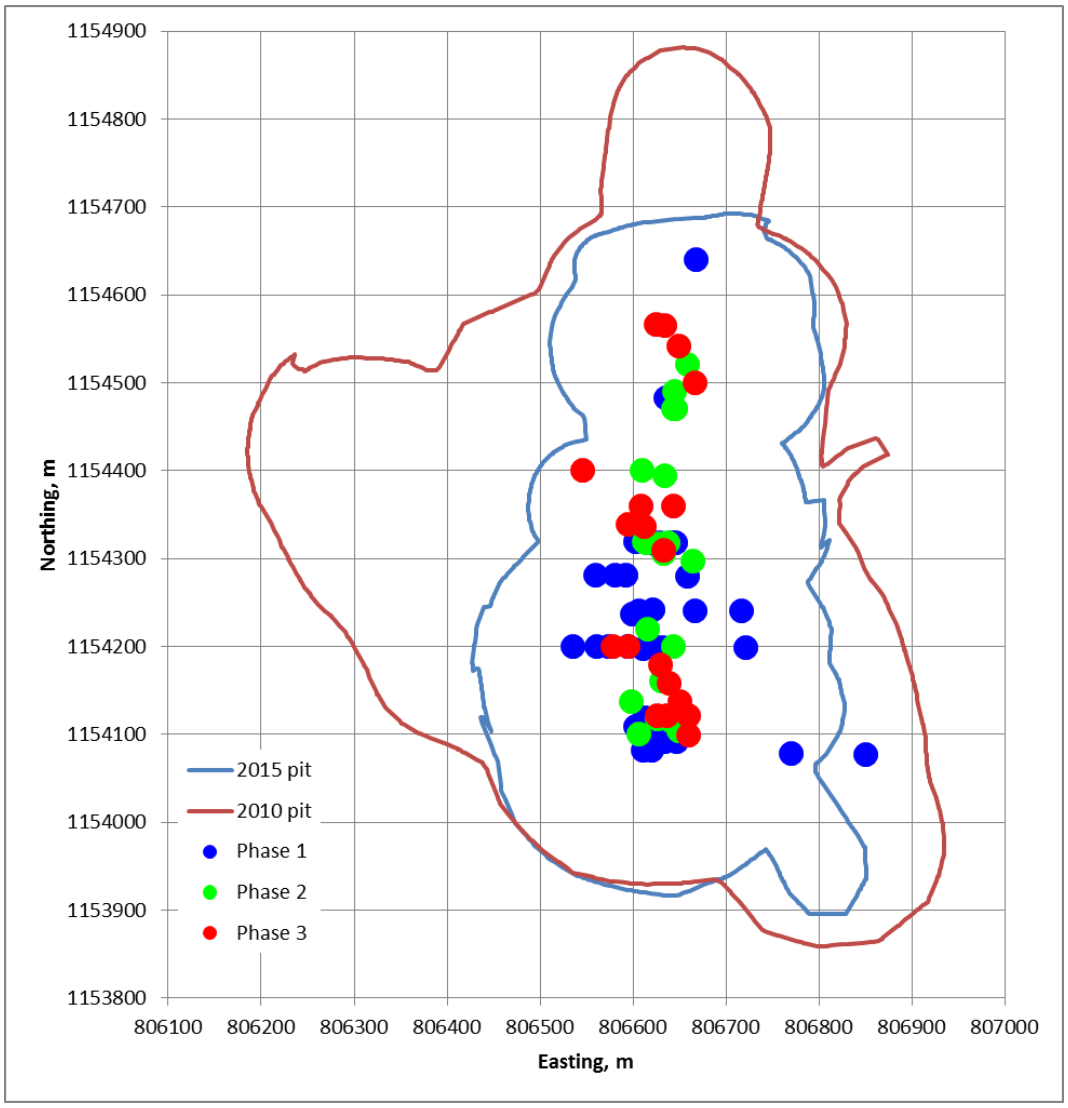
13.3.4 Variability Samples

A total of 60 variability samples were generated from the Sissingué core samples over the various phases of the Project:

- Phase 1: 2 x oxide, 5 x lithology, 6 x grade, 5 x depth.
- Phase 2: 8 x oxide, 1 x transition, 12 x primary.
- Phase 3: 6 x oxide, 1 x transition, 14 x primary.

The locations of each sample used for variability testwork are shown in Figure 13.3.4.

Figure 13.3.4 Variability Sample Location



13.3.5 Head Assay

Ore Type Composites

The head assays of the primary / sulphide ore composites and oxide ore composites are shown in Table 13.3.4 and Table 13.3.5.

Table 13.3.4 Primary / Sulphide Ore Composite Head Assay

Analyte	Unit	Primary Ore	North Pit Sulphide	South Pit Sulphide	Sulphide Master
$Au_{(1)}$	g/t	1.52	1.98	2.12	2.14
$Au_{(2)}$	g/t	3.29	2.16	3.94	1.94
$Au_{(3)}$	g/t	5.24	n/d	n/d	n/d
$Au_{(4)}$	g/t	3.65	n/d	n/d	n/d
$Au_{(ave)}$	g/t	3.42	2.07	3.03	2.04
Ag	g/t	0.5	0.4	0.5	0.3
As	ppm	2044	3810	4370	4300
Bi	ppm	<10	<10	<10	<10
Hg	ppm	<0.1	<0.1	0.1	0.1
C_{TOTAL}	% w/w	1.47	1.17	<10	0.99
$C_{ORGANIC}$	% w/w	<0.03	0.03	0.06	0.06
S_{TOTAL}	% w/w	1.02	1.36	1.46	1.14
$S_{SULPHIDE}$	% w/w	0.77	1.16	1.26	1.12
Sb	ppm	<5	4.2	2.6	3.40
True SG	t/m^3	2.78	2.78	2.80	2.80

The following conclusions are drawn from Table 13.3.4:

- There is a fair degree of variability in the gold assays, particularly the 'primary ore' composite, that is likely due to discrete gold particles / composites.
- Arsenic and sulphide sulphur are present in quantity indicating the possible presence of arsenopyrite which can be a refractory ore. Solubilisation of arsenic may occur and is to be investigated by testwork (refer Section 13.8.4).
- There should be no mercury handling issues.
- Organic carbon is low suggesting there is unlikely to be 'preg-robbing' issues.
- There is no correlation between gold and either arsenic or $S_{sulphide}$ assays.
- Silver content is low and will not compete with gold when loading onto activated carbon.

Table 13.3.5 Oxide Ore Composite Head Assay

Analyte	Unit	Oxide Ore	North Pit Oxide	South Pit Oxide	Oxide Master
$Au_{(1)}$	g/t	4.45	3.27	1.27	2.18
$Au_{(2)}$	g/t	4.48	5.51	1.92	2.57
$Au_{(3)}$	g/t	n/d	5.63	n/d	n/d
$Au_{(4)}$	g/t	n/d	n/d	n/d	n/d
$Au_{(ave)}$	g/t	4.47	4.80	1.60	2.38
Ag	g/t	0.5	0.4	0.4	0.3
As	ppm	823	710	520	670
Bi	ppm	<10	<10	<10	<10
Hg	ppm	<1	<0.1	<0.1	<0.1
C_{TOTAL}	% w/w	0.03	<0.03	<0.03	0.03
$C_{ORGANIC}$	% w/w	<0.03	<0.03	<0.03	0.06
S_{TOTAL}	% w/w	0.07	<0.02	<0.02	<0.02
$S_{SULPHIDE}$	% w/w	0.02	<0.03	<0.03	<0.02
Sb	ppm	<5	4.3	2.90	3.50
True SG	t/m3	2.77	2.75	2.78	2.77

The following conclusions are drawn from Table 13.3.5:

- Variability in gold assays indicate the possible present of discrete gold particles.
- Arsenic and sulphur levels are markedly lower than primary ore as expected.
- Silver assays are low and will not compete with gold for adsorption onto loaded carbon.
- Organic carbon is low suggesting there is unlikely to be 'preg-robbing' issues.

Variability Samples

The head assay of the variability samples are given in Table 13.3.6, Table 13.3.7 and Table 13.3.8. As with the composite samples there is no correlation between gold and $S_{sulphide}$ assay.

Table 13.3.6 Phase 1 (DFS) Variability Head Assay

Sample	Lithology	Weathering	Au g/t	Ag g/t	As ppm	Bi ppm	$C_{organic}$ %	$S_{sulphide}$	True SG
Lithology 1	Porphyry	Primary	0.44	<2	1521	<10	<0.03	0.32	2.74
Lithology 2	Sediment	Primary	1.95	<2	4256	<10	<0.03	1.66	2.83
Lithology 3	Sediment	Primary	1.55	<2	2007	11	<0.03	1.28	2.80
Lithology 4	Granite	Primary	0.06	<2	434	<10	<0.03	1.04	2.83
Lithology 5	Granite	Primary	1.42	<2	2452	<10	<0.03	0.79	2.78
0.8 g/t Gold	Mixed	Primary	0.78	<2	1095	<10	0.04	0.57	2.77
1.2 g/t Gold	Mixed	Primary	0.76	<2	1736	<10	<0.03	0.46	2.78
2.0 g/t Gold	Mixed	Primary	3.75	<2	1953	<10	0.11	0.99	2.79
3.0 g/t Gold	Mixed	Primary	1.75	<2	3790	<10	<0.03	0.85	2.80
4.0 g/t Gold	Mixed	Primary	2.73	<2	2302	<10	0.06	0.87	2.77
5.0 g/t Gold	Mixed	Primary	2.04	<2	2953	<10	<0.03	0.96	2.87
Depth 1	Mixed	Oxide	2.03	<2	270	<10	<0.03	0.00	2.77
Depth 2	Mixed	Transitional	1.29	<2	877	<10	<0.03	0.00	2.80
Depth 3	Mixed	Primary	1.86	<2	2761	<10	<0.03	0.76	2.77
Depth 4	Mixed	Primary	2.10	<2	3946	<10	<0.03	1.26	2.79
Depth 5	Mixed	Primary	1.68	<2	5369	<10	<0.03	1.02	2.81
Oxide 1	Porphyry	Oxide	5.18	0.4	1068	<10	<0.03	0.02	2.78
Oxide 2	Sediment	Oxide	3.17	<0.3	757	<10	<0.03	0.02	2.77

Table 13.3.7 Phase 2 Variability Head Assay

Sample	Lithology	Weathering	Au g/t	Ag g/t	As ppm	Bi ppm	$C_{organic}$ %	$S_{sulphide}$	True SG
PCL04-1	Porphyry	Oxide	2.435	<0.3	640	<10	<0.03	0.01	2.82
PCL08-01	Sediment	Oxide	2.055	<0.3	670	<10	0.06	0.01	2.77
PCL08-02	Sediment	Oxide	1.02	< 0.3	570	<10	0.03	0.01	2.76
PCL12-1	Sediment	Oxide	2.185	<0.3	860	<10	<0.03	0.01	2.77
PCL12-2	Sediment	Oxide	3.41	<0.3	750	<10	0.03	0.01	2.72
PCL16-1	Granite	Primary	2.02	<0.3	3070	<10	<0.03	1.12	2.73
PCL16-2	Sediment	Oxide	2.055	<0.3	1350	<10	<0.03	0.08	2.78
PCL20-1	Granite	Primary	2.125	<0.3	3570	<10	<0.03	1.24	2.78
PCL20-2	Sediment	Oxide	15.1	1.2	620	<10	<0.03	0.01	2.80
PCL24-1	Sediment	Transition	83.25	2.3	1160	<10	<0.03	0.2	2.76
PCL24-2	Sediment	Oxide	1.56	0.4	420	<10	<0.03	0.04	2.75
PCL30-1	Sediment	Primary	4.565	0.3	6600	<10	<0.03	1.56	2.79
PCL36-1	Sediment	Primary	1.435	<0.3	225	<10	<0.03	0.1	2.75
PCL42-1	Sediment	Primary	2.115	<0.3	1050	<10	0.03	0.58	2.77
PCL42-2	Sediment	Primary	1.265	<0.3	2050	<10	0.03	0.72	2.76
PCL45-1	Sediment	Primary	6.235	0.5	3025	<10	0.09	1.2	2.79
PCL45-2	Granite	Primary	1.69	<0.3	3225	<10	<0.03	1.3	2.71
PCL48-1	Granite	Primary	1.965	<0.3	5300	<10	<0.03	1.14	2.78
PCL48-2	Sediment	Primary	2.14	<0.3	4125	<10	<0.03	1.48	2.77
PCL51-2	Sediment	Primary	4.755	<0.3	6375	<10	<0.03	1.68	2.81
PCL54-1	Sediment	Primary	0.49	<0.3	1375	<10	<0.03	1.22	2.79

Table 13.3.8 Phase 3 Variability Head Assay

Sample	Lithology	Weathering	Au g/t	Ag g/t	As ppm	Bi ppm	$C_{organic}$ %	$S_{sulphide}$	True SG
VC#1	Sediment	Primary	0.53	0.4	1400	<10	0.06	0.68	2.78
VC#2	Sediment	Primary	3.615	0.7	3160	<10	0.09	0.82	2.79
VC#3	Sediment	Oxide	0.32	0.5	360	<10	0.06	0.01	2.76
VC#4	Sediment	Oxide	27.9	3.2	790	<10	<0.03	0.01	2.78
VC#5	Granite	Primary	2.145	0.8	4800	<10	0.06	1.16	2.78
VC#6	Granite	Primary	11.825	0.5	3780	<10	0.06	1.36	2.82
VC#7	Granite	Primary	8.94	0.4	2050	<10	0.06	0.74	2.80
VC#8	Granite	Primary	5.015	1.1	4240	<10	0.03	1.34	2.79
VC#9	Sediment	Primary	3.495	0.5	13200	<10	0.12	1.6	2.86
VC#10	Sediment	Primary	4.11	0.6	8320	<10	0.06	2	2.86
VC#11	Sediment	Primary	2.93	0.7	3460	<10	0.06	1.42	2.82
VC#12	Sediment	Primary	2.495	0.5	1500	<10	0.18	0.82	2.79
VC#13	Sediment	Primary	5.155	0.5	700	<10	0.06	0.34	2.82
VC#14	Sediment	Oxide	5.55	0.5	1520	<10	<0.03	0.02	2.78
VC#15	Sediment	Primary	0.855	0.5	6230	<10	0.06	1.1	2.77
VC#16	Sediment	Oxide	1.36	0.4	740	<10	<0.03	0.01	2.84
VC#17	Sediment	Primary	1.585	< 0.3	7800	<10	0.06	1.34	2.79
VC#18	Sediment	Oxide	4.645	0.5	1180	<10	<0.03	0.01	2.79
VC#19	Sediment	Primary	2.81	0.4	4520	<10	0.06	1.5	2.79
VC#20	Laterite	Oxide	0.045	0.5	500	<10	0.09	0.01	2.82
VC#21	Sediment	Oxide	4.215	0.6	530	<10	<0.03	0.01	2.90

13.3.6 Mineralogy

Mineralogical examination was conducted on the Primary Ore composite used for the DFS testwork using the following methods of investigation:

- QEMSCAN and XRD analysis on gravity concentrate and tail.
- Optical / SEM examination on whole ore sample.

QEMSCAN and XRD

A sample of the Primary Ore composite, ground to a P_{80} of about 150 µm underwent gravity concentration consisting of a single pass through a Knelson gravity concentrator and panning of the Knelson concentrate. Panning tails were combined with Knelson tailings to form a single tailing fraction.

The mass splits to gravity concentrate and tail are summarised in Table 13.3.9.

Table 13.3.9 Mass split of gravity separation

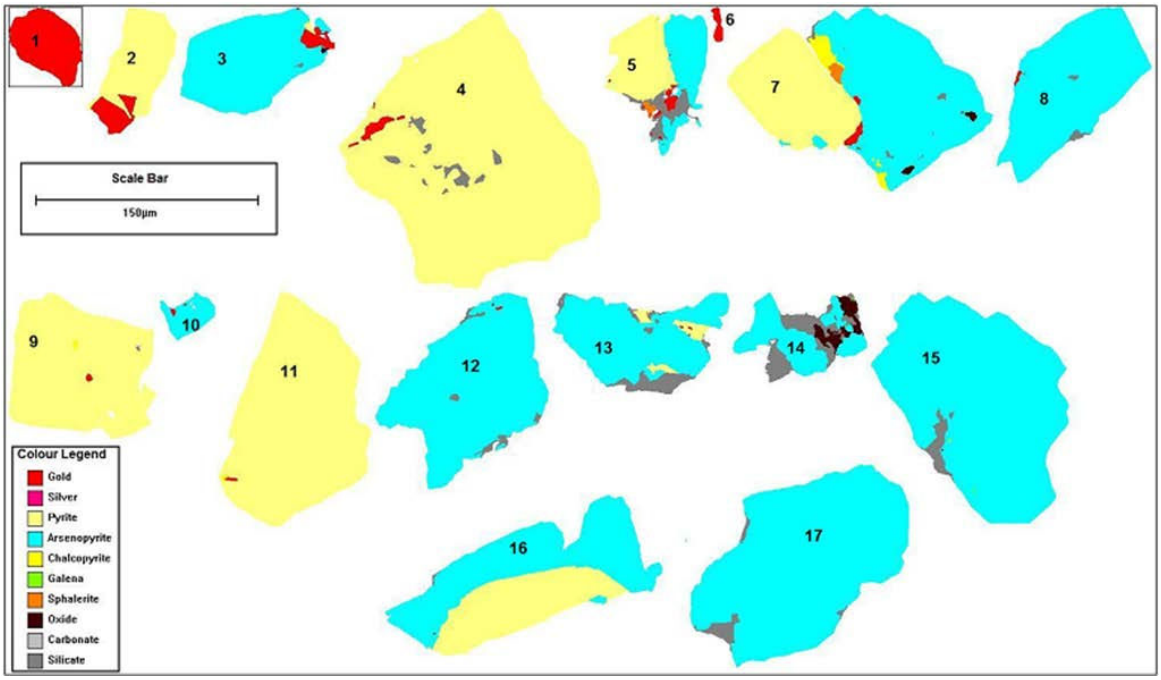
Sample Name	Sample Type	Mass % Retained	Mineralogical Sample Number
Primary Ore	Gravity concentrate	0.5	MIN 357A1A
	Gravity tail	99.5	MIN 357A2A

An XRD trace of the concentrate and tailings samples was collected. The major constituents for each sample were:

- concentrate: pyrite (35% w/w), alpha quartz (20% w/w), albite (19% w/w), muscovite (11% w/w), arsenopyrite (8% w/w), ankerite (5% w/w) and miscellaneous others (2% w/w)
- tails: albite (35% w/w), alpha quartz (28% w/w), muscovite (27% w/w), ankerite (6% w/w) and miscellaneous others (4% w/w).

Quantitative Automated Mineralogical analysis was performed on the two samples. An MLA Sparse Phase Search (XSPL) was conducted on the concentrate sample. Seventeen particles containing gold were identified and are shown in Figure 13.3.5.

Figure 13.3.5 Mineral Map of Particles Containing Gold



Salient comments from the data include the following:

- Seventeen particles identified.
- Gold size varied between ~59 µm x 36 µm to ~1 µm x 1 µm – with the largest particle (fully liberated) containing 57% of gold present and 90% of gold present accounted for by the six largest particles.
- Gold is amenable to leaching, with 61% being liberated, 31% being free and 8% locked in other minerals.
- Spot x-ray analysis shows most gold containing 10-12% silver and no copper. One gold grain reported 21% silver and two particles had almost no silver. One low silver gold grain had porous texture indicating it is of supergene origin.
- Gold is strongly associated with pyrite and arsenopyrite in approximately equal proportions.
- Some gold may be present as solid solution in sulphide minerals (pyrite and arsenopyrite). The amount of gold in this category is constrained by the relatively low abundance of both pyrite (1.2%) and arsenopyrite (0.5%) and because high concentrations of gold in these minerals can be excluded based on their EDS X-ray spectra analysis.
- No silver minerals were observed.

- Pyrite and arsenopyrite are the most abundant sulphide minerals present and both were well liberated at the grind size tested (P_{80} = 150 µm), indicating they are coarse grained in the unbroken rock.

- Flotation followed by regrinding and cyanidation may be an effective way to recover fine (but not gold in solid solution) gold that is encapsulated by pyrite and arsenopyrite, should that be encountered.

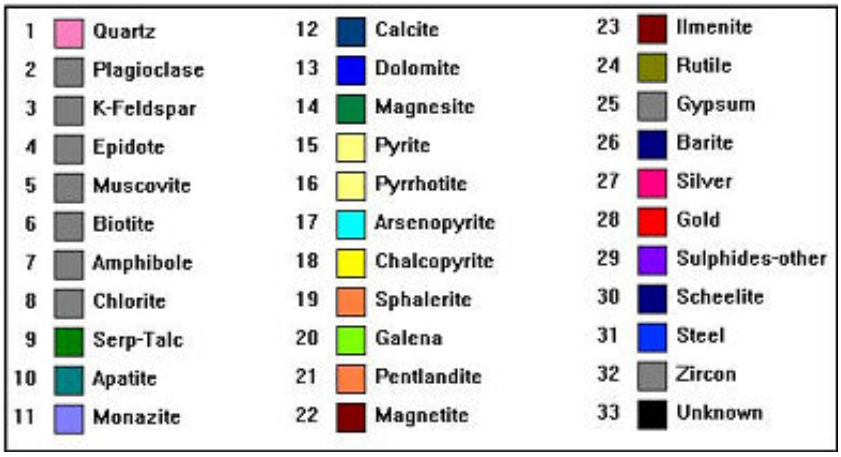
Mineral liberation analysis (MLA) of the tail sample showed that:

- main minerals are quartz (35%), plagioclase (32%), muscovite, calcium rich carbonate(9%) and calcium-free carbonate (1.3%)

- main sulphides present are pyrite and arsenopyrite. As with the concentrate sample they are well liberated. There are few other sulphides indicating that modal abundance of pyrite and arsenopyrite can be calculated from chemical assay for arsenic and sulphur with reasonable accuracy.

The mineral maps of all gravity tails, pyrite in gravity tails and arsenopyrite in gravity tails are shown in Figure 13.3.6, Figure 13.3.7 and Figure 13.3.8.

Figure 13.3.6 Mineral Map of Gravity Tails



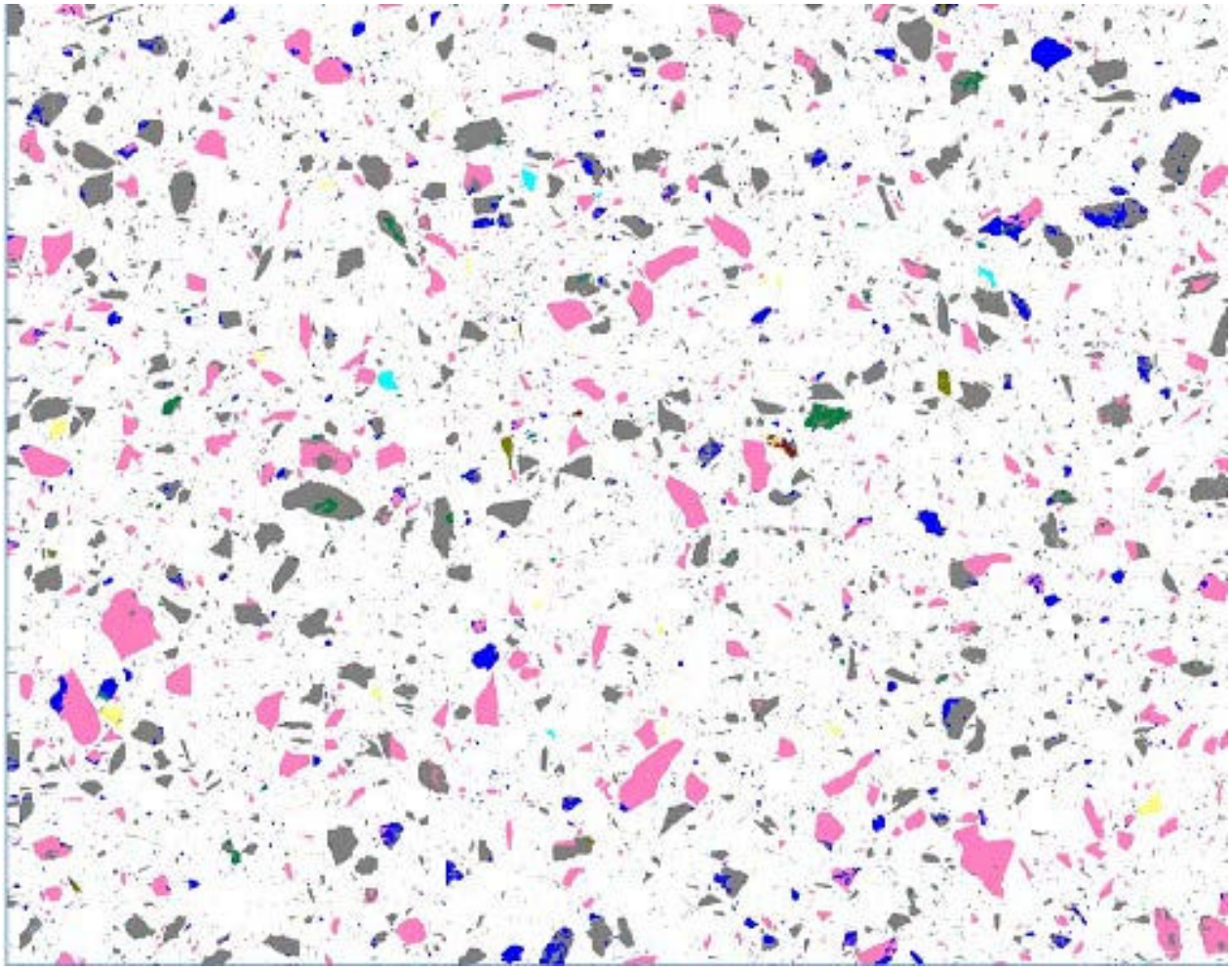


Figure 13.3.7 Mineral Map of Pyrite (Yellow) in Gravity Tails

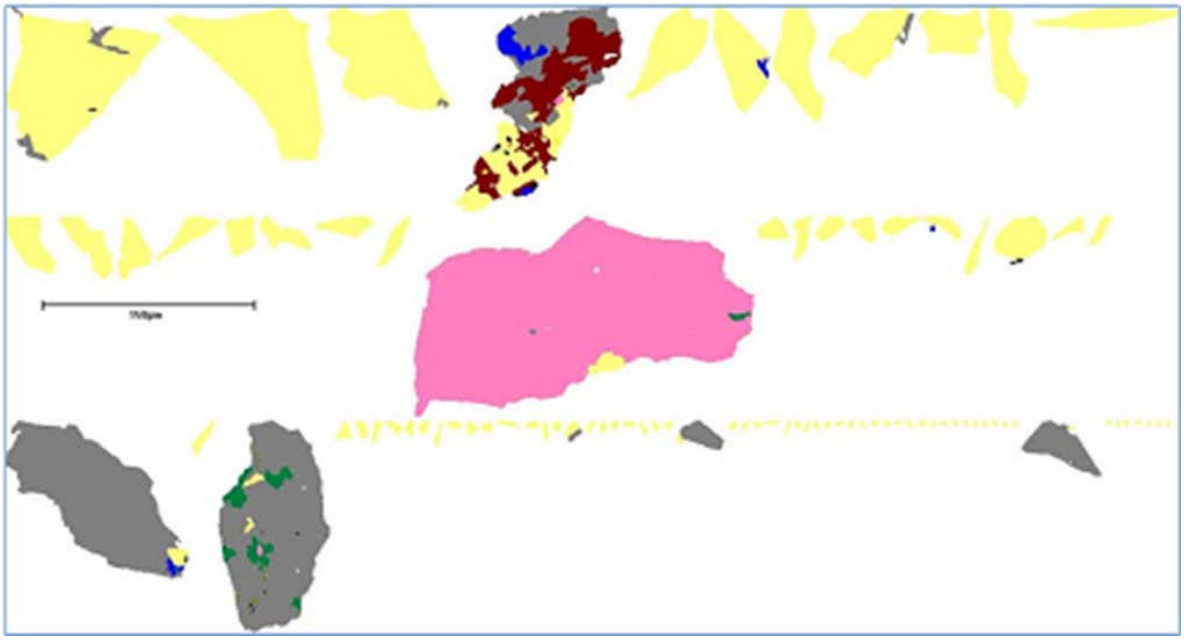
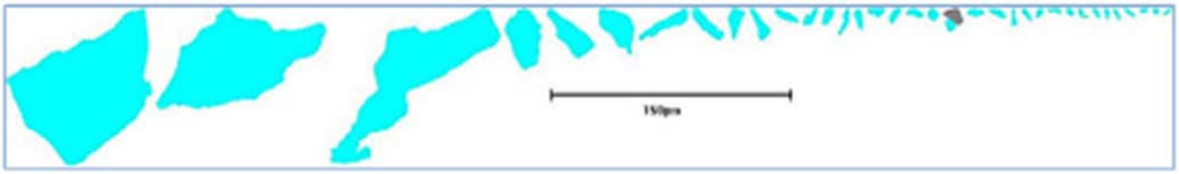


Figure 13.3.8 Mineral Map of Arsenopyrite in Gravity Tails



Low assays for base metals, bismuth, tellurium, antimony and organic carbon indicate there should be little interference with cyanide leaching.

Optical / SEM

A sample of ground Primary ore composite was sent to Roger Townend and Associates for mineralogical examination by optical and Scanning Electron Microscopy (SEM) methods. The sample was screened at 106 µm with both fractions being subject to heavy liquid separation. The 'sinks' fraction were mounted on a polished section and examined.

Salient comments for the +106 µm 'sinks' fraction include:

- Pyrite (dominant) and arsenopyrite (major) are the dominant sulphide minerals present. They are dominantly liberated with grain sizes up to 1 mm. Composites are uncommon. There is small percentage of both sulphides as inclusions in coarser silicate gangue.

- Other sulphides are rare, chalcopyrite as inclusions in pyrite or arsenopyrite, very rare discrete pyrrhotite and rare fine galena in pyrite and arsenopyrite and also included in quartz.

- One gold particle was observed. It was coarse 450 µm x 100 µm and contained 9% silver and was composite with quartz.

Salient comments for the -106 µm 'sinks' fraction include:

- Pyrite (dominant) and arsenopyrite (minor) were the major sulphide minerals present with trace amounts of chalcopyrite and galena.
- Sulphides and oxides are essentially liberated.
- No gold was detected.
- Auriferous pyrite arsenopyrite concentrates can have solid solution gold, particularly when pyrite is arsenical. SEM / EDS analyses showed pyrite containing 1-2% arsenic.

In general the optical / SEM examination supported the observations made from the QEMSCAN and XRD analysis.

13.4 Comminution Testwork

All comminution testwork occurred during the 2010 DFS phase and included the following:

- Unconfined Compressive Strength (USC) determination.
- Bond Impact Crushing Work Index (CWI) determination.
- Bond Rod Mill Work Index (RWI) determination.
- Bond Ball Mill Work Index (BWI) determination.
- Bond Abrasion Index (Ai) determination.
- JK Drop Weight testwork.
- SMC testwork.

13.4.1 UCS

The UCS is a measure of a rock's resistance to compression. Table 13.4.1 provides a summary of the descriptive terms over the range of all UCS values.

Table 13.4.1 UCS Descriptive Strength Terms

UCS (MPa)	Strength
<6	Very weak
6–20	Weak
20–60	Moderately strong
60–200	Strong
>200	Very strong

The results of the UCS tests are presented in Table 13.4.2.

Table 13.4.2 UCS Results

Sample	Lithology	Unit	Value
Comm 001 specimen 1	Granite	MPa	173
Comm 001 specimen 2	Granite	MPa	156
Comm 001 specimen 3	Sediment	MPa	107
Comm 001 specimen 4	Sediment	MPa	115
Comm 001 specimen 5	Sediment	MPa	92
Comm 001 specimen 6	Sediment	MPa	177
Comm 001 specimen 7	Granite	MPa	135
Comm 001 specimen 8	Granite	MPa	150
Comm 001 specimen 9	Granite	MPa	47
Comm 001 specimen 10	Sediment	MPa	141

Salient comments include:

- granite ores are moderately strong to strong
- sediment ores are strong
- minimum UCS value is 47 MPa
- maximum UCS value is 173 MPa
- the 85th percentile value is 167 MPa.

13.4.2 CWI

The results for the bond crushing work index (CWI) tests are presented in Table 13.4.3.

Table 13.4.3 CWI Results

Sample	Lithology	Unit	Value
Comm 002 Specimen 1	Sediment	kWh/t	5.0
Comm 002 Specimen 2	Sediment	kWh/t	6.7
Comm 002 Specimen 3	Sediment	kWh/t	8.3
Comm 002 Specimen 4	Sediment	kWh/t	10.0
Comm 002 Specimen 5	Sediment	kWh/t	5.0
Comm 002 Specimen 6	Granite	kWh/t	6.7
Comm 002 Specimen 7	Granite	kWh/t	6.7
Comm 002 Specimen 8	Granite	kWh/t	13.4
Comm 002 Specimen 9	Granite	kWh/t	6.7
Comm 002 Specimen 10	Granite	kWh/t	8.3

Salient comments include:

- granite and sediment ores have similar properties and generally considered to have moderate energy demand
- minimum CWI value is 5.0 kWh/t
- maximum CWI value is 13.4 kWh/t
- the 85th percentile value is 9.4 kWh/t.

13.4.3 RWI

The results for the bond rod mill work index (RWI) tests are presented in Table 13.4.4. The tests were completed with a closing screen of 1,180 µm.

Table 13.4.4 RWI results

Sample	Lithology	Unit	Value
Comm 003 Var 1	Granite	kWh/t	16.5
Comm 003 Var 2	Granite	kWh/t	17.1
Comm 003 Var 3	Granite	kWh/t	16.5
Comm 003 Var 4	Granite / Porphyry	kWh/t	20.2
Comm 003 Var 5	Granite	kWh/t	16.4
Comm 003 Var 6	Sediment	kWh/t	23.7
Comm 003 Var 7	Sediment	kWh/t	22.6
Comm 003 Var 8	Sediment	kWh/t	20.0
Comm 005 Oxide / Transition	Oxide / Transition	kWh/t	3.7

Salient comments include:

- oxide / transition ore is soft
- granite and sediment ores have slightly different properties, with sediment ore (hard) having a higher energy demand than granite (average)
- minimum RWI = 16.4 kWh/t (granite) and 20.0 kWh/t (sediment)
- maximum RWI = 20.2 kWh/t (granite) and 23.7 kWh/t (sediment)
- the 85th percentile value is 22.4 KWh/t (18.3 kWh/t for granite and 23.4 kWh/t for sediment).

13.4.4 BWI

The results for the bond ball mill work index (BWI) tests are presented in Table 13.4.5. The tests were completed with a closing screen of 125 µm with the exception of Comm 003 Var 1 which was done on a 150 µm screen.

Table 13.4.5 BWI Results

Sample	Lithology	Unit	Value
Comm 003 Var 1	Granite	kWh/t	14.3
Comm 003 Var 2	Granite	kWh/t	16.4
Comm 003 Var 3	Granite	kWh/t	15.2
Comm 003 Var 4	Granite / Porphyry	kWh/t	17.8
Comm 003 Var 5	Granite	kWh/t	15.4
Comm 003 Var 6	Sediment	kWh/t	18.4
Comm 003 Var 7	Sediment	kWh/t	16.4
Comm 003 Var 8	Sediment	kWh/t	16.7
Comm 005 Oxide / Transition	Oxide / Transition	kWh/t	5.4

Salient comments include:

- oxide / transition ore is soft
- granite and sediment ores have similar properties and described as moderately hard
- minimum BWI = 14.3 kWh/t
- maximum BWI = 17.8 kWh/t
- the 85th percentile value is 17.6 KWh/t.

13.4.5 Abrasion Index

The results for the abrasion index (Ai) tests are presented in Table 13.4.6.

Table 13.4.6 Ai Results

Sample	Lithology	Value
Comm 003 Var 3	Granite	0.48
Comm 003 Var 6	Sediment	0.39
Comm 005 Oxide / Transition	Oxide / Transition	0.003

The data shows that the oxide ore is not abrasive, while granite and fresh ores have average abrasiveness[1].

13.4.6 JK Drop Weight Test

A single JK Drop Weight Test was conducted on a sample of the Primary Composite. The results are presented in Table 13.4.7.

Table 13.4.7 JK Drop Weight Test Results

Sample	A	b	A x b	t_a	SG
Comm 004	65.3	0.48	31.3	0.31	2.81

The data describes the ore as being competent.

13.4.7 SMC Test®

Results for the SMC (semi-autogenous mill comminution) Test® are presented in Table 13.4.8. The table presents only the JK Drop Weight Test parameter, A x b and the apparent ore specific energy. The values for the other parameters measured; drop weight index (DWI), coarse ore work index (Mia), high pressure grinding rolls work index (Mih), crushing ore work index (Mic) and JK abrasion parameter (ta) are recorded in the JKTech Pty Ltd report *10001/P24 DW and SMC Test Report on Ten Samples from Tengréla Project – March 2010*.

[1] Not abrasive <0.1, slightly abrasive 0.1-0.4, average abrasive 0.4-0.6, abrasive 0.6-0.8, and very abrasive >0.8

Table 13.4.8 SMC Test® Results

Sample	Lithology	A x b	Category	SG
Comm 003 Var 1	Granite	32.1	Competent	2.77
Comm 003 Var 2	Granite	32.6	Competent	2.76
Comm 003 Var 3	Granite	36.4	Competent	2.75
Comm 003 Var 4	Granite / Porphyry	32.6	Competent	2.75
Comm 003 Var 5	Granite	33.8	Competent	2.77
Comm 003 Var 6	Sediment	24.0	Extremely competent	2.77
Comm 003 Var 7	Sediment	24.7	Extremely competent	2.79
Comm 003 Var 8	Sediment	32.7	Competent	2.75
Comm 004 Primary Ore	Mixed	35.4	Competent	2.79

Salient comments include:

- granite are competent and sediment ores are competent to very competent
- minimum Axb = 32.1 (granite) and 23.7 (sediment)
- maximum Axb = 36.4 (granite) and 32.7 (sediment)
- the 15th percentile value = 32.4 (granite) and 24.2 (sediment).

The high competency exhibited by the sediment ores in particular indicate a propensity for critical size build-up within SAG and ball mill circuits (SAB). This can be managed by either secondary crushing or by pebble crushing, both of which are more energy efficient methods of treating critical size material.

13.4.8 Comminution Design Criteria

The comminution circuit ore characteristics selected from the testwork program for use in comminution circuit design are summarised in Table 13.4.9.

Table 13.4.9 Ore Characteristics for Comminution Circuit Design

Parameter	Unit	Oxide	Granite / Porphyry	Sediment	Source
Ore SG	t/m3	2.78	2.78	2.77	Average of Table 13.4.8
A x b		100[1]	32.4	24.2	85th percentile of Table 13.4.8
CWI	kWh/t	5[2]	13.4	10	Maximum of Table 13.4.3
RWI	kWh/t	3.7	18.3	23.4	85th percentile of Table 13.4.4
BWI	kWh/t	5.4	17.0	17.9	85th percentile of Table 13.4.5
Ai		0.0026	0.48	0.39	Table 13.4.6

[1] A x b parameter for oxide is an assumed value
[2] Crushing work index for oxide is an assumed value

13.5 Gravity Concentration

Gravity concentration testwork was carried out on a sample of the Primary ore composite in Phase 1 test programme, using the following procedure:

- 9.5 kg of Primary ore composite was ground to a P_{80} of 850 µm and processed through a 4.5 inch Knelson concentrator to recover the liberated, coarse free gold.
- The 'coarse' Knelson concentrate was intensively leached for 24 hour in a rolling bottle with 80 kg/t of Leachwell and 200 kg/t of sodium cyanide.
- The 'coarse' concentrated leach residue was recombined with the Knelson tailings, ground to a P_{65} of 75 µm then passed through a 4.5 inch Knelson concentrator to recover the liberated, intermediate gold.
- The 'intermediate' Knelson concentrate was intensively leached for 24 hour in a rolling bottle with 80 kg/t of Leachwell and 200 kg/t of sodium cyanide.
- The 'intermediate' concentrated leach residue was recombined with the Knelson tailings, ground to a P_{80} of 75 µm then passed through a 4.5 inch Knelson concentrator to recover the liberated, fine gold.
- The 'fine' Knelson concentrate was intensively leached for 24 hour in a rolling bottle with 80 kg/t of Leachwell and 200 kg/t of sodium cyanide.

The results of the gravity testwork are presented in Table 13.5.1.

Table 13.5.1 Gravity Recoverable Gold Testwork – Primary Ore

Sample	Grind size (µm)	Mass pull (%)	Concentrate Grade (g/t)	Recovery (%)	Equiv. Head Assay (g/t)
Stage 1	P_{80} 850	0.94	17.5	32.6	0.66
Stage 2	P_{65} 75	1.06	20.3	42.6	0.86
Stage 3	P_{80} 75	0.90	3.5	6.2	0.13
Total				**81.4**	**2.03**

The results indicate that:

- the ore is amenable to gravity concentration
- approximately one third of the feed gold was gravity recoverable at a coarse grind (P_{80} = 850 µm)
- a further 43% was recoverable by grinding to intermediate size (P_{65} = 75 µm).

The results supported the mineralogical examination and a gravity circuit was included in the process flowsheet.

13.6 Cyanide Composite Testwork

Cyanidation tests, most including a gravity recovery stage, have been completed in each stage of metallurgical testwork.

13.6.1 Grind Optimisation

Comparative gravity / cyanide leach tests were conducted in Phase 1 of the testwork programme, to investigate the effect of grind size on gold recovery for the Primary ore composite, with the results summarised in Table 13.6.1.

The standard conditions for each leach tests were:

- grind to P_{80} of 850 µm and pass through 4.5 inch Knelson concentrator
- amalgam concentrate with mercury, and return residue to Knelson tails
- grind to target grind size
- 40% solids with Perth tap water in agitated leach vessel
- initial pH of 10 adjusted with lime (60% CaO)
- 20 g/L attritioned fresh carbon
- oxygen sparge
- initial cyanide concentration of 1,000 ppm maintained at > 500 ppm
- 48 hour leach with solution and carbon assay at 1, 2, 4, 8, 24 and 48 hours.

Table 13.6.1 Grind Optimisation – Primary Ore

Parameter	Unit	P_{80} 212 µm	P_{80} 150 µm	P_{80} 106 µm	P_{80} 75 µm
Test number		WH2078	WH2079	WH2080	WH2081
Calc head grade	g/t Au	4.49	4.12	2.61	2.82
	g/t Ag	0.53	0.53	0.52	0.49
Gold recovery		-	-	-	-
• gravity	% Au	34.1	49.4	38.3	24.0
• 1 hour	% Au	56.1	74.0	68.9	54.2
• 2 hour	% Au	57.9	77.0	70.9	59.7
• 8 hour	% Au	62.0	84.4	78.4	68.2
• 24 hour	% Au	72.9	89.8	85.7	78.7
• 48 hour	% Au	90.8	92.7	91.2	92.2
	% Ag	81.2	81.2	80.8	79.5
	g/t Au	0.41	0.30	0.23	0.22

Results from the data, indicate that:

- gold head grade and gravity recovery is variable within the Primary ore composite sample
- gold recovery is sensitive to grind size above P_{80} 106 µm. This is based on the residue solid assay rather than calculated recovery, the data for which is clouded by the high and variable head grades of the coarser grind tests
- there is a slow leaching component in the ore with leaching continuing at each grind size to 48 hours.

Silver grades were low and recovery consistent at about 80%.

An incremental cost benefit model was developed and reported in the 2011 Definitive Feasibility Study Report, to predict the cost benefit of targeting a grind size P_{80} smaller than 150 µm. It concluded that a P_{80} of 106 µm was optimal.

13.6.2 Multi-Stage Sequential Diagnostic Gold Analysis

A total of fourteen multi-stage sequential diagnostic gold analysis tests were conducted on samples during Phase 1, 2 and 3 of the testwork program:

- Phase 1: Primary ore composite and 4 x variability samples.
- Phase 2: five x variability leach test solid residues.
- Phase 3: four x variability samples.

The variability based samples were selected on the basis of leach performance as discussed in Section 13.7. The results are summarised in Table 13.6.2.

Table 13.6.2 Multi–Stage Sequential Diagnostic Gold Analysis

Sample	Weathering	Lithology	Gravity gold (g/t)	Cyanide recoverable gold (g/t)	Gold locked in carbonate (g/t)	Gold locked in arsenopyrite (g/t)	Gold locked in pyrite (g/t)	Gold locked in silicate (g/t)	Total gold locked (g/t)
Primary Composite	sulphide	mixed	2.94	0.34	0.02	0.09	0.008	0.004	0.12
2.0 g/t Gold	sulphide	mixed	1.33	1.40	0.01	0.06	0.453	0.443	0.97
3.0 g/t Gold	sulphide	mixed	0.18	1.24	0.01	0.06	0.381	0.379	0.83
5.0 g/t Gold	sulphide	mixed	0.97	1.95	0.03	0.04	0.230	0.238	0.54
Depth 5	sulphide	mixed	0.19	1.46	0.02	0.24	0.463	0.457	1.18
PCL20-1	sulphide	granite	0.22	0.02	0.01	0.08	0.024	0.015	0.13
PCL30-1	sulphide	sediment	0.10	0.01	0.01	1.29	0.352	0.413	2.05
PCL42-2	sulphide	sediment	0.31	0.01	0.00	0.04	0.051	0.05	0.14
PCL51-2	sulphide	sediment	0.02	0.03	0.04	1.83	0.188	0.144	2.20
PCL54-1	sulphide	sediment	0.98	0.02	0.00	0.12	0.25	0.024	0.39
Var comp #2	sulphide	sediment	5.94		0.01	0.03	0.034	0.033	0.11
Var comp #9	sulphide	sediment	1.44		0.01	0.01	0.272	0.163	0.46
Var comp#14	transition	sediment	3.98		0.02	0.01	0.035	0.016	0.08
Var comp#20	oxide	laterite	0.08		0.03	0.01	0.00	0.003	0.04

The diagnostic leach test data shows that:

- primary ore composite should be free milling with good gold recovery expected
- gold occurrence is variable within sulphide (or primary) sediment ores, ranging from:
  - free milling (PCL20-1, PCL42-2, and Var comp#2)
  - containing significant gold locked in arsenopyrite with lesser amounts locked in pyrite and silicates (PCL30-1 and PCL51-2)
  - intermediate gold locked in arsenopyrite and pyrite (PCL54-1)
  - intermediate gold locked in arsenopyrite, pyrite and silicates (Depth 5)
  - intermediate gold locked in pyrite and silicates (2.0 g/t Gold, 3.0 g/t Gold, 5.0 g/t Gold and Var comp#9).

The variable nature of the gold deportment suggests the gold is not in solid solution with sulphide minerals but present as small discrete particles within sulphides or silicates with no contact with the host particle edge.

No tests were done to determine whether the gold locked in the various mineral types could be recovered by finer grinding.

13.6.3 Comparative Bottle Roll Tests

A series of four 1 kg bottle roll leach tests were completed on the Primary ore composite during Phase 1 of the test work programme. The aim was to compare leach performance of sealed bottle leach vessels, with that of the agitated leach vessel used in the 'standard' test (refer test WH2080).

The first test (WH2465) was carried out at the following conditions:

- Grind to P_{80} = 106 µm.
- 40% solids with Perth tap water in sealed bottle roll vessel.
- Initial pH of 10 adjusted with lime (60% CaO).
- Initial cyanide concentration of 1,500 ppm maintained at > 500 ppm.
- 48 hour leach with solution assay at 1, 2, 4, 8, 24 and 48 hours.

The second test (WH2467) included a gravity stage and oxygen sparging. The third test (WH2668) included gravity stage only.

The results of the tests are summarised in Table 13.6.3 and includes results from the 'standard' test WH2080 for comparison.

Table 13.6.3 Direct Leach Bottle Roll Tests – Primary Ore

Parameter	Unit	WH2465	WH2467	WH2468	WH2080
Condition		leach	gravity/leach/O_2	gravity/leach	gravity/CIL/O_2
Gold recovery		-	-	-	-
• gravity	% Au	n/a	22.4	28.7	38.3
• @ 8 hour	% Au	91.0	68.4	77.9	78.4
• @ 24 hour	% Au	92.9	76.3	83.7	85.7
• @ 48 hour	% Au	93.6	92.6	90.5	91.2
• Consumption			-	-	-
• Lime	kg/t	0.4	0.34	0.35	0.30
• NaCN	kg/t	0.78	0.43	0.45	1.60

From the data it is apparent that:

- cyanide consumption in a bottle roll leach test is significantly lower than the agitated leach test for the same ore. It is thought that this is due to HCN gas stripping in the open agitated vessel. The estimated cyanide consumption used in plant operating costs will be based on the bottle roll test data
- a gravity stage did not increase overall recovery
- leach kinetics is reduced with a gravity stage included and is contrary to expectation
- elevated dissolved oxygen levels (oxygen addition) does not increase kinetics or overall recovery
- the best result is achieved with direct leach only, i.e. no gravity stage and no oxygen addition
- results from bottle roll and agitated leach test are comparable with the exception of cyanide consumption noted above.

Data from the test log-sheets show that dissolved oxygen (DO) was consistent throughout the bottle roll tests without oxygen addition, maintaining DO levels of approximately 8 ppm. This confirms the observations recorded in Section 13.8.1. When oxygen was sparged into the vessel, DO level increased to 30 ppm. The slower leach kinetics of the tests with gravity stage cannot be attributed to oxygen starvation.

13.6.4 Cyanide Dosage Optimisation

Three comparative bottle roll tests were performed on samples of the Primary ore composite during Phase 1 of the test programme, to determine the effect of cyanidation concentration on leach performance.

The tests were conducted to the same conditions as WH2465 in Section 13.6.3, but with the following variations:

- 0.05% w/v initial cyanide and maintaining 0.025% w/v.
- 0.08% w/v initial cyanide and maintaining 0.04% w/v.
- 0.12% w/v initial cyanide and maintaining 0.06% w/v.

The results of the tests are summarised in Table 13.6.4.

Table 13.6.4 Cyanide Dosage Optimisation – Primary Ore

Parameter	Unit	WH2699	WH2700	WH2701
CN dose	% w/v	0.05	0.08	0.12
Gold recovery		-	-	-
• @ 36 hour	% Au	87.1	92.9	91.9
• @ 48 hour	% Au	92.0	93.5	94.0
• Consumption		-	-	-
• Lime	kg/t	0.24	0.20	0.20
• NaCN	kg/t	0.18	0.33	0.55

The results show that:

- cyanide consumption increases linearly with initial cyanide concentration
- cyanide concentration below 0.08% w/v can affect leach kinetics and leach extraction up to 48 h.

Initial cyanide concentrations of between 0.08%w/v and 0.09% w/v should be targeted for Primary ore when leaching for 48 hours.

13.6.5 Leach Kinetics – Oxide

A direct leach bottle roll test was completed on a sample of the Oxide ore composition during Phase 1 of the test programme. The test was conducted at the same conditions as WH2465 in Section 13.6.3 with exception that the initial cyanide strength was higher at 0.15% w/v and maintained at 0.075% w/v.

The result is summarised in Table 13.6.5.

Table 13.6.5 Leach Kinetics – Oxide Ore

Parameter	Unit	WH2678
Phase		1
Gold recovery		-
• @ 4 hour	% Au	70.3
• @ 8 hour	% Au	77.8
• @ 24 hour	% Au	86.7
• @ 48 hour	% Au	93.6
• Consumption		
• Lime	kg/t	0.76
• NaCN	kg/t	0.39

The results indicate:

- there is a slow leaching component to the oxide ore
- very good recoveries can be expected after 48 h of leaching.

13.6.6 Bulk Gravity / CIL Tests

Four bulk gravity CIL tests have been performed on ore samples from Phase 1 and 3 of the testwork program:

- Phase 1: Primary ore composite and oxide ore composite.
- Phase 3: Sulphide master composite and oxide master composite.

The Phase 1 tests were conducted at the conditions outlined in Section 13.6.1 and a grind P_{80} size of 106 µm. The Phase 3 tests were done at the same conditions with the exception that of the initial cyanide concentration which was increased to 1,500 ppm.

The tests were conducted to provide samples for geochemical, tailings, slurry rheology and arsenic precipitation testwork.

The results are summarised in Table 13.6.6.

Table 13.6.6 Bulk Gravity / CIL Tests

Parameter	Unit	Primary Ore	Oxide Ore	Sulphide Master	Oxide Master
Test No		WH2669	WH2692	WH4551	WH4550
Phase		1	1	3	3
Gold recovery		-	-	-	-
• gravity	% Au	22.6	18.5	20.9	58.2
• @ 8 hour	% Au	88.3	85.1	84.7	96.4
• @ 24 hour	% Au	90.8	91.7	83.4	96.0
• @ 48 hour	% Au	93.0	95.2	84.1	96.6
• Consumption		-	-	-	-
• Lime	kg/t	0.26	0.50	0.36	1.06
• NaCN	kg/t	2.13	1.88	2.25	1.58

13.6.7 Pre-Oxygenation and Lead Nitrate

A series of tests were completed during Phases 2 and 3 of the test programme to assess the effect of pre-oxygenation and lead nitrate addition on gold recovery:

- Phase 2: 1 pre-oxygenation, 1 x lead nitrate, 1 x (pre-oxygenation plus lead nitrate) test on one sample (PCL20-1).
- Phase 3: 3 x lead nitrate addition tests on four composites samples (Pit North Oxide, Pit North Sulphide, Pit South Oxide and Pit South Sulphide).
- Phase 3: pre-oxygenation tests on four composites samples.

Phase 2

Diagnostic leach analysis on the residues of four selected low recovery Phase 2 variability samples showed that half had significant (0.22 g/t to 1.00 g/t) unrecovered free / cyanide recoverable gold after the 48 hours of leaching. One of these samples, PCL20-01, was selected to assess the effect of pre-oxygenation and lead nitrate addition on gold recovery.

Four tests were planned:

- Test 1: Base line (repeat of Phase 2 variability test WH3139).
- Test 2: Baseline plus four hour pre-oxygenation.
- Test 3: Baseline plus 50 g/t lead nitrate.
- Test 4: Test 2 plus 50 g/t lead nitrate.

The results of the tests are summarised in Table 13.6.7.

Table 13.6.7 Pre-Oxygenation and Lead Nitrate Addition

Parameter	Unit	Test 1	Test 2	Test 3	Test 4
Test No		WH3346	WH3347	WH3348	WH3349
Pre-oxygenation		no	4 hour	no	4 hour
Lead nitrate	g/t	0	0	50	50
Calc head grade	g/t Au	4.49	4.12	2.61	2.82
Gold recovery		-	-	-	-
• gravity	% Au	15.0	17.0	15.5	14.0
• @ 1 hour	% Au	64.1	68.3	66.4	59.8
• @ 2 hour	% Au	69.0	72.7	71.5	62.8
• @ 8 hour	% Au	74.9	81.2	76.0	71.1
• @ 24 hour	% Au	77.5	87.1	82.8	78.7
• @ 48 hour	% Au	80.9	91.1	91.0	90.3
• Consumption		-	-	-	-
• Lime	kg/t	0.35	0.50	0.33	0.58
• NaCN	kg/t	1.7	1.56	1.63	1.56

From the results it is apparent that:

- the baseline test replicated the previous data (refer WH3139 in Table 13.7.2)
- 4 h of pre-oxygenation increased leach kinetics and overall gold recovery by ≈10%
- 50 g/t lead nitrate achieved the same overall gold recovery increase as per-oxygenation but with marginally slower kinetics
- there was no synergy in adding lead nitrate addition with pre-oxygenation
- lime addition was lower in this series of tests than with WH3139
- cyanide consumptions are essentially unchanged.

The results from these tests triggered further work in Phase 3 of the test programme.

Phase 3

Four composite samples, Pit North Oxide, Pit North Sulphide, Pit South Oxide and Pit South Sulphide, were subjected to series of tests to assess the effect of a four hour pre-oxygenation stage and varying lead nitrate additions (20 g/t 50 g/t and 70 g/t on gold recovery. All other conditions were kept to those noted in Section 13.6.1, with the exception of the initial cyanide strength, which was increased to 0.15% w/v.

The results are presented in Table 13.6.8, Table 13.6.9, Table 13.6.10 and Table 13.6.11.

Table 13.6.8 Pre-Oxygenation and Lead Nitrate Addition – Pit North Oxide

Parameter	Unit	Base	Test 1	Test 2	Test 3	Test 4
Test No		WH4525	WH4552	WH4513	WH4514	WH4515
Pre-oxygenation		no	4 hours	no	no	No
Lead nitrate	g/t	0	0	20	50	70
Calc head grade	g/t Au	1.63	1.67	2.68	2.14	1.54
Residue grade	g/t Au	0.19	0.12	0.12	0.14	0.08
Gold recovery		-	-	-	-	-
• gravity	% Au	18.2	27.9	25.1	27.5	15.2
• @ 24 hour	% Au	88.3	89.0	93.2	90.9	89.1
• @ 48 hour	% Au	88.3	92.8	95.5	93.5	94.8
• Consumption		-	-	-	-	-
• Lime	kg/t	0.86	1.60	0.95	0.92	0.91
• NaCN	kg/t	1.53	1.45	1.53	1.32	1.46

Table 13.6.9 Pre-Oxygenation and Lead Nitrate Addition – Pit South Oxide

Parameter	Unit	Base	Test 1	Test 2	Test 3	Test 4
Test No		WH4526	WH4553	WH4516	WH4517	WH4518
Pre-oxygenation		no	4 hours	no	no	No
Lead nitrate	g/t	0	0	20	50	70
Calc head grade	g/t Au	3.97	1.64	2.04	2.32	1.94
Residue grade	g/t Au	0.15	0.12	0.03	0.10	0.13
Gold recovery		-	-	-	-	-
• gravity	% Au	39.2	40.0	46.2	38.9	19.0
• @ 24 hour	% Au	72.8	92.7	90.1	92.0	88.9
• @ 48 hour	% Au	96.2	92.7	98.5	95.7	93.3
• Consumption		-	-	-	-	-
• Lime	kg/t	1.03	1.39	0.89	1.14	1.43
• NaCN	kg/t	1.50	1.45	1.38	1.53	1.42

Table 13.6.10 Pre-Oxygenation and Lead Nitrate Addition – Pit North Sulphide

Parameter	Unit	Base	Test 1	Test 2	Test 3	Test 4
Test No		WH4527	WH4554	WH4519	WH4520	WH4521
Pre-oxygenation		no	4 hours	no	no	No
Lead nitrate	g/t	0	0	20	50	70
Calc head grade	g/t Au	2.26	2.06	1.77	1.95	2.03
Residue grade	g/t Au	0.22	0.17	0.17	0.25	0.22
Gold recovery		-	-	-	-	-
• gravity	% Au	29.5	26.1	15.0	13.0	14.3
• @ 24 hour	% Au	82.7	85.5	80.5	79.8	80.6
• @ 48 hour	% Au	89.4	89.4	90.4	87.2	89.2
• Consumption		-	-	-	-	-
• Lime	kg/t	0.34	0.69	0.35	0.35	0.36
• NaCN	kg/t	1.56	1.43	1.55	1.72	1.51

Table 13.6.11 Pre-Oxygenation and Lead Nitrate Addition – Pit South Sulphide

Parameter	Unit	Base	Test 1	Test 2	Test 3	Test 4
Test No		WH4528	WH4555	WH4522	WH4523	WH4524
Pre-oxygenation		no	4 hours	no	no	No
Lead nitrate	g/t	0	0	20	50	70
Calc head grade	g/t Au	3.36	4.85	2.51	3.59	3.19
Residue grade	g/t Au	0.60	0.64	0.59	0.77	0.59
Gold recovery		-	-	-	-	-
• gravity	% Au	36.7	34.0	8.0	27.0	39.7
• @ 24 hour	% Au	76.0	71.9	66.0	71.0	78.8
• @ 48 hour	% Au	82.2	86.8	76.5	78.5	81.5
• Consumption		-	-	-	-	-
• Lime	kg/t	0.33	0.59	0.49	0.49	0.44
• NaCN	kg/t	1.42	1.71	1.35	1.42	1.55

The calculated head grades across each sample were variable with the exception of the Pit North Sulphide, making interpretation of the recovery results difficult. The following conclusions are based on analysis of the leach residue solids grade:

- Pit north oxide: sample responds to both pre-oxidation and lead nitrate addition. The best result was achieved with high (70 g/t) lead nitrate addition.
- Pit north sulphide: some benefit in recovery with pre-oxidation and low (20 g/t) lead nitrate addition.

- Pit south oxide: sample responds to both pre-oxidation and lead nitrate addition. The best result was achieved with low (20 g/t) lead nitrate addition.

- Pit south sulphide: no appreciable change in leach solid residue grade.

The four hour pre-oxidation step was included in the 'standard' test used for the Phase 3 variability testwork.

13.7 Cyanide Variability Testwork

Cyanidation variability test work has been completed during Phase's 1, 2 and 3 of the test programme.

13.7.1 Phase 1

A total of eighteen samples were subjected to the same test conditions noted in Section 13.6.1 (gravity/CIL/oxygen sparging) with a grind P_{80} size of 106 µm and an initial cyanide concentration of 0.15% w/v and maintained above 0.05% w/v:

- 2 x oxide sample.

- 5 x lithology.

- 6 x grade.

- 5 x depth.

The results are presented in Table 13.7.1.

Table 13.7.1 Phase 1 Cyanidation Variability Tests

Sample	Lithology	Weathering	Test	Calc head grade, Au g/t	Gravity Recovery, %	24 hour leach recovery, %	48 hour leach recovery, %	Lime Usage, kg/t	NaCN Usage, kg/t
Lithology 1	Primary	Porphyry	WH2388R	2.09	15.4	89.0	95.9	0.37	1.33
Lithology 2	Primary	Sediment	WH2389R	1.96	19.0	85.5	86.3	0.31	1.51
Lithology 3	Primary	Sediment	WH2390R	1.81	11.2	83.4	88.1	0.36	1.60
Lithology 4	Primary	Granite	WH2391R	0.19	6.8	86.1	87.2	0.33	1.72
Lithology 5	Primary	Granite	WH2392	2.45	9.1	82.2	92.2	0.30	1.72
0.8 g/t Gold	Primary	Mixed	WH2382R	1.47	22.8	78.0	88.8	0.30	1.40
1.2 g/t Gold	Primary	Mixed	WH2383	1.02	32.0	83.3	91.6	0.32	1.41
2.0 g/t Gold	Primary	Mixed	WH2384R	3.68	36.1	69.6	74.2	0.33	1.59
3.0 g/t Gold	Primary	Mixed	WH2385R	2.10	8.2	63.3	68.0	0.34	1.61
4.0 g/t Gold	Primary	Mixed	WH2386R	2.58	32.3	88.4	92.3	0.31	1.47
5.0 g/t Gold	Primary	Mixed	WH2387R	3.59	27.1	80.6	81.6	0.37	1.65
Depth 1	Oxide	Mixed	WH2393R	1.89	17.1	84.9	90.2	0.37	1.69
Depth 2	Transitional	Mixed	WH2394R	1.35	25.7	85.1	89.6	0.35	1.68
Depth 3	Primary	Mixed	WH2395	1.92	10.0	85.3	89.6	0.82	1.77
Depth 4	Primary	Mixed	WH2396R	3.21	54.5	86.4	89.6	0.79	1.56
Depth 5	Primary	Mixed	WH2397R	3.59	6.8	54.3	59.7	0.34	1.73
Oxide 1	Oxide	Porphyry	WH2676	4.48	21.2	96.8	96.5	0.51	1.93
Oxide 2	Oxide	Sediment	WH2677	2.78	17.8	94.9	95.7	0.51	1.97

The results indicate that:

- oxide ore gold recoveries range from 90.2% to 96.5%, gravity recovery was consistent at 17-20%
- fresh porphyry gold recovery was excellent at 95%
- fresh sediment gold recovery is lower than oxide but amenable to cyanidation, with recoveries between 86% to 88%
- fresh granite gold recovery is higher than sediment but lower than oxide at 87.2% to 92.2%
- unexpectedly low recoveries were obtained for three of the grade variability tests 2.0 g/t Gold, 3.0 g/t Gold and 5.0 g/t Gold. Diagnostic leach analysis shows that the gold in tails is predominantly locked in pyrite and silicate particles in roughly equal proportion. No tests were performed to determine if the gold could be recovered by further liberation
- gold recovery seems unchanged with depth (ore type excluded) until the very bottom of the deposit, where the Depth 5 sample returned a poor recovery of 59.7%. Diagnostics leaching showed that of the 1.11 g/t gold in solid residue ≈20% is locked in arsenopyrite, ≈40% locked in pyrite and ≈40% locked in silicates. No tests were performed to determine if the gold could be recovered by further liberation
- there is significant leaching (>2%) occurring during the last 24 h of the 48 h leach in thirteen of the eighteen samples – indicating a slow leaching fraction of gold.

13.7.2 Phase 2

A total of twenty samples were subjected to the same test conditions noted in Section 13.6.1 (gravity / CIL / oxygen sparging) with a grind P_{80} size of 106 µm and an initial cyanide concentration of 0.15% w/v and maintained above 0.025% w/v:

- 8 x oxide sample.
- 1 x transition sample.
- 4 x fresh granite samples.
- 8 x sediment samples.

Note mercury amalgamation period was two hours not five hours as per the Phase 1 tests. The results, sorted by ore type are presented in Table 13.7.2.

Table 13.7.2 Phase 2 Cyanidation Variability Tests

Sample	Lithology	Weathering	Test	Calc head grade, Au g/t	Gravity Recovery, %	24 hour leach recovery, %	48 hour leach recovery, %	Lime Usage, kg/t	NaCN Usage, kg/t
PCL08-1	Oxide	sediment	WH3133	1.99	73.2	96.0	96.0	3.95	1.62
PCL20-2	Oxide	sediment	WH3140	15.18	38.7	95.7	95.9	0.95	1.55
PCL24-2	Oxide	sediment	WH3142	4.14	56.4	87.0	95.2	1.41	2.10
PCL12-1	Oxide	sediment	WH3135	1.69	18.2	89.2	94.1	1.06	1.62
PCL04-1	Oxide	porphyry	WH3132	2.35	15.1	90.2	90.2	0.87	1.48
PCL12-2	Oxide	sediment	WH3136	4.22	16.3	81.8	86.0	1.83	1.48
PCL16-2	Oxide	sediment	WH3138	2.30	4.6	78.7	85.2	0.70	1.45
PCL08-2	Oxide	sediment	WH3134	1.48	6.3	72.9	85.1	0.81	1.45
PCL24-1	Transition	sediment	WH3141	73.5	76.7	98.9	99.0	0.81	1.46
PCL45-2	Sulphide	granite	WH3148	3.59	19.1	72.5	93.0	0.45	1.26
PCL16-1	Sulphide	granite	WH3137	2.51	8.2	86.4	90.7	0.39	1.47
PCL48-1	Sulphide	granite	WH3149	2.22	5.9	87.0	87.0	0.45	1.61
PCL20-1	Sulphide	granite	WH3139	2.26	18.5	77.8	81.4	1.11	1.39
PCL36-1	Sulphide	sediment	WH3144	2.71	75.6	93.3	95.6	0.72	1.66
PCL45-1	Sulphide	sediment	WH3147	6.29	26.8	69.2	86.3	0.66	1.25
PCL48-2	Sulphide	sediment	WH3150	2.48	10.5	80.0	83.5	0.33	0.51
PCL42-1	Sulphide	sediment	WH3145	1.82	34.8	79.3	82.4	0.40	1.54
PCL42-2	Sulphide	sediment	WH3146	2.03	9.6	71.1	73.9	0.48	1.54
PCL54-1	Sulphide	sediment	WH3152	2.98	24.5	57.1	59.7	0.34	1.40
PCL30-1	Sulphide	sediment	WH3143	4.73	7.4	39.2	47.6	0.50	1.51
PCL51-2	Sulphide	sediment	WH3151	5.01	0.7	31.0	40.0	0.43	1.65

The results indicate that:

- Oxide ore:
  - Gravity recoverable gold content is highly variable (average 28.6% with a range of 4.6% to 73.2%) and does not increase with head grade.
  - Total gold recovery is variable with a range of 85.1% to 96.0%. The average recovery is 91.0% and is similar to that recorded from Phase 1 testwork.
  - High recovery oxide ores leached very quickly. Test log-sheets show leaching complete in less than four hours.
  - Low recovery oxide ores tended to have low gravity recovery and a slow leaching component, with leaching continuing to the end of the test at 48 h. This may indicate that gold is finer and may not be adequately liberated at P_{80} 106 µm.
  - Transition ore should have good gold recovery, however only one sample was tested.
- Primary granite:
  - Gravity recoverable gold content is low averaging 12.9% with a range of 5.9% to 19.1%.
  - Total gold recovery is variable, average = 88.0% and a range of 81.4% to 93.0%.
  - Leaching is continuing to the test completion on three of the four samples suggesting a slow leaching component to the ore.
- Diagnostic analysis (refer Table 13.6.2) of the leach residue for lowest recovery granite sample (PCL20-1) showed that:
  - 60%, or 0.22 g/t, was free gold
  - 6%, or 0.02 g/t, was cyanide recoverable
  - 34% of the gold, 0.14 g/t, was locked in carbonates, pyrite, arsenopyrite and silicates and considered unrecoverable at this grind size.
- Primary sediment:
  - Gravity recoverable gold content is highly variable, average = 23.7% with a range of 0.7% to 75.6% and does not appear to be related to head grade.

- Total gold recovery is variable, average = 71.1% with a range of 39.9% to 95.6%.

- Leaching (>2% in last 24 h) is continuing to the test completion in all samples indicating a slow leaching component.

- As with the oxide ore the lower recover samples tended to have low gravity recoverable gold content.

- Diagnostic analysis (refer Table 13.6.2) of the three lowest recovery test residues show that in:

  - PCL54-1, 84%, or 0.98 g/t of the gold was free, 0.12 g/t was locked in arsenopyrite, and 0.05g/t locked in pyrite and silicates. The total amount of gold locked is 0.17 g/t and this is considered unrecoverable at the current grind size. Assuming the free gold in residue was recovered then recovery on this sample would increase to 94%

  - PCL30-1, the majority of the gold was locked in arsenopyrite (1.29 g/t) with lesser amounts locked in pyrite (0.35 g/t) and silicates (0.41 g/t). The total locked gold is 2.07 g/t and is considered unrecoverable at the current grind size

  - PCL51-2 gold deportment is similar to PCL30-1 in that the majority of gold is locked in arsenopyrite (1.83 g/t) with lesser amounts locked in pyrite (0.19 g/t) and silicates (0.14 g/t). The total locked gold is 2.20 g/t and is considered unrecoverable at the current grind size.

13.7.3 Phase 3

Twenty one batch / gravity CIL tests were completed during Phase 3 of the metallurgical testwork programme. The samples included:

- 6 x oxide
- 1 x transitional
- 4 x fresh granite
- 10 x fresh sediment.

The test conditions were as per the previous phase but modified to include a four hour pre-oxygenation stage prior to cyanide addition and the commencement of the 48 hour leach. The results of the tests, grouped by lithology, are presented in Table 13.7.3.

Note one sample VC#20 had a calculated head grade of 0.12 g/t Au. This is well below the deposit cut-off grade and is excluded in the discussion of the results.

Table 13.7.3 Phase 3 Cyanidation Variability Tests

Sample	Lithology	Weathering	Test	Calc head grade, Au g/t	Gravity Recovery, %	24 hour leach recovery, %	48 hour leach recovery, %	Lime Usage, kg/t	NaCN Usage, kg/t
VC#20	Oxide	laterite	WH4548	0.12	15.5	66.3	66.3	3.14	1.45
VC#4	Oxide	sediment	WH4532	14.61	87.4	98.4	98.9	0.79	1.35
VC#21	Oxide	sediment	WH4549	2.53	68.1	93.8	96.8	1.05	1.32
VC#16	Oxide	sediment	WH4544	1.56	38.2	89.4	94.9	1.10	1.48
VC#3	Oxide	sediment	WH4531	0.45	16.6	91.1	91.1	1.26	1.28
VC#18	Oxide	sediment	WH4546	4.51	5.6	86.7	88.2	0.85	1.24
VC#7	Sulphide	granite	WH4535	2.49	24.7	87.2	94.4	0.53	1.33
VC#8	Sulphide	granite	WH4536	3.59	31.6	83.7	92.5	0.53	1.08
VC#5	Sulphide	granite	WH4533	2.17	10.5	86.6	86.6	0.61	1.40
VC#6	Sulphide	granite	WH4534	7.55	28.8	69.6	75.1	0.70	1.43
VC#2	Sulphide	sediment	WH4530R	6.30	64.9	85.9	94.3	0.85	1.22
VC#13	Sulphide	sediment	WH4541	4.96	13.0	81.6	92.1	0.62	1.34
VC#12	Sulphide	sediment	WH4540	2.76	12.9	88.9	91.0	0.75	1.34
VC#10	Sulphide	sediment	WH4538	3.44	8.0	84.7	86.3	0.83	1.50
VC#11	Sulphide	sediment	WH4539	2.22	6.2	79.2	82.7	0.69	1.62
VC#19	Sulphide	sediment	WH4547	2.28	17.4	75.3	81.6	0.66	1.44
VC#1	Sulphide	sediment	WH4529	1.46	5.5	79.5	81.5	0.67	1.20
VC#17	Sulphide	sediment	WH4545	2.12	20.9	80.2	80.2	0.53	1.13
VC#15	Sulphide	sediment	WH4543	1.07	6.2	72.0	76.6	0.63	1.24
VC#9	Sulphide	sediment	WH4537	3.10	5.4	46.5	46.5	0.72	1.50
VC#14	Transition	sediment	WH4542R	4.31	44.0	85.3	92.2	1.28	1.49

The results indicate that:

- Oxide ore:
  - Gravity recoverable gold content is highly variable (average 43.2% with a range of 5.6% to 87.4%).
  - Total gold recovery is variable with an average of 94.0% and a range of 88.2% to 98.9%. The results are generally similar to previous test phases.
  - As with Phase 2, the lowest recovery oxide ores had low gravity recovery and slow leaching component, however on this occasion leaching was complete after 36 hours.
  - Has variable leach kinetics, some with leaching complete after 4 hours and others still leaching at the completion of the 48 hour test.
  - Transition ore should have good gold recovery, however only one sample was tested.
- Primary granite:
  - Gravity recoverable gold content is variable averaging 23.9% with a range of 10.5% to 31.6%.
  - Total gold recovery is variable, average = 87.2% and a range of 75.1% to 94.4%.
  - Leaching is continuing to the test completion on two of the four samples suggesting a slow leaching component to the ore.
- Primary sediment:
  - Gravity recoverable gold content is highly variable, average = 18.6% with a range of 5.4% to 64.9% and would appear to have some relation to head grade – with the two highest recoveries coinciding with the two highest grade samples.
  - Total gold recovery is variable, average = 82.3% with a range of 46.5% to 94.3%.
  - Leaching (>2% in last 24 h) is continuing to the test completion in seven of ten samples indicating a slow leaching component.
  - As with the oxide ore the lower recover samples tended to have low gravity recoverable gold content.

- Diagnostic analysis (refer Table 13.6.2) of VC#9 (46.5% recovery) showed that:
  - 59.6% of the gold is locked in pyrite
  - 35.7% of the gold is locked in silicates
  - the remainder evenly spread between carbonate and arsenopyrite
  - diagnostic analysis on three other samples (refer Table 13.6.2) showed consistent trend of little gold (0.04g/t to 0.11g/t) 'locked' in either carbonate, arsenopyrite, pyrite and silicates.

13.8 Ancillary Testwork

13.8.1 Oxygen Uptake Rate

One oxygen uptake test was completed on a sample of the primary ore composite during Phase 1 of the test work programme. The results are presented in Table 13.8.1.

Table 13.8.1 Oxygen Uptake Rate

Time (h)	Unit	Primary Ore
0	mg/L/min	0
1	mg/L/min	-0.0129
2	mg/L/min	0.0099
3	mg/L/min	-0.0138
4	mg/L/min	-0.0093
5	mg/L/min	-0.0094
6	mg/L/min	-0.0115
24	mg/L/min	-0.0251

The primary ore composite does not have a high oxygen demand.

13.8.2 Viscosity

Slurry viscosity was measured on two ores sample during Phase 1 of the testwork programme:

- Primary ore composite at pulp densities of 40% and 55% w/w solids and pH 10.5.
- Oxide ore composite at pulp densities of 40%, 50% and 55% w/w solids and pH 10.5.

The results are presented in Table 13.8.2 and Table 13.8.3.

Table 13.8.2 Slurry Viscosity – Primary Ore

Shear Rate $(sec)^{-1}$	Viscosity (cps) at Various Pulp Densities		
	40% w/w	50% w/w	55% w/w
67.4	16	n/d	16
119.4	26	n/d	28
209.5	42	n/d	47

Table 13.8.3 Slurry Viscosity – Oxide Ore

Shear Rate $(sec)^{-1}$	Viscosity (cps) at Various Pulp Densities		
	40% w/w	50% w/w	55% w/w
4.2	412	599	1,422
7.4	255	425	1,275
13.1	156	312	960
21.9	108	208	632
38.9	77	137	412
67.4	68	105	271
119.4	69	98	195
209.5	88	110	185

13.8.3 Carbon Equilibrium and Adsorption Test Work

Approximately 9 kg of the Primary ore composite was subjected to a 48 hour leach test (no gravity recovery stage and no added carbon) in Phase 1 of the test programme. The test was done with a grind P_{80} size of 106 µm and oxygen sparging. The results are presented in Table 13.8.4.

Table 13.8.4 CIL Test for Carbon Adsorption Kinetics

Parameter	Unit	WH2720
Phase		1
Sample		Primary ore
Gold recovery		-
• @ 2 hour	% Au	45.3
• @ 8 hour	% Au	79.7
• @ 24 hour	% Au	90.7
• @ 48 hour	% Au	95.7
• Consumption		
• Lime	kg/t	0.25
• NaCN	kg/t	1.22

The test confirmed the presence of a slow leaching component and returned a higher recovery than any of the previous tests on the same sample that included a gravity stage.

The resultant slurry from WH2720 was used for sequential triple contact carbon adsorption testwork. The Fleming model was used to define the carbon kinetics. The Fleming constants for the Primary ore sample were determined to be:

- Rate constant, k = 296 h^{-1}
- Time constant, n = 0.63.

The data shows that gold will load onto carbon rapidly.

13.8.4 Arsenic Precipitation

The leach solution from the Phase 3 bulk gravity / CIL tests discussed in section Table 13.6.6 were used for a series of tests to assess the efficiency of arsenic precipitation at various Fe:As molar ratio's; 2.5, 5, 10, 20 and 40.

The results of the tests are summarised in Table 13.8.5.

Table 13.8.5 Arsenic Precipitation from Leach Solutions

Fe:As Molar Ratio	Oxide Master (WH4550)			Sulphide Master (WH4551)		
	Solution As (mg/L)	Solution Fe (mg/L)	Lime Dosage (g/g Fe)	Solution As (mg/L)	Solution Fe (mg/L)	Lime Dosage (g/g Fe)
0	0.6	0.6	0	169	36.1	0
2.5:1	0.7	0.6	0	4.5	44.9	1.43
5:1	0.8	0.3	0	0.4	32.7	1.15
10:1	0.7	0.4	0	0.1	23.3	0.85
20:1	0.7	0.5	0	<0.1	7.3	0.84
40:1	0.6	0.5	0	<0.1	1.8	0.85

From the results:

- arsenic dissolution in oxide ore was low and precipitation using ferric sulphate is ineffective
- ferric sulphate addition is effective in precipitating As in the sulphide ore, Fe:As molar ratio of 2.5:1 and 5:1 produced precipitation rates of 97.3% and 99.8%, respectively.

Samples of the proposed tailings storage facility floor were drilled to recover clay for arsenic adsorption tests and to provide holes for TSF floor permeability testwork. The TSF floor clays were transported to Perth for:

- constant head infiltration tests (CHIT)
- arsenic adsorption tests.

The results of the tests showed that:

- TSF floor permeability's ranged from 3.4×10^{-8} ms^{-1} to 3.4×10^{-11} ms^{-1}
- TSF floor has potential for an arsenic sink with an arsenic adsorption potential range of 73 mg_{As}/kg_{clay} to 148 mg_{As}/kg_{clay}.

Worley Parson used the results to generate a contaminant transport model to estimate the seepage rate of arsenic in TSF water. The model predicted a very slow release of arsenic bearing solution, such that after six years of operation and 200 years of lying dormant an arsenic plume would have extended 250 m beyond the footprint of the TSF and that a plume in excess of World Health Organisation drinking levels would not reach the Bagoe River for 8,700 years.

An arsenic precipitation circuit has not been included in the process flow sheet.

13.8.5 Thickener Testwork

Settling rate testwork was carried out on leach residue samples from the Phase 3 composite samples: Pit South Sulphide, Pit North Sulphide, Pit South oxide and Pit North Oxide.

Initial flocculant screening testwork was carried out on each blended sample to select the most appropriate flocculant for the thickener testwork program. The flocculant tested were from the BASF and SNF range; Magnafloc 10, Magnafloc 342, Magnafloc 5250 and SNF AN923, AN910SH and AN905SH. Magnafloc 10 was selected as it gave the best performing flocculant in terms of free settling rate and overflow clarity.

The results from the dynamic thickener tests are presented in Table 13.8.6.

Table 13.8.6 Thickener Testwork Results

Parameter	Unit	Pit North Sulphide	Pit South Sulphide	Pit North Oxide	Pit South Oxide
Settling Rate	t/m^{2}h	1.00	1.00	1.00	1.00
Feed Slurry Density	%w/w	22	20	16	14
Slurry pH		1.06	1.06	10.6	10.6
Flocculant Dose	g/t	30	10	30	30-50
Overflow Clarity	ppm	~350	<100	<100	<100
Underflow Density	%w/w	63-65	63-65	53-55	49-51
Yield Stress	Pa	28-39	28-43	35-120	28-65

The CIL circuit has a design density of 52% solids w/w for the slurry being leached. The data shows that that the ores can be thickened up to this figure – although only just for the pit south oxide sample.

Centrifugal pumping systems are generally designed with a slurry yield stress of less than 30 Pa. The sulphide ores are just reaching this value at 63% solids so there should be no pumping issues at the lower density. The oxide sample yield stresses indicate that pumping will be problematic at the design CIL density, therefore a lower density of 45% solids w/w is recommended when processing oxides.

13.9 Flotation

Four laboratory batch flotation tests were conducted on samples of the Primary ore composite. Grind P_{80} size was varied (212 µm, 150 µm, 106 µm and 75 µm), with all other conditions remaining constant, as follows:

- 2.5 L Agitair flotation cell.
- Natural pH with Perth tap water.
- Add 25 g/t PAX and condition for 1 minute.
- Recover two consecutive concentrates at 30 seconds and 60 seconds.
- Add 10 g/t PAX and condition for 1 minute.
- Recover three consecutive concentrates at 1 minute, 1 minute and 2 minute.

The results of the tests are summarised in Table 13.9.1.

Table 13.9.1 Flotation Grind / Recovery Result – Primary Ore

Parameter	Unit	Test 1	Test 2	Test 3	Test 4
Test No		WH2012	WH2013	WH2014	WH2015
Grind size, P_{80}	µm	212	150	106	75
Calc head grade	g/t Au	2.01	2.85	3.91	2.00
Recovery		-	-	-	-
• mass	%	3.7	3.4	3.2	3.8
• gold	%	93.3	93.6	96.8	96.2
• sulphur	%	95.1	98.0	97.0	97.0
• arsenic	%	91.9	94.3	94.8	95.0
Tails grade	g/t Au	0.14	0.19	0.13	0.08

While the variability in gold head grade (attributed to liberated free gold particles in the concentrate samples) does somewhat mask the results it is evident that:

- the ore responds well to flotation
- gold recovery decreases with increasing grind size (optimum grind size is difficult to differentiate, either P_{80} = 75 µm based on tailings assays and 106 µm based on recovery)
- sulphur recovery is essentially complete at P_{80} grind sizes of 150 µm and less
- arsenic recovery did not significantly increase below 106 µm.

No cyanidation testwork was conducted on the flotation concentrate. Oxide ore was not expected to respond as favourably to flotation as it does to whole of ore cyanidation and further evaluation of flotation as a processing route was discontinued.

13.10 Expected Metallurgical Performance

13.10.1 Oxide Ore

Gravity and total gold extraction data for all the leach tests conducted on Sissingué oxide ore samples have been collated in Table 13.10.1. Where multiple tests have been conducted on the same sample the results have been averaged.

Table 13.10.1 Gravity / CIL Results – Oxide Ore

Sample	Calc. Head	Gold Recovery		
	Au, g/t	Gravity, %	24 h leach, %	48 h leach, %
Oxide Var 1	4.48	21.2	96.8	96.5
Oxide Var 2	2.78	17.8	94.9	95.7
Oxide Composite	4.05	9.2	89.2	94.4
Variability (Depth) - 1	1.89	17.1	84.9	90.2
PCL04-1	2.35	15.1	90.2	90.2
PCL08-1	1.99	73.2	96.0	96.0
PCL08-2	1.48	6.3	72.9	85.1
PCL12-1	1.69	18.2	89.2	94.1
PCL12-2	4.22	16.3	81.8	86.0
PCL16-2	2.3	4.6	78.7	85.2
PCL20-2	15.18	38.7	95.7	95.8
PCL24-2	4.14	56.4	87.0	95.2
VC#16	1.56	38.2	89.4	94.9
VC#3	0.45	16.6	91.1	91.1
VC#4	14.61	87.4	98.4	98.9
VC#18	4.51	5.55	86.7	88.2
VC#21	2.53	68.1	93.8	96.8
Pit North - Oxide	1.93	22.8	90.1	93.0
Pit South - Oxide	2.38	36.7	87.3	95.3
Average [1]		35.8	91.6	94.5

[1] Excludes PCL08-2, PCL12-2, PCL16-2 and VC#18

In approximately 40% of the samples, leaching was essentially complete within 24 hours, with the remaining samples showing a slow leaching component to 48 hours.

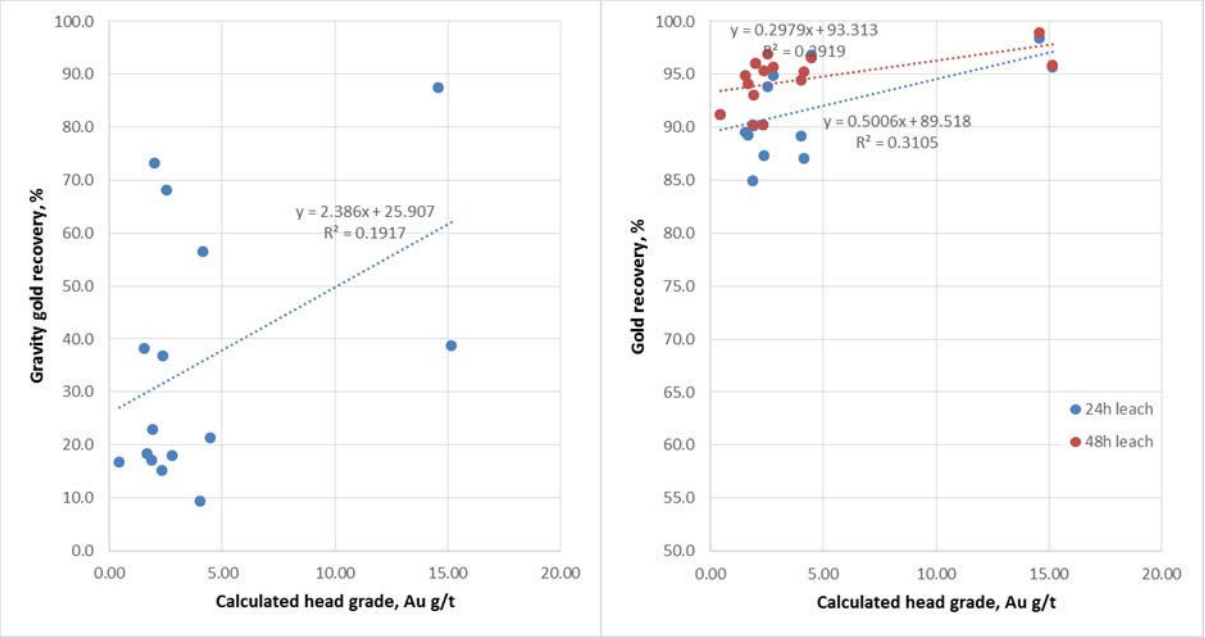
All but four of the samples reported total gold extractions of greater than 90%. The lowest recovery samples were:

- PCL08-2: 85.2%
- PCL12-2: 86.0%
- PCL16-2: 85.2%
- VC#18: 88.2%

The position of theses samples within the pit was determined from drill hole collar co-ordinates, RL, azimuth, dip and the sample interval. The samples[1] were found to be clustered within a 12 m long by 20 m wide by 23 m high block in the centre of the pit. This volume equates to 10,000 tonnes assuming 1.8 t/m^3 for insitu density. The results are excluded from recovery predictions as they represent only 0.6% of the total oxide tonnes planned for processing. Note a fifth sample[2] taken from within the same cluster returned a gold recovery of 95.7%.

The remaining results were plotted to show gravity / leach extraction with head grade as show in Figure 13.10.1.

Figure 13.10.1 Gravity and CIL Extractions – Oxide Ore



[1] PCL08-2: 1154318 m north, 806614 m east and RL352
PCL12-2: 1154306 m north, 806633 m east and RL334
PCL16-2: 1154316 m north, 806628 m east and RL330
VC#18: 1154309 m north, 806634 m east and RL332

[2] Oxide Var 2: 1154317 m north, 806624 m east and RL337

The data shows that apart from a general increasing trend with very high grade, there is no clear relationship between head grade and gravity / total recovery at the grades expected to report as mill feed. In view of this averaged value are used for predicted oxide ore gold recovery:

- Gravity gold recovery 30%.
- Total recovery 91% with 24 h leaching and 94% with 48 h leaching.

At the design oxide throughput rate of 150 t/h, oxide ore gold recovery is expected to be 92%.

13.10.2 Primary Ore

Granite

Gravity and total gold extraction data for all the leach tests conducted on Sissingué primary granite ore samples have been collated in Table 13.10.2.

Table 13.10.2 Gravity / CIL Results – Primary Granite Ore

Sample	Calc. Head	Grade	Grade	Gold Recovery		
	Au, g/t	$S_{sulphide}$ %	As %	Gravity, %	24 h leach, %	48 h leach, %
Variability (Lith) - Granite 1	0.19	0.043	1.04	6.8	86.1	87.2
Variability (Lith) - Granite 2	2.45	0.245	0.79	9.1	82.2	92.2
PCL16-1	2.51	0.307	1.12	8.2	86.4	90.7
PCL20-1	2.26	0.357	1.24	16.0	80.8	86.9
PCL45-2	3.59	0.323	1.30	19.1	72.5	93.0
PCL48-1	2.22	0.530	1.14	5.9	87.0	87.0
VC#5	2.17	0.480	1.16	10.5	86.6	86.6
VC#6	7.55	0.378	1.36	28.8	69.6	75.1
VC#7	2.49	0.205	0.74	24.7	87.2	94.4
VC#8	3.59	0.424	1.34	31.6	83.7	92.5
Average [1]				14.6	83.6	90.1

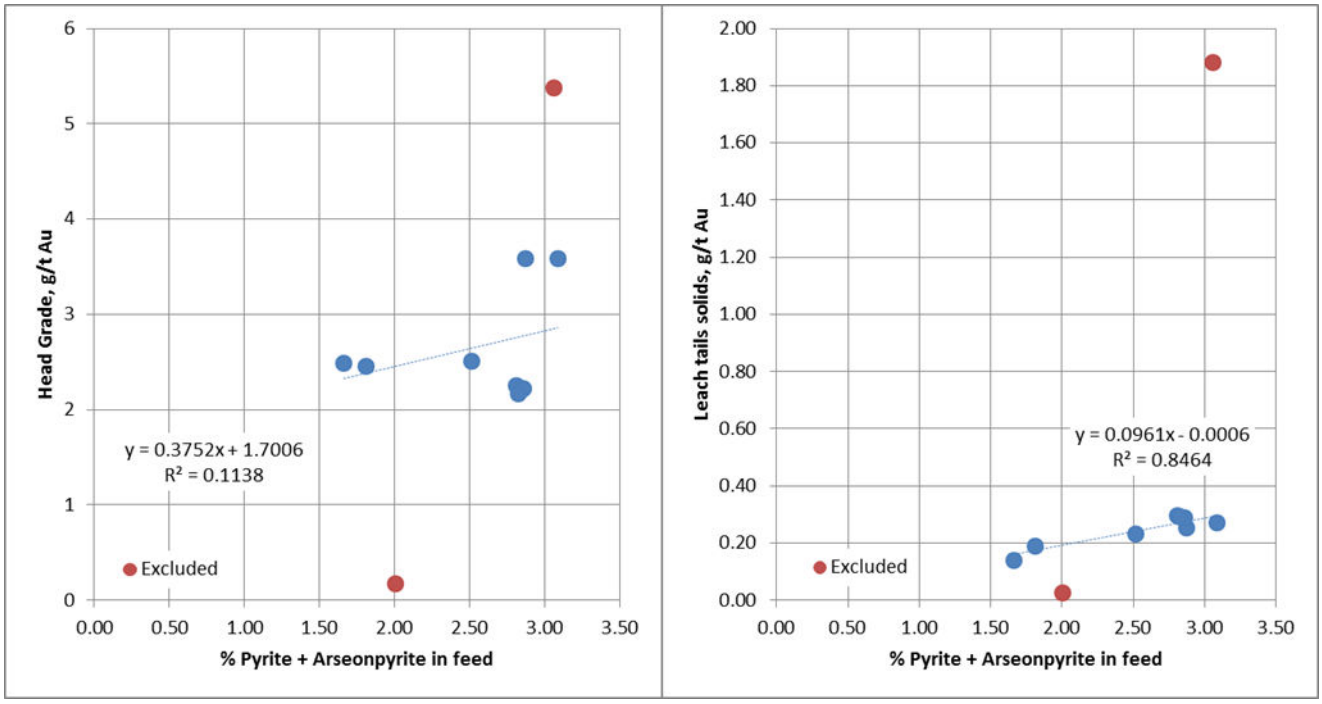
The calculated head grade for the 'Variability (Lith) – Granite 1' sample is well below the mine cut-off grade and it is excluded from analysis, however it is worth noting it recorded a comparatively high recovery of 87%.

Sample 'VC#6' exhibits the same relationship between gold residue grade and sulphide mineral content as the Type 2 sediments discussed below.

The remaining results exhibited a poor relationship between gold in feed and pyrite and arsenopyrite content[1], but a very strong relationship between gold in leach tails solids and pyrite and arsenopyrite content as demonstrated by Figure 13.10.2.

[1] VC#6 excluded

Figure 13.10.2 Feed and Tails Gold with Pyrite and Arsenopyrite in Feed – Granite Ore



The relationship between pyrite and arsenopyrite content and leach solids residue grade can be used to model expected granite gold recovery, as shown in the following equation:

Granite recovery = 100 (Au – 0.1797Ss – 0.1316As +0.006) / Au

Where:

- Au is gold head grade
- S_s is sulphide sulphur grade
- As is arsenic grade

A comparison of actual and modelled recovery is shown in Table 13.10.3.

[1] The abundance of pyrite and arsenopyrite content is calculated by assuming there are no other sulphide minerals present, with arsenopyrite calculated from arsenic grade and pyrite content calculated from $S_{sulphide}$ assay minus the amount of S associated with arsenopyrite.

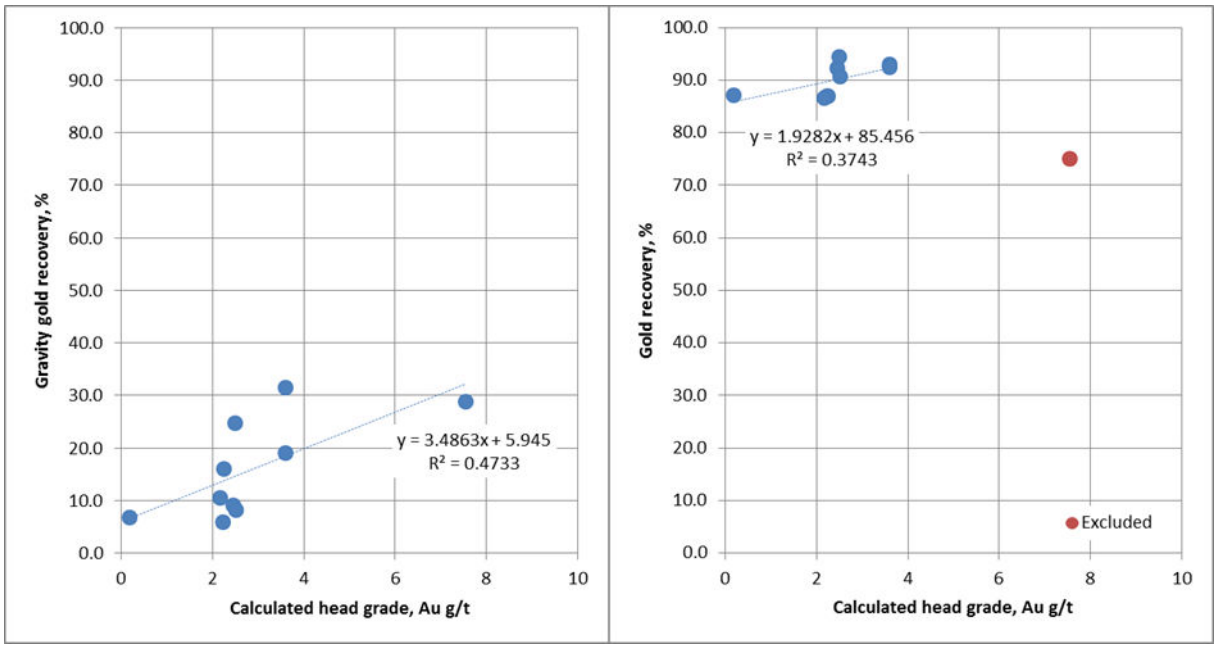
Table 13.10.3 Sulphide Model Recovery Comparison – Primary Granite Ore

Sample	Gold Recovery	
	Actual, %	Model, %
Variability (Lith) - Granite 1	92.2	92.9
Variability (Lith) - Granite 2	90.7	90.4
PCL16-1	86.9	88.1
PCL20-1	93.0	92.3
PCL45-2	87.0	87.7
PCL48-1	86.6	87.5
VC#5	94.4	93.6
VC#7	92.5	91.8
VC#8	90.4	90.5
Average [1]	92.2	92.9

The model is an accurate estimate of performance, with the differential between model and actual varying between 0.1% and 1.3% of the test result. Unfortunately the geology database does not contain sufficient S_s and As information to allow the use model for predictive purposes and an alternate model is required.

Figure 13.10.3 is a plot of the show gravity / leach extraction with head grade.

Figure 13.10.3 Gravity and CIL Extractions – Granite Ore



[1] VC#6 excluded

The gravity gold and total gold recovery trends with head grade are not strong. In view of this, averaged values are used for predicted granite ore gold recovery:

- gravity gold recovery 10%[1].
- total recovery 90%.

Porphyry

Gravity and total gold extraction data for the leach test conducted on Sissingué primary porphyry ore samples are shown in Table 13.10.4.

Table 13.10.4 Gravity / CIL Results – Primary Granite Ore

Sample	Calc. Head	Grade	Grade	Gold Recovery		
	Au, g/t	$S_{sulphide}$ %	As %	Gravity, %	24 h leach, %	48 h leach, %
Variability (Lith) - Porphyry	2.09	0.320	0.152	15.3	88.6	95.5

The recovery is similar to the granite ores and the recovery using the granite sulphide recovery model is calculated at 96.3%.

The porphyry ore only data base is limited to a single drill hole and continuous interval. Extrapolating this performance across all porphyry ores in the deposit may overstate recovery. Until such time as additional porphyry samples are tested and prove otherwise the porphyry ores will be assigned granite recovery.

Sediment

Gravity and total gold extraction data for all the leach tests conducted on Sissingué primary sediment ore samples are shown in Table 13.10.5.

[1] The test average figure has been discounted by 5% (absolute) to account for plant operation where less than 100% of mill feed is passed through the gravity concentrator

Table 13.10.5 Gravity / CIL Results – Sediment Granite Ore

Sample	Calc. Head	Grade	Grade	Gold Recovery		
	Au, g/t	$S_{sulphide}$ %	As %	Gravity, %	24 h leach, %	48 h leach, %
VC#15	1.07	1.10	0.623	6.2	72.0	76.6
VC#1	1.46	0.68	0.140	5.5	79.5	81.5
PCL36-1	2.71	0.10	0.023	75.6	93.3	95.6
Variability (Lith) - Sediment 1	1.96	1.66	0.426	19.0	85.5	86.3
Variability (Lith) - Sediment 2	1.81	1.28	0.201	11.2	83.4	88.1
PCL48-2	2.48	1.48	0.413	10.5	80.0	83.5
VC#10	3.44	2.00	0.832	8.0	84.7	86.3
VC#2	6.3	0.82	0.316	64.9	85.9	94.3
PCL42-1	1.82	0.58	0.105	34.8	79.3	82.4
VC#11	2.22	1.42	0.346	6.2	79.2	82.7
VC#12	2.76	0.82	0.150	12.9	88.9	91.0
VC#17	2.12	1.34	0.780	20.9	80.2	80.2
VC#19	2.28	1.50	0.452	17.4	75.3	81.6
VC#9	3.1	1.60	1.320	5.4	46.5	46.5
PCL51-2	5.01	1.68	0.638	0.7	31.0	39.9
PCL30-1	4.73	1.56	0.660	7.4	39.2	47.6
PCL54-1	2.98	1.22	0.138	24.5	57.1	59.7
PCL42-2	2.03	0.72	0.205	9.6	71.1	73.9
VC#13	4.96	0.34	0.070	13.0	81.6	92.1
PCL45-1	6.29	1.20	0.303	26.8	69.2	86.3
Average				19.0	73.1	77.8

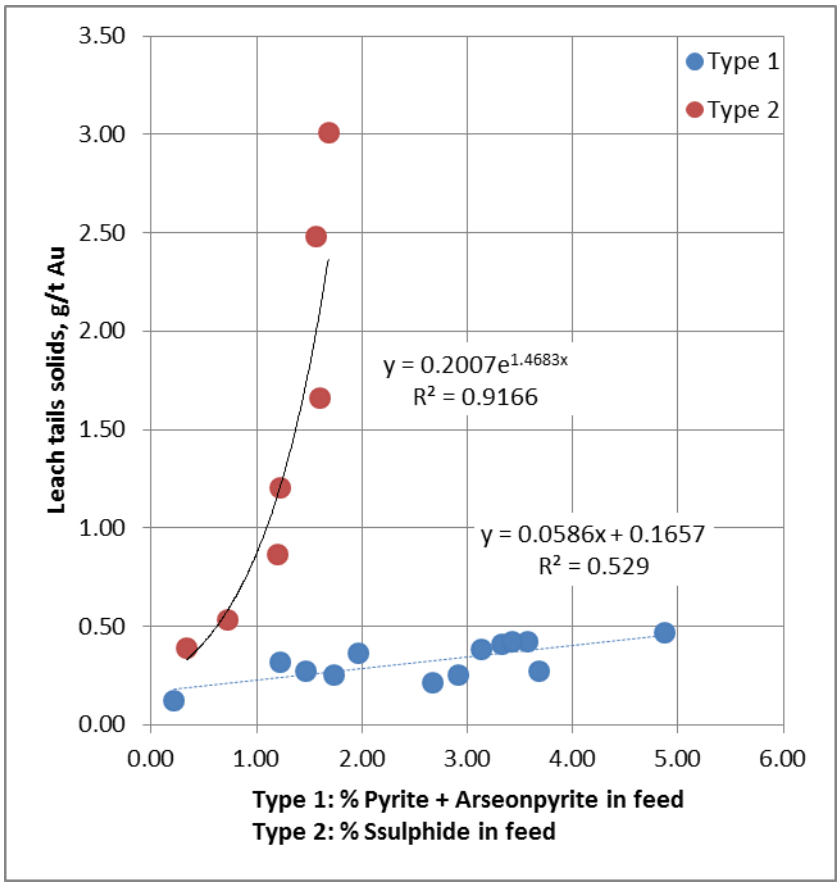
The data shows variable metallurgical performance across this lithology, with gravity recovery varying between 0.7% to 75.6% and total gold recovery varying between 39.9% and 95.6%.

Closer analysis of the data shows there appears to be at least two types metallurgical responses exhibited by the sediment ores:

- Type 1: gold in leach tails solids increases linearly with pyrite and arsenopyrite content.
- Type 2 gold in leach tails solids increases exponentially with $S_{sulphide}$ content.

The relationships are shown graphically in Figure 13.10.4.

Figure 13.10.4 Relationship of Tails Gold with Pyrite and Arsenopyrite Content / Sulphide Assay

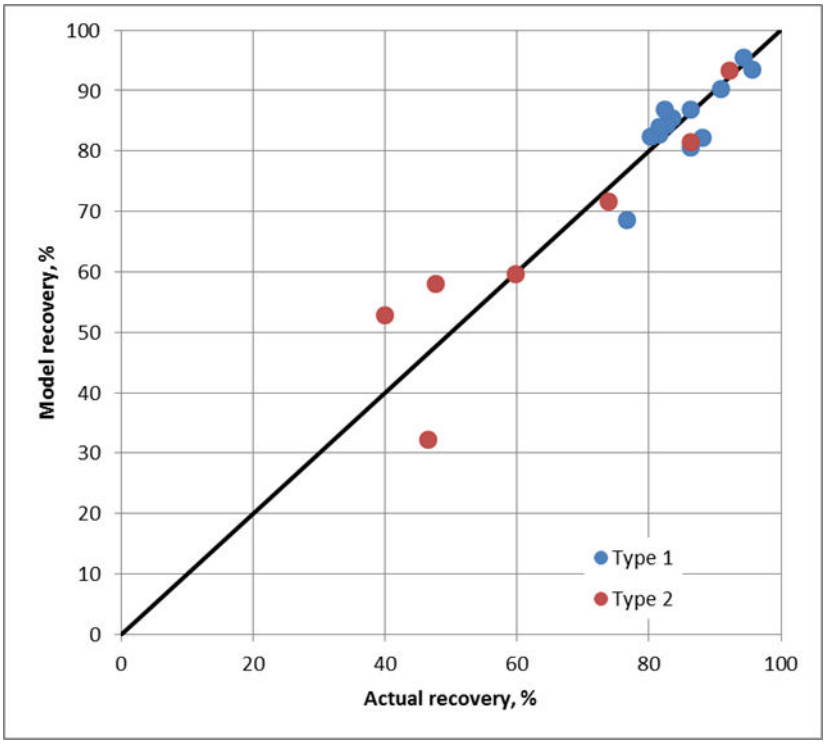


The relationships can be used to generate the recovery models:

- Sediment Type 1 recovery = 100 (Au – 0.1096S_s – 0.0803As - 0.1657) / Au.
- Sediment Type 2 recovery = 100 (Au – 0.2007(EXP(1.4668S_s)) / Au.

A comparison of actual to model is provided in Figure 13.10.5.

Figure 13.10.5 Sediment Sulphide Model Comparison



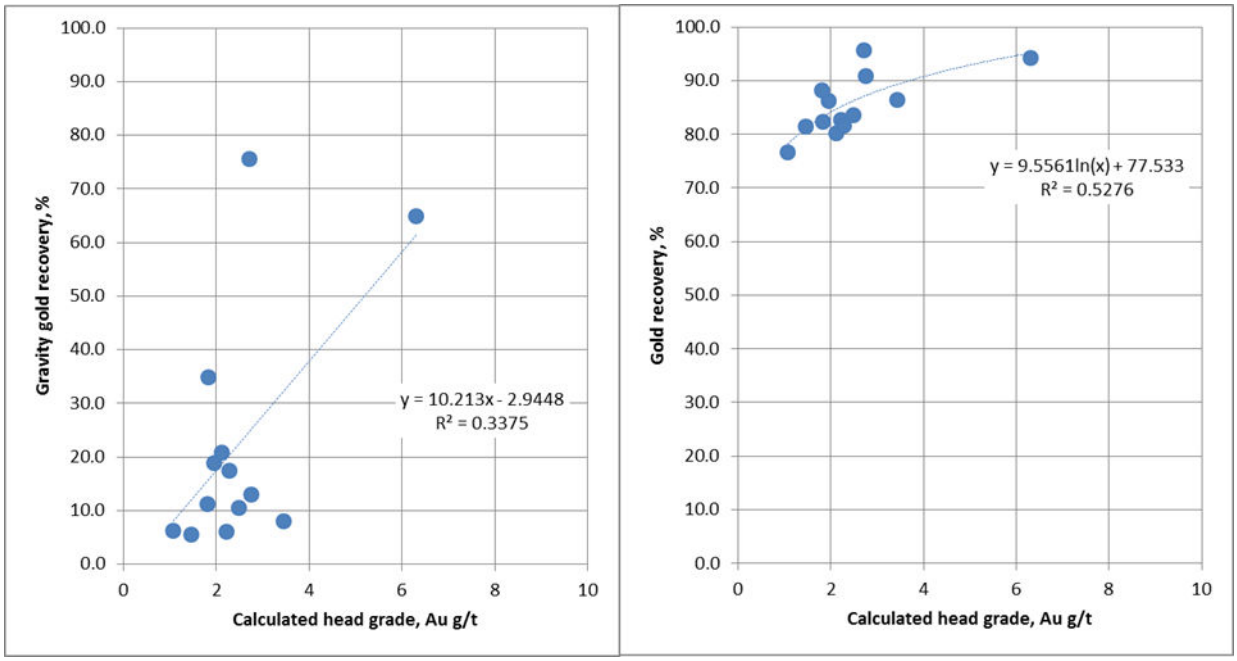
As with the granite these models cannot be used for predictive purposes due to insufficient data in the geology database.

There are seven sediment Type 2 samples; one (VC#9) is located inside the proposed pit and the remaining six (VC#13, PCL51-2, PCL30-1, PCL42-2, PCL54-1 and PCL45-1) are located outside of the proposed pit. Including VC#6 the number of 'Type 2 sediment' ores within the pit shell is two.

The Type 2 sediment ores located within the pit are allocated recovery based on the formula; recovery = 50.3 x Head grade +3.2.

Figure 13.10.6 shows all gravity and leach extraction data for the sediment Type 1 ores.

Figure 13.10.6 Type 1 Sediment Ore Gravity / CIL Extractions



Both the gravity recovery and overall recovery increase with head grade. The gravity gold relationship is variable and not very strong. The head grade and recovery relationship is more pronounced. In view of this predicted sediment ore gold recoveries are:

- gravity gold recovery 17%[1]
- total recovery = 9.556 ln (Au) +77.53.

Summary

The expected gold recovery from each ore type is summarised in Table 13.10.6.

Table 13.10.6 Gold Recovery by Ore Type

Ore type	Recovery, %	
	Gravity	Total
Oxide	30	91 @ 24h / 94 @ 48h
Granite	10	90
Porphyry	10	90
Sediment – Type 1	17	9.556 ln (HG_{Au}) + 77.53
Sediment - Type 2	17	50.3 x HG_{Au} + 3.2

[1] The test average figure has been discounted by 5% (absolute) to account for plant operation where less than 100% of the mill feed is passed through the gravity concentrator.

13.10.3 Reagent Consumption

Cyanide

The cyanide consumption data from the agitated leach vessel tests are excluded from review as they are prone to HCN gas stripping and only the data from bottle roll leach tests are considered.

From the testwork the cyanide consumptions are as follows:

- Oxide ore 0.39 kg/t.
- Primary ore 0.18 kg/t to 0.78 kg/t with 0.50 kg/t selected.

Lime

Lime consumption data from all tests are summarised into Table 13.10.7.

Table 13.10.7 Lime Consumption Results

Weathering	Lime Consumption		
	Minimum	Average	Maximum
Oxide	0.50	1.14	3.95
Transition	0.97	0.81	1.28
Sulphide	0.20	0.46	1.11

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14.0 MINERAL RESOURCE ESTIMATES

The Mineral Resource estimate has been prepared by Snowden in October 2014, using CAE Studio (Datamine) software. No additional drilling was carried out over the Mineral Resource area since the completion of the estimate.

The Mineral Resource estimate was prepared using the following steps:

- Data validation.
- Data preparation.
- Geological interpretation and modelling.
- Establishment of block models.
- Compositing of assay intervals.
- Drill hole grade data analysis.
- Analysis of top cuts.
- Variogram analysis.
- Derivation of kriging plan and boundary conditions.
- Grade interpolation.
- Validation of grade estimates.
- Classification of estimates.
- Resource tabulation and resource reporting.

In the opinion of the author, the key assumptions, parameters and methods used, as described below, are appropriate for the style of mineralisation and to industry best practice, as laid out by the CIM council (2003).

14.1 Domain Analysis, Data Analysis and Compositing

The exploration drill dataset was composited to 2 m downhole with the composite lengths adjusted to include all intervals and avoid the loss of residual samples. Over 99 % of the raw samples are 2 m or less in length. The mineralisation domains and weathering domains were used as hard boundaries for compositing purposes.

A statistical and geological analysis was carried out to determine the boundary conditions between the various geology, mineralisation and the weathering domains. This analysis was used to determine which data to combine for statistical analysis, variography and estimation.

The granite, sediments and dykes were kept separate for statistical analysis, variography and estimation as they are different geological units with mostly hard boundaries between them. The southern and northern granite and sediment domains were also kept separate as there is a physical separation and a difference in grade tenor between the two areas, particularly for the granite (higher in the southern area, Figure 14.1.1).

Review of the individual dykes showed that they are statistically similar and, given the relatively small amount of data in the individual dykes, the dykes were combined for statistical analysis and variography (Figure 14.1.2). The dykes were kept separate for estimation as there are hard boundaries between them.

Figure 14.1.1 Log-probability Plots For Southern and Northern Granites (after Snowden, 2014)

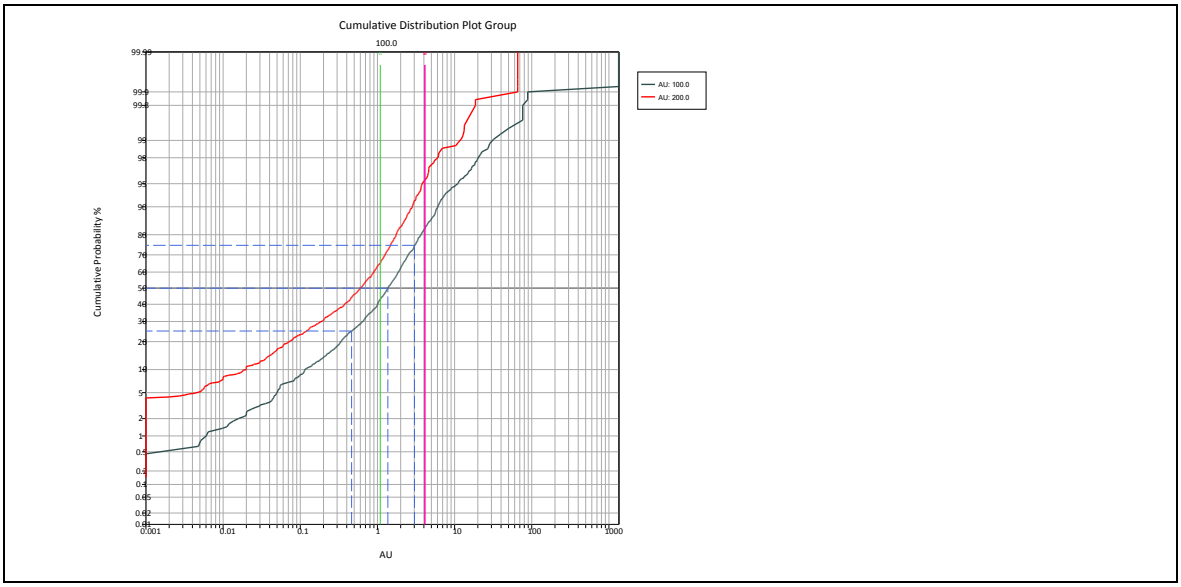
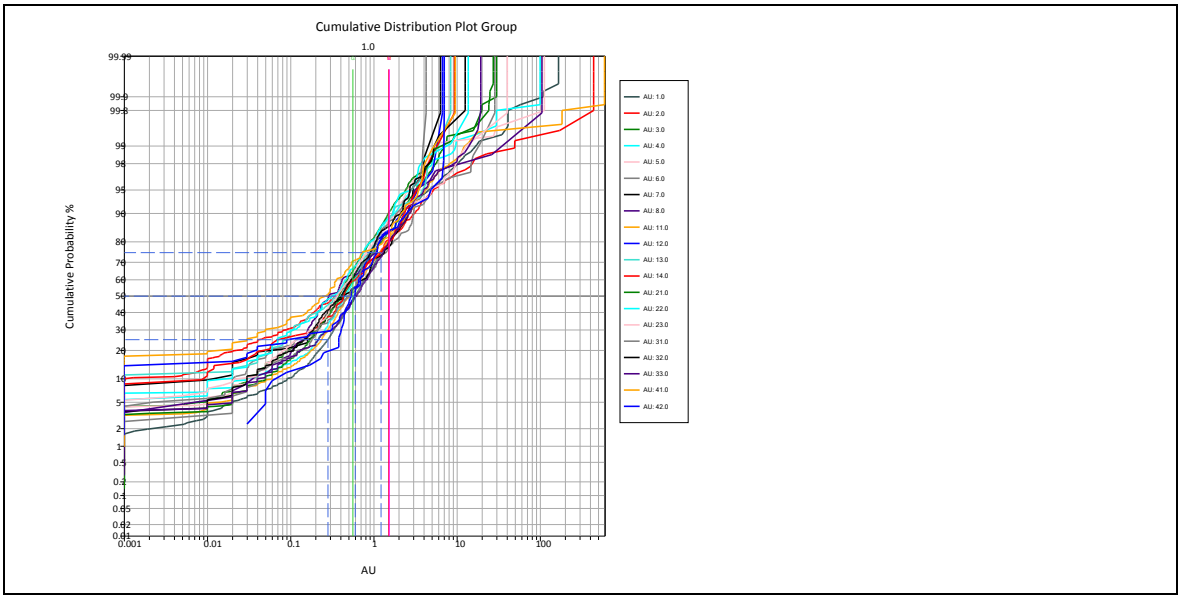


Figure 14.1.2 Log-probability Plots For All Dykes (after Snowden, 2014)

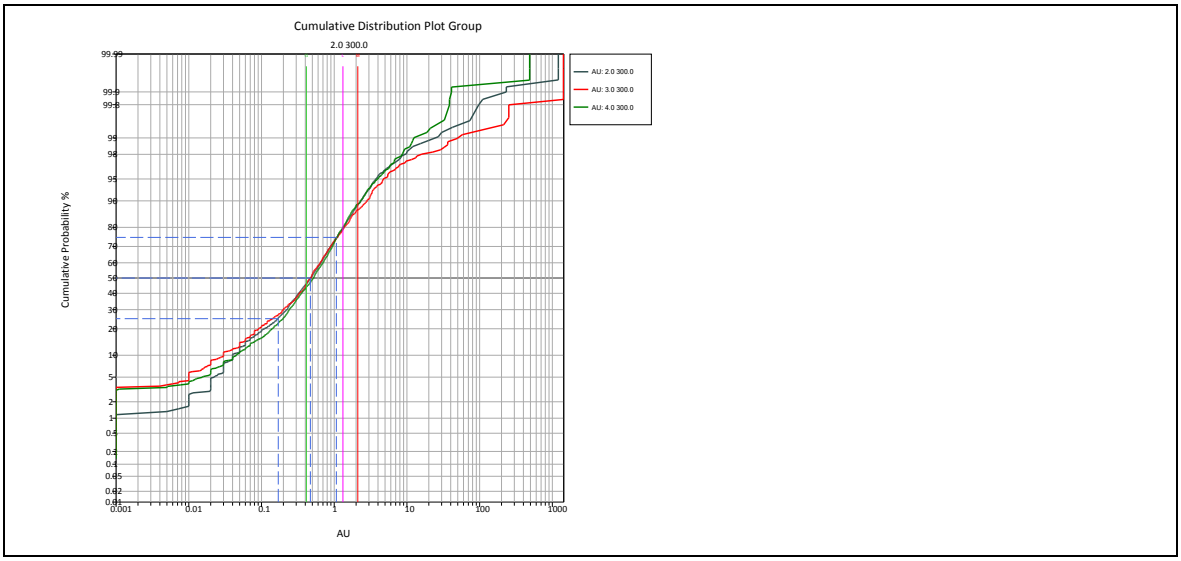


Review of the oxide, transitional and fresh weathering domains within the granites, sediments and dykes indicates that the grade distributions have similar shapes, with typically higher grades in the transitional domain. Figure 14.1.3 shows an example of the oxide, transitional and fresh log probability plots for the northern granite.

This pattern of grade distribution fits with the geological model discussed in Section 7.4, which suggests that gold has been depleted in weathered (oxidised) material and descending waters resulted in gold enrichment in the transitional zone.

The transitional domain is typically narrow and contains limited data compared to the oxide and fresh domains.

Figure 14.1.3 Log-probability Plots For Oxide (2), Transitional (3) and Fresh (4) Domains in the Northern Granite (after Snowden, 2014)



Boundary plots have been generated to assess the change in grade across the weathering domains for each of the mineralisation domains, except for the southern granite which contains little data outside of the fresh domain. Except for the southern sediments where the transition zone is clearly enriched, the boundary plots of all domains show strong differences between the oxide and fresh domains.

Based on the statistical analysis and boundary conditions, the oxide, transitional and fresh domains were kept separate for statistical analysis, however, they were combined for variography. The oxide and transitional data were used to estimate the oxide, while the transitional and fresh data were used to estimate the transitional and fresh domains.

The laterite domain has very little data in it. Visual inspection of the drillhole grades between the laterite and the oxide does not show evidence of a depleted zone. As a result of the small dataset, the laterite was combined with the oxide for estimation. The laterite data was not used for variography.

Statistical analysis was carried out on the gold composite data to determine the grade characteristics and to ensure that the domains are adequate for estimation. Based on the boundary analysis carried out initially, all dykes were combined for statistical analysis.

The data exhibits a bias as a result of data clustering. As a result, all statistics were declustered using cell weighted declustering with a cell size of 40 mE by 40 mN by 2 mRL.

Summary statistics (Table 14.1.1, Table 14.1.2 and Table 14.1.3), histograms and log-probability plots were generated for each mineralisation and weathering domain.

All gold grade distributions are positively skewed with high coefficient of variation (CV). The granite and sediment domains in particular contain significant outliers with three grades above 1,000 g/t Au. Review of the histograms and log-probability plots for each domain indicates that there is some mixing of grade populations. As a result of the mixed populations, together with the high CV values and outliers evident in the granite and sediment domains, multiple indicator kriging was selected as an appropriate estimation method. As result, no top cuts were required for these domains.

For the lower grade dykes, ordinary kriging was selected as appropriate, with top cuts to limit the influence of the higher grade composites. A top cut of 30 g/t Au was selected for the dykes in all weathering domains, based on review of the histograms and natural breaks in the populations in the log-probability plots. Top cut statistics are presented at the bottom of Table 14.1.3.

Table 14.1.1 Summary Statistics by Mineralisation Domain and Weathering Domain - Granites

Statistic		Southern Granite			Northern Granite			
		Total	**Transition**	**Fresh**	**Total**	**Oxide**	**Transition**	**Fresh**
Samples		1,095	37	1,058	690	57	73	558
Minimum		0.00	0.02	0.00	0.00	0.00	0.00	0.00
Maximum		1,336.00	51.78	1,336.00	65.82	5.13	65.82	18.65
Mean		4.33	3.33	4.38	1.03	1.23	259	0.85
Std Dev.		49.38	7.09	50.47	2.57	1.28	7.00	1.54
CV		11.40	2.13	11.53	2.49	1.04	2.70	1.81
Variance		2,438.00	50.30	2,547.00	6.59	1.63	49.00	2.39
Skewness		26.65	5.68	26.09	16.08	1.47	8.18	5.61
Percentile	10 %	0.10	0.35	0.10	0.01	0.02	0.00	0.01
	20 %	0.27	0.58	0.26	0.05	0.07	0.12	0.05
	30 %	0.47	0.95	0.44	0.13	0.37	0.63	0.12
	40 %	0.77	1.24	0.75	0.26	0.82	0.76	0.22
	50 %	1.15	1.56	1.11	0.45	0.91	1.24	0.38
	60 %	1.59	1.78	1.55	0.69	1.00	1.71	0.58
	70 %	2.13	2.14	2.12	1.00	1.32	2.23	0.88
	80 %	3.18	3.59	3.17	1.46	2.03	2.82	1.30
	90 %	5.48	4.89	5.48	2.39	2.94	4.54	2.00
	95 %	7.88	8.86	7.72	3.60	3.63	6.93	3.15
	97.50 %	16.28	18.34	15.14	4.64	5.05	11.13	4.35
	99 %	26.97	18.34	26.97	10.42	5.13	12.21	6.21

Table 14.1.2 Summary Statistics by Mineralisation Domain and Weathering Domain - Sediments

Statistic		Southern Sediment				
		Total	Laterite	Oxide	Transitional	Fresh
Samples		5,087	101	987	454	3,545
Minimum		0	0	0	0	0
Maximum		286	3.75	60.48	137.7	286
Mean		1.29	0.8	1.34	1.46	1.28
Std Dev.		5.93	0.84	3.75	5.85	6.37
CV		4.59	1.04	2.81	4	4.97
Variance		35.14	0.7	14.08	34.21	40.58
Skewness		31.59	1.96	9.13	18.71	32.12
Percentile	10 %	0.02	0.16	0.03	0.03	0.02
	20 %	0.07	0.21	0.07	0.08	0.07
	30 %	0.16	0.28	0.13	0.24	0.16
	40 %	0.28	0.43	0.26	0.4	0.27
	50 %	0.43	0.5	0.43	0.55	0.42
	60 %	0.64	0.65	0.67	0.74	0.62
	70 %	0.95	0.88	1	1.02	0.93
	80 %	1.48	1.08	1.57	1.59	1.46
	90 %	2.59	2.02	2.9	3.05	2.49
	95 %	4.06	2.93	4.81	4.74	3.96
	97.50 %	6.32	3.18	7.72	8.16	6.14
	99 %	12.34	3.72	15.79	9.18	12.34

Table 14.1.2 Summary Statistics by Mineralisation Domain and Weathering Domain – Sediments (continued)

Statistic		Northern Sediment				
		Total	Laterite	Oxide	Transitional	Fresh
Samples		4,091	209	1,796	629	1,457
Minimum		0	0.02	0	0	0
Maximum		1,420.55	23.22	1,209.20	1,420.55	487.31
Mean		1.7	0.88	1.33	2.34	1.88
Std Dev.		21.36	1.91	18.83	35.1	18.98
CV		12.55	2.16	14.13	15	10.11
Variance		456.1	3.64	354.4	1,232.00	360.3
Skewness		44.73	6.94	60.36	37.82	24.98
Percentile	10 %	0.04	0.07	0.04	0.03	0.05
	20 %	0.13	0.15	0.11	0.09	0.14
	30 %	0.23	0.24	0.21	0.19	0.24
	40 %	0.33	0.32	0.31	0.3	0.34
	50 %	0.46	0.38	0.46	0.43	0.51
	60 %	0.63	0.43	0.63	0.53	0.68
	70 %	0.84	0.64	0.82	0.72	0.9
	80 %	1.22	1.14	1.22	1.04	1.28
	90 %	2.19	1.98	2.03	2.3	2.28
	95 %	3.3	2.4	3	4.38	3.58
	97.50 %	5.94	3.78	4.01	5.87	6.72
	99 %	11.64	9.69	8.82	16.05	13.62

Table 14.1.3 Summary Statistics by Mineralisation Domain and Weathering Domain - Dykes

Statistic		Dykes				
		Total	Laterite	Oxide	Transitional	Fresh
Samples		7,288	459	3,591	1,009	2,229
Minimum		0.00	0.00	0.00	0.00	0.00
Maximum		593.80	105.03	437.60	99.72	593.80
Mean		1.27	1.15	1.18	1.17	1.48
Std Dev.		9.93	6.29	9.48	4.09	12.56
CV		7.79	5.46	8.02	3.50	8.48
Variance		98.56	39.51	89.90	16.69	157.70
Skewness		43.08	13.54	39.76	18.62	40.10
Percentile	10 %	0.04	0.03	0.03	0.05	0.05
	20 %	0.13	0.08	0.11	0.16	0.16
	30 %	0.23	0.17	0.22	0.25	0.25
	40 %	0.33	0.23	0.32	0.35	0.36
	50 %	0.44	0.33	0.43	0.46	0.47
	60 %	0.59	0.44	0.58	0.61	0.64
	70 %	0.83	0.60	0.81	0.90	0.87
	80 %	1.21	0.86	1.19	1.32	1.29
	90 %	2.15	1.57	2.07	2.26	2.33
	95 %	3.39	2.42	3.07	3.68	3.88
	97.50 %	5.25	3.72	4.70	5.38	6.20
	99 %	10.10	10.19	9.36	10.86	11.96
Top cut		30	30	30	30	30
Top cut count		17	2	5	2	8
Top cut mean		1.02	0.89	0.96	1.08	1.12
Top cut Std Dev.		2.38	2.84	2.22	2.28	2.54
Top cut CV		2.33	3.21	2.32	2.12	2.27
Top cut variance		5.68	8.08	4.94	5.22	6.45

14.2 Variogram Analysis

Indicator variograms were generated and modelled for the southern and northern sediments using the combined oxide, transitional and fresh data. The southern and northern granites contained insufficient data for full indicator variography and as such the variogram models for the sediments were also used for the granite estimates.

Three extreme outliers were removed from the datasets prior to variography as they appear to be part of a very small, extremely high grade population which is not representative of the main population. These values are 1,420.55 g/t Au and 1,209.2 g/t Au in the northern sediments (next highest grade in domain is 487) and 1,336.00 g/t Au in the southern granite (next highest grade in domain is 88).

Twelve indicator thresholds were defined based on a combination of ensuring an even spread of sample numbers and metal within each bin. Linear models between each bin were reviewed on the cumulative probability plots to ensure that the linear models honoured the statistical distribution within each bin.

Thresholds were defined as gold cut-off grades and the same cut-off grades were used for all domains, as there appears to be a consistent change in population at certain grades in all domains. These changes in population occur at around 0.3 g/t Au and 10.0 g/t Au.

Figure 14.2.1 and Figure 14.2.2 show the log-probability plots with the indicator thresholds highlighted and the cumulative distribution plots showing the linear model fit between the indicator thresholds, for the southern and northern sediments respectively.

Figure 14.2.1 Log-probability Plot Showing Indicator Thresholds and Cumulative Distribution Plot Showing Linear Model Fit Between Indicator Thresholds - Southern Sediments (after Snowden, 2014)

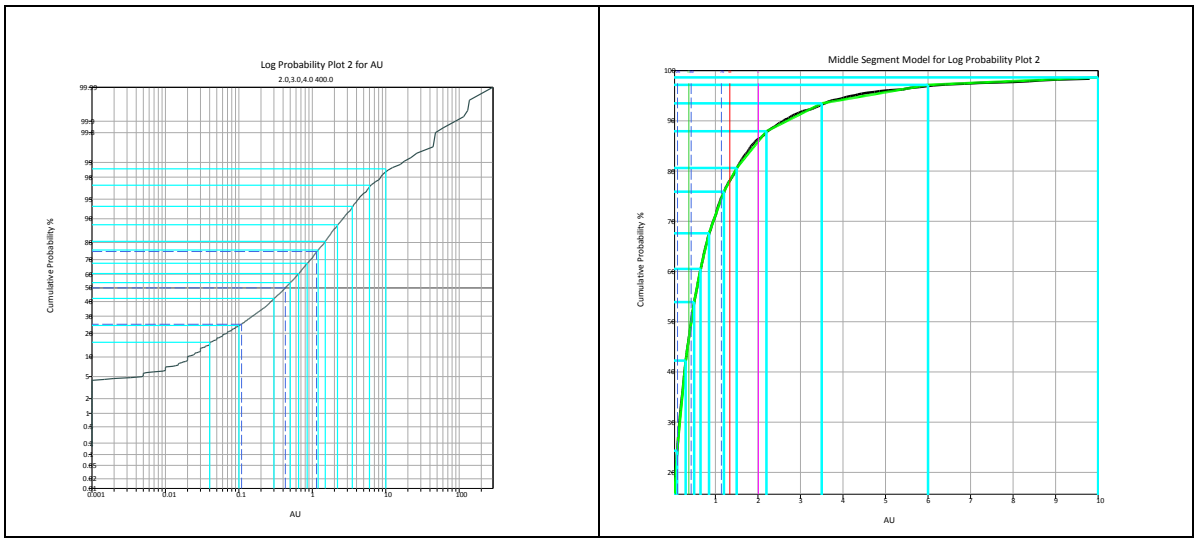
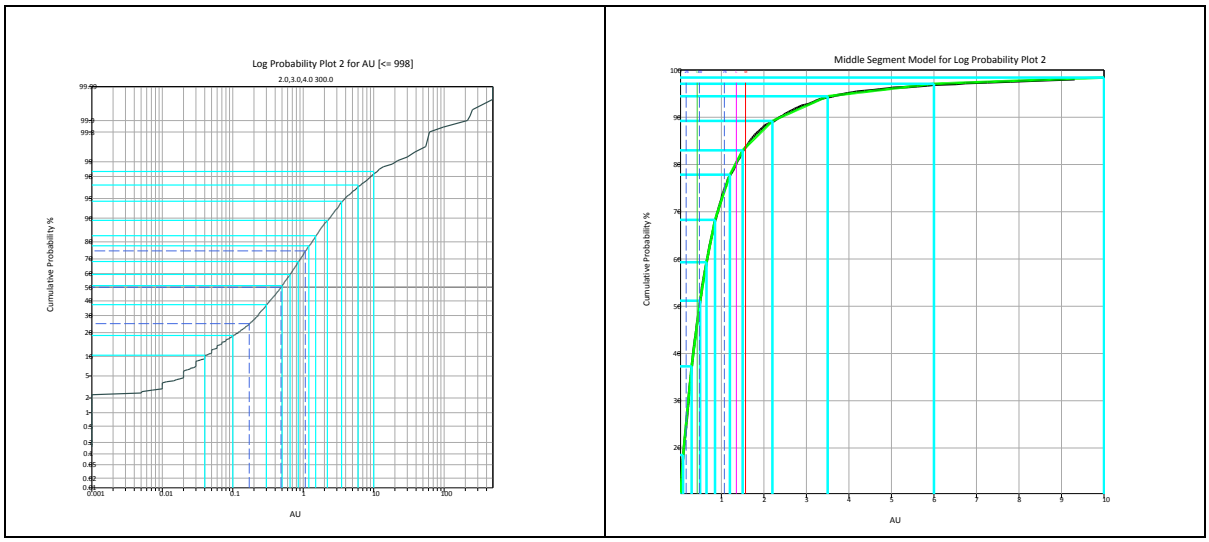


Figure 14.2.2 Log-Probability Plot Showing Indicator Thresholds and Cumulative Distribution Plot Showing Linear Model Fit Between Indicator Thresholds - Northern Sediments (after Snowden, 2014)



Variogram fans were assessed to determine the dominant orientation of the mineralisation at each threshold. Results of the analysis for both southern and northern sediments indicate that the lower grade mineralisation, below 0.3 g/t Au, trends to north-northwest, dipping sub vertically to the west, in line with the main geological trend. At the higher grade above 0.3 g/t Au, the mineralisation is plunging to the east-southeast, in line with expectation based on the orientation of high grade vein sets in these areas.

A downhole variogram was modelled for each threshold to determine the nugget, followed by directional variography. The variograms for the upper two or three thresholds were poorly structured and the patterns seen in the previous thresholds were used to predict the variogram models at these higher grades.

Variogram models for each threshold are detailed in Table 14.2.1 and Table 14.2.2. Figure 14.2.3 and Figure 14.2.4 illustrate the grade continuity ellipse at low grade thresholds (0.1 g/t Au) and higher grade thresholds (0.5 g/t Au) respectively.

Table 14.2.1 Indicator Variogram Models for Gold – Southern Sediments

Domain	Percentile	Cut-off	Orientation	Nugget	Structure 1		Structure 2	
					Sill	Range	Sill	Range
Southern Sediments	15.65	0.04	00→350	0.49	0.24	25	0.27	145
			-70→260			25		145
			20→260			10		70
	24.30	0.10	00→350	0.49	0.28	10	0.23	70
			-70→260			10		70
			20→260			10		40
	42.25	0.30	00→350	0.55	0.32	10	0.13	60
			-70→260			10		60
			20→260			6		30
	53.90	0.50	-42→119	0.55	0.32	7	0.13	30
			34→173			7		30
			-30→240			5		15
	60.50	0.65	-42→119	0.55	0.31	6	0.14	25
			34→173			6		25
			-30→240			5		10
	67.60	0.85	-42→119	0.56	0.28	5	0.16	20
			34→173			5		20
			-30→240			5		10
	75.88	1.20	-42→119	0.59	0.20	5	0.21	20
			34→173			5		20
			-30→240			5		10
	80.60	1.50	-42→119	0.66	0.13	5	0.21	20
			34→173			5		20
			-30→240			3		5
	87.92	2.20	-42→119	0.67	0.12	5	0.21	20
			34→173			5		20
			-30→240			3		5
	93.47	3.50	-42→119	0.67	0.12	5	0.21	15
			34→173			5		15
			-30→240			3		5
	97.15	6.00	-42→119	0.67	0.12	4	0.21	10
			34→173			4		10
			-30→240			3		5
	98.64	10.00	-42→119	0.67	0.12	3	0.21	5
			34→173			3		5
			-30→240			3		5

Table 14.2.2 Indicator Variogram Models for Gold – Northern Sediments

Domain	Percentile	Cut-off	Orientation	Nugget	Structure 1		Structure 2	
					Sill	Range	Sill	Range
Northern Sediments	10.35	0.04	00→350	0.54	0.19	30	0.27	100
			-70→260			30		100
			20→260			10		70
	18.52	0.10	00→350	0.54	0.19	20	0.27	80
			-70→260			20		80
			20→260			6		60
	37.28	0.30	00→350	0.59	0.21	20	0.20	80
			-70→260			20		80
			20→260			6		60
	51.17	0.50	-42→119	0.59	0.21	15	0.20	60
			34→173			15		60
			-30→240			6		30
	59.32	0.65	-42→119	0.61	0.21	10	0.18	40
			34→173			15		40
			-30→240			6		25
	68.29	0.85	-42→119	0.61	0.21	10	0.18	20
			34→173			10		20
			-30→240			6		20
	77.85	1.20	-42→119	0.58	0.26	5	0.16	10
			34→173			5		10
			-30→240			5		10
	83.00	1.50	-42→119	0.58	0.26	5	0.16	10
			34→173			5		10
			-30→240			5		10
	89.23	2.20	-42→119	0.57	0.27	4	0.16	8
			34→173			4		8
			-30→240			4		8
	94.45	3.50	-42→119	0.57	0.27	3	0.16	5
			34→173			3		5
			-30→240			3		5
	97.09	6.00	-42→119	0.57	0.27	3	0.16	5
			34→173			3		5
			-30→240			3		5
	98.40	10.00	-42→119	0.57	0.27	3	0.16	5
			34→173			3		5
			-30→240			3		5

Figure 14.2.3 Orthogonal View Showing Grade Continuity Ellipse at 0.1 g/t Au (image supplied by Snowden)

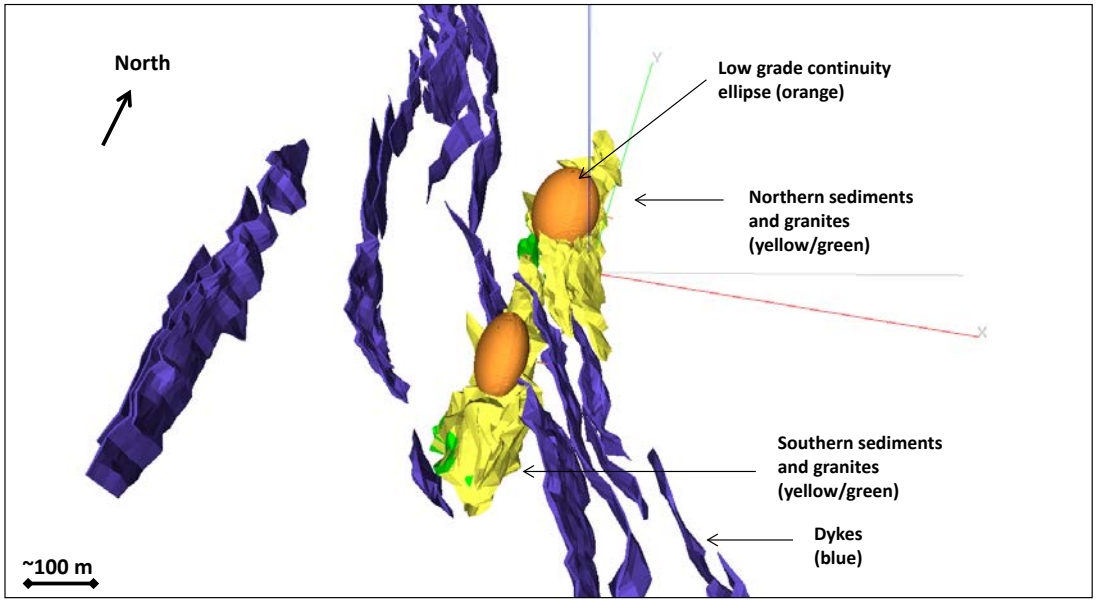
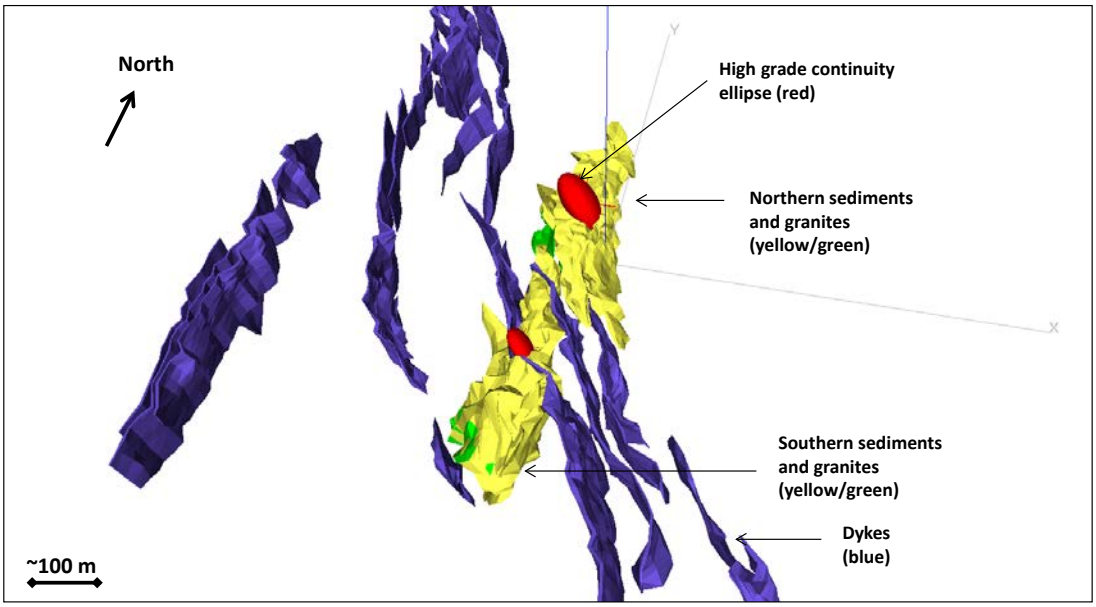


Figure 14.2.4 Orthogonal View Showing Grade Continuity Ellipse at 0.5 G/T Au (Image Supplied By Snowden)



For the dykes, variogram analysis was carried out using the oxide, transitional and fresh data combined.

Variogram fans were assessed to determine the dominant orientation of the mineralisation. Results of the analysis indicate that the mineralisation not surprisingly follows the main north-northwest trend of the dykes, dipping sub vertically to the east. The orientation of the dykes

changes along the strike length of the deposit. As a result of this Snowden elected to generate and model variograms in the dominant orientation. The orientation of the model was then adjusted in different parts of the deposit using the Dynamic Anisotropy method in Datamine to match the local trends. Dynamic anisotropy involves estimating a local dip and dip direction into each block and using these values to locally orientate the variogram and search ellipses.

Due to the positively skewed nature of the grade distributions, normal scores variograms were modelled. A downhole variogram was modelled to determine the nugget, followed by directional variography. All normal scores variogram models were back-transformed prior to estimation. Normal scores and back-transformed variogram models are tabulated in Table 14.2.3 and Table 14.2.4, respectively.

Table 14.2.3 Normal Scores Variogram Models for Dykes

Domain	Orientation	Nugget	Structure 1		Structure 2	
			Sill	Range	Sill	Range
Dykes	00→350	0.42	0.35	35	0.23	92
	-80→080			8		17
	-10→260			5		6

Table 14.2.4 Back-transformed Variogram Models for Dykes

Domain	Orientation	Nugget	Structure 1		Structure 2	
			Sill	Range	Sill	Range
Dykes	00→350	0.85	0.13	35	0.02	92
	-80→080			8		17
	-10→260			5		6

14.3 Block Modelling

Snowden carried out a kriging neighbourhood analysis (KNA) on several test areas within the granite and sediments to determine the optimal parent block size and number of informing samples for estimation. Test estimates were run in Snowden's Supervisor software using the 0.5 g/t Au threshold indicator variograms. Kriging efficiency, slope of regression, and number and sum of negative weights were calculated and reviewed.

Block sizes were varied from 5 mE by 5 mN by 2.5 mRL up to 20 mE by 20 mN by 10 mRL. Search parameters for the block size tests were based on the orientation and ranges of the variograms.

Based on this analysis, a parent block size of 10 mE by 10 mN by 5 mRL was selected as appropriate. In the southern sediments, a smaller block size gives a marginally better result, while in the northern sediments a larger block size gives a marginally better result. The selected block size appears to give reasonable results in both areas.

Using the 10 mE by 10 mN by 5 mRL parent cell, a second test was run to look at the impact of changing the number of informing samples. Based on this analysis, a minimum of 8 and a maximum of 24 samples were selected for estimation.

Table 14.3.1 summarises the prototype used for the block model. The block model was coded by mineralisation domain, weathering domain and area.

A waste model was created outside of the mineralisation area to provide sufficient area around the mineralisation for pit optimisation. Sub-celling was used to more accurately reflect the wireframe volumes.

Table 14.3.1 Block Model Parameters

	Easting (X)	Northing (Y)	Elevation (Z)
Origin	805847	1152819	0
Limit	807547	1155849	400
Parent block size (m)	10	10	5
Number of blocks	170	303	80
Minimum sub-cell size (m)	2.5	2.5	1.25

14.4 Grade Interpolation and Boundary Conditions

Due to the highly skewed nature and mixed populations evident in the granites and sediments, multiple indicator kriging (MIK) was used to estimate gold grades. Twelve indicator thresholds were selected as discussed above. Datamine software was used to estimate the probability of grade being above or below each threshold into parent blocks.

As discussed above, the laterite, oxide and transitional data were used to estimate the laterite and oxide, while the transitional and fresh data were used to estimate the transitional and fresh domains. Hard boundaries were used between the granite and sediments and the southern and northern areas.

The initial search pass used ranges equivalent to the ranges of continuity seen in the variograms at around 90 % to 100 % of the variance in the 0.3 g/t Au to 0.5 g/t Au threshold variograms, with the search ellipse orientated as per the higher grade thresholds (0.5 g/t Au and above) as detailed in Table 14.4.1.

Blocks were estimated using a minimum and maximum number of samples of 8 and 24 respectively based on the KNA work. A maximum of 6 composites was allowed per drill hole for estimation. This restriction was set to prevent over smoothing in the downhole direction and help retain the grade trends seen in the drilling data.

If the initial search failed to find the minimum number of samples required, then a second search was conducted using double the search radii. Where required a third search using eight times the search radii was used with the minimum number of samples reduced to 2 and the maximum reduced to 8.

Table 14.4.1 Search Parameters – Multiple Indicator Kriging

Domain	Search Orientation	Search Dimensions (m)		
		Pass 1	Pass 2	Pass 3
Dykes	-42→119	60	120	480
	34→173	60	120	480
	-30→240	30	60	240

Post processing of the MIK probability estimates was carried out in GSLIB software. Post processing was used to carry out order relation corrections using an averaging approach, and to calculate the e-type grade for each block. The e-type grades were calculated using a linear model between indicator thresholds and a hyperbolic or power model for the upper and lower tails (above the top threshold and below the bottom threshold). A minimum of 0.001 was used and a maximum based on the distribution of the input data used to estimate each domain. Table 14.4.2 summarises the lower and upper tail models for each domain.

Table 14.4.2 Lower and Upper Tail Modelling

Domain	Lower Tail			Upper Tail		
	Minimum	Model Type	Parameter	Maximum	Model Type	Parameter
Southern granites Laterite, oxide	0.001	power	0.60	51.78	power	0.49
Southern granites Transitional, fresh	0.001	power	0.60	88.63	hyperbolic	1.70
Northern granites Laterite, oxide	0.001	power	0.45	66.00	power	0.24
Northern granites Transitional, fresh	0.001	power	0.45	66.00	power	0.21
Southern sediments Laterite, oxide	0.001	power	1.07	137.70	hyperbolic	1.36
Southern sediments Transitional, fresh	0.001	power	0.60	286.00	hyperbolic	1.14
Northern sediments Laterite, oxide	0.001	power	1.13	257.00	hyperbolic	1
Northern sediments Transitional, fresh	0.001	power	0.65	487.40	hyperbolic	1

The three extreme grades which were removed for variography, were included in the probability estimates but were removed for the upper tail modelling. A sensitivity model was run, changing the maximum value in the two affected domains (northern sediments and southern granite) to 500 g/t Au. There was no significant change (0.01 g/t Au change) in the average grade and as a result the original estimate was accepted.

Ordinary kriging into parent cells was used to estimate gold grades for the dykes using a discretisation of 5 east by 5 north by 3 elevation. Again, the laterite, oxide and transitional data were used to estimate the laterite and oxide, while the transitional and fresh data were used to estimate the transitional and fresh domains. Hard boundaries were used between the dykes.

A dynamic anisotropy approach was used, whereby the true dip and azimuth of the mineralised lodes was estimated into each block in the model and the search and variogram orientations were locally adjusted to reflect the geological orientation. This method allows the estimate to better reflect the changing orientation and undulating nature of some of these dykes along strike.

The initial search pass used ranges equivalent to the ranges of continuity seen in the variograms at around 90 % to 100 % of the variance as detailed in Table 14.4.3.

Blocks were estimated using a minimum and maximum number of samples of 8 and 24 respectively based on the KNA work. A maximum of 6 composites was allowed per drill hole for estimation. This restriction was set to prevent over smoothing in the downhole direction and help retain the grade trends seen in the drilling data.

If the initial search failed to find the minimum number of samples required, then a second search was conducted using double the search radii. Where required a third search using eight times the search radii was used with the minimum number of samples reduced to 2 and the maximum reduced to 8.

Table 14.4.3 Search Parameters – Ordinary Kriging

Domain	Search Orientation	Search Dimensions (m)		
		Pass 1	Pass 2	Pass 3
Dykes	Strike	40	80	240
	Dip	40	80	240
	Cross-strike	20	40	120

14.5 Density

A total of 770 bulk density measurements were taken from HQ and NQ drill core: 380 from oxide material, 132 from transitional material and 258 from fresh material. Sample length was nominally 10 cm. Bulk densities were determined by the water immersion method using a scale with a precision of ±0.02 g. Oxide and transitional material was wrapped in foil to avoid intake of water and sample disintegration. The wrapper was afterwards weighted independently and deducted from the measured total weight.

Calculated means for the transition and fresh weathering domains as well as the lateritic horizon within the oxide domain (i.e. in the top 8 to 10 vertical metres) were assigned to the block model.

The oxide domain below the lateritic horizon, however, shows a gradient to higher densities with increasing depth (Figure 14.5.1) and it is considered that a single value for the oxide domain would not be adequate. Instead, the bulk density for oxide blocks has been estimated using vertically orientated, inverse distance squared interpolation.

Of the 770 bulk density measurements, 625 were taken from sediments as the predominant domain, 39 from granite, 28 from porphyritic dykes and 78 from the lateritic horizon. Differences between the respective means exist across the various lithologies. However, as the differences across domains are smaller than the differences within individual domains (for the respective weathering zone), bulk density assignment has been identical for all lithologies (other than the lateritic horizon).

No variation in bulk density between mineralised and non-mineralised material was assumed. The raw measurements and assigned bulk densities used in the resource estimate are listed in Table 14.5.1 and Table 14.5.2, respectively.

Figure 14.5.1 Bulk Density versus Depth in Oxide

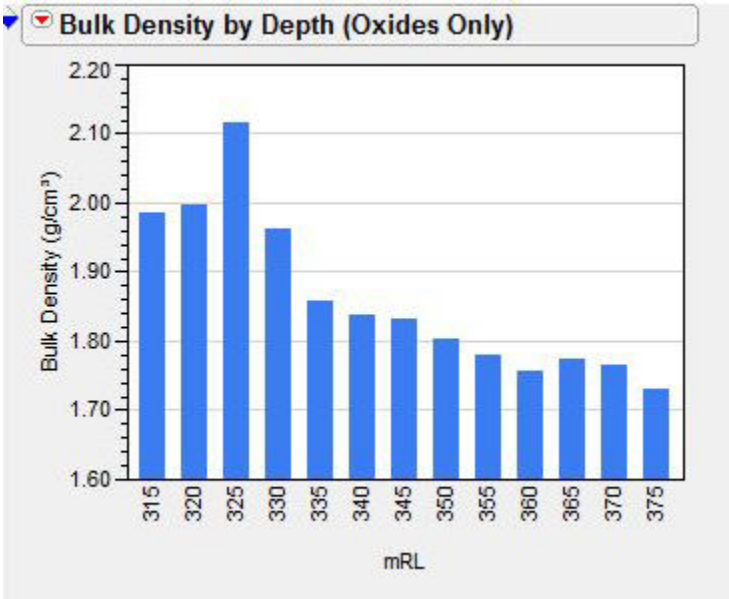


Table 14.5.1 Bulk Density Measurements

Weathering	Bulk Density g/cm3			
	# Samples	Mean	Minimum	Maximum
Completely oxidised	89	1.91	1.49	2.53
Strongly oxidised	293	1.78	1.42	2.26
Transition (moderately oxidised)	74	2.02	1.62	2.57
Transition (weakly oxidised)	56	2.31	1.71	2.75
Fresh, weak oxidation along fractures	42	2.68	2.47	2.81
Fresh	216	2.74	2.36	2.96

Table 14.5.2 Bulk Density Values Assigned to Resource Estimate

Field	Value	Description	Bulk Density g/cm3
weath_n	1	Laterite	1.90
	2	Oxide	1.70-2.20
	3	Transitional	2.30
	4	Sulphide	2.73

14.6 Model Validation

Snowden validated the Mineral Resource model by:

- Global statistical comparison of top cut input grades with tonnage weighted output grades.
- Visual inspection of the model against the input composites.
- Comparison of moving window input and output statistics.
- Global change of support to check level of selectivity in model.

14.6.1 Global Statistical Comparison

The final grade estimates were validated statistically against the input drillhole composites. Table 14.6.1 provides a comparison of the estimated grades compared to the input declustered, top cut grades for the global estimate. This statistical comparison shows that all domains validate reasonably well globally although the southern granite and northern sediments appear to be underestimated as a result of the three extreme grades in these domains. These three grades have been purposefully not honoured in the estimate as they are not indicative of the surrounding mineralisation. To provide a more meaningful comparison in the domains estimated by MIK, a top cut of 500 g/t Au has been applied to these statistics.

Table 14.6.1 Statistical Comparison between Estimated Grades and Input Data

	Southern Granites	Northern Granites	Northern Sediments	Southern Sediments	Dykes
Samples	1,095	690	4,091	5,087	7,288
Top cut data	3.20	1.03	1.58	1.29	1.02
Estimate	2.72	0.96	1.24	1.26	0.92
Difference	-15%	-7%	-22%	-2%	-10%

14.6.2 Visual Validation

The final grade estimates were visually compared to the input drillhole data on section. This comparison shows that the estimate honours the input drillhole data. Example cross sections are illustrated in Figure 14.6.1 and Figure 14.6.2.

Figure 14.6.1 Example Cross Section Showing Estimated Gold Grades versus Input Drillhole Data – 1154240 mN (±10 m) (image supplied by Snowden)

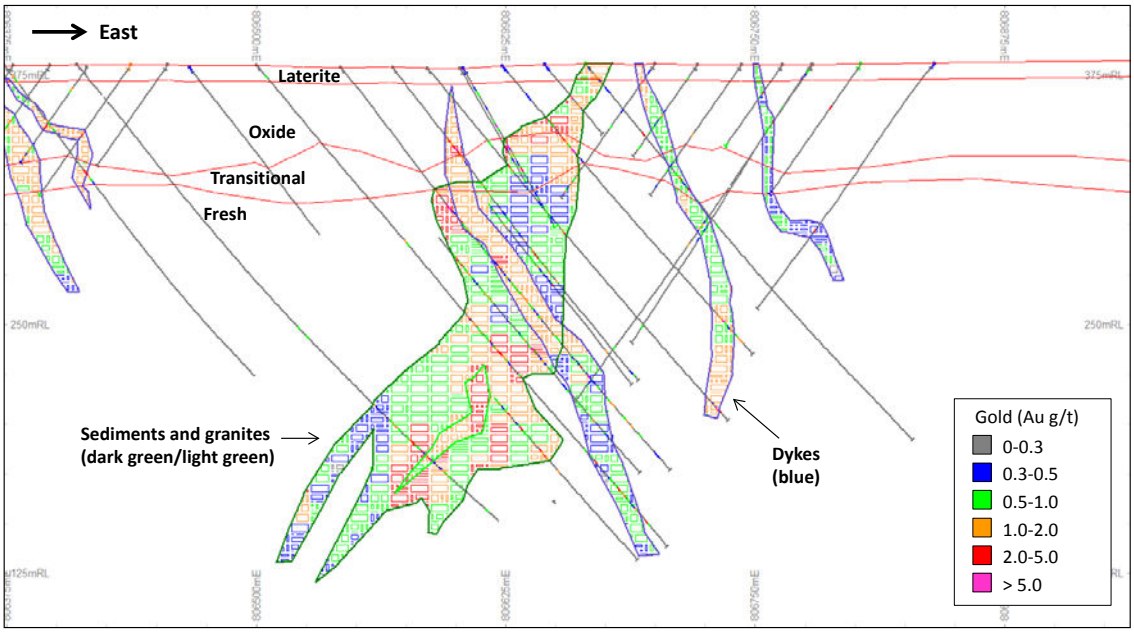
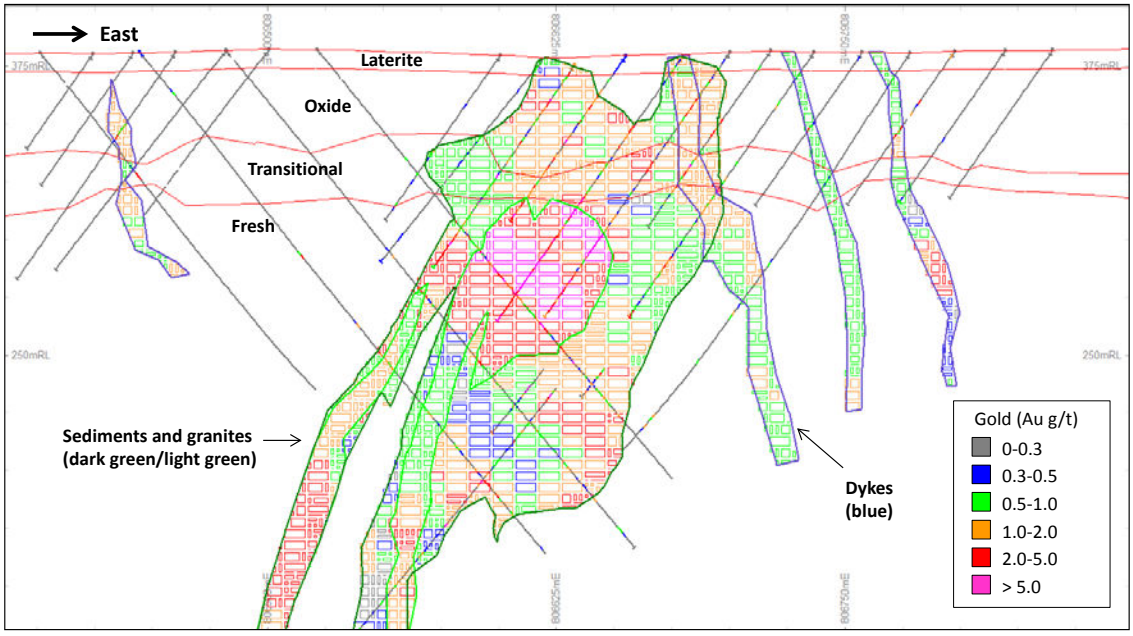


Figure 14.6.2 Example Cross Section Showing Estimated Gold Grades versus Input Drillhole Data – 11542180 mN (±10 m) (image supplied by Snowden)



14.6.3 Moving Window Validation

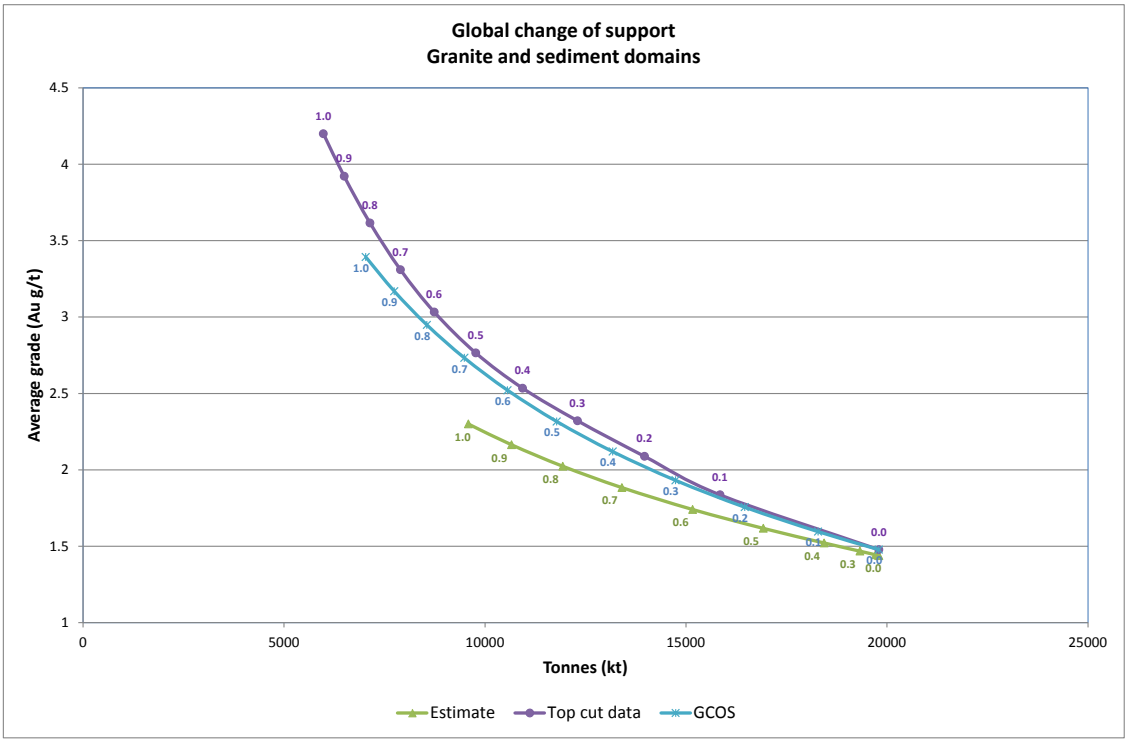
Sectional validation graphs were created for each mineralised domain to assess the reproduction of local means and to validate the grade trends in the model. These graphs compare the mean of the estimated grades to the mean of the input grades (declustered, not cut) within model slices (bins) for gold. Analysis of the resulting graphs indicates that there is good local reproduction of the input grades in both the horizontal and vertical directions.

14.6.4 Global Change of Support

Snowden carried out a global change of support (GCOS) on the declustered input data to assess the selectivity in the MIK estimate. This was carried out on the granite and sediment data only. The three extreme grades were removed from the dataset prior to running the global change of support to provide a more realistic result.

The theoretical grade tonnage curve at the parent block size was compared to the estimated grade tonnage curve (Figure 14.6.3). The results show that the MIK estimate is over smoothed compared to the theoretical result, however Snowden considers the result to be acceptable.

Figure 14.6.3 Global Change of Support



14.7 Mineral Resource Classification

Snowden reviewed the classification applied to the previous estimate (Widenbar, 2013) and concluded that the classification is appropriate with respect to confidence in tonnes and grade estimates; however recent information on recovery and mining factors has resulted in a trial optimisation being run to determine the potential for economic extraction, in particular for the lower grade and peripheral mineralisation.

The Sissingue Mineral Resource has been classified in the Measured, Indicated and Inferred categories, in accordance with the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC Code – 2012 edition). A range of criteria has been considered in determining this classification including:

- Geological continuity.
- Data quality.
- Drillhole spacing.
- Estimation properties including kriging neighbourhood analysis to determine appropriate block size and search strategy.
- Potential for economic extraction.

14.7.1 Geological Continuity

Geological and grade continuity is understood with reasonable confidence. The classification reflects this level of confidence. Granite and sediment domains were able to be assigned to Measured, Indicated or Inferred categories according to other parameters such as data distribution. Porphyry lithologies (dyke domains) were limited to Inferred and Indicated categories due to the mostly narrow nature of the mineralised bodies and consequently their sensitive geometry.

14.7.2 Data Quality

Resource classification is based on information and data provided from the Perseus database. Descriptions of drilling techniques, survey, sampling/sample preparation, analytical techniques and database management/validation provided indicate that data collection and management is well within industry standards. The database represents an accurate record of the drilling undertaken at the project.

14.7.3 Drilling Spacing

Drillhole location plots have been used to ensure that local drill spacing conforms to the minimum expected for the resource classification.

The Measured category requires a drill spacing of 20 m by 20 m or closer.

The Indicated category is confined to areas where drill spacing is greater than 20 m by 20 m, but nominally up to 20 (east-west) by 40 m (north-south) spacing. In the dyke domains, in additional to drill spacing constraints, a minimum of two drillhole intercepts per drill section is required.

The Inferred category is assigned to all other estimated blocks within the potentially economic areas of the deposit.

14.7.4 Estimation

The resource model was generated using an ordinary kriging and MIK interpolation method with a minimum number of 8 samples and a maximum number of 24 samples used based on a kriging neighbourhood analysis. The number of samples used and the kriging variance were all stored in the block model. Snowden considers the estimation technique and parameters appropriate for this style of mineralisation.

14.7.5 Potential for Economic Extraction

A Whittle optimisation was carried out to determine the limits of potentially economically mineable material for the purposes of classifying the open pit Mineral Resource estimate.

It is usual in this case to use an elevated metal price that could be achieved in a 10 to 20 year timeframe. In this case the optimisation was based on a USD2400/oz gold price (the spot price at the time was USD1200/oz).

The mining inventory derived from the optimisation was based on the preliminary coding of the Measured, Indicated and Inferred Mineral Resource classification.

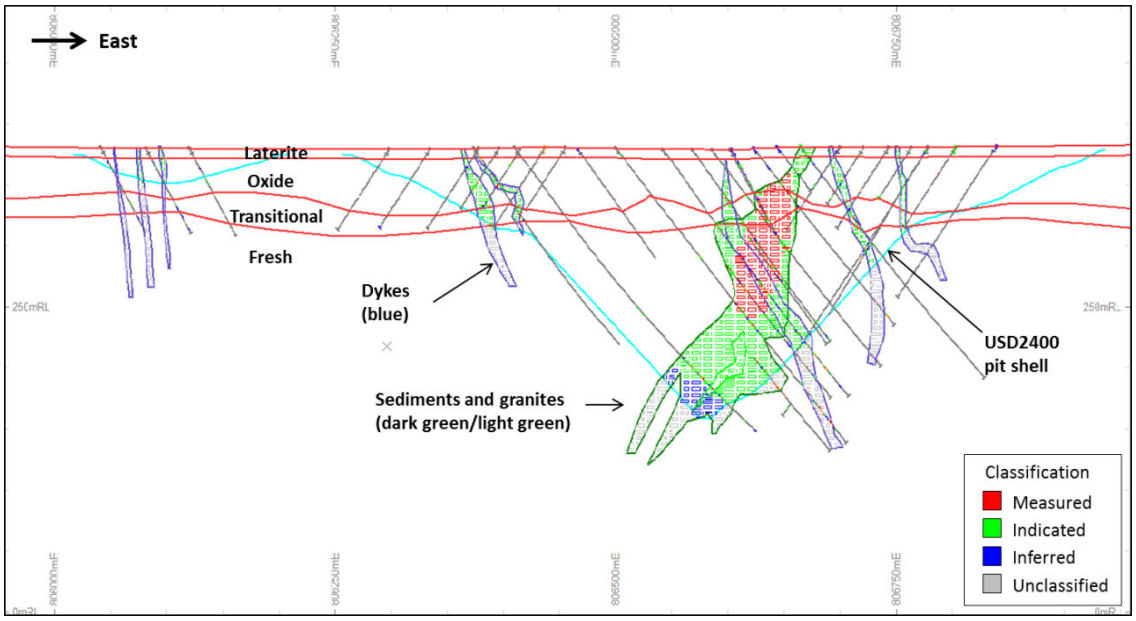
14.7.6 Final Classification

The above parameters were used in combination to arrive at an initial resource classification. Two wireframes were created: one for the Indicated category, and the other for the Measured category. The appropriate category was then assigned to the block model cells. All other cells were initially set to Inferred.

Subsequent to this the USD2400 pit shell was used to constrain the base of potentially economic material. All blocks outside this shell are unclassified.

A typical cross-section view is provided in Figure 14.7.1. Volumetrically, approximately 28 % of the Sissingue Mineral Resource has been classified as Measured, 65 % as Indicated and 7 % as Inferred.

Figure 14.7.1 Cross Section View of Resource Classification (±10 m) (image supplied by Snowden)



14.8 Mineral Resource Reporting

Table 14.8.1 summarises the resources by category and area reported above a 0.6 g/t Au cut-off grade. The classification categories of Inferred, Indicated and Measured under the JORC Code (2012). They are equivalent to the CIM categories of the same name (CIM, 2010).

The reporting cut-off is based on optimisation studies carried out as part of the 2010 Feasibility Study, which have suggested that the deposit can be economically extracted at a gold cut-off in the range 0.4 g/t to 0.6 g/t. Trial optimisation has been run at a USD2400 gold price to define the base of potentially economic material.

Metallurgical studies have confirmed that the Sissingué ore is amenable to processing by conventional CIL methods.

Table 14.8.1 October 2014 Sissingué Mineral Resource Reported at a 0.6 g/t Au Cut-off

Category	Area	Tonnage (kt)	Grade (g/t Au)	Contained Gold (oz)
Measured	Oxide	1,000	1.8	59,000
	Transitional	650	2.3	49,000
	Fresh	3,200	2.5	260,000
Total Measured		**4,800**	**2.4**	**370,000**
Indicated	Oxide	3,100	1.3	130,000
	Transitional	800	1.5	38,000
	Fresh	7,100	1.5	350,000
Total Indicated		**11,000**	**1.4**	**510,000**
Measured + Indicated	Oxide	4,100	1.4	190,000
	Transitional	1,400	1.9	87,000
	Fresh	10,000	1.8	600,000
Total Measured + Indicated		**16,000**	**1.7**	**880,000**
Inferred	Oxide	310	1.2	12,000
	Transitional	54	1.2	2,100
	Fresh	760	2.0	49,000
Total Inferred		**1,100**	**1.7**	**63,000**

Notes: Mineral Resources are inclusive of Mineral Reserves. Mineral Resources are reported to two significant figures. Rounding may cause minor discrepancies in the table. Oxide includes small portions of laterite (571 kt total).

14.9 Comparison to Previous Model

Snowden compared the updated Mineral Resource with the previously reported Mineral Resource (Widenbar, 2013) (Table 14.9.1). The comparison shows that the updated estimate is more selective, resulting in higher grades in the central, Measured portion of the estimate. This has resulted in an additional 60 koz in the Measured category.

The increased selectivity in the updated estimate is a result of using MIK estimation to honour the highly skewed, mixed distribution in the granite and sediment domains. The previous estimate used top cut ordinary kriging in these domains, which tends to over smooth in these styles of strongly skewed grade distributions.

There is a decrease in Indicated and Inferred Resources as a result of the application of the pit shell to define the base of potentially mineable material.

Table 14.9.1 Widenbar 2013 Sissingué Mineral Resource Reported at 0.6 g/t Au Cut-off

Category	Area	Tonnage (kt)	Grade (g/t Au)	Contained Gold (oz)
Measured	Oxide	925	1.6	48,000
	Transitional	600	2.0	39,000
	Fresh	2,700	2.5	217,000
Total Measured		**4,225**	**2.2**	**305,000**
Indicated	Oxide	4,600	1.2	171,000
	Transitional	1,300	1.3	56,000
	Fresh	8,850	1.4	394,000
Total Indicated		**14,750**	**1.3**	**620,000**
Measured + Indicated	Oxide	5,520	1.2	218,000
	Transitional	1,900	1.6	95,000
	Fresh	11,560	1.6	612,000
Total Measured + Indicated		**18,980**	**1.5**	**925,000**
Inferred	Oxide	950	1.0	31,000
	Transitional	650	1.0	21,000
	Fresh	5,400	1.4	239,000
Total Inferred		**7,000**	**1.3**	**291,000**

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15.0 MINERAL RESERVE ESTIMATE

This section summarises the main considerations in relation to preparation of Mineral Reserves and provides references to the sections of the study where more detailed discussions of particular aspects are covered.

The Mineral Reserves were compiled with reference to the NI 43-101 and JORC (2012). The qualified person responsible for the reporting of open pit Mineral Reserves is Mr. Joe McDiarmid, who is a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy, and is a full-time employee of RungePincockMinarco Ltd. Mr Joe McDiarmid has sufficient experience, which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he has undertaken to be a Qualified Person, as defined by NI 43-101.

15.1 Mineral Reserve Statement

Mineral Reserves were estimated by RungePincockMinarco Ltd (RPM). As shown in Table 15.1.1, a total of 5.5 Mt of Open Cut Mineral Reserves at 2.4 g/t gold were estimated as at 1 February 2015.

Table 15.1.1 February 2015 Open Pit Mineral Reserve

Classification	Tonnes dry Mt	Au Grade g/t	Gold koz
Proved	3.4	2.8	312
Probable	2.1	1.7	115
Proved + Probable	5.5	2.4	429

The estimates have been rounded to two significant figures to reflect accuracy.

15.2 Mineral Reserve Cut-off Grades

The gold cut-offs used for reserving was provided by Perseus based on the latest mineral processing studies. These cut-offs, which vary by material type, are shown in Table 15.2.1.

Table 15.2.1 Ore Cut-off Grade

Ore Type	Units	Au Cut-off
Oxide	g/t	0.6
Transitional	g/t	0.8
Granite/Porphyry	g/t	0.8
Sediment	g/t	1.0

15.3 Mineral Reserve Dilution and Ore Loss

The smallest mining unit (SMU) defines the smallest size of material that can be selectively mined using a style and size of excavator. Based on the excavators selected for this project a SMU of 2.5 m east x 5.0 m west x 5.0 m high was selected. Modelling of the SMU has been done by re-blocking the geological model so that no block size is below the SMU. The resulting SMU model has ore loss and dilution built in. For this project the SMU model was created using RPM's BLOCK AGG software.

Once the SMU blocks have been created they are classified as 'ore' or 'waste rock' based on the required grade confidence class and ore cut-off grade. For this project, only material classified as Measured or Indicated Resources could potentially be classified as 'ore'.

15.4 Mineral Reserve Pit Determination and Modifying Factors

The economic pit limits were determined by using the Whittle 4X pit limit optimisation software ("Whittle 4X Optimiser"). It is termed 'optimisation' as the pit shell generated defines the theoretical optimal economic mining limits for the given input parameters. It does not 'optimise' the Project development as practical elements such as blending strategy; water management, etc. are excluded.

The general approach to identifying the final economic pit limits is:

- Identify the physical constraints to mining.
- Define the Project economic input parameters, for example, mining costs and ore price.
- Estimate mining modifying factors, for example, ore loss and dilution (or use model re-blocked to smallest mining unit to account for loss and dilution.
- Estimate metallurgical modifying factors, for example, process plant ore recovery.
- Define the pit slope design requirements (overall slope including ramps, catch berms etc.).
- Import all above parameters, plus including geological model into the pit limit optimisation software.
- Run pit limit optimisation software to produce a series of nested pit shells at increasing product selling prices.
- If applicable, for deposit type, define minimum mining width.
- Analyse results.
- Select a preferred pit shell in conjunction with Perseus.

15.4.1 Physical Mining Constraints

Physical constraints are typically surface features which limit the allowable extent of mining. Examples include critical infrastructure, mining titles, property ownership and environmentally sensitive areas.

After consultation with Perseus and reviewing the available technical reports, no physical constraints to the pit limits were identified and applied.

15.4.2 Geological Block Model and Topography

The Surpac format block model supplied by Perseus ("ssest1306a.mdl") was prepared by Snowden Mining Consultants ('Snowden'). Snowden built the original model in Datamine format and subsequently converted it to Surpac format at the request of Perseus. The block model was constrained by the topography (Topo1.dtm) and exported to Whittle 4X model format.

15.5 Pit Optimisation Input Parameters

15.5.1 Geotechnical Parameters

A geotechnical study was completed by Coffey Mining Ltd in October 2010. The pit optimisation and design work Base Case pit slope parameters are presented in Table 15.5.1.

Table 15.5.1 Geotechnical Slope Angles

ID	Domain	Pit Sector		Slope Dip Direction		IRSA		Batter Angle	Batter Height	Berm Width
				From (deg)	To (deg)	Angle (deg)	Height (m)	(deg)	(m)	(m)
S1	Oxide (Saprolite)	All slopes		All slopes		38	50	55	10	6
T1	Transition (rock)	All Slopes		All slopes		47	20	60	20	7
F1	Fresh (rock)	1	NE	220	270	51		65	20	7
		2	SE	270	330	51		65	20	7
		3	N,W,S	0 330	220 360	53		70	20	8
Interim fresh rock pit slopes where best practice blasting is not applied						48.5		65	20	8.5
A double berm width (12m) is recommended at the base of oxidation between the Oxide Domain and the Transition Rock Domain. De-couple the fresh rock slope every 100m (i.e. every 5th bench) with either placement of the haul ramp or a double-berm width (16 m).										

An initial optimisation case was completed using the inter-ramp slope angles to assess the potential size, shape and depth of the pit. Based on this result an estimate was made of the number of ramp segments needed in the various pit walls for mining. The pit slopes were then adjusted to angles that were inclusive of ramps. These ramp-inclusive angles were used in the final series of pit optimisations.

15.5.2 Metallurgical Factors

Metal recovery was estimated based on the values shown in Table 15.5.2. Note that the recoveries for oxide, transitional and fresh granite / porphyry are fixed percentages while the recovery for fresh sediment is based on the diluted ore gold grade.

Table 15.5.2 Processing Recovery

Description	Units	Oxide	Trans	Fresh - GR/PY	Fresh - SD
Gold Recovery	%	92%	91%	90%	78%

All recoveries were sourced from Perseus and are based on the latest processing study.

15.5.3 Operating and Sustaining Capital Costs

Rates for mine operating costs are based on a mining contractor quotation prepared for Perseus and made available to RPM. These unit rates include contractor capital costs and anticipated replacement capital for the term of the contract. In addition to the contractor costs all owner costs relating to the supervision of the contract have also been provided by Perseus and added to the unit operating costs. No Perseus capital costs were included as major Project infrastructure expenditure should be considered a sunk cost. Processing costs were provided by Perseus based on planned overhead costs and the current processing study.

The waste and ore mining costs are summarised in Table 15.5.3. Load and haul costs used in the pit optimisation were on a $/bcm mined basis. An average material density has been applied in this table to show the equivalent costs in $/t mined. The processing and other ore costs are presented in Table 15.5.4.

Table 15.5.3 Pit Optimisation Mining Costs

Description	Units	Oxide	Transition	Fresh
Waste Mining Cost				
Drill & Blast	US$/t	0.00	0.69	1.11
Load & Haul	US$/t	1.58	1.62	1.53
Haulage Cost Per Vertical Metre	US$/t per m	0.007	0.006	0.006
Load & Haul	US$/bcm	3.09	3.73	4.18
Haulage Cost Per Vertical Metre	US$/bcm per m	0.014	0.013	0.017
Other Mining Costs	US$/t	1.98	1.98	1.98
Sub-Total	US$/t	3.55	4.29	4.62
Ore Mining Cost				
Drill & Blast	US$/t	0.00	0.69	1.11
Load & Haul	US$/t	1.58	1.62	1.53
Haulage Cost Per Vertical Metre	US$/t per m	0.007	0.006	0.006
Load & Haul	US$/bcm	3.09	3.73	4.18
Haulage Cost Per Vertical Metre	US$/bcm per m	0.014	0.013	0.017
Other Mining Costs	US$/t	2.22	2.22	2.22
Subtotal	**US$/t**	**3.79**	**4.53**	**4.86**

Table 15.5.4 Processing and Other Ore Costs

Description	Units	Oxide	Trans	Fresh - GR/PY	Fresh - SD
Processing	US$/t ore	12.17	16.77	18.44	19.71
Rehandle	US$/t ore	0.60	0.60	0.60	0.60
Overhead & Ore Costs	US$/t ore	6.40	7.20	7.50	8.10
Total	**US$/t ore**	**19.18**	**24.58**	**26.54**	**28.41**

15.5.4 Metal Selling Price and Royalties

The price received for the final product (refer Table 15.5.5), royalty rates and refining costs (refer Table 15.5.6) were provided by Perseus.

Table 15.5.5 Metal Selling Price

Description	Units	Value
Metal Price - Gold	US$/tr.oz	1,200

Table 15.5.6 Royalty and Refining Cost

Description	Units	Values
Royalty	% Metal Price	4.50%
Bullion refining cost	US$/tr.oz	3.60

15.5.5 Mining and Processing Rate

The ore processing rate is based on the processing plant operating for 8,000 hours per year applied to the hourly throughput rate for each ore type. These hourly throughput rates are as follows:

- Oxide 150 t/h
- Transition 132 t/h
- Fresh - GR/PY 128 t/h
- Fresh – SD 118 t/h

15.5.6 Other Assumptions

No mining or processing cut-off grades were applied within the pit optimiser. Selection of material for processing was made based on the value of the contained metal and associated mining costs.

15.5.7 Shell Selection for Mine Planning

The selection of the preferred pit shell was undertaken in conjunction with Perseus. Key criteria used to select the pit shell included:

- Maximise resource recovery.
- Practical pit size and shape.
- Balance between maximising ore and minimising incremental strip ratio.
- Increased cashflow.

The 100% revenue factor pit shell was selected as the overall preferred result. Smaller satellite pits generated by the optimiser at the periphery of the deposit were reviewed and discarded if they were found to too small for provide for practical mining.

15.6 Mineral Reserves Sensitivity

The economic sensitivity of the project has been reviewed at a high level using pit optimisation software and in detail in Item 22 Economic Analysis. As in any mineral project, the results of the economic analysis represent forward looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Areas of uncertainty that may materially impact mineral reserve estimation include:

- Commodity price and exchange rate assumptions.
- Capital and operating cost estimates.
- Geotechnical slope designs for pit walls.

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16.0 MINING METHODS

16.1 Mining Method Selection

In assessing and selecting a preferred mining method, the deposit characteristics, operating objectives, as well as other salient factors have been considered.

The key considerations in relation to the selection of a preferred mining method are as follows:

- The key deposit characteristics affecting the mining method are:
  - near surface location
  - high value ore
  - ore occurs in zones including high grade zones
  - generally low strength.
- High processing cost warrants mining method which minimises dilution.
- Nominal ore production target of 1 Mt per year.
- A selective mining method will be required to minimise ore loss and dilution and control risks to the processing plant.
- Based on the above requirements and characteristics, the preferred approach is for open cut selective mining. To support this approach is recommended that the method employs hydraulic excavators mining in 5 m benches loading standard off-highway rear dump trucks hauling to surface ore stockpiles and waste disposal dumps. These would be supported by front-end loaders for stockpile rehandle.
- Other features of the mining method include:
  - free dig oxides, except for limited areas to be ripped by dozer
  - separation of various grades into stockpiles to facilitate preferential high grade feed to the process plant
  - a detailed grade control system to ensure material is directed to the correct stockpile
  - initial waste dumping primarily on surface dumps.
- The pit will be accessed by a 20 m wide haul road incorporating a 10 percent gradient and two lane ramps. At the pit floor the final benches will be accessed using single lane ramps.

16.2 Mining Strategy

The mining development strategy was determined by examining the outcomes of the Whittle 4X pit optimisation analysis (Refer Section 15) and considering the project objectives. A three phase mining cutback strategy. The first cutback focussed on near surface higher grade zones to maximise return and capital payback. The second phase then expanded the pit from north to south following the strike of the ore zone and increasing the depth to balance the strip ratio with the final phase.

16.3 Geotechnical Pit Design Properties

The pit design parameters presented in Table 16.3.1 are based on the Coffey Mining geotechnical study discussed in Section 15.4. It reflects the rock mass being categorised into three geotechnical domains based on weathering (i.e. the Oxide (saprolite) Domain, the Transition Rock Domain, and the Fresh Rock Domain).

Table 16.3.1 Pit Design Parameters

ID	Domain	Pit Sector		Slope Dip Direction		IRSA		Batter	Batter	Berm
				From (deg)	To (deg)	Angle (deg)	Height (m)	Angle (deg)	Height (m)	Width (m)
S1	Oxide (Saprolite)	All slopes		All slopes		38	50	55	10	6
T1	Transition (rock)	All Slopes		All slopes		47	20	60	20	7
F1	Fresh (rock)	1	NE	220	270	51		65	20	7
		2	SE	270	330	51		65	20	7
		3	N,W,S	0 330	220 360	53		70	20	8
Interim fresh rock pit slopes where best practice blasting is not applied						48.5		65	20	8.5
A double berm width (12m) is recommended at the base of oxidation between the Oxide Domain and the Transition Rock Domain. De-couple the fresh rock slope every 100m (i.e. every 5th bench) with either placement of the haul ramp or a double-berm width (16m).										

16.4 Pit and Waste Dump Design

Detailed pit designs were completed for each of the phase cutbacks. The detailed designs are shown in Figure 16.4.1 for Phase 1, Figure 16.4.2 for Phase 2 and Figure 16.4.3 for Phase 3. A cross section through the three phases is shown in Figure 16.4.4.

The ramp exit points for each design were placed on the east to provide the shortest hauls to the ROM pad and waste dump. The pit will be accessed by a 20 m wide haul road incorporating a 10 percent gradient and two-lane haulage. At the pit floor the final benches will be accessed using single lane ramps.

The waste rock dump was located to the east of the pit and adjacent to the processing plant. This design included 6 m wide berms every 10 m in height and with the dump faces at a slope of 20 degrees. The final design reached a height of approximately 50 m to elevation 430 mRL. The waste dump design, with adjacent pit and process plant, is shown in Figure 16.4.5.

Figure 16.4.1 Phase 1 Design



Figure 16.4.2 Phase 2 Design



Figure 16.4.3 Phase 3 Design (Final Pit)



Figure 16.4.4 Cross Section through the Pit Phases

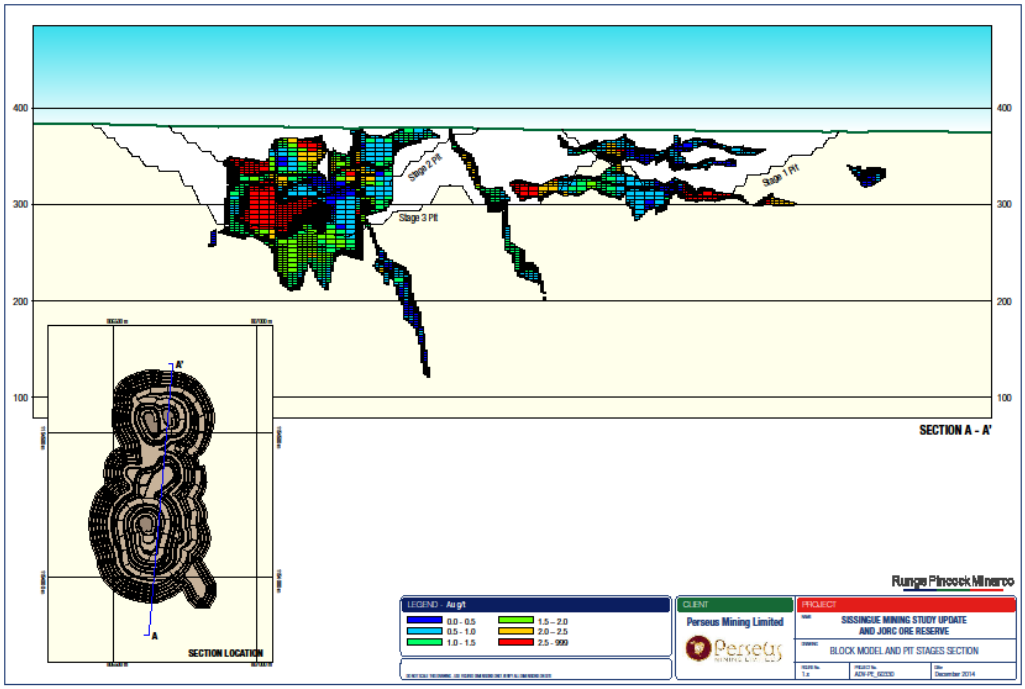
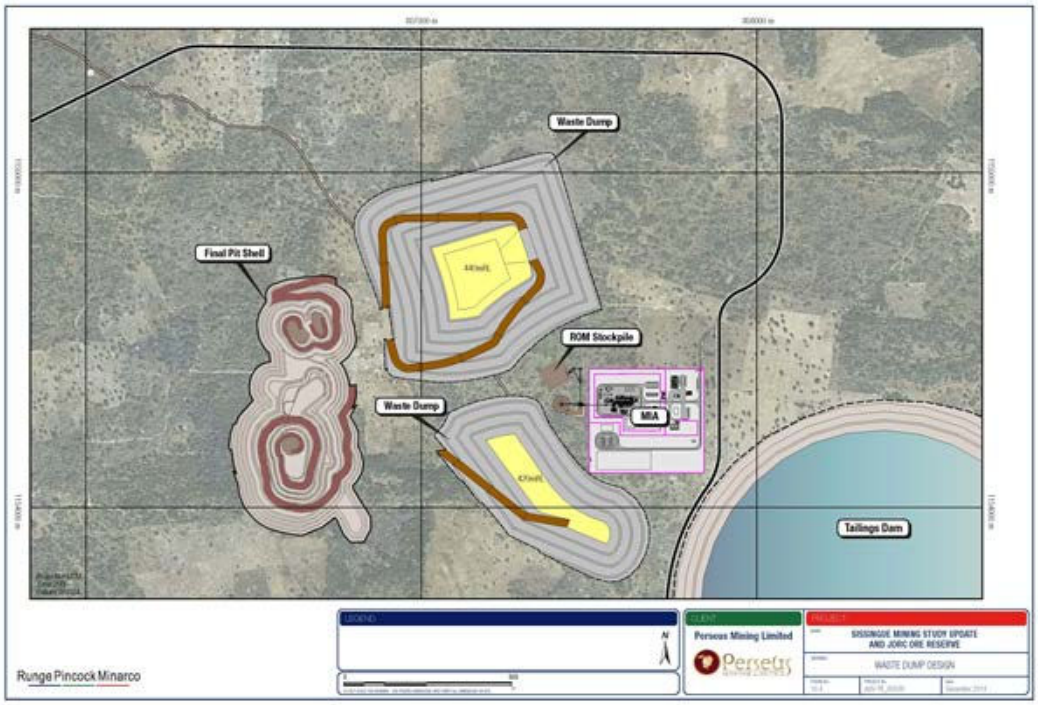


Figure 16.4.5 Waste Dump Design



16.5 Mineable Quantities

The resulting ore and waste mineable quantities for the three phase designs are presented in Table 16.5.1.

Table 16.5.1 Mineable Quantities

Pit Phase	Weathering Type	Waste Mt	Ore Mt	Strip Ratio t/t	Ore Grade Au g/t
1	Oxide	3.3	0.5	6.2	2.1
	Transition	0.4	0.2	1.8	2.7
	Fresh	0.3	0.4	0.6	2.1
Phase 1 Total		**3.9**	**1.2**	**3.3**	**2.2**
2	Oxide	3.2	0.8	4.2	1.4
	Transition	0.3	0.3	1.1	1.7
	Fresh	0.1	0.5	0.3	3.6
Phase 2 Total		**3.6**	**1.5**	**2.4**	**2.1**
3	Oxide	6.1	0.7	8.3	1.4
	Transition	1.6	0.3	5.2	2.4
	Fresh	2.5	1.8	1.4	3.1
Phase 3 Total		**10.2**	**2.8**	**3.6**	**2.6**
Total of All Phases		**17.7**	**5.5**	**3.2**	**2.4**

16.6 Mine Scheduling

The production schedules were generated using RPM's Open Pit Metals Solution (OPMS) scheduling software. The software was used to generate the excavation sequence, dump sequence and stockpile movements. The life of mine production was scheduled in monthly periods. Each period's material ore destination allocation was optimised using OPMS' Product Optimiser module.

The production schedule key drivers were:

- Maximise recovered ounces.
- Elevated rate at the start of the mine life to:
  - provide access to high grade material.
  - allow for potential backfilling of phase one by the subsequent phases.
  - minimise haulage requirements.

16.6.1 Scheduling Assumptions

Mine

The following assumptions were made regarding the mine plan:

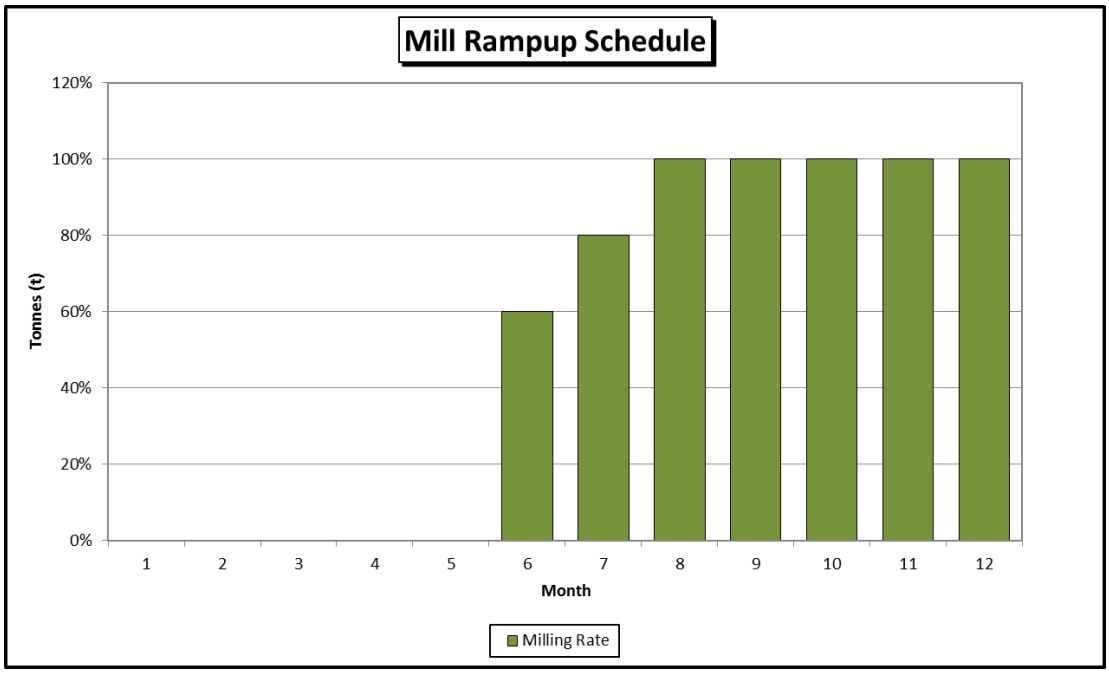
- Maximum mining rate of 6 Mtpa.
- Maximum change in production on a period to period basis of 15%.
- 100% of ore is rehandled in the stockpile.

Process Plant

The following assumptions were made regarding the mill:

- Plant ramp up is to occur over a period of three months, beginning in Month 6 as illustrated in Figure 16.6.4.
- Mill throughput rates as shown in Section 15.
- Processing recoveries as shown in Section 15.

Figure 16.6.1 Processing Plant Ramp-up Schedule



16.6.2 Schedule Results

The following factors influence the mining sequence:

- Phase 3 stripping requirements.
- Contractor capacity.
- Accelerated development of Phase 1.

The annual production is summarised in Table 16.6.1 and Figure 16.6.2 details the timing of major activities in the mine life.

Table 16.6.1 Annual Production

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Total Mined	Mt	6.0	6.0	5.5	3.2	1.7	0.7	23.2
Strip Ratio	t/t	7.7	2.7	3.9	3.8	1.0	0.2	3.2
Total Waste Mined	Mt	5.3	4.4	4.4	2.6	0.9	0.1	17.7
Total Ore Mined	Mt	0.7	1.6	1.1	0.7	0.8	0.6	5.5
Total Ore Grade	g/t	2.0	1.8	2.3	3.1	3.3	2.8	2.4
Total Milled	Mt	0.6	1.1	1.0	1.1	1.0	0.6	5.5

Figure 16.6.2 Major Activity Timing Summary

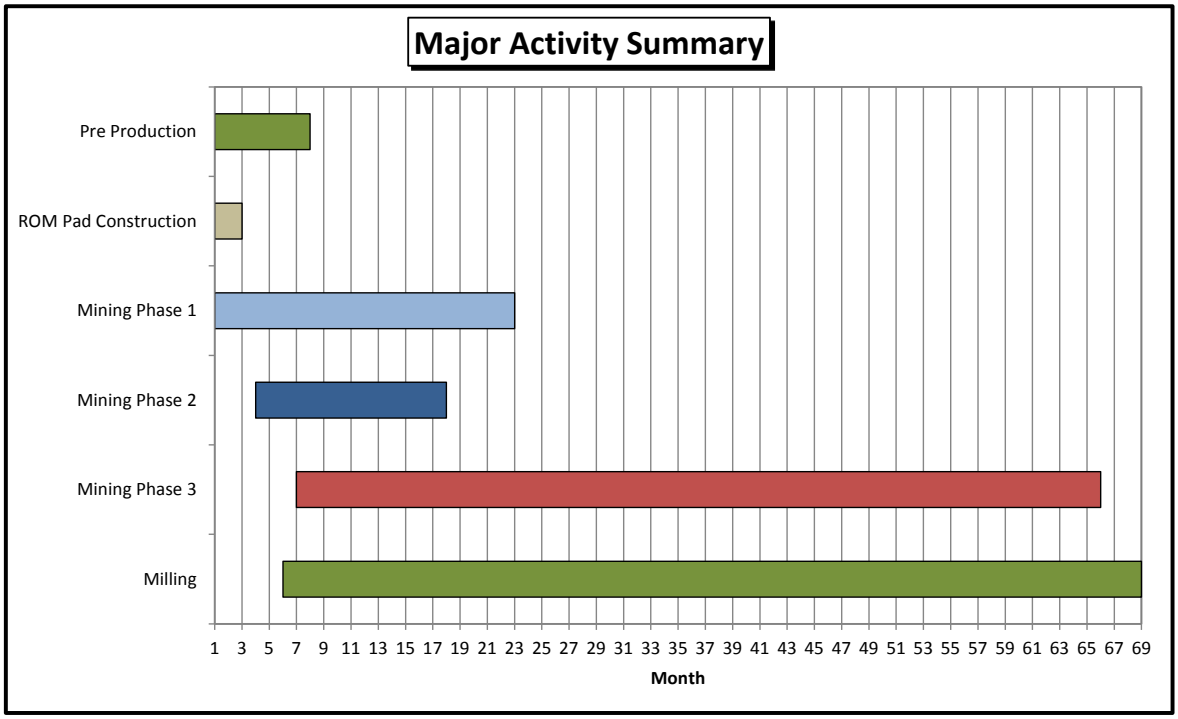
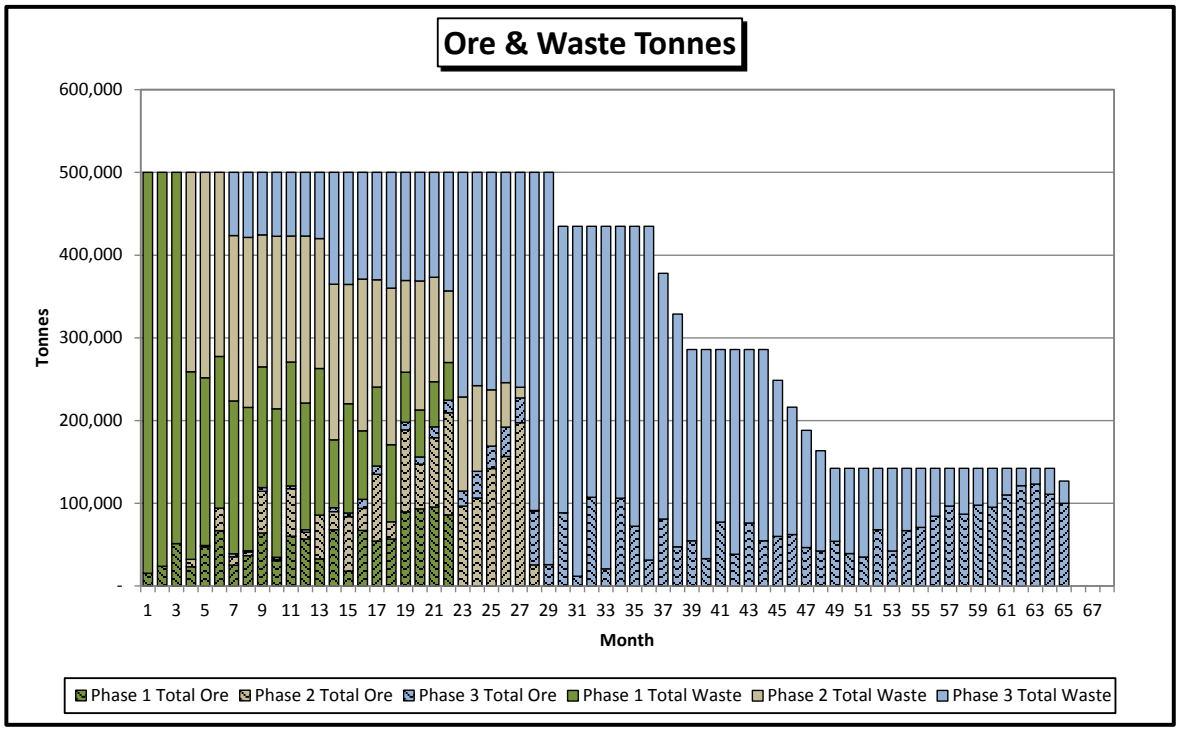


Figure 16.6.3 shows the total material movement for each phase. It demonstrates the progressive transition between each cutback with the first two phases completed in the first two years.

Figure 16.6.3 Total Material Movement by Phase



The tonnage processed by ore type is shown Figure 16.6.4 while Figure 16.6.5 shows the recovered ounces by ore. These illustrate that gold production ranges from ~55 koz to ~100 koz per year and that processing of stockpiles continues for eight months beyond the mining activity.

Figure 16.6.4 Processing Plant Tonnage by Ore Type

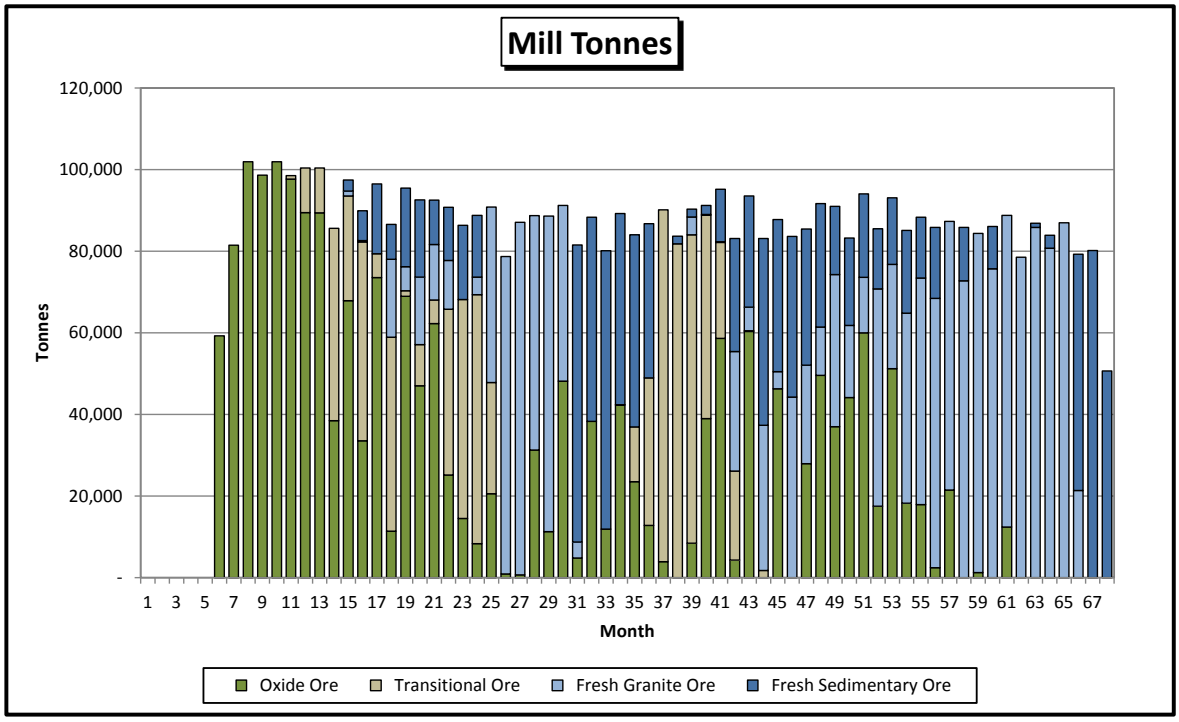
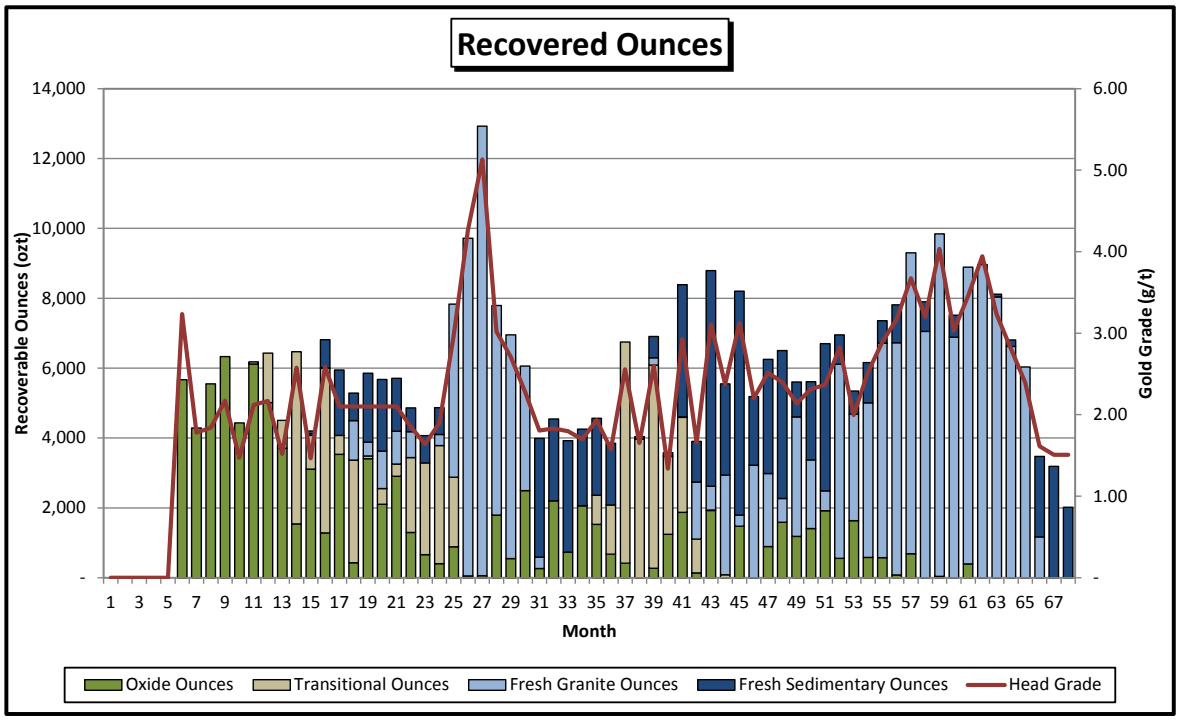


Figure 16.6.5 Recovered Gold by Ore Type



The key outcomes of the production schedule are:

- Six year mine life.
- 1 Mtpa peak processing rate.
- 6 Mtpa peak mining rate for the first half of the mine life.
- Stepped decline in the mining rate for the second half on the mine life with the final mining rate dropping to 1.7 Mtpa.
- Ore grades increase in the last three years.

16.7 Equipment Fleet

The detailed equipment types and classes selected for use at the Sissingué Project are shown in Table 16.7.2.

All haulage calculations were performed within OPMS. OPMS uses the Travel Time calculation object from RPM's Talpac software to calculate truck travel times. This travel time object uses rimpull and retard curve data to perform detailed travel time estimates. All haul distances were calculated between the excavation block centre and destination block centre based on digitised road paths. Table 16.7.1 outlines the assumptions made with regards to haulage calculations.

Table 16.7.1 Haulage Calculation Assumptions

Description	Units	Maximum Speed
Site Speed Limit	kph	45
Ramp Speed Limit	kph	25
Switchback Speed Limit	kph	10
On Bench \ On Lift Speed Limit	kph	20
Rolling Resistance	%	2

Table 16.7.2 Major and Minor Equipment

Equipment Numbers	Units	Ave.	Max.	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6
Drill Rig - 287 kW	no.	1	1	0	1	1	1	1	1
Excavator - 90 t	no.	2	2	2	2	2	2	2	1
Dozer - 228 kW	no.	3	3	3	3	3	3	3	3
Grader - 200 kW	no.	2	2	2	2	2	2	2	2
Dumper - 65 t	no.	8	9	8	8	9	9	9	4
Water Cart - 30 kl	no.	1	1	1	1	1	1	1	1
Wheel Loader - 6,4 cu.m	no.	2	2	2	2	2	2	2	2
Light Plants - 30-ft tower	no.	7	7	7	7	7	7	7	6
Lube/Fuel Truck	no.	1	1	1	1	1	1	1	1
Service Truck	no.	1	1	1	1	1	1	1	1
Tire Truck	no.	1	1	1	1	1	1	1	1
IT - IT38H	no.	1	1	1	1	1	1	1	1
Backhoe - 430E	no.	1	1	1	1	1	1	1	1
Bobcat - 246C	no.	1	1	1	1	1	1	1	1
Pickups - Unknown	no.	12	12	12	12	12	12	12	11
Compactor - 100 kW	no.	1	1	1	1	1	1	1	1

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17.0 RECOVERY METHODS

17.1 Process Design

17.1.1 Design Philosophy

The proposed process plant design for the Sissingué Gold Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is constructed from unit operations that are well proven in industry.

The key criteria for equipment selection are the suitability for duty and the initial short mine life of the operation without unnecessarily compromising reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a compact footprint to minimise construction costs.

The Sissingué plant will process a range of ore types (oxide, transition and primary porphyry, granite and sediment ores) with variable ore characteristics, gold grades and metallurgical treatment requirements. The primary ores are significantly more competent than the oxide ores and require a longer leach time in order to optimise gold recovery.

Perseus has advised that ores will be mined so that predominantly oxide / transitional ore will be processed in two phases; during the first year and then again for a five month period in Year 3. Some blending with primary ores will be required in the final two years. Oxide ore will be processed at a rate of up to 1.2 Mt/y using a two stage crushing and single stage SAG mill circuit. Primary ore will be processed at a rate of 1.0 Mt/y using a three stage crushing and ball mill circuit, where the SAG mill has been re-configured to operate as a ball mill.

The key project and ore specific design criteria that the plant design must meet are:

- 800,000 t/y of primary blend ore; 78% granite / porphyry and 22% sediment.
- 1,200,000 t/y of oxide ore.
- Crushing plant mechanical availability of 70%.
- Mechanical availability for the remainder of the plant of 91.3% supported by crushed ore storage and standby equipment in critical areas.
- Sufficient automated plant control to minimise the need for continuous operator intervention and allow manual override and control if and when required.

A process design criteria document has been prepared incorporating the engineering and key metallurgical design criteria derived from the results of metallurgical testwork and comminution circuit modelling. The design document forms the basis for the design of the processing plant and required site services.

The selected mill has surplus of available grinding power and it is planned to treat primary ore at rates faster than the design rate of 0.8 Mt/y. Granite ores will be treated at approximately 1.0 Mt/y and sediment ores at 0.9 Mt/y. The leach feed density will be increased to compensate for the higher throughput rates and cyanide will be added to the first tank if required.

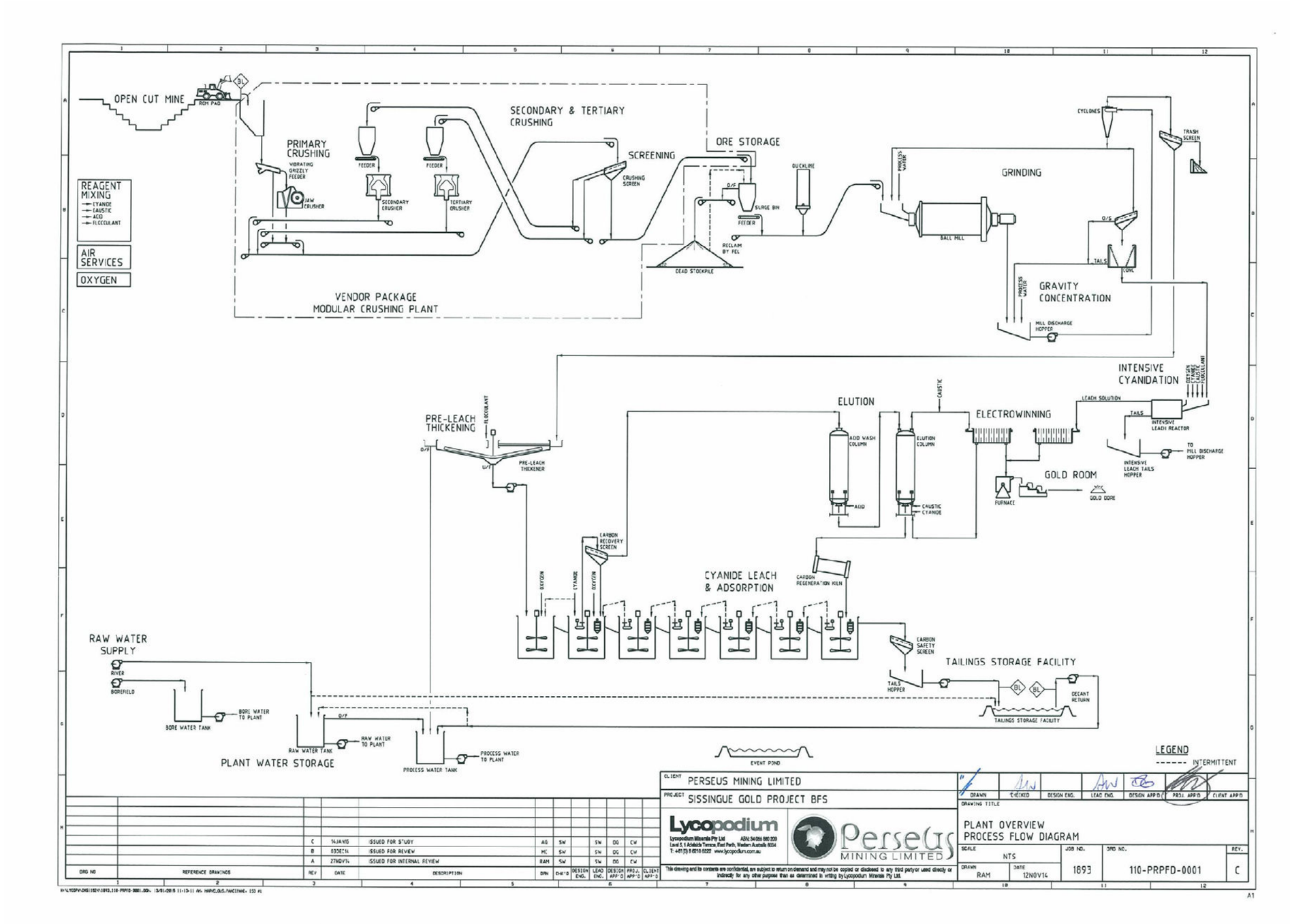
17.1.2 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Initially two stage crushing of oxide ore with open circuit primary crushing and closed circuit secondary crushing to produce a crushed product size of 100% passing (P_{100}) 50 mm.
- Thereafter three stage crushing of primary ores with open circuit primary crushing and closed circuit secondary and tertiary crushing to produce a crushed product size of 100% passing (P_{100}) 14 mm.
- A crushed ore surge bin with a nominal capacity of 100 t. Surge bin overflow will be conveyed to a dead stockpile. Ore from the dead stockpile will be reclaimed by front end loader (FEL) to feed the mill during periods when the crushing circuit is off-line.
- Initial single stage SAG milling of oxide ore in closed circuit with hydrocyclones to produce a P_{80} grind size of 106 µm.
- Thereafter, ball milling of primary ore in closed circuit with hydrocyclones to produce a P_{80} grind size of 106 µm.
- Gravity concentration and removal of coarse gold from the milling circuit recirculating load and treatment of gravity concentrate by intensive cyanidation and electrowinning to recover gold to doré.
- Pre-leach thickener to increase the slurry density to the carbon in leach (CIL) circuit to minimise CIL tankage, improve slurry mixing characteristics and reduce overall reagent consumption.
- Initial leach / CIL circuit for oxide ore incorporating one leach tank and six CIL tanks containing carbon for gold adsorption.
- Thereafter pre-oxidation / CIL circuit incorporating one pre-oxidation tank with oxygen addition to reduce leach oxygen requirements and cyanide consumption and six CIL tanks containing carbon for gold adsorption.
- Four tonne pressure Zadra elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré.
- Tailings pumping to the tailings storage facility (TSF).

A simplified flow diagram depicting the unit operations incorporated in the selected process flowsheet is shown in Figure 17.1.1.

Figure 17.1.1 Plant Overview Process Flow Diagram



17.1.3 Plant Design Basis

The key issues considered in the process and equipment selection are outlined in this section.

Process Plant

The plant design has been based on a nominal capacity of 1.2 Mt/y of oxide ore or 0.8 Mt/y ore primary ore, based on the selected 78% granite / porphyry and 22% sediment blend. The plant design is based on initial separate treatment of predominantly oxide ore followed by reconfiguration of the comminution circuit as the proportion of primary ore in the plant feed increases.

ROM Pad

The ROM pad will be used to provide a buffer between the mine and the plant. The ROM stockpile will allow blending of feed stocks and ensure a consistent feed type and rate to the plant. A mobile rock breaker will be used to break oversize rocks on the ROM pad.

Comminution Circuit Selection

After consideration of three comminution circuit configurations, a hybrid comminution circuit that can be configured for both the low competency and potentially difficult to handle oxide ores and the competent primary ores was selected.

The comminution data was provided to Orway Mineral Consultants (OMC) for comminution circuit modelling and mill sizing. Details of the OMC work are summarised below with full details in the OMC report 7586.30.

Comminution Circuit Design Basis

The Sissingué ores have variable comminution characteristics but can be classified into two main groups:

- Oxides ores with typically low competency and below average grinding energy requirements and low abrasivity.
- Primary ores (granite / porphyry and sediment) exhibiting high competency with above average grinding energy requirements and are moderate to highly abrasive.
- The primary sediment ore in particular has a higher competency, resistance to impact breakage and grinding energy requirement than the primary granite / porphyry ore. It is unlikely a simple primary crushing / SAG milling circuit will be suitable for the primary sediment ore and some form of pre-crushing will be required.

Due to the different comminution characteristics of the oxide and primary ores, OMC identified that different milling conditions were required for the two ore types and that mill throughput on oxide ore should be raised to limit overgrinding.

The Sissingué mine schedule produces predominantly oxide and transition ores during the first 12 months of operation and again for a five month period in Year 3. The proportion of primary ores is then predicted to increase rapidly. A blend of primary granite / porphyry and sediment ores will be treated with the proportion of granite / porphyry primary increasing towards the end of the Project. After reviewing the mine schedule, Perseus advised that the milling circuit design should be based on a throughput of 800,000 t/y of a primary ore blend of 78% granite / porphyry and 22% sediment. This provided a circuit with a nominal throughput on oxide ore of 1,200,000 t/y.

OMC evaluated four comminution circuit options based on achieving the target P_{80} grind size of 106 µm:

- Option 1: Primary crushing, single stage SAG milling and recycle crushing (1C SS SAC). This option was used to evaluate whether a grate discharge 5.50 m diameter x 9.15 m effective grinding length (EGL) SAG mill with 4,500 kW motor available to Perseus and in storage could be used for the Project.

- The mill was deemed unsuitable for either the oxide or primary ores at the nominated throughput rates. On oxide ore, the mill is too large (both dimensionally and installed power) leading to overgrinding and inefficient operation. When treating the highly competent sediment primary ore the mill diameter is too small to generate sufficient impact breakage of primary crushed ore resulting in the throughput target not being achieved.

- Option 2: Primary and secondary crushing, single stage SAG milling and recycle crushing using a grate discharge SAG mill 4.80 m diameter x 6.80 m EGL with 2,500 kW motor.

- Although this circuit suits the low competency oxide ore on which it is expected to be energy efficient and easily operable, the circuit will become less efficient and more difficult to operate when treating predominantly primary ore.

- Option 3: Primary, secondary and tertiary crushing, single stage ball milling (3C, SS Ball) incorporating a 2,050 kW, 4.50 m diameter x 6.4 m EGL overflow ball mill.

- While this circuit is suitable and is expected to be the most energy efficient and easily operable for the high competency primary ores it will be difficult to operate when treating oxide ores with materials handling problems in the crushing and screening circuits and overgrinding expected whilst producing a large amount of scats.

- Option 4: 2C SS SAG with conversion to 3C SS Ball using a grate discharge SAG mill 4.80 m diameter x 6.8 m EGL later modified to an overflow ball mill 4.8 m diameter x 7.10 m EGL.

- This circuit combines the operability of Option 2 while treating oxide ore and the energy savings of Option 3 while treating primary ore.

Given the variation in the comminution characteristics of the primary ore, Perseus advised that the mill should be oversized to prevent the milling circuit becoming throughput limiting when treating the most competent primary ores. The selected installed mill power of 2,500 kW will be sufficient to process 0.92 Mt/y of the highly competent sediment primary ore.

OMC recommended that the mill be fitted with high capacity pulp lifters and a high (>8%) open area grate design to mitigate the risk of slurry pooling and a full variable speed (VVVF) drive so that the mill speed can be varied to suit the variable ore types and blends.

The key design criteria used in modelling of the comminution circuit and the key results for the selected circuit are summarised in Table 17.1.1. The comminution average power and consumables requirements are summarised in Table 17.1.2 and are used in the estimation of the process plant operating cost.

Selected Comminution Circuit

When treating the low competency (and possibly wet and sticky) oxide ores, two stages of closed circuit crushing to a coarse screen size followed by SAG milling will be used

For blends high in primary ores the circuit will comprise closed circuit three stage crushing followed by a ball mill operating in closed circuit with hydrocyclones.

Table 17.1.1 Milling Circuit Design Criteria

Parameters	Condition	Units	Oxide	Design Blend	Granite / Porphyry	Sediment
Design Blend		%	0	100	78	22
CWi		kWh/t	5.0	12.7	13.4	10.0
BWi		kWh/t	5.4	17.2	17.0	17.9
RWi		kWh/t	3.7	19.4	18.3	23.4
Axb		-	150*	30.3	32.4	24.4
Milling Feed F_{80}		mm	32.0	9.5	9.5	9.5
Milling Product P_{80}		µm	106	106	106	106
Mill Specific Energy	@ P_{80} 106µm	kWh/t	6.2	17.5	17.1	19.0
Mill Pinion Power	Nominal	kW	930	1,750	1,750	1,750
Grinding Circuit Throughput	Nominal	t/h	150	100	102	92
	Nominal	Mt/y	1.20	0.80	0.82	0.74
Mill Pinion Power	Nominal Max	kW		2,188	2,188	2,188
Grinding Circuit Throughput	Nominal Max	t/h		125	128	115
	Nominal Max	Mt/y	0.00	1.00	1.02	0.92
Mill Diameter		m	4.80	4.80		
Mill EGL		m	6.80	7.10*		
Mill Dia x EGL		ft x ft	15.7 x 22.3	15.7 x 23.3		
L:D Ratio			1	1		
Discharge Arrangement			Grate	Overflow		
Speed	Design	% Nc	60	75		
	Range	% Nc	0 - 80**	0 - 75**		
Liner Thickness	New	mm	80	80		
Top Ball Size		mm	80	80		
Milling Density		% Solids	65	72		
Ball Charge	Nominal	% vol	10	21		
	Nominal Max			28		
	Maximum	% vol	24	32		
Total Load	Nominal	% vol	10	-		
	Maximum	% vol	32	-		
Pinion Power	Nominal	kW	930	1,750		
	Nominal Max	kW		2,188		
	Maximum	kW	2,390***	2,390****		
Installed Power		kW	2,500	2,500		

Notes:

* To account for grate removal

** Full variable speed VVVF drive recommended

***Maximum conditions of 24% ball charge, 32% total load and 80% Nc.

****Maximum conditions of 32% ball charge and 75% Nc.

Table 17.1.2 Comminution Consumables by Ore Type (@ Design Throughputs)

Equipment / Parameter	Unit	Oxide	Design Blend	Granite / Porphyry	Sediment
Primary Crusher					
Liner Consumption – Fixed Jaw	hours/unit	4000	1276	1100	1900
Liner Consumption – Moving Jaw	hours/unit	6000	1825	1550	2800
Power Consumption	kWh/t milled	0.14	0.25	0.26	0.23
Secondary Crusher					
Liner Consumption	hours/unit	8500	1060	950	1450
Power Consumption	kWh/t milled	0.22	0.52	0.54	0.46
Tertiary Crusher					
Liner Consumption	hours/unit	-	730	650	1000
Power Consumption	kWh/t milled	-	0.64	0.66	0.55
SAG Mill					
Media Consumption	kg/t milled	0.07	-	-	-
Liner Consumption	kg/t milled	0.02	-	-	-
Power Consumption	kWh/t milled	6.7	-	-	-
Ball Mill					
Media Consumption	kg/t milled	-	1.17	1.15	1.23
Liner Consumption	kg/t milled	-	0.15	0.15	0.16
Power Consumption	kWh/t milled	-	17.4	16.9	19.3

Crushing Circuit

The crushing circuit has been sized on the basis of achieving 70% utilisation at a feed rate 30% above that of the milling throughput. Excess crushed ore will be stockpiled and fed to the process plant using a FEL via the crushed ore surge bin during periods of crusher maintenance.

A vendor supplied package crushing plant comprising 'off the shelf' modules has been selected in order to minimise capital cost and the Project site works schedule. The crushing circuit parameters including ore characteristics and required crushed ore size were provided to a number of vendors who then proposed suitable circuits.

A grizzly with an aperture of 800 mm will be installed on the ROM bin to minimise oversize material entering the bin and causing down-stream blockages.

A vibrating grizzly feeder will withdraw ore from the ROM bin and will allow crusher product sized ore to bypass the jaw crusher, reducing the load and wear on the jaw crusher.

A triple deck vibrating screen will be used to size the crushed material into secondary crusher and tertiary crusher feed. Bottom deck screen undersize will be crushing circuit product.

When processing oxide ore, the required SAG mill feed (-50 mm) can be achieved with secondary crushing only and the tertiary crushing circuit will not be used. The deck apertures in the vibrating screen will be configured to achieve this. When processing primary ore, the ball mill feed (-14 mm) will required secondary and tertiary crushing to achieve the required crush size.

Milling and Classification

A milling circuit has been selected to reduce crushed product to the nominal circuit P_{80} size of 106 µm.

The mill will be equipped with a variable speed VVVF drive and will be capable of operating between 55% and 80% of critical speed. In the first year of operation while treating oxide ores, the mill will be configured as a grate discharge SAG mill based on a mill feed top size of 50 mm and will operate at a lower mill speed and ball charge. As the proportion of primary ore increases, the mill will be re-configured as an overflow ball mill based on a mill feed top size of 14 mm and will operate at a higher mill speed and ball charge.

Relatively large diameter cyclones have been selected for the classification duty to minimise the number of cyclones and reduce the potential for spigot blockages from any coarse mill discharge material.

Selection of a pre-leach thickener to increase leach feed density allows operation of the milling and classification circuit to be optimised by operating cyclones at feed densities that increase classification efficiency, thereby reducing circulating load and overall circuit power consumption.

Gravity Concentration

Gravity testwork indicated that high gravity gold recoveries can be expected from the Sissingué ores and that removal of gravity gold prior to CIL improves overall gold recoveries on the primary ores. This justified the inclusion of a gravity recovery circuit with intensive cyanidation of the concentrates. A separate electrowinning circuit will be provided for the intensive cyanidation solution to assist with metallurgical accounting and eliminate any potential impact on the operation of the carbon elution and electrowinning circuit.

Pre-Leach Thickening

As well as allowing the grinding circuit to operate at optimum classification densities, inclusion of a pre-leach thickener ahead of the CIL circuit achieves higher and more constant leach feed density. The higher leach feed density reduces the overall tankage volume required to achieve the target residence time. Additionally a higher CIL density improves slurry mixing characteristics and reduces reagent costs. Pre-leach thickening also recovers process water for re-use in the plant.

Thickening testwork showed high settling rates for both the oxide and primary ores however the viscosity of the oxide ore at the target grind size increased significantly as the thickened slurry solids concentration increased. At a solids concentration of 51% w/w, the thickened oxide slurry exceeded the centrifugal pumping limit of 40 Pa. A thickener underflow density maximum of 45% w/w solids was selected for the oxide ores.

No viscosity issues were noted for the primary ores and thickener underflow density maximum of 52% w/w solids was selected for the primary ore.

Leach and Adsorption Circuit

Metallurgical testwork indicated that:

- the Sissingué ores do not show any indication of 'preg-robbing' characteristics
- the initial leach kinetics for the oxide ores are very fast, with more than 95% of total gold extraction being achieved in the first four hours of leaching
- the leach kinetics of the primary ores are initially reasonably fast, with about 85% of total gold extraction being achieved in the first four hours of leaching
- primary ore gold extraction, however, continued through to 48 hours with improved leach kinetics demonstrated with the inclusion of four hours of slurry conditioning with lime and oxygen prior to leaching.

On this basis, a circuit configuration comprising one conditioning tank and six adsorption tanks has been adopted with six CIL tanks required to achieve acceptable stage efficiency and target solution tails grades. The tanks will be identical in size with cyanide added to the conditioning tank discharge and the CIL tanks as required. The CIL circuit has been designed for a residence time of 48 hours at 100 t/h solids at 52% w/w density. Oxygen will be sparged to all tanks to oxygenate the slurry and oxidise any cyanicides in the conditioning tank and to maintain an adequate dissolved oxygen level for leaching in the CIL tanks.

When processing oxide ore the conditioning tank will be used as a leach tank followed by six CIL tanks. Cyanide will be added to the leach feed distribution box and first tank as well as downstream if required. Oxygen will be sparged to all tanks. The leach / CIL circuit will have a residence time of 31 hours at 45% w/w density.

Elution

The average daily movement of carbon has been calculated based on the design feed grade and maximum CIL gold extraction (assuming no gravity gold recovery). On this basis, a four tonne capacity pressure Zadra elution circuit has been selected requiring seven strips per week for the oxide ore and 3 - 4 strips per week for the primary ore.

A vendor supplied package desorption and refining facility has been selected in order to minimise capital and project construction schedule. The desorption circuit parameters were provided to the vendor who then proposed a suitable facility. The facility includes the elution and electrowinning circuits, the goldroom and the carbon regeneration kiln. The stripping circuit contains separate rubber liner acid wash and stainless steel elution wash columns.

Raw Water

Water for the Project will be sourced from the Bagoe River and from bores within the Project boundary.

The Bagoe River flows for a minimum of six months each year and water sufficient for annual plant usage will be pumped from the river during that six months. The water will be stored in the TSF which has been designed as a combined tailings and water storage facility. Water from the TSF will be recovered via the decant and pumped to the plant. Potentially this water may contain a low amount of cyanide and lime and will be not be used in areas that this would not be suitable (elution circuit, goldroom and potable water feed).

TSF decant return water will be pumped to both the raw and process water tanks. The process water tank will also receive pre-leach thickener overflow. A separate raw water tank has been included in the design (rather than a combined raw / process water tank) to ensure that that the raw water supply is not contaminated by fine solids that may be present in the thickener overflow.

Bore water will be sourced from three bores and will be pumped to the plant bore water tank for use in the elution circuit and goldroom and as the feed stream for the potable water treatment plant.

17.1.4 Key Process Design Criteria

The key process design criteria listed in Table 17.1.3 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17.1.3 Summary of Key Process Design Criteria

		Oxide	Primary Blend	Source [1,2,3,4]
Ore Blend			78% Porphyry/ Granite 18% Sediment	Perseus
Crushing Circuit Capacity	t/y	1,200,000	1,000,000	Perseus
Plant Throughput - Design	t/y	1,200,000	800,000	Perseus
Plant Throughput – Nominal Max.	t/y		1,000,000	Perseus
Gold Head Grade	g Au/t	2.50	3.75	Perseus
Design Gold Recovery	%	92	90	Perseus
Crushing Plant Utilisation	%		70.0	Lycopodium
Plant Availability	%		91.3	Lycopodium
ROM Ore Top Size	mm	600	750	Assumed / OMC
Crushing Work Index (CWi)	kWh/t	5.0	12.7	Testwork / OMC
Bond Ball Mill Work Index (BWi)	kWh/t	5.4	17.2	Testwork / OMC
Axb		150	30.3	Assumed / OMC
SG		2.60	2.77	Testwork / OMC
Abrasion Index (Ai)		0.0026	0.459	Testwork / OMC
Comminution Circuit		2° Crush & SAG	3° Crush & Ball	Perseus / OMC
Crush Size, P_{80}	mm	32	9.5	OMC
Grind Size, P_{80}	µm	106	106	Perseus
Mill Pinion Power	kW	620	1,750	Perseus
Grinding Media Consumption	kg/t	0.07	1.17	OMC
Gravity Gold Recovery	%	25	25	Assumed
Pre-leach Thickener Solids Loading	$t/m^2.h$	1.0	1.0	Testwork
Leach /CIL Residence Time	hrs	31	48	Perseus
Leach Slurry Density	% w/w	45	52	Testwork
Number of Pre-Oxidation Tanks		0	1	Perseus
Number of Leach Tanks		1	0	Perseus
Number of Adsorption Tanks		6	6	Perseus
Cyanide Consumption[5]	kg/t	0.51	0.59	Perseus
Quicklime Consumption[6]	kg/t	0.76	0.32	Perseus
Elution Circuit Type			Zadra	Perseus
Elution Circuit Size	t		4	Lycopodium
Frequency of Elution	strips / week	7	4	Lycopodium

Notes:

1. ‘Perseus’ refers to advice from Perseus Mining Limited.
2. 'Lycopodium' refers to Lycopodium experience or generally accepted practice.
3. 'Testwork' refer to metallurgical testwork conducted.
4. 'OMC' refers to advice from Orway Mineral Consultants.
5. Cyanide consumption makes allowance for 100ppm residual cyanide in the CIL tail solution.
6. Lime consumption based on 90% CaO.

17.2 Process and Plant Description

The process and plant description should be read in conjunction with the simplified process plant flowsheet provided (Figure 17.1.1) and the process plant general arrangement drawing Figure 17.2.1.

The configuration of some circuits / equipment will vary depending on the ore being treated, as outlined above:

- The crushing circuit will initially be configured for two stage crushing to P_{100} 50 mm for processing oxide ore then reconfigured for three stage crushing to P_{100} 14 mm when processing primary ore. Reconfiguration will require only the changing of screen decks on the crusher screen.

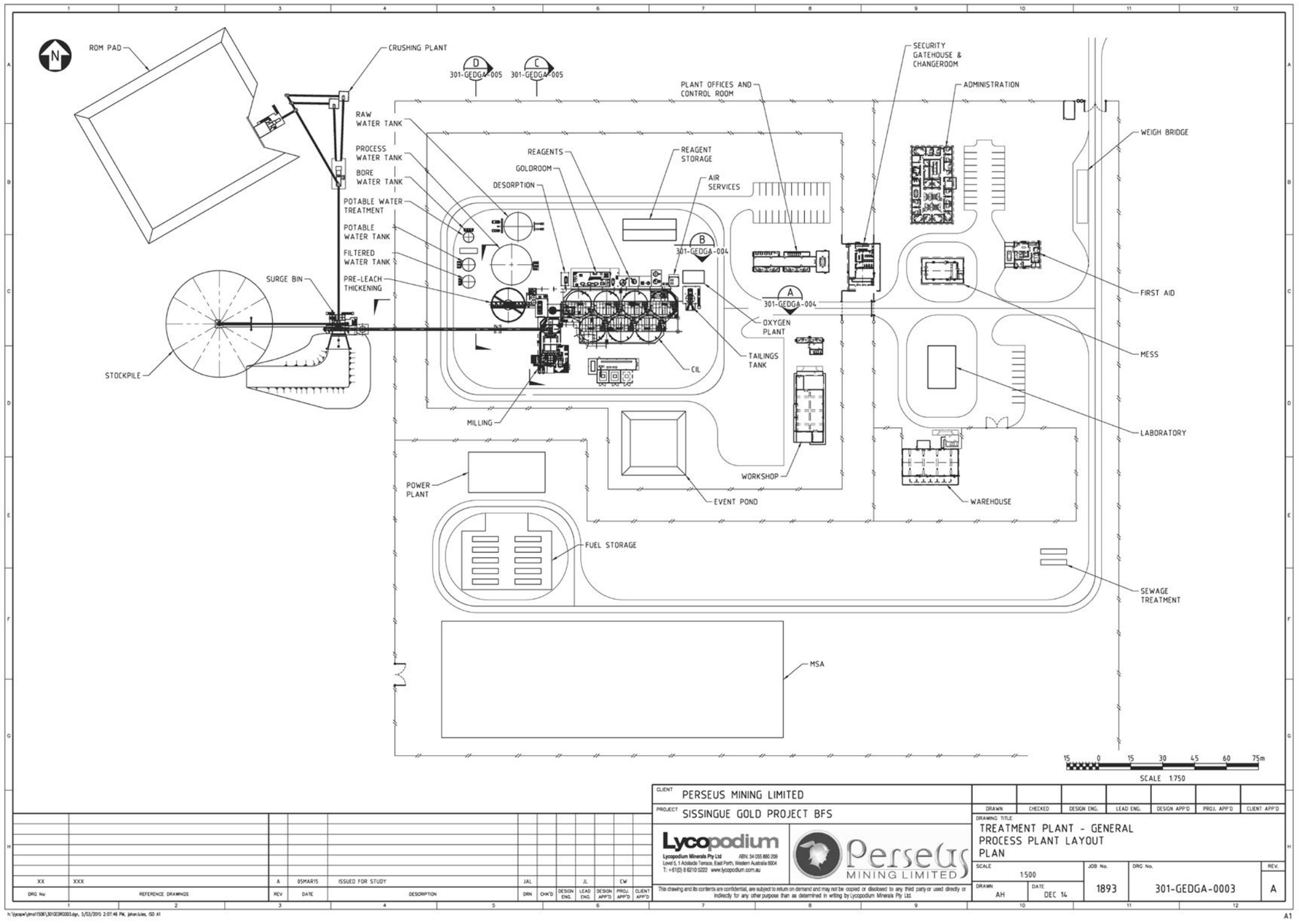
- The selected mill will be configured as a SAG mill for processing oxide ore then reconfigured as a ball mill for processing primary ore. Reconfiguration will require only a change of mill liners and will be carried out during a routine mill liner change.

- The leach and adsorption circuit will be configured as a leach / CIL train when processing oxide ore and reconfigured as a pre-oxidation / CIL train when processing primary ore. Reconfiguration will require only the turning off of cyanide addition to the leach tank.

None of these reconfigurations require large capital expenditure or a significant period of plant downtime.

For the purpose of simplicity and ease of understanding the following process description will be restricted to the primary ore duty.

Figure 17.2.1 Process Plant General Arrangement Drawing



17.2.2 Run-of-Mine (ROM) Pad

Haul trucks operating directly from the pit will deliver run-of-mine (ROM) ore to the ROM pad where it will be dumped in blending 'finger' stockpiles arranged by ore gold grade and lithology. A front end loader (FEL) will be used to reclaim and tram ore from the various stockpiles to the ROM bin. Ore will be blended under the guidance of mine geologists and process personnel to maintain a relatively constant feed grade to the process plant.

17.2.3 Crushing Circuit

ROM ore will be loaded into the crushing circuit feed bin (ROM bin) by FEL from the ROM pad stockpiles. A grizzly will be fitted to the ROM bin to protect the downstream equipment from oversize material. A mobile rock breaker will be utilised to break oversize rocks.

ROM ore will be drawn from the ROM bin at a controlled rate by a vibrating grizzly feeder. The grizzly oversize will report to the jaw crusher. The grizzly undersize and jaw crusher discharge will gravitate to the primary crushing conveyor which will discharge onto the crushing screen feed conveyor feeding directly to the crushing screen. A weightometer beneath the crushing screen feed conveyor will indicate the primary crushing tonnage. A fixed electromagnet suspended above the conveyor will remove magnetic tramp metal from the crushing screen feed. A metal detector will provide protection against non-magnetic metals entering and damaging the downstream crushing equipment.

The triple deck vibrating crushing screen will size crushed material. Screen top and middle deck oversize will be conveyed via the secondary crushing feed conveyor to the secondary crusher feed bin. The secondary crusher will be 'choke' fed by a vibrating feeder from the feed bin. The secondary crushed material will discharge to the secondary crusher discharge conveyor and will report to the crushing screen for sizing via the crushing screen feed conveyor.

Screen bottom deck oversize will be conveyed via the tertiary crushing feed conveyor to the tertiary crusher feed bin. The tertiary crusher will be 'choke' fed by a vibrating feeder from the feed bin. The tertiary crushed material will to the discharge to the tertiary crusher discharge conveyor and will report to the crushing screen for sizing via the crushing screen feed conveyor.

The screen bottom deck undersize will discharge to the surge bin feed conveyor as crushing circuit product and will be conveyed to the surge bin. A weightometer beneath the surge bin feed conveyor will indicate the crushing circuit product tonnage.

Coarse spillage in the crusher area will be cleaned up by FEL and transported to the ROM pad for drying or fed directly to the primary crusher.

Water sprays will be installed for dust suppression as required.

The crushing circuit will be controlled manually. The FEL driver will ensure feed is maintained to the crushing circuit and will communicate with the crushing operator using a two way radio to supply information on crusher feed operation. The speed of the vibrating grizzly feeder will be adjusted by the crushing operator as required.

17.2.4 Ore Storage and Reclaim

Under normal operating conditions, the crushing rate into the surge bin will exceed the rate of withdrawal of ore to the milling circuit. Crushed ore will overflow the surge bin and be directed on to the conveyor feeding the dead stockpile. When required, ore from the dead stockpile will be loaded by FEL into the surge bin to maintain mill feed when the crushing circuit is off line.

Crushed mill feed will be withdrawn from the surge bin at a controlled rate by a variable speed belt feeder and fed via the mill feed conveyor directly to the ball mill. A weightometer will indicate the instantaneous and totalised mill feed tonnage.

Quicklime, used for pH control in the leach circuit, will be added directly onto the mill feed conveyor. Quicklime will be stored in a lime silo and will be metered onto the belt using a variable speed rotary feeder. The speed of the rotary feeder will be varied according to the mill feed tonnage. Quicklime will be delivered to site in bulk bags and transferred to the lime silo using a hoist and lifting frame to a bag breaker mounted at the top of the silo. The silo will be fitted with a dust collector.

Spillage in the surge bin area will be cleaned up by FEL and loaded into the circuit via the surge bin.

Water sprays will be installed for dust suppression as required.

The reclaim circuit will be controlled from the main control room. Area shift operators will also monitor the circuit.

17.2.5 Grinding and Classification Circuit

The grinding circuit will consist of a ball mill in closed circuit with hydrocyclones. Crushed ore will be fed directly to the ball mill. Process water will be added to the ball mill to achieve the required milling density.

The ball mill will discharge via the trommel into the mill discharge hopper and will be diluted with process water and pumped to the classifying hydrocyclone (cyclone) cluster for classification. Ball mill scats (trommel oversize) will discharge to the scats bunker and will be periodically removed by skidsteer loader.

The cyclone underflow will be collected in the launder and gravitate to the feed chute of the ball mill with a portion feeding the gravity circuit. The combined overflow stream will gravitate to the trash screen.

Cyclone overflow will be screened on a vibrating trash screen to remove any misreporting coarse ore particles, wood fragments, organic material and plastics that may adversely affect the performance of the circuit carbon and 'peg' the intertank screens.

Grinding media will be added into the ball mill via kibbles. The kibbles will be charged with grinding media and lifted to the ball charging level and discharged into the ball mill.

17.2.6 Gravity Circuit

Feed for the gravity circuit will be taken as a bleed from the cyclone underflow. The circuit will consist of a scalping screen and a centrifugal concentrator.

The gravity circuit feed stream will be diluted with process water and will gravitate to the scalping screen to remove coarse material and fragments of broken mill balls. The scalping screen oversize will gravitate to the mill discharge hopper for further processing. The screen undersize stream will report to an automated centre discharge centrifugal concentrator. The tails slurry from the centrifugal concentrator will gravitate to the mill discharge hopper. The concentrator will be operated on a semi-batch basis with the periodic discharge of the coarse, high specific gravity material (gravity concentrate) via a sealed and protected pipeline to the concentrate storage hopper in the goldroom.

17.2.7 Pre-leach Thickening

Trash screen underflow from the grinding circuit will be thickened in a high rate thickener. The feed slurry will be de-aerated in the thickener feed box prior to entry into the thickener. Flocculant will be added in the feed launder and feed well. Flocculant will be diluted with water in a static mixer to ensure good dispersion throughout the feed stream. Thickener underflow will be pumped to the leach feed distribution box and thickener overflow will gravitate to the process water tank.

17.2.8 Leach and Carbon Adsorption Circuit

Pre-leach thickener underflow stream will be pumped to the leach and carbon adsorption circuit. The circuit will consist of seven tanks interconnected with launders and slurry will flow by gravity through the tank train. The first tank will be used as a conditioning tank to condition the slurry with oxygen prior to leaching and the six subsequent tanks will operate as CIL tanks for gold adsorption onto carbon.

Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing and the six CIL tanks will each be fitted with a mechanically swept woven wire intertank screen to retain the carbon. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for agitator or screen maintenance.

Quicklime added directly to the mill feed conveyor will ensure that the slurry pH is suitable for cyanidation. Sodium cyanide solution will be metered into the conditioning tank discharge and CIL tanks from a ring main system. Oxygen gas from the pressure swing adsorption (PSA) plant will be distributed to all tanks and sparged down the agitator shafts to oxidise any cyanicides and to provide oxygen to the leach.

Fresh / regenerated carbon will be returned to the circuit at CIL Tank 6 and will be advanced counter current to the slurry flow by pumping slurry and carbon from CIL Tank 6 to CIL Tank 5 and so on. The intertank screen in CIL Tank 5 will retain the carbon whilst allowing the slurry to flow by gravity back to Tank 6. This counter-current process will be repeated until the carbon eventually reaches CIL Tank 1, the first adsorption tank. A recessed impeller pump will be used to transfer slurry to the loaded carbon recovery screen mounted above the acid wash column in the elution circuit. The carbon will be washed and dewatered on the recovery screen prior to reporting to the acid wash column. The associated slurry and wash water will return to CIL Tank 1.

Slurry from the last CIL tank (CIL tails) will gravitate to the carbon safety screen to recover any carbon leaking from worn screens or overflowing tanks. Screen underflow will gravitate to the tails hopper. Screen oversize (recovered carbon) will be collected in the fine carbon bin for potential return to the circuit.

Barren carbon returning to the adsorption circuit from the carbon regeneration kiln will be screened on the carbon sizing screen to remove fine carbon and quench water. The sized and regenerated carbon will report directly to CIL Tank 6.

17.2.9 Elution and Goldroom Operations

The following operations will be carried out in the elution and goldroom areas:

- Acid washing of carbon.
- Stripping of gold from loaded carbon using the Zadra method.
- Electrowinning of gold from pregnant solution.
- Smelting of electrowinning products.

An in-line leach reactor (ILR) for the gravity concentrate and a dedicated electrowinning cell will also be located in this area.

The elution and goldroom areas can operate seven days per week, if required, with the majority of loaded carbon preparation and stripping occurring during day shift.

Acid Wash

Loaded carbon will be recovered on the loaded carbon recovery screen and directed to the rubber lined acid wash column. Transfer and fill operations will be controlled manually. All other aspects of the acid wash and the pumping sequence will be automated.

Acid washing of the carbon will commence after carbon transfer is complete.

The acid wash solution, 3% w/w HCl in bore water, will be prepared prior to use and stored in the acid mixing / storage tank.

During acid washing the dilute solution of hydrochloric acid will be pumped through the column in an up-flow direction to remove contaminants, predominantly carbonates, from the loaded carbon. This process improves the elution efficiency and has the beneficial effect of reducing the risk of calcium-magnesium 'slagging' within the carbon during the regeneration process.

After an acid soak period the carbon bed will be rinsed with bore water to displace any residual acid from the carbon. Dilute acid and rinse water will be disposed of in the tailings hopper. Acid-washed carbon will be transferred to the elution column for stripping.

Elution and Electrowinning

Sodium hydroxide and sodium cyanide will be metered into the strip solution tank and the tank filled with bore water. The strip solution will then be pumped through a recuperative heat exchanger and a diesel fired in-line heater before entering the base of the elution column.

The recuperative heat exchanger recovers heat from the return eluate and preheats the incoming barren strip solution. The diesel fired in-line solution heater raises the temperature of the incoming barren solution to 130°C.

The heated, barren strip solution will flow upwards, in a plug flow regime, through the bed eluting gold and silver from the loaded carbon. The pregnant solution exiting the top of the column will be cooled through the recuperative heat exchanger and in the process pre-heating the incoming barren strip solution.

Once cooled below flashpoint the pregnant solution is directed to two electrowinning cells in parallel. Direct current will be passed through stainless steel anodes and stainless steel mesh cathodes within the electrowinning cell and electrolytic action will cause gold in solution to form a gold rich sludge on the cathodes. The cells will contain sufficient cathodes in order to provide a high cell pass efficiency to ensure a minimum gold tenor in the spent electrolyte returning to the strip solution tank. The elution / electrowinning cycle will continue until the solution exiting the electrowinning cells is depleted of gold and silver values.

Goldroom

The electroplated silver and gold will be removed from the cathodes by washing with high pressure water jets. The resulting slurry will be filtered in vacuum filters and the solids dried in an oven. The sludge will then be direct smelted with fluxes in a diesel-fired furnace to produce doré bars. Slag from smelting operations will be returned to the milling circuit.

Fume extraction equipment will be provided to remove gases from the electrowinning cells, oven and barring furnace.

Carbon Regeneration

After completion of the elution process, the barren carbon will be transferred from the elution column to the carbon dewatering screen to dewater the carbon prior to entering the feed hopper of the horizontal carbon regeneration kiln. In the kiln feed hopper any residual and interstitial water will be drained from the carbon before it enters the kiln. Kiln off-gases will also be used to dry the carbon prior to entering the kiln.

The carbon will be heated to 650 - 750°C and held at this temperature for 15 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched and transferred to the carbon sizing screen. Carbon sizing screen oversize (regenerated, sized carbon) will return to the CIL circuit and screen undersize (transfer water containing some fine carbon) will gravitate to the quench hopper for re-use with a bleed stream gravitating to the tails hopper via the carbon safety screen for disposal in the TSF.

Gravity Concentrate Intensive Leach and Electrowinning

Gravity concentrates from the centrifugal concentrator will be processed in batches through the ILR. The concentrates will be leached under intensive cyanidation conditions in the reactor. The resulting liquor will be pumped to a small storage tank prior to the solution being electrowon in a dedicated electrowinning cell. Reactor tails will be pumped back to the mill discharge hopper for additional milling to recover any remaining entrained gold. The cathodes from the gravity electrowinning cell will be treated separately to assist in metallurgical accounting. Spent electrolyte will be recycled to the head of the CIL circuit.

17.2.10 Tails Disposal

Carbon safety screen undersize (CIL tails) and other miscellaneous waste streams from the process plant will be combined in the tails hopper and pumped to the TSF.

Tailings will be deposited into the TSF using peripheral discharge and decant method. Cyclic spigot deposition at various locations will be practised to allow consolidation and drying of deposited material into beaches to direct supernatant water to a pond around the decant tower.

17.2.11 Decant Return

The TSF will be used as a combined tailings and water storage facility. Water from the Bagoe River will be pumped to the TSF during the six months the river is flowing and will be stored in the TSF until required in the plant.

Supernatant water will be recovered from the TSF and returned as process water decant return to the plant. The decant water will be pumped from the TSF to the plant raw water tank and process water tank.

17.2.12 Reagents

Quicklime

Quicklime will be delivered to site in one tonne bulk bags. Quicklime will be added to the mill quicklime storage silo using an electric hoist and enclosed bag breaker. Quicklime will be metered via a rotary valve directly onto the mill feed conveyor for circuit pH control. A dust collector will minimise dust emissions during loading of quicklime into the storage silo.

Cyanide

Cyanide will be delivered to site in one tonne boxes containing bulka bags of cyanide briquettes. Cyanide briquettes will be added to the cyanide mixing tank using an electric hoist and enclosed bag breaker and dissolved in process water to achieve the required solution strength.

The cyanide solution will be transferred to the storage tank for use in the process. Cyanide will be reticulated to the CIL circuit via a ring main and dosed to the leach and CIL tanks as required. A dedicated pump will provide cyanide solution to the elution circuit and intensive leach reactor as required.

Caustic Soda

Caustic soda (sodium hydroxide) will be delivered to site in 25 kg bags of 'pearl' pellets. Caustic bags will be added by hand to the mixing tank via a bag breaker on the receiving hopper. The pellets will be discharged into the mixing tank via a rotary vane feeder to prevent splash back from the tank and dissolved in bore water to the required solution strength.

The caustic solution will be pumped to the elution circuit and intensive leach reactor as required.

Hydrochloric Acid

Concentrated hydrochloric acid (32% w/w) will be delivered to site in 1,000 L isotainers. The concentrated hydrochloric acid will be pumped into the acid mixing / storage tank where it will be diluted with the correct quantity of bore water to achieve the required acid wash concentration.

The dilute acid solution will be pumped to the elution circuit as required.

Activated Carbon

Activated carbon will be delivered to site in 500 kg bulk bags. Carbon will be added to the carbon quench tank as required for carbon make-up to the CIL inventory. Carbon will be added directly to the last adsorption tank, as required, for carbon make-up or via the regeneration kiln to allow fines removal on the sizing screen.

Grinding Media

Grinding balls will be delivered to site in 200 L steel drums. Balls will be charged to the ball mill as required to achieve the target power draw.

Grinding balls will be charged to the ball mill by loading the balls using a fork lift with a hydraulic drum tipper attachment into ball charging kibbles and lifting the kibble via a hoist into the ball mill feed box.

Flocculant

Flocculant will be delivered to site in 25 kg bags. Flocculant will be added to the flocculant plant storage hopper manually. The vendor supplied package flocculant mixing plant will automatically mix batches of flocculant with filtered water and transfer the mixed flocculant to the aging tank after each mixing cycle is complete.

The flocculant solution will be pumped to the pre-leach thickener and intensive leach reactor as required.

Leach Aid

Leach aid for the gravity concentrate intensive cyanidation circuit will be delivered to site in 10 kg plastic containers. Leach aid will be added by hand to the intensive cyanidation circuit.

Diesel

Diesel will be delivered to site by bulk tankers and transferred to the multiple double skinned diesel storage tanks. The diesel will be used in the power station, the mine, the process plant and to refuel site vehicles.

Diesel will be pumped from the storage tanks to the plant for use in the strip solution heater, carbon regeneration kiln and smelting furnace.

Reagents Storage

Reagents will be received on site in shipping containers, with a minimum of one months' capacity stored on site, to ensure that supply interruptions due to port, transport or weather delays do not restrict production. Reagent containers will be offloaded from the delivery truck by the site crane and stacked in a lay-down area until required. Empty containers will be returned with the next delivery.

17.2.13 Services

Raw Water

Water will be pumped from the TSF decant to the plant raw water tank. The raw water tank will have sufficient capacity to minimise the impact of short term supply interruptions. Duty / stand-by water pumps will be provided for the raw water distribution to the plant.

Fire Water

Fire water for the process plant will be drawn from the raw water tank. Suctions for other water services fed from the raw water tank will be at an elevated level to ensure a fire water reserve always remains in the raw water tank.

The fire water pumping system will contain:

- an electric jockey pump to maintain fire ring main pressure
- an electric fire water delivery pump to supply fire water at the required pressure and flowrate
- a diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system.

Fire hydrants and hose reels will be placed throughout the process plant, fuel storage and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

Filtered Water

Filtered water for the process plant will be produced by treating raw water in the filtered water treatment plant. The treatment plant will consist of clarification through flocculant addition, sand filtration, carbon filtration and biocide dosing. Filtered water will be report to the filtered water storage tank and will be distributed to the plant as required using duty / stand-by filtered water pumps.

Gland Water

Water from the filtered water storage tank will be distributed as gland service water using duty / stand-by gland water pumps.

Process Water

Water will be pumped from the TSF decant to the plant process water tank. The plant process water will consist of pre-leach thickener overflow, TSF decant return water and raw water tank overflow. The process water tank will be located so that the raw water tank overflows to the process water tank allowing the raw water tank to be kept full at all times.

Duty / standby process water pumps will be provided for the plant process water supply. A separate water pump will be provided for fluidisation water supply to the gravity concentrator.

Antiscalant will be added as required to condition the water and reduce fouling of pipelines, spray nozzles and screen decks.

Bore Water

Bore water will be pumped from the bores to the plant bore water tank. The bore water tank will have sufficient capacity to minimise the impact of short term supply interruptions. Duty / stand-by bore water pumps will be provided for the bore water distribution to the plant.

Potable Water

Bore water will be supplied to the plant potable water treatment plant. The water treatment facility will include sand filtration, micro filtration, ultra-violet sterilisation and chlorination. Potable water will be stored in the plant potable water tank and will be reticulated to the site ablutions, safety showers and other potable water outlets. Additional ultra-violet sterilisation units will be installed on outgoing potable water distribution headers.

Plant and Instrument Air Supply

Plant and instrument air will be supplied from duty / standby air compressors. The air will be filtered and dried before distribution with separate plant and instrument air receivers. A check valve on the instrument air supply will ensure the integrity of instrument air supply such that air from the plant air system serves as a back-up for instrument air, but plant air cannot draw down the instrument air system.

Oxygen

A Pressure Swing Adsorption (PSA) oxygen plant will be installed adjacent to the CIL tanks. The oxygen rich stream (90% O_2) will discharge to the oxygen receiver and then be distributed to the conditioning / leach and CIL tanks. The oxygen plant will be supplied with clean air from a dedicated high pressure screw compressor. The compressed air will have water and oil removed prior to the PSA unit using mist eliminators, filters, a carbon tower and desiccant dryers.

17.3 Plant Area Design

17.3.1 General

The civil, mechanical and electrical design of the plant facilities is based on industry standard practice and Lycopodium's extensive experience in gold plant design and project implementation in West Africa and elsewhere.

The mechanical design of all process tanks will be in accordance with API 650 with due consideration of the seismicity applicable to the plant location, with appropriate materials selection and corrosion allowances.

Materials handling, containment and bunding in all plant areas has been designed to minimise the possibility of spillages to the environment and to meet legislative requirements.

Due consideration will be given to the nature of the various types of process and reagent slurries being pumped around the plant in terms of settling, wear and scaling.

Design of the plant has been undertaken with consideration for future expansion. The dry plant design will accommodate an increased throughput without impacting on the design or equipment. Future addition of a ball mill and duplication of the classification and CIL circuits will allow for increased throughput to be achieved in the wet plant. Space has been provided in the plant layout for this future expansion. Similarly, all ponds have been sized to provide additional storage in the event of plant expansion.

17.3.2 Primary Crushing

The ROM pad will be located at the NW corner of the plant site and isolates heavy vehicle movements to this area and the mine services area. This location also ensures dust picked up by the prevailing winds from the ROM pad will not be blown over other plant or infrastructure.

The crushing facility, ending at the surge bin feed conveyor discharge chute, will be a compact and modular design, provided by the one supplier as a complete package. It will be fit for purpose with minimal automation and basically consist of a primary crusher module, a screening module and a secondary and tertiary crusher module. All conveyors will use a straight profile with walkways provided on one side only and around the headchute. Conveyor inclines of up to 18 degrees will be used on coarser material only. Minimal footings, slabs and civil works will be required.

The crushing facility will be located at the NE side of the ROM pad. Reinforced concrete retaining walls, approximately 6 m high, will be recessed within the embankment to create a chamber in which to locate the crushing plant dump hopper. The 41 tonne capacity lined steel dump hopper will be supported on a steel structure and ore will be reclaimed via a primary vibrating grizzly feeder. The jaw crusher will be supported on an intermediate steel platform over the crusher discharge conveyor

The secondary and tertiary crusher module will be located near the tail end of the screen feed conveyor and all crushed product including primary crushed material will be delivered on this conveyor to the double deck screen. A magnet and metal detector will be provided on the screen feed to protect the screen and downstream equipment.

No fixed maintenance cranes will be provided. All major maintenance activities will require use of a mobile crane.

Replaceable abrasion resistant linings will be provided in all chutes and conveyor skirtboards at all wear points.

Dust collection is not provided. Dust suppression sprays will be used to control dust emissions.

17.3.3 Surge Bin and Stockpile

The mill surge bin feed conveyor, supplied as part of the part of the crushing facility, will be inclined at 10 degrees to minimise spillage. The 100 tonne surge bin height will be designed as low as practical, thereby minimising the cost of the concrete and earthworks required for the front-end loader ramp.

Additional stockpile capacity as required, will be created by pushing out of the 500 tonne capacity conical stockpile formed by the stockpile feed conveyor.

17.3.4 Grinding and Classification Circuit

A lime silo will be provided on the mill feed conveyor. An electric hoist will be provided for lifting of the one tonne bulka bags to the bag breaker on top of the bin.

The grinding and classification circuit building will be a combined concrete and steel structure, supporting a single grinding mill with single pinion drive, mill discharge hopper, classification cyclone cluster, and trash screen.

Access to the main mill level platform will be via two stairways from ground level; one at the mill discharge end and one at the feed elbow floor. Intermediate operating levels including the mill feed conveyor head platform and cyclone / trash screen platform may be reached via a stairway from the feed elbow floor. Secondary access will also be possible via the mill feed conveyor walkway.

Mill ball charging will be performed using a dedicated electric hoist and kibble discharging into the mill feed chute.

Servicing of the 2.5 MW mill drive, the mill discharge trommel and cyclone feed pumps will rely on mobile cranage for all lifting requirements, road access for which will be available adjacent to the structure on three sides. The mill will use lightweight polymet liners and therefore a mill liner handler will not be required for mill relining. The cyclone / trash screen platform will be serviced by a one tonne davit crane.

A trommel overflow chute provides the ability to discharge pebbles from the mill to a drive-in scats bunker. The bunker will be cleared using a front-end loader or skid-steer loader. A sump pump will collect floor spillage in the vicinity of the discharge end of the mill, bunker drainage water and pump scuttling slurry and solids from the duty / standby cyclone feed pumps. A second sump pump and drive-in sump will be located at the feed end of the mill and will collect local floor spillage and hopper overflows.

17.3.5 Trash Screening and Pre-leach Thickening

The trash screen will be located on the milling building upper level to allow height for gravity launder collection. Oversize material from the screen will be diverted to a bunker at ground level, undersize will flow by gravity into the pre-leach thickener feed box.

The 15 m diameter above-ground pre-leach thickener will be located in a bunded area with process water tank adjacent to grinding and classification. The thickener underflow pumps will serve as leach feed pumps and will be located outside the thickener wall for ease of maintenance and mobile crane access.

A single stairwell will allow access from ground level to the thickener bridge at the feed box end. Similarly, access to the bridge can be gained from the second level of the mill building.

The concrete slab will be drained to a floor area sump pump for spillage collection. In the case of a thickener emergency dump, the bund will fill and overflow to the event pond.

17.3.6 Leach and Carbon Adsorption Circuit

The CIL plant will consist of seven 1,040 m³ agitated tanks. One tank will initially serve as a pre-oxidation tank and later on will be fitted with an intertank screen and carbon advance pump to serve as an additional CIL tank. A CIL feed box and launder diverter gates will be located at top of tank platform level and feed box inspection will be accessible by stairway from that level. The carbon recovery screen and will also be located at top of tank level.

There will be three means of access to the top of tanks, stairway from ground level at the southern end, stairway through the elution area and high level walkway from the carbon safety screen platform.

The top of tank equipment will be serviced by the plant mobile crane. A 70 tonne all-terrain crane will be provided for this and other maintenance duties. The intertank screens, carbon transfer pumps and launder gates will be located below the platform steel to reduce potential trip-hazards on the platform area above the tanks. Removable grating panels will provide safe access to submerged platform levels to inspect equipment and temporary handrails will make the CIL platform area safe.

The CIL tanks will be bunded for spillage and splash, but not for containment of a tank volume. Accidental or deliberate dumping of the contents of a CIL tank will cause the bund to overflow to the event pond. Two floor area sump pumps will be located in the bund at accessible positions adjacent the bund wall to collect and return spillage to the circuit. The CIL bund floor will slope to each of these sump positions.

17.3.7 Elution, Carbon Regeneration and Goldroom

For the elution circuit, carbon regeneration and goldroom, a modular, packaged plant provided by one supplier has been included. This will be a compact, fit-for-purpose plant that requires minimal civil works and installation time but provides adequate operation and maintenance access

All the elution and carbon regeneration equipment will be under roof and wall cladding to protect the regeneration kiln shell from differential temperature changes and to keep instrumentation and control panels sheltered.

The carbon regeneration kiln and quench tank will be fitted with an overhead monorail hoist used regularly for new carbon addition but will also be available for maintenance of the kiln drive, shell and rollers. Additional lifting facilities will not be required in this area as maintenance of other equipment will be performed in situ.

The acid wash column will have a partitioned area of the area bund, separated such that any acid spillage cannot mix with cyanide solutions and will be separately disposed of by a dedicated sump pump.

The goldroom building and eluate tanks will be located inside a security area for restricted access by authorised personnel and vehicles only. The security area will have separate personnel and vehicle access gates. The strongroom will be enclosed by additional secure walls and roof to provide additional security. The building will be fresh air ventilated with separate extraction of dust, fumes and heat from the smelting furnace and electrowinning cells. There will be stairway access to the upper level as well as an alternative emergency egress.

The electrowinning cells, cathode wash trough and anode support frame will be located on the upper level and serviced by an overhead gantry crane.

The pregnant solution tank and strip solution tank will be located within a concrete containment bund.

17.3.8 Tailings Disposal

The carbon safety screen will be located on a steel structure adjacent to, and below CIL Tank No. 6 to allow for gravity feed of the screen. Access to the screen will be via a stairway from the top of CIL or ground level. Screen undersize will gravity flow to the tailings hopper whereas oversize will be diverted into the fine carbon bin situated at the base of the structure. A concrete slab and bund will contain any spillage and direct overflow to the event pond.

The tailings pumps will transport the tailings slurry via an HDPE pipe approximately 1.8 km to the discharge point of the tailings storage facility located to the south-east of the plant.

All equipment will be readily accessible by mobile crane to conduct maintenance.

17.3.9 Reagents

The reagent storage / mixing area will be located on the northern end of the CIL tanks and adjacent to the reagent store. Hydrochloric acid, caustic and the flocculant mixing plant will be grouped on one common bunded area with appropriate partitioning for floor spillage, whilst a second separate bunded area will be used for sodium cyanide.

Cyanide

Sodium cyanide briquettes in one tonne bulka bags will be laid down next to the cyanide mixing plant. A steel structure with access stairs from ground level will house the enclosed bag breaker, a mixing tank and a dedicated bag lifting hoist. The structure is fitted with a roof to provide protection to the bag breaking operation from the weather. Mixed cyanide is pumped to the cyanide storage tank.

Spillage in the area is pumped via a sump pump into the leach feed box.

Caustic Soda

Caustic soda in 25 kg bags will be loaded into a bag breaker located on the top of a small mixing and storage tank. Stair access to the top of tank will be provided.

Hydrochloric Acid

The hydrochloric acid facility is to be located within a partitioned area of the reagents bund and has a dedicated sump pump which will dispose of any spillage to the tailings thickener.

Delivery of acid will be via 1,000 L isotainers and unloaded into the mixing and storage tank using a floor-mounted peristaltic pump.

Flocculant

Dry flocculant will be transported by forklift from the reagents store to the flocculant mixing area adjacent to the pre-leach thickener. The 25 kg bags will be loaded into a bag breaker located on the top of a small mixing and storage tank. Stair access to the top of tank will be provided.

The bag breaker and dry hopper structure will be partially clad for rain and wind protection.

17.3.10 Air, Oxygen and Water Services

All compressed air services will be housed in a compressor station adjacent the reagents area consisting of a fully clad steel structure and concrete slab floor. The building will have one permanently open panel for forklift access should any of the package unit compressors and blowers need to be removed for major overhaul. Normal maintenance will be expected to be carried out with the units in situ.

A compact, packaged PSA oxygen plant will be located adjacent to the compressor station. Weather protection is not required.

Water services include raw water, process water, fire water, potable water, bore water and filtered (non-potable quality). All tanks, pumps and the potable water treatment plant will be located at the north-western corner of the process plant. The sewage treatment plant will be located at the south-western corner of the plant site which is also the lowest elevation within the plant site. The plant water and sewage treatment plants will also serve the camp. No weather protection or permanent lifting facilities have been allowed for water services, although the potable water and sewage treatment plants will be containerised.

Raw water will be supplied to the raw water tank from the tailings storage facility (TSF) decant return pumps. The TSF will be supplied with raw water from river pumps located on the Bagoe River approximately one kilometre from the TSF. Additional raw water will also be supplied from three nearby bores as required.

The raw water tank will also hold a volume of 288 m^3 of water reserved for firewater use. The fire protection skid of electric with backup diesel pumps and jockey pump will draw from the base of the raw water tank. The raw water distribution pump set will draw from higher up in the tank to prevent depletion of the fire water supply reserve.

Gland water will be pumped from the filtered (non-potable) water tank.

Process water consists predominantly of pre-leach thickener overflow, raw water tank overflow and TSF decant return.

17.3.11 Large Volume Spillage Containment

All bunded areas in the plant will overflow to the event pond for spillage as well as slurry from the tailings pipeline drain. The contained slurry will be recovered by submersible pumps discharging to the mill discharge hopper.

17.4 Electrical Design

17.4.1 Installed Load and Maximum Demand

The plant site-wide electrical power requirements for infrastructure, mining and processing were calculated on the basis of preliminary equipment sizing. The installed load and maximum demand for the Project is summarised below.

Table 17.4.1 Installed Load and Maximum Demand

Installed Load		Maximum Demand		Average Demand	
(MW)	(MVA @ 0.85pf)	(MW)	(MVA @ 0.85pf)	(MW)	(MVA @ 0.85pf)
6.51	7.66	5.05	5.94	4.15	4.88

The maximum demand is based on an average load factor of 80% for all areas except the mill drive, which was given a load factor of 95%. The 2,500 kW ball mill represents approximately 47% of the total load.

17.4.2 11 kV Switchboards

One 11 kV switchboard has been allowed for in the plant and one will be supplied with the onsite power plant. The 11 kV switchboards will be fully withdraw-able design complete with protection, metering and earthing facilities.

The design fault level and circuit breaker ratings adopted are:

- 11 kV switchboard busbar 2,000 A, 25 kA at 3 sec.
- 11 kV circuit breakers 630 A.

Protection will be provided by microprocessor based protection relays.

17.4.3 Electronic Variable Speed Drives and Soft Starters

LV variable speed drive (VSD) units and soft starter (SS) ratings range from 0.18 kW up to 315 kW. These are floor or wall mounted (dependent upon size) along the internal wall of the LV substation.

17.4.4 415 V Motor Control Centre

The LV MCCs will double-sided (back to back) and housed in the LV switchroom. Construction of all MCCs will have Form 4 segregation, Type 2 coordination. Starters in MCCs will be of demountable design and main incoming circuit breakers will be of withdraw-able design complete with protection. All motor starters will be equipped with smart overload relays as per Lycopodium engineering standard. The LV MCCs will supply power to the low voltage motors, low voltage variable speed drives and low voltage distribution boards.

17.4.5 Earth Fault Protection

Earth leakage protection will be applied to circuits with GPOs (General Purpose Outlets, i.e. power points) and for lighting circuits.

17.4.6 Fire Protection

The HV switchroom, LV switchroom and the plant control room will be provided with fire detection systems. Signals from the fire detection system will be wired to the respective fire indication panel (FIP) in the switchrooms and all signals will be monitored by a master fire detection panel (MFIP) in the security / emergency services control room. Each FIP will also be wired to a local siren with beacon to warn staff of the fire detection. The same fire and smoke activation alarm signals detected by the fire detection system will also be monitored in the plant control room.

17.4.7 Cable Ladders

Cable ladders will generally be laid horizontally, with vertical ladders used in areas where spillage may occur. Hot dip galvanised / epoxy painted type cable ladder will be used. However, stainless steel cable ladders will be used in areas exposed to corrosive particles / fumes / mists.

Cables of different voltage groups will be installed on separate ladders. If they need to be installed on the same ladder, then complete segregation of the ladders will be provided. Ladder routes will follow the mechanical pipe racks.

17.4.8 Cables

Direct buried cables will be provided with armouring.

Cables up to 16 mm^2 will be PVC insulated and bigger cables will be XLPE insulated.

VSD cables will be multiple core 3 x phase and 3 x earth cables symmetrically laid out within an overall shielded cable.

Cables within the plant area will be installed above ground, on cable ladders and follow the mechanical pipe racks wherever possible.

17.4.9 Lighting

All lighting around the process plant will be design in a fit for purpose manner to suit the operational requirement for each area.

17.4.10 Earthing System and Lightning Protection

The earthing system within the plant will be designed in accordance with relevant Australian Standards (i.e. AS 3000, AS 3007 and Australian Communications Authority (ACA)). The following method of system earthing will be implemented at various voltage levels:

11 kV	Earthed via earthing transformers
415 V	Solidly earthed system / Multiple Earthed Neutral (MEN) / T-N-C-S
Note:	T – Terre (French for earth)
	N – Neutral
	C – Combined
	S - Separate

Lightning protection will be provided for all plant building structures. Plant substations / switchrooms and structural high points will be fitted with lightning masts of sufficient height and quantity to ensure that all exposed points will be covered as per 'Rolling Sphere Method' of AS 1768. Lightning protection systems will have their own independent earthing electrodes and will be interconnected with the power earthing system.

17.5 Control System

17.5.1 General Overview

The general control philosophy for the plant will be one with a low level of automation and remote control facilities. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room, which will be located adjacent to the plant offices, will house two PC based operator interface terminals (OIT). Both of the OITs will act as the control system supervisory control and data acquisition (SCADA) servers as well as configuration / operator stations. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

The process control system that will be used for the plant will be a programmable logic controller (PLC) and SCADA based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

In general, the plant process drives will report their ready, run and start pushbutton status to the PCS and will be displayed on the OIT. Local control stations will be located in the field in proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) pushbuttons which will be hard-wired to the drive starter. Plant drives will predominantly be started by an operator in the field after inspecting the local equipment.

The OITs will allow drives to be selected to Auto, Local, Remote, Maintenance or Out-of-Service modes via the drive control popup. Statutory interlocks such as emergency stops and thermal protection will be hardwired and will apply in all modes of operation. All PLC generated process interlocks will apply in Auto, Local and Remote modes. Process interlocks will be disabled or bypassed in Maintenance mode with the exception of critical interlocks such as lubrication systems on the mill.

Local selection will allow each drive to be operated by the operator in the field via the local start pushbutton which is connected to a PLC input. Remote selection will allow the equipment to be started from the control room via the drive control popup. Maintenance selection will allow each drive to be operated by maintenance personnel in the field via the local start pushbutton which is connected to a PLC input. A PLC output will be wired to each drive starter circuit for starting and stopping drives. Status indication of process interlocks as well as the selected mode of operation will be displayed on the OIT.

Vendor supplied packages will use vendor standard control systems as required throughout the Project. Vendor packages will generally be operated locally with limited control or set-point changes from the PCS system. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

A remote control panel will be utilised for the following vendor packages:

- Pre-leach thickener rake mechanism.
- Centrifugal gravity concentrator.
- ILR, intensive cyanidation reactor unit.
- Flocculant mixing system.
- Potable water treatment system.
- Compressed air system.
- Instrument air dryers.

17.5.2 Drive Controls

Each drive will be supplied from a Motor Control Centre (MCC) switchboard. All drive control circuits will be hardwired. In the field, each drive will be provided with a stop / start push-button control station.

Variable Speed Control units will be Variable Voltage Variable Frequency (VVVF) utilising Pulse Width Modulated (PWM) technology. The drive will be mounted in a free standing cubicle. The drive will be provided with an integral control panel for programming and operation at the VVVF unit for commissioning and emergency running.

17.5.3 Control Loops

Regulatory control loops will be provided around the milling, pre-leach thickening and tails disposal circuits.

There will be two modes for loop controlled set points available in the OIT. These are 'Loop Auto Mode' and 'Loop Manual Mode'. In Loop Auto Mode (analogous to cascade control), the set point will be predominantly controlled by the applicable 'master' PID loop (e.g. for thickener underflow pumping control the bed pressure PID controller output will supply a set point for the thickener underflow flow control loop which ultimately controls the speed of the thickener underflow pump). In Loop Manual Mode, set point may be entered manually from the loop set point pop up in the OIT.

Where required, analogue set points from the PCS to vendor supplied control panels (e.g. desired rectifier output voltage) can be done either via the OIT or via vendor control panels. The selection of whether the set point will come from the OIT or from the vendor control panel will be made in the OIT as a PLC digital output.

17.5.4 Crushing Circuit

The crushing circuit will be a vendor supplied package with its own control system and MCC which will house all the motor starters for the area. The crushing circuit will be manually controlled with warning and status indicators being displayed on the MCC. No allowance has been made for remote indication back to the PCS.

17.5.5 Milling

A level detector in the surge bin will indicate the availability of mill feed at all times.

Ore feed to the ball mill will be measured on the mill feed conveyor with a belt weigh scale. The speed of the variable speed reclaim belt feeder which recovers ore from the surge bin will be modulated to achieve the desired mill feed rate. Process water addition to the ball mill feed will be ratioed to the ore feed rate in order to maintain a relatively constant mill feed slurry density to optimise grinding efficiency. Process water addition to the mill discharge hopper will be automatically controlled to maintain a constant cyclone feed density.

Flowmeters will be provided on the mill feed and mill discharge hopper water addition lines. A nucleonic density gauge located on the cyclone feed line will indicate the cyclone feed density.

A level element will measure the slurry level in the mill discharge hopper. A PID controller will be used to control the speed of the duty mill discharge pump to maintain the required cyclone pressure. The cyclone pressure, hopper level and pump speed (output) will be trended and recorded by the PCS.

17.5.6 Pre-leach Thickening

Pre-leach Thickener Control

Control of the pre-leach thickener rakes will be carried out using the vendor supplied thickener control panel and the Plant Control System. Rakes will be controlled to automatically lift when rake torque exceeds a pre-determined set point and will automatically be lowered in stages as rake torque decreases.

The pre-leach thickener rake torque, rake height and hydraulic oil temperature will be trended and recorded by the PCS.

Pre-leach Thickener Underflow Control

The speed of the variable speed pre-leach thickener underflow pump will be automatically controlled by selecting one of the following regulatory control loops:

- Bed Pressure Control - A bed pressure transmitter in the pre-leach thickener will measure the bed pressure of the slurry within the thickener and a bed pressure PID controller will be used to send a remote set point to the flow controller which controls the speed of the pre-leach thickener underflow pump.

- Underflow Density Control - A density element will measure the slurry density of the thickener underflow slurry and a PID density controller will be used to send a remote set point to the flow controller which controls the speed of the pre-leach thickener underflow pump.

Output from the PID controller selected for thickener control will cascade to the flow controller which will regulate the speed of the pre-leach thickener underflow pump to maintain the set point. A minimum flow rate will be programmed into the flow controller to ensure that sanding of the pre-leach line to the CIL does not occur.

The pre-leach thickener underflow slurry density, volumetric flow rate, calculated mass flow rate and pump speed (output) will be trended and recorded by the PCS.

Pre-leach Thickener Bed Level and Flocculant Dosing Control

The slurry bed level within the pre-leach thickener will be measured using an ultrasonic bed level detector. A PID bed level controller will send a remote set point to the flocculant PID flow controller which will adjust the speed of the pre-leach thickener flocculant dosing pump to maintain the flow set point.

The pre-leach thickener bed level and flocculant flow rate will be trended and recorded by the PCS.

17.5.7 Leaching

All aspects of the CIL circuit, with the exception of the pH control loop, will be manually controlled or measured. The main functions of the operators will be as follows:

- pH - measured on pulp in Tank 1 by a hand-held pH meter that will be used to cross-check the automatic pH meter.
- Dissolved oxygen level - measured in tanks by hand-held dissolved oxygen meter.
- Cyanide level in leach solutions measured on filtrate by titration.
- Carbon level - measured by volume from dip sample.
- Collection of composite head and tail samples - for assay in laboratory.
- Observation of slurry level in tanks and condition of intertank screens.
- CIL feed density - measured by Marcy scale.
- Monitor performance of carbon recovery screen.
- Monitor the leach feed grind size and provide feedback to the grinding circuit control.
- Temperatures, vibration, etc. of mechanical equipment.

The pH of the slurry in Leach / Conditioning Tank or CIL Tank 1 will be continuously measured by a submerged pH probe which can be selected to control the ratio set point on the lime addition controller. The pH probe will be relocated as required to ensure that if CIL Tank 1 is bypassed pH measurement is still available.

17.5.8 Desorption

All aspects of the Zadra desorption and electrowinning circuit will be manually controlled with the exception of the acid rinse pump, caustic and cyanide dosing pumps and the strip solution heater:

- The acid rinse pump will operate on a timer to prevent excessive water usage.
- The positive displacement caustic and cyanide metering pumps will operate on timers to ensure the correct dosage of reagents to the strip solution tank.
- The eluate temperature will be measured by a thermocouple located in the eluate line to the elution column. A temperature controller mounted in a control panel adjacent to the elution heater will modulate the input of energy from the heater to maintain the desired eluate temperature.

The critical elution process parameters (temperature and elution column pressure) will be trended by the PCS to assist the operator in the monitoring of the process.

Electrowinning will be manually initiated. An operator will be required to perform the following duties:

- Supervision of electrowinning cell operations.
- Sample collection.

The stripping plant will strip up to seven batches of carbon per week.

Electrowinning parameters (voltage, current) will be displayed on the OIT for recording and trending purposes.

The carbon regeneration kiln temperature will be controlled via a temperature controller mounted in a local control panel. The speed of the screw feeder will be varied to ensure the correct feed rate.

17.5.9 Tailings Disposal

Carbon safety screen underflow slurry will be discharged into the tailings hopper at a controlled rate. A level element will measure the slurry level in the tailings hopper. A PID controller will be used to control the speed of the duty tailings pump set to maintain the required flow rate cascaded with the level control loop.

17.5.10 Services

The PSA oxygen plant will be controlled by a vendor supplied PLC. The rate of oxygen addition to each CIL tank will be controlled manually. Oxygen pressure and flow from the main storage receiver will be trended and recorded by the PCS.

Sensors will detect the water levels in the raw, water and raw water tanks. High and low level alarms will register in the control room.

The river pumps and bore pumps will be controlled manually.

The potable water treatment system will be controlled from a local control panel. System faults will be alarmed on the PCS. Sensors will detect the water levels in the potable water tanks.

Low pressure alarms will be provided for plant and instrument air, process water, raw water, potable water and fire water.

17.5.11 Control Interfaces

Main Control Room

The main control room will contain an operator desk and the Operator Interface Terminals (OIT) and associated computers.

An audible alarm will be provided. Alarms will be accepted via the OIT keyboard.

Operator Interface Terminals

Operator interface terminals will be provided in the main plant control room mounted on top of a standard table.

SCADA software running on the OITs will allow control, monitoring and trending of the process plant. Communication between PLC's and the supervisory SCADA software will be via fibre optic network to ensure system performance and reliability.

17.6 Metallurgical Accounting

Weightometers will be located on the indicated conveyors:

- Crushing screen feed conveyor will measure crushing screen feed tonnage.
- Surge bin feed conveyor will measure crushed ore tonnage.
- Mill feed conveyor will measure mill feed tonnes.

The tonnage of crushed ore reporting to the dead stockpile can be estimated from the difference between the crushed ore tonnage and the mill feed tonnes.

A dedicated electrowinning cell will be provided for recovery of the gold leached by intensive cyanidation of gravity concentrate. Recovered gravity gold sludge can be smelted separately to determine the gravity gold component of the ore.

Routine manual sampling of the leach feed stream and the final leach tailings will ensure reliable composite shift samples for leach head grade and tails solution and residue grades. Inclusion of the gravity gold component will allow estimation of the plant head grade.

Density and flowmeters on the leach feed and tailings lines will allow the dry tonnage of solids pumped to the CIL and TSF to be determined as a cross check on the mill feed tonnage determined from the mill feed weightometer. In conjunction with the leach feed and tails samples, the mass flow measurements will allow the gold recovered in the CIL to be calculated.

Regular gold 'in circuit' surveys will allow reconciliation of precious metals in feed compared to doré production.

Water supplied and used in the various areas will be continuously monitored.

Reconciliation of the amount of reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates of cyanide and elution reagents and diesel flow rates to unit operations will be measured (L/min) and accumulated (m^3) using flowmeters.

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18.0 PROJECT INFRASTRUCTURE

18.1 Site Development

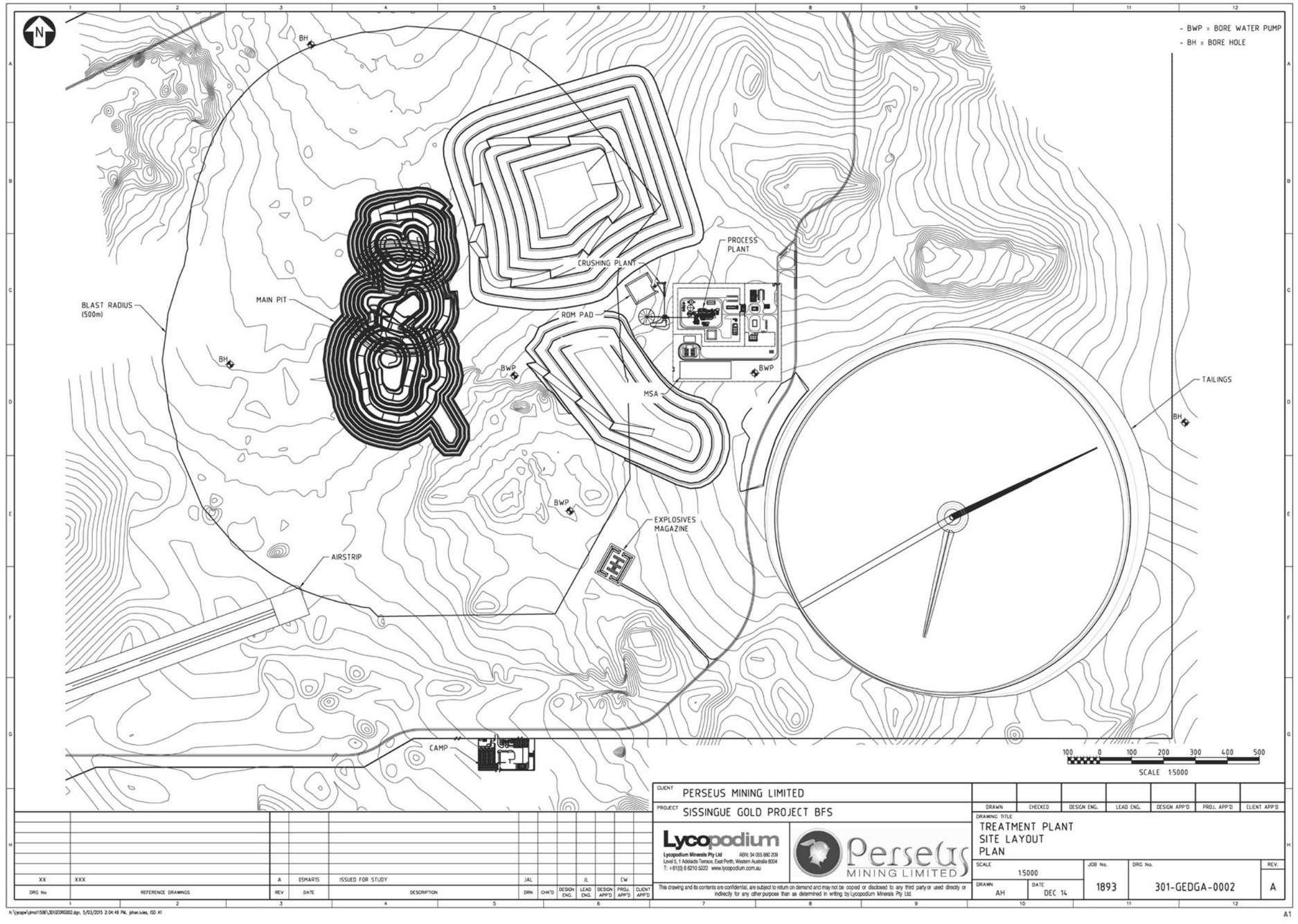
The overall site development plan is shown in Figure 18.1.1. The drawing shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, airstrip, mine services area, mine open pit and mine waste dump.

The process plant and tailings storage facility are located on the eastern side of the Sissingué open pit, just outside of the 500 m blast zone. The accommodation camp and airstrip are located south/southwest of the process plant.

The main access road approaches the site from the west and the layout provides easy access for personnel and material movements.

The site as a whole will be fenced to clearly delineate the mine area, prevent animal access and deter access by unauthorised persons. Road access into the fenced area will be through a manned checkpoint. Security fencing will surround the accommodation camp and general site infrastructure. Monitored high security fencing will surround the process plant.

Figure 18.1.1 Overall Site Development Plan



18.2 Roads

18.2.1 Main Access Road

The existing road, approximately 23 km long, between Tengrela and Sissingué will be upgraded to a six metre wide gravel, all weather, and free draining carriageway to provide access for the delivery of equipment, materials and services to the site. Culverts will be installed as appropriate on creek crossings. Fill material for the upgrade will be obtained from excavation alongside the road where possible.

The existing road is trafficable but has sections which are prone to failure in the wet season. Alignment is curvilinear and appears to have been formed historically through usage. No information is available on vertical alignment but terrain is relatively flat and it is likely to be similar to the horizontal. The estimated design speed is in the region of 70 to 80 km / hour with slower sections based on tighter alignment. The road passes directly through the settlements of Tialaka, Basso and Sissingué. A bypass road approximately 1.8 km long will be constructed around the western side of Sissingué up to the main entrance gate of the Project site.

Treatment of the access road will vary over different sections from that of a light regrading to full construction. Where possible the alignment is to be improved and this will also include vertical geometry once level data is available. Details of the existing road are sparse thus assumptions have been made based on the available information.

The adopted design parameters for the main access road is summarised in Table 18.2.1 below.

Table 18.2.1 Main Access Road Design Parameters

Parameter	Value
• Pavement width	6 m (2 x 3 m lanes with no shoulders)
• Crossfall	3%
• Road surface	Unsealed gravel
• Pavement	300 mm base course
• Design vehicle speed	80 km/h
• Minimum radius	300 m desirable
• Superelevation	4% (5% maximum)
• Turning vehicle at junctions	Semi trailer

18.2.2 Plant Roads

The main access road will continue from the entrance gate along the northern site perimeter fence for a further 4.2 km up to the process plant.

Plant internal roads will provide access between the administration area, plant site facilities, fuel storage, power plant and mine services area. These roads will generally be 6 m wide and will be constructed flush with the bulk earthworks pad to ensure that storm water sheet flow is achieved across the site, thereby avoiding the need for deep surface drains and culvert crossings within the plant area.

Access to the accommodation camp, located approximately 2.2 km south west of the process plant, will be by a 6 m wide gravel road. Heavy vehicle traffic on the camp access road will be much lower than on the main access road and a reduced pavement thickness of 150 mm has been adopted in the design.

The airstrip is located approximately 2 km west of the accommodation camp. A 4 m wide road will provide access for the transport of personnel from the airstrip to the camp.

18.2.3 Access Tracks

A number of new tracks will be constructed to access infrastructure such as the tailings storage facility, sediment control structures and water bore pumps remote from the plant site. The access tracks will be cleared and graded natural earth tracks. Exact routes will be determined during construction of the Project to best fit local terrain and vegetation density.

18.3 Airstrip

A 1,200 m long x 20 m wide all weather gravel airstrip will be provided for secure transport of bullion to Abidjan for on-shipment to a refinery. The airstrip can also be used for emergency medivac.

The air strip has clearance zones of 100 m at the ends and 35 m on the sides. No allowance has been made for lighting for night landings or takeoffs.

The airstrip will be located to the southwest of the plant.

18.4 Tailings Storage Facility

18.4.1 Introduction

The TSF design, site sediment control and water balance were undertaken by Worley Parsons.

Site hydrogeological studies were undertaken prior to the 2012 report.

The Tailings Storage Facility (TSF), located about 300 m south east of the plant, will receive the tailings slurry at about 43% solids (oxide) and 48% solids (primary). The TSF will also be used to store water recovered from the open pit and Bagoe River. Return water from the TSF will therefore provide 100% of the plants total process water make-up requirements.

The TSF has been designed for a storage life of in excess of five years with a nominal diameter of 1,100 m occupying a storage surface area of approximately 100 ha. The construction of the TSF will be executed two stages. Stage 1, which provides storage for 4.5 Mt, will occur concurrently with the pit pre-strip and plant construction work. Stage 2 will occur in Year 4 and will provide storage for up to a further 4.2 M, or 8.7 Mt in total.

18.4.2 TSF Type and Location

The TSF will be located approximately at grid point 808500 mE and 1153500 mN and adjacent to the south east of the process plant.

The TSF will comprise a circular, single cell, facility and will be surrounded by mine waste to form an Integrated Waste Landform (IWL). The facility is a geotechnically engineered clay structure designed to contain tailings and minimise seepage losses.

The embankment will comprise a compacted clayey mine waste zone constructed in one stage on the upstream (inner) side of an outer traffic compacted mine waste zone.

A toe drainage system is located at the downstream toe of the perimeter embankment below RL 380. Any seepage collected within this drain will be discharged into an HDPE lined sump with the water recovered returned to the TSF. The volume of water recovered is expected to be small and a solar pump fitted with low level and high level float switches will be used to evacuate the sump.

The TSF has been designed with a storage capacity for 5.5 Mt over a life of five to six years although, additional wall raises could provide future capacity over 15 Mt.

The TSF will be constructed in two stages using mine pre-strip materials as part of the construction. The Stage 1 embankment crest will be constructed to RL 385.5. On the south western side of the TSF, where the natural ground is above RL 385.5 there is a 1 m high containment embankment to allow tailings deposition from this area of the TSF. Stage 2 lifts the crest to RL388.

18.5 Sediment Control

A clay lined sediment pond is provided to the north-west of the TSF to contain solids washed by rainfall from the processing facilities.

18.6 Water Balance

A life of mine water balance model was developed. The objectives of the modelling were to:

- Develop an understanding of the site water network.
- Determine the water surplus / deficit of the site.

- Determine how much, if any, water should be taken from the Bagoe River.
- Determine the water management requirements for the project.

Key inputs to the water balance model include:

- An estimated 80 m³/h water will be available from pit dewatering.
- The TSF has been designed as a water storage facility.
- Water will be available from the Bagoe River if the water stored on the TSF falls below a nominated amount.

The water balance determined that the volume of water in the TSF is expected to increase consistently over the life of the project for the low average and wet scenarios modelled. The site water balance and storage volumes will therefore need to be carefully monitored and managed over the life of the project.

The model is sensitive to the amount of water coming from the main pit dewatering activities. This is a key risk especially if pit dewatering falls below 30 m³/h.

In the worst case approximately 40 m³/h of mine dewatering may need to be discharged from the site rather than sent to the TSF.

The site is expected to have sufficient water without abstraction from the Bagoe River, however river abstraction has still been incorporated into the design as a 'fall back' position for water supply.

Providing sufficient water can be accumulated in the TSF pond prior to commencing mill operations (a minimum of 50,000 m³) there will be adequate water to run the process plant even if production commences during the dry season.

18.7 Power

The closest connection point to grid power is the 225 kV substation at Boudiali. As a connection is dependent on providing additional infrastructure to connect Tengrela to the national grid this is not an attractive option. Power will therefore be generated by a dedicated project power station.

18.7.1 Power Generation

Power supply to the site will be from a diesel fuelled power station located adjacent to the fuel storage facility. The power station will supply the main HV switchroom inside the processing plant from which power will be distributed.

The proposed configuration of the power station is:

- 4 x 1.6 MW 415 V high speed generators (3 duty, 1 standby).
- Step up transformers 415 V / 11 kV.

- 11 kV switchroom.
- Black start generator.
- The 2.5 MW mill at the process plant is by far the largest single load on the power station. The mill has been specified with a VVVF (variable speed) drive to provide a 'soft start' capability to reduce the load surge during start-up and minimise the need for 'spinning reserve' in the power station.

18.7.2 Electrical Distribution

The electrical system is based on 11 kV distribution and 415 V working voltage.

The 11 kV feeder from the power station will feed the site distribution 11 kV switchboard.

For the process plant the 11 kV supply will be stepped down from 11 kV to 415 V at the switchrooms using three separate 11 kV / 433 V distribution transformers fed from the HV distribution board.

Switchrooms in the crushing area and process plant area will house the 415 V motor control centres (MCCs). Outdoor control panels and distribution boards have been allowed for plant lighting and small power distribution and UPS power distribution.

The 11 kV supply will be stepped down to 3.3 kV for the 2,500 kW mill drive.

Power will be supplied to the mine services area with the mining contractor responsible for distribution to their facilities.

Approximately 6 km of 11 kV overhead power line between the power station and various remote facilities (TSF pumps, bore pumps, etc.) has been allowed. Three off 11 / 0.433 kV 1,500 kVA, two off 11 / 0.433 kV 500 kVA, and three off 50 kVA transformers are required at the various sites.

Accommodation camp power will be supplied from a 750 kVA kiosk substation fed from the 11 kV overhead line and will have a 300 kW emergency backup generator.

18.7.3 Electrical Buildings

To minimise site works electrical equipment will generally by procured pre-installed in prefabricated switchroom buildings:

- Two HV switchrooms for the power station and process plant, the latter adjacent to the plant LV switchroom).
- Two LV switchrooms for the crushing and process plant areas.
- A plant control room with the control system equipment pre-installed and installed adjacent to the plant offices.

The electrical buildings will be installed with air-conditioners and suitably sealed to prevent ingress of dust.

18.7.4 Transformers and Compounds

All the 11 kV / 433 V distribution transformers will be of ONAN cooling configuration and vector group Dyn11.

Fire rated concrete walls will be constructed around the pad mounted transformers.

Outdoor rated 11 kV / 433 V kiosk substations will be used to provide power to the mine services area and accommodation camp.

18.8 Potable Water

The potable water demand for the Project has been calculated on a per capita usage basis and is summarised in Table 18.8.1 below.

Table 18.8.1 Potable Water Demand

Area	No. of Personnel	Usage (L/person/day)	Demand (m^3/day)
Accommodation Camp	140	300	42
Plant	240	70	17
Other	-	-	10
Total			**69**

Raw water will be sourced from bores around the site and pumped to a raw water storage tank located adjacent to the potable water treatment facility at the process plant.

A vendor package modular potable water treatment plant including filtration, ultra-violet sterilisation and chlorination will be installed. Potable water will be stored in the plant potable water tank and will be reticulated to the plant buildings, site ablutions, safety showers and other potable water outlets.

Potable water for the accommodation camp will be pumped from the water treatment plant at the plant site to a water storage tank at the camp. The tank will provide two day reserve storage in the event of supply line disruptions. Water will be delivered into the camp reticulation system using a constant pressure variable flow pump system. The pump skid will include a UV disinfection unit to provide additional security against contamination.

18.9 Sewage

Effluent from all water fixtures in the process plant, mine services area and accommodation camp will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewer pump station from where it will discharge via a pressure main to a vendor package sewage treatment plant system located at the process plant.

Treated effluent will be discharged into leach drains or the plant tails hopper. Treatment pant sludge will be suitable for direct landfill burial in unlined pits.

18.10 Solid and Hydrocarbon Wastes

Wastes will be sorted and reused or recycled as far as the limited access to recycling facilities allows.

Waste lubricating oils will be returned to the supplier for recycling.

General solid wastes will be deposited into a landfill at the toe of the mine waste dump and promptly covered to deter vermin and scavengers.

Materials such as cyanide packaging will be decontaminated and buried, under supervision, on site beneath mine waste to prevent unauthorised use.

18.11 Communication System Infrastructure

Internal communications and IT services shall be via a site wide fibre optic network.

One of the local mobile phone providers will be contracted to install facilities on site and provide a link into the local, national and international telecommunication network.

A radio network will be established with dedicated operational, security and emergency channels.

A local ground station will be installed to provide global satellite voice and data connection.

18.12 Fuel Supply

Bulk fuel storage for the power station and mining fleet will be owned and managed by the fuel supplier.

18.13 Explosive Storage and Handling

Explosive storage and handling will be the responsibility of the mining contractor. Perseus shall ensure that the contractor's facilities meet the required safety and security standards.

18.14 Security and Fencing

Site security is based on concentric lines of fencing / control.

The entire site will be enclosed within a patrolled fence line to discourage casual entry. The main point of entry will be where the main access road from Tengrela enters the site. This point of entry will be provided with a gate and manned security post.

Main facilities within the site will be enclosed by a further line of patrolled fencing with manned access control points. Entry into the administration accommodation and similar areas will require a mine identity card and/or proof of legitimate business beyond that point.

The process plant itself will be enclosed by a double line of security fencing monitored by closed circuit cameras. Entry will be via a single monitored security post and will be strictly controlled. Exit from the plant area will be subject to a search of vehicles, toolboxes and 'pat down' and/or metal detector search of all persons.

Access to the goldroom within the plant will be restricted and strictly controlled. Extensive camera surveillance will be installed and all personnel allowed into the area will be accompanied and monitored by members of the security team. Persons leaving the area will be subject to a comprehensive search of themselves and any tools or equipment leaving the building.

18.15 Site Buildings

Site buildings will be 'fit for purpose' industrial type structures. The workshop and warehouse will be constructed of a concrete slab on ground with structural steel frame and metal cladding. Offices and amenity buildings will be prefabricated structures.

The following facilities will be located in a fenced area adjacent to the process plant:

- Gatehouse with turnstile and entry boom gate control.
- Weighbridge for control of fuel and consumable deliveries.
- Main administration building.
- First aid / clinic.
- Administration junior staff mess.
- Warehouse and stores.
- Contract laboratory.
- Power station.
- Fuel storage facility.
- Mine services area (facilities to be provided by the mining contractor).

The following buildings will be located inside the process plant high security area:

- Security access building and change room including laundry.
- Plant offices, training room, junior staff mess and ablutions.
- Plant workshop including small store, welding bay and overhead crane.
- Engineering offices and ablutions.

- Reagent storage area.
- Goldroom.

18.16 Workforce Accommodation

18.16.1 Permanent Accommodation Camp

The accommodation camp will be located approximately 2.2 km south west of the process plant and will provide accommodation for salaried and security staff not originating from the local area.

The camp will consist of the following:

- Three bedroom general managers house.
- 48 modular 2 bedroom units with en-suite.
- Two barrack style units with accommodation for 20 people in each. A shared ablution block will be located at the end of each unit.
- Dry mess with food storage and preparation, kitchen and dining facilities.
- Wet mess and camp administration office with a bar, TV area, store room and camp administration offices.
- Combined laundry building / gymnasium incorporating ablution facilities for camp staff.

Costing has been based on the use of several modular two bedroom units and an ablution block already purchased by Perseus and available on site.

18.16.2 Temporary Construction Accommodation

The existing exploration camp supplemented with the available two bedroom units will be used for pioneer accommodation pending early completion of elements of the permanent camp.

The permanent camp will then be used for Perseus and EPCM contractor staff with any surplus capacity made available to senior contractor personnel if available.

Contractors will be required to provide their own accommodation for their construction workforce. An area will be set aside for temporary contractor camps adjacent to the construction site. Alternatively some contractors may choose to source temporary accommodation in nearby towns and villages.

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19.0 MARKET STUDIES AND CONTRACTS

19.1 Gold Market

A gold price of US$1,200/oz based on analyst consensus has been used for the Mineral Reserve estimate and the economic analysis. Silver levels are so low that it has no appreciable impact on either the Mineral Reserve estimation or economic outcomes and has been ignored in both cases.

Sissingué will produce gold doré which is readily marketable on an 'ex-works' or 'delivered' basis to a number of refineries in Europe and Africa. There are no indications of the presence of penalty elements that may impact the price or render the product unsalable.

Payment terms are widely available in the public domain and vary little from refinery to refinery. Perseus has a contract in place for the refining of gold doré from its Edikan operation in Ghana and it has been assumed that Sissingué doré will attract similar terms.

19.2 Contracts

Before the commencement of operations, contracts will be entered into for mining, supply of all consumables (including fuel and reagents), camp management and catering services, freight delivery and refining of gold.

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20.0 ENVIRONMENTAL STUDIES, PERMITTING, SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Studies, investigations and research, conducted by consultants from CECAF International commenced in 2010 and culminated in the publishing of the Environmental and Social Impact Assessment Final Report in December 2011. Although the ESIA report was based on the earlier concept of a 1.6 Mtpa operation it remains relevant for the scope of the proposed 1.0 Mtpa operation.

20.2 Baseline Studies

Baseline studies for the ESIA collected data on:

- Climate.
- Air quality.
- Sound levels.
- Hydrology.
- Hydrogeology.
- Chemical and biological status of surface and underground waters.
- Fauna and flora.
- Soil classification.
- Archaeology.
- Cadastral surveys.
- Economic and social conditions.
- Community health.

The studies were conducted by Ivorian consultants using appropriate scientific methods with data systematically collected on that basis.

Sampling exercises took place in April - May and July - August 2010 in order to correspond respectively with the dry and rainy seasons. This ensured that the study covered both seasons and took into account the seasonal climate changes. There were no environmental issues identified that would impact on the project activities.

The project area is characterized by two seasons, the rainy season from mid-April to end October and the dry season that covers November to mid-April. During the rainy season, the biggest rainfalls occur from July to September with the maximum (370 mm) in August and some annual rainfalls between 1,000 mm and 1,600 mm. There is a total absence of precipitation during December, January and February. Evaporation is very high in the dry season with a maximum in January (255 mm) corresponding to the Harmattan. Evaporation is less (<80 mm) during the rainy season. The relative average humidity varies from 31% to 77% with a minimum in December and January and a maximum from June to September. The monthly maximum and minimum average wind speed is 3.1 m/s (11.16 km/h) and 2.1 m/s (7.56 km/h) respectively.

The area is entirely rural with no industrial source of atmospheric pollution other than motor vehicle exhausts. However every dry season the Harmattan winds transport enormous quantities of fine dust particles to the region. The results of the chemical pollutant (COVT, CO, SO2 and H2S) analysis illustrates that the registered pollutant levels in the area are extremely weak.

The environment in the study area is noisy during the daytime, but the registered sound levels are generally below the 70dB (A). The daytime sounds are mainly caused by vehicle and motorbike engines and the highest values recorded range between 60dB(A) and 80dB(A). After 20.00 hrs, the noise levels fall, with some values around 50 dB (A).

The surface water in the project area shows different characteristics during the wet and dry seasons. The values of pH obtained are slightly basic (7.23-8.58). The groundwater is characterized by a weak hardness and mineralisation and a pH close to neutral. This water contains weak nitrate concentrations and nitrite traces. Iron, manganese and aluminium were found to have concentrations lower than the potable standards of the World Health Organisation (WHO). The microbiological quality of ground water revealed general contamination of well water in the project area, yet compared with the surface waters, the ground water is better quality. The surface waters are subjected to more pollution as a result of direct exposure to human activities. The flow analysis at the hydrometric stations in the area shows only one climax period, in September. The average annual flow of the Bagoé River in Papara, to the north of the Project site, is 33.11 m^3/s for a specific flow of 0.0037 $m^3/s/km^2$.

Sixty phytoplankton taxa were identified (45 taxa in the dry season and 30 in the rainy season). The class of the Conjugatophyceae is the best represented with 23 taxa. The least represented class is the Dinophyceae with one species. Only four phytoplankton species inventoried in the Bagoé River are often associated with eutrophic conditions, which indicate that the river is not polluted. Only Diaphanosoma excisum was collected during both sampling periods. During the two study periods, six benthic organisms belonging to the taxonomic groups of insects, myriapods and worms were identified. The low diversity is explained by the sandy substratum of Bagoé River that is incompatible with the development of these organisms because of its high porosity. A total of 57 fish species belonging to 17 families and 37 genera were identified. Quantitatively, 1,008 fish specimens with a total weight of 23045.49 g were collected during the two study periods.

The genesis of the soil depends on several factors including regional climate, topography, parent material and vegetation type. The topographical relief is low with a set of plateaus and hills above. The natural drainage system is very loose and the Bagoe River has a few tributaries and a flood plain, which is rather large in some sectors. The natural vegetation is the savannah, the undergrowth sparse. The main landscape units identified in the study area are divided into three large sequence sets following the topography of the environment and nine main biotopes were identified on the project site, being clear forests, forest galleries, arboreous savannahs, arbustive savannahs, wooded savannah, grassy savannah, pools, lakes and fallows / crops.

The flora study identified 417 plant species belonging to 269 genera and 84 families. The most represented genera are Ficus (12 species) and Combretum (9 species). The most diversified families are the Poaceae (62 species) and Fabaceae (32 species). From the registered species, two are reported as endemic to West Africa (GCW) according to the lists from Aké-Assi (2001 - 2002). Those are Anthostema senegalense (Plot 4) and Moghania faginea (Plots 2 and 13). The Sissingué site flora as inventoried is relatively poor in endemic species. Fifty-seven species are used in traditional medicines preparation and twenty-three species are used as food.

The captured fauna species are monospecific in both forest and savannah habitat types. Only the species Praomys rostratus was collected in the forest whilst two (Dasymys incomptus and Tatera valida) were collected in the savannah. With 139 bird species (18.39% of the bird registered in Côte d'Ivoire), the study area supports a number of species, although none of concern (UICN, 2008). Three West African endemic species, being the youyou parrot (Poicephalus senegalus), the violet Touraco (Musophaga violacea) and the Barbarous Gonolek (Laniarus barbatus) were inventoried in the dry season.

The mining lease covers an area of about 44,600 hectares and there are approximately 1,219 agricultural plots on the lease. Those plots cover a total surface of 4895.18 hectares, representing about 10.98% of total surface area. The remaining part of the area is covered with fallow, arbustive, arboreous savannahs and some forest reserves (448.6ha). The arbustive savannah largely dominates the project area. Local farmers burn clearings to create their plantations. The agricultural system mainly uses familial manpower and carts pulled by oxen. During the rainy season some food crops (maize, sorghum, ground nuts etc.) are sowed to help meet the family food needs and cashew seedlings are also planted. These are supplemented by diverse vegetable crops including peppers, okra, garden eggs etc.

The Department of Tengrela is located in the extreme North of Côte d'Ivoire. It is bordered to the north, east and west by the Republic of Mali, in the south by the Department of Kouto, in the southeast by the Department of Korhogo and in the southwest by the Department of Odienné. The Tengrela Department covers a surface about 2,200 km^2 and comprises three sub-prefectures, being Tengrela, Kanakono and Débété. According to the data from the Population and Habitat General Census (RGPH, 1998), the population in the Tengrela Department is estimated up to 63,644. The main economic activity of the Department is agriculture. Fishing is practiced traditionally on the Bagoé and Mahandianabani rivers and some pastoral dams in the Department. There are five main types of houses being earth mounted geo-concrete houses (41%), hard houses (19.10%), earth beaten houses (20.10%), cement mounted geo-concrete houses (18.36%) and shacks (1.45%). The Tengrela department is equipped with fourteen health clinics of which two are non-functional. There is only one private health clinic. The main morbidity causes are

attributed to twelve ailments which include malaria, malnutrition, chronic diarrhoea, typhoid, urinary bilharzia, buruli ulcer, meningitis and leprosy.

The collected archaeological artefacts characterizing the human activity in the past, concern average Pleistocene and the quaternary one. The archaeological investigations revealed a cultural diversity during prehistoric times in this area and contribute to the enrichment of the national archaeological collection.

20.3 ESIA and Environmental Permits

The ESIA was submitted in September 2011 and was approved by the Agence Nationale De l'Environnement (National Environment Agency) in January 2012. The Environmental Permit was granted in February 2012.

The Project will obtain all other permits and approvals required under law prior to the commencement of relevant project activities. This includes water extraction permits, building permits, road permits, quarry permits and permits pertaining to the transport, storage and use of cyanide and hydrocarbons.

20.4 Management Plans

An early activity of the Project will be the development of comprehensive Environmental and Social Management Plans setting out the standards and outlining procedures and practices that shall be adopted during development and operation of the Project.

A preliminary Closure Plan outlining the closure and rehabilitation of the site was prepared for the ESIA. The total closure cost calculated for the ESIA is USD3,549,420. This includes capping of the tailings storage facility, land forming, spreading topsoil and revegetating areas on site, ripping up any roads no longer required, removing relevant infrastructure and ongoing environmental monitoring programs. Some of these costs will be incurred during operations as the plan is to progressively rehabilitate areas as they become available.

A more detailed Closure Plan will be developed during the early stages of operations, following consultation with relevant stakeholders. This will identify what infrastructure needs to be removed from site and what will remain, determine if there are any preferred revegetation species (e.g. cash crops), and may also determine final uses for the flooded mine pits and other areas on site.

20.5 Potential Impacts and Mitigation Measures

Areas identified in the ESIA report as in need of mitigation measures and the approach to be adopted are summarised below. Environmental monitoring will be undertaken on a regular basis during both construction and operations, and will include monitoring for potential impacts on the surface and groundwater (e.g. water quality, groundwater levels and pumping data), air quality (e.g. dust, noise and air blast), ground vibration, climate (e.g. temperature, rainfall, evaporation and wind), flora and fauna (e.g. revegetation success) and soils. The relevant aspects will continue to be monitored following site closure and decommissioning and this monitoring will continue for approximately two years. Samples requiring laboratory analysis will be sent off site to a suitable external laboratory.

20.5.1 Control of Atmospheric Emissions

Mitigation measures include control of blasting operations, limitation of the speed and density of traffic on unsealed roads and the use of water tankers for dust suppression on haul roads and access roads.

20.5.2 Control of Noise and Vibration

Only blasting operations are predicted to have any impact on surrounding communities. Signage will be posted advising of the dates and times that blasting will take place and a building survey will be conducted prior to the commencement of blasting to monitor potential vibration damage to nearby structures.

20.5.3 Conservation of Water Quality and the Aquatic Environment

All practical measures will be taken to minimise the impact of the Project on surface and ground waters. Major earthworks will be concentrated in the dry season and sediment will be removed from run-off from disturbed areas by appropriate sediment control structures. No process water will be released to the environment and impoundments containing process water, such as the tailings storage facility, will be fenced and signposted. A site water balance will also be developed to monitor water use and predict ongoing water supply requirements.

20.5.4 Impact on Flora and Fauna

Site clearing will be strictly controlled and carried out only when necessary. Topsoil recovered from cleared areas will be stockpiled for future rehabilitation of the site. The hunting and unnecessary disturbance of fauna on the site will be strictly prohibited.

20.5.5 Employee Health and Safety

Written procedures, training and personal protective equipment shall be provided to ensure a safe working environment. A site medical clinic will be provided and appropriately staffed to provide primary health care and emergency trauma treatment.

20.5.6 Cyanide Control

Training and strict controls, including emergency response procedures, shall be put in place to ensure cyanide is safely transported to site and its use does not compromise safety or the environment.

20.5.7 Storage of Residues and Other Wastes

The tailings storage facility (TSF) design complies with local and international standards and the TSF is located close to the process plant. Tailings will be pumped to the TSF, and ponded water at the TSF will be pumped back to the process plant for re-use. Ongoing independent monitoring shall occur during operation and the detailed site Closure Plan will ensure the long term stability of the structure.

Waste rock will be stored on designated waste dumps which have been designed to minimise the visual impact and erosion. Sediment control structures will be installed at the base of the waste dumps if required. The waste dumps will be progressively shaped and rehabilitated.

Other identified waste streams will be managed according to the 'reduce, re-use and recycle' philosophy and will be disposed of in an internationally accepted manner if disposal is the only option available. Wastes may be sent off site for treatment and/or disposal (e.g. waste hydrocarbons), may be re-used (e.g. oil drums, heavy equipment tyres, plastic bulky boxes), may be recycled if economically viable (e.g., scrap metal, paper and cardboard, plastics etc) or otherwise disposed of on site. Dedicated pits will be excavated for the disposal of wastes on site.

20.5.8 Identification and Rectification of Adverse Socioeconomic Impacts

Although the positive benefits of the Project should significantly outweigh the negative, ongoing monitoring will be required to ensure unforseen adverse impacts are identified and addressed. This may relate to issues such as the blocking of access or rights of way, ensuring employment opportunities are offered to local people, and minimising any negative impact of an influx of migrants to the area looking for employment opportunities. As a social partner the company commits to maintaining an ongoing dialogue with stakeholders in the area to identify and address concerns and issues as they arise. The Company may also be required to contribute an amount to a Community Development Fund on an annual basis.

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21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

The overall study capital cost estimate was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this study report. Mine capital costs (developed by RPM and Perseus) are included in the estimate tables below.

Worley Parsons provided quantities for the tailings storage facility and sediment control dam with rates provided by Lycopodium to derive the capital estimate.

All costs are expressed in US dollars ($) unless otherwise stated and based on 1Q2015 pricing. The estimate is deemed to have an accuracy of ±15%.

Where costs used in the estimate were provided in other than US dollars the following exchange rates were used:

- 1 US$ = 1.250 AUD.
- 1 US$ = 11.11 ZAR.
- 1 US$ = 0.633 GBP.

The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Perseus for scope and accuracy.

The methodology used and resulting estimate accuracy are in accordance with AACE Class 3 estimates.

21.1.1 Summary Capital Costs

The capital estimate is summarised in Table 21.1.1. The initial project capital cost was estimated at US$105.98 million.

Table 21.1.1 Initial Capital Cost Estimate Summary (US$, 1Q2015, ±15%)

Main Area	Initial Capital (US$M)
Mining	11.89
Construction Indirects	8.01
Treatment Plant	21.75
Reagents and Services	8.43
Infrastructure & Tailings	22.86
Owners Costs	14.46
EPCM Costs	9.18
Contingency	9.40
Project Total	**105.98**

21.1.2 Mining Capital Costs

Mining costs were prepared by RPM on the basis of contract mining with the contractor establishing their facilities on a site provided and serviced by the Project. Capital costs for contractor establishment on site ($14.2M) are principally recovered through the contractor's rates.

Mining contractor costs pre-dating first ore to the mill ($11.89M), principally being mobilisation costs and early mining costs are capitalised.

21.1.3 Plant and Infrastructure Capital Costs

The capital cost estimate was prepared in accordance with Lycopodium's standard estimating procedures and practices.

The process plant was broken down into unit operation areas with quantity take-offs benchmarked against similar facilities from previous projects to provide the additional scope and level of confidence needed to confirm a DFS level estimate was achieved.

Overall plant layout and equipment sizing was prepared with sufficient detail to permit an assessment of the engineering quantities for the majority of the facilities for earthworks, concrete, steelwork, and mechanical items. The layouts enabled preliminary estimates of quantities to be taken for all areas and for interconnecting items such as piperacks.

Unit rates for labour and materials were derived from responses to BQRs sent to fabricators and contractors experienced in the scale and type of work in the region.

Budget pricing for equipment was obtained from reputable suppliers with the exception of low value items which were costed from Lycopodium's database of recent project costs.

For the accommodation camp, offices, workshops and similar items appropriate budget pricing was obtained from reputable suppliers of similar prefabricated designs with erection / installation costs derived from solicited contractor rates.

For the tailings storage facility and sediment control facilities bills of quantities were provided by Worley Parsons based on their preliminary designs. Appropriate construction rates were then applied to the quantities by Lycopodium.

21.1.4 Contingency

The purpose of contingency is to make specific provision for uncertain elements of cost within the Project scope. Contingencies do not include allowances for scope changes, escalation or exchange rate fluctuations.

Contingency is an integral part of an estimate and has been applied (after careful analysis) to all parts of the estimate, i.e. direct costs, indirect costs, services costs, etc.

21.1.5 Owner's Costs

The following items are included in the Owner's costs:

- Owner's project management team.
- Owner's team costs including project insurances.
- Operating and capital spares holdings.
- First fill and opening stocks of reagents and consumables.
- Cost of preproduction labour, training and operational readiness.

21.1.6 Exclusions

The following is excluded from the overall project capital costs:

- Duties / taxes.
- Project sunk costs (allowance made in the Cash Flow Model).
- Project escalation.

21.2 Operating Costs

21.2.1 Mining Costs

RPM prepared a Mining Specific Cost Model within its XERAS financial modelling software for a Contractor Mining scenario. That is, the battery limits for the economic model is the delivery of ore to the process plant hopper and placement of waste rock. It excludes the estimation of operating and capital costs for non-mining tasks such as:

- ore processing
- environmental management and mine closure
- water management
- tailings management
- on-site infrastructure, including construction of all facilities (such as power and water), process plant, roads and distribution of services
- rehabilitation and mine closure
- off-site infrastructure such as mine access and connection of services.

The key underlying data for the mining cost model is the mining contractor quote developed by DTP Mining on behalf of Perseus for the Sissingué Project. The quotation covers all mining costs related (including capital expenditure, operating expenditure and contractor margins) to the Project, excluding grade control and mine planning. The mining costs are considered to have a ±25% level of accuracy.

Perseus has also provided assumptions for contractor grade control for a consistent comparison.

As the schedule of rates provided by the contractor are based on an outdated mining schedule. RPM applied the schedule of rates provided by the contractor to the study mining schedule. Additionally diesel costs has been added to the rates, assuming a price of US$1.00 litre to determine an overall contractor mining cost as shown in Table 21.1.1.

Table 21.2.1 Adjusted Contractor Estimate (Contractor SOR & RPM New Schedule)

RPM Contractor Estimate						
Item	**Item Description**	**Units**	**Driver**	**Rate**	**Units**	**Total US$M**
1	Mobilisation, Site Est & Demob	No.	1	3,609	US$k	3.6
2	Fixed Monthly Fee	Months	65	512	US$k	33.3
3	Area Preparation	km^2	796	0.88	US$	0.7
4	Drilling	kdrm	172	22.40	US$	3.8
5	Blasting	kholes	21.8	173	US$	3.8
6	Excavate, Load, Haul and Dump	kbcm	10,872	3.63	US$	39.5
7	Stockpile Rehandle	kt	5,525	0.60	US$	3.3
8	Dewatering	Months	65	32.19	US$k	2.1
9	Grade Control	ore kt	5,525	0.24	US$	1.3
Total						**91.4**

Note – Inclusive of grade control assumption provided by Perseus.

Over the life of the Project, this equates to an average cost of $16.55 per ore tonne or $8.42/bcm or $3.95/tonne rock.

Of the $91.4M, $3.6M of mobilisation and $7.45M of pre-stripping is capitalised.

Over the life of mine in addition to the above costs, Owners mine management and supervision costs total $5.99M of which $0.53M is capitalised during pre-stripping.

21.2.2 Operating Costs Process and Administration

Process plant operating costs for the Sissingué Gold Project have been developed jointly with Perseus.

The operating costs have been compiled from information sourced by Lycopodium and Perseus:

- Manning levels and pay rates advised by Perseus.
- Consumable prices from supplier budget quotations or the Lycopodium database.
- Cyanide and lime consumptions and gold extractions based on laboratory testwork results.
- Modelling by Orway Mineral Consultants (OMC) for crushing and grinding energy and consumables, using ore characteristics measured during the testwork.
- First principle estimates based on typical operating data.

The following exchange rates, as advised by Perseus, have been used for the preparation of the operating cost estimate:

- US$1.00 = A$1.250 (Australian Dollar).
- US$1.00 = 526 XAF (CFA Franc).
- US$1.00 = 11.11 ZAR (South African Rand).

The operating cost estimates are presented at summary and detail level. The estimate is considered to have an accuracy of ±15%, is presented in US dollars (US$) and is based on prices obtained during the fourth quarter of 2014 (4Q14).

The operating cost estimate includes all direct costs to allow production of gold bullion. All costs associated with mining are excluded from this part of the estimate with the exception of mining personnel camp and catering costs and grade control laboratory costs, which are included in the general and administration cost and laboratory cost.

The treatment plant availability includes allowance for scheduled and unscheduled maintenance.

Provision was be made in the financial model for the initial commissioning and production ramp-up period.

The battery limits for the process plant operating costs are as follows:

- Ore delivered to the ROM bin (ROM pad FEL costs are included in the mining costs).
- Tailings discharge from the tails pipeline to the tailings storage facility (TSF).
- Gold bullion in plant goldroom safe.

Operating costs have been developed for the oxide, transition, primary granite / porphyry and primary sediment ores.

Process plant and administration operating costs were developed using the parameters specified in the Process Design Criteria based on 'design' annual throughputs of 1,200,000 t/y for oxide ore, 1,000,000 t/y for transition ore 818,713 t/y for primary granite / porphyry ore and 736,842 t/y primary sediment ore based on anticipated LOM blends at that time with the additional constraint that throughput on blends of primary ores was not to exceed 800,000 t/y. These costs are summarised in Table 21.2.2.

Table 21.2.2 Summary Operating Costs by Ore Type at Design Throughput

Cost Centre / **Plant Throughput**	**Oxide** 1,200,000 t/y		**Transition** 1,000,000 t/y	
	US$/y	**US$/t**	**US$/y**	**US$/t**
Power	6,095,249	5.08	7,309,918	7.31
Operating Consumables	4,173,072	3.48	4,439,864	4.44
Maintenance	875,047	0.73	1,007,602	1.01
Laboratory	1,052,460	0.88	1,052,460	1.05
Process & Maintenance Labour	3,101,865	2.58	3,101,865	3.10
Plant Feed Rehandle	336,240	0.28	280,200	0.28
Total Processing Cost	**15,633,933** US$176 /oz	**13.03**	**17,191,908** US$157 /oz	**16.91**
Abidjan Office Labour	186,754	0.16	186,754	0.19
Administration Labour	2,698,227	2.25	2,698,227	2.70
General & Administration Costs	4,754,747	3.96	4,754,747	4.75
Total General & Administration	7,639,728	6.37	7,639,728	7.64
Total Cost including G&A	**23,273,661** US$262 /oz	**19.39**	**24,831,637** US$226 /oz	**24.55**

Cost Centre / **Plant Throughput**	**Granite / Porphyry Primary** 818,713 t/y		**Sediment Primary** 736,842 t/y	
	US$/y	**US$/t**	**US$/y**	**US$/t**
Power	7,842,960	9.58	7,842,960	10.64
Operating Consumables	4,544,944	5.55	4,241,779	5.76
Maintenance	1,106,435	1.35	1,106,435	1.50
Laboratory	1,052,460	1.29	1,052,460	1.43
Process & Maintenance Labour	3,101,865	3.79	3,101,865	4.21
Plant Feed Rehandle	229,404	0.28	206,463	0.28
Total Processing Cost	**17,878,067** US$201 /oz	**21.84**	**17,551,962** US$220 /oz	**23.82**
Abidjan Office Labour	186,754	0.23	186,754	0.25
Administration Labour	2,698,227	3.30	2,698,227	3.66
General & Administration Costs	4,754,747	5.81	4,754,747	6.45
Total General & Administration	7,639,728	9.33	7,639,728	10.37
Total Cost including G&A	**25,517,795** US$287 /oz	**31.17**	**25,191,690** US$315 /oz	**34.19**

The General and Administration costs in Table 21.2.2 and Table 21.2.3 exclude company contributions to a Community Development Fund of 0.5% of gold sales less transport and refining charges. This has been allowed for in the Project Financial Analysis.

Costs were split into fixed and variable components to enable then to be used to derive annual costs for changing ore blends and/or throughput with further development of the mine plan. The split of fixed and variable costs by ore type are presented in Table 21.2.3.

Table 21.2.3 Summary of Fixed and Variable Plant Operating Costs by Ore Type

Cost Centre	Oxide			Transition		
	Fixed US$/y	Variable US$/t	% Fixed	Fixed US$/y	Variable US$/t	% Fixed
Power	3,478,813	2.18	57%	3,526,774	3.78	48%
Operating Consumables	834,614	2.78	20%	887,973	3.55	20%
Maintenance Materials	670,736	0.17	77%	780,416	0.23	77%
Laboratory	947,214	0.09	90%	947,214	0.11	90%
Process & Maintenance Labour	3,101,865	0.00	100%	3,101,865	0.00	100%
Plant Feed Rehandle	0	0.28	0%	0	0.28	0%
Total Processing Cost	**9,033,242**	**5.50**	**58%**	**9,244,241**	**7.95**	**54%**
Abidjan Office Labour	186,754	0.00	100%	186,754	0.00	100%
Administration Labour	2,698,227	0.00	100%	2,698,227	0.00	100%
General & Administration Costs	4,754,747	0.00	100%	4,754,747	0.00	100%
Total General & Administration	7,639,728	0.00	100%	7,639,728	0.00	100%
Total including G&A	**16,672,970**	**5.50**	**72%**	**16,883,969**	**7.95**	**68%**

COST CENTRE	Granite / Porphyry Primary			Sediment Primary		
	Fixed US$/y	Variable US$/t	% Fixed	Fixed US$/y	Variable US$/t	% Fixed
Power	3,526,774	5.27	45%	3,526,774	5.86	45%
Operating Consumables	908,989	4.44	20%	848,356	4.61	20%
Maintenance Materials	862,180	0.30	78%	862,180	0.33	78%
Laboratory	947,214	0.13	90%	947,214	0.14	90%
Process & Maintenance Labour	3,101,865	0.00	100%	3,101,865	0.00	100%
Plant Feed Rehandle	0	0.28	0%	0	0.28	0%
Total Processing Cost	**9,347,021**	**10.42**	**52%**	**9,286,388**	**11.22**	**53%**
Abidjan Office Labour	186,754	0.00	100%	186,754	0.00	100%
Administration Labour	2,698,227	0.00	100%	2,698,227	0.00	100%
General & Administration Costs	4,754,747	0.00	100%	4,754,747	0.00	100%
Total General & Administration	7,639,728	0.00	100%	7,639,728	0.00	100%
Total including G&A	**16,986,749**	**10.42**	**67%**	**16,926,116**	**11.22**	**67%**

Subsequently the fixed / variable splits were used to determine costs based on throughput rates of 150 t/h oxide, 132 t/h transition, 128 t/h granite / porphyry and 118 t/h for sediment ores. These were used to match predicted mill feed blends by year as shown in Table 21.2.4 to enable annualised operating costs to be estimated.

Table 21.2.4 Summary of Sissingué Mill Feed by Year and Ore Type

Position	% of Total Plant Feed Processed						
	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM
Oxide	84	28	28	38	7	0	36
Transition	14	23	35	2	0	0	14
Primary Granite / Porphyry	0	39	5	29	85	10	30
Primary Sediment	2	10	32	31	8	90	20
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**

In most years the predicted feed blend is such that the milling circuit has sufficient power to achieve a grind size of P_{80} 106 µm at throughput rates in excess of 800,000 t/y. Scheduled plant throughput was therefore increased to as much as 1 Mt/y on predominantly primary ore where the blend grinding characteristics so allowed. The fixed and variable components of the operating cost were used to estimate the process plant operating costs at the higher primary ore throughputs.

Life of mine process plant and administration operating costs were estimated and are presented in Table 21.2.5.

Table 21.2.5 Life of Mine Process Plant and Administration Operating Costs

Cost Centre	LOM	
	US$	US$/t
Power	39,867,729	7.22
Operating Consumables	25,389,577	4.60
Maintenance	6,236,076	1.13
Laboratory	4,240,832	0.77
Process & Maintenance Labour	14,180,385	2.57
Plant Feed Rehandle	1,546,694	0.28
Total Processing Cost	**91,461,293**	**16.56**
Abidjan Office (DGA) Labour	1,273,507	0.23
Administration Labour	13,539,305	2.45
General & Administration Costs	27,738,812	5.02
Total General & Administration	**42,551,624**	**7.70**
Total Cost including G&A	**134,012,917**	**24.26**

The operating cost estimate includes the following:

- Abidjan Office labour and expenses (in the general and administration cost).
- Import duties on consumable unit costs (in the consumables cost).
- Costs for the preparation and assaying of 100 mine grade control samples per day and routine laboratory tests on the site water samples (in the laboratory cost).

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22.0 ECONOMIC ANALYSIS

22.1 Summary

The Sissingué Project is an economically viable Project at current gold prices. Applying a long term gold price of US$1,200/oz on a flat line basis from the commencement of production, estimated pre-tax cashflows generated are US$112.4M and the payback of the Project is estimated to occur after 2.7 years. The mine life is estimated at 5.3 years.

An internal rate of return of 27.0% and net present value ('NPV') of US$52.5M, based upon real, post-tax cash flows, calculated using a 10% discount rate are estimated.

Life of mine average C1 cash costs of production are US$570/oz, while all-in sustaining costs are US$632/oz.

Project returns are mostly sensitive to gold prices and head grade/recovery. For a US$100/oz increase in gold prices, a further US$24.3M NPV is added whilst for a US$100/oz decrease in gold prices, there is a US$26.1M reduction in NPV.

22.2 Valuation Methodology

The Sissingué Project economics has been evaluated using the discounted cashflow method, by taking into account yearly milled tonnages, grades and associated recoveries for the ore, gold price, operating costs, bullion transport and refining charges, royalties and capital (both initial and sustaining). Construction is completed in Project Year 1 and operations commence in Project Year 2.

The project has been evaluated as stand-alone and 100% equity financed with no debt financing considered.

22.3 Key Assumptions

Construction Capital Expenditure

Capital expenditure payments are assumed to have been paid in Project Year 1.

Royalties

The State of Côte d'Ivoire is entitled to production royalties as follows:

Royalty	Gold Price
3%	Up to US$1,000
3.5%	US$1,000 – US$1,300
4%	US$1,300 – US$1,600
5%	US$1,600 – US$2,000
6%	Over US$2,000

The royalty is based on the gross revenue from gold produced, after deduction of transportation and refining costs.

Franco-Nevada receives a royalty of 0.5% of the value of all minerals recovered from the Tengrela Exploration Permits less refining and smelting costs.

The royalty granted to Leo IC is an amount of US$0.80 per ounce of refined gold produced from the Tengrela East Research Permit 145 (the whole of the current area of the Exploitation Permit is part of the area covered by Research Permit 145), payable monthly.

Income Tax

A five year tax holiday on revenue from the date of first production. After that period the corporate tax rate is 25%. Income tax losses may be carried forward for a maximum of five years. Sunk costs have been taken into account when calculating deductible capital allowance for tax purposes.

Other Taxes

An exemption from VAT on the acquisition of foreign and Ivorian services and goods and sales in connection with the operations, this exemption extends to contractors; an exemption from import duties on imported plant and parts during construction and any expansion phase; and an exemption from import duties on fuel, lubricants, chemical products, etc. during the operations phase.

Marketing

The gold bullion bars produced may be sent to any of the active gold refiners in the world for refining. Perseus is already operating another gold mine in West Africa and it is highly likely that the gold produced at the Project will be shipped to the same refiners.

The refining process is completed within three working days after receiving the bullion. The refiner will charge costs to cover transport, melting, assaying and refining. For the purpose of this study 99.95% of the gold contained in the bullion will be returned to Perseus for sale on the open market.

Bullion transport, insurance and refining costs were assumed to be $2.80/oz.

Closure / Rehabilitation Costs

Both closure and rehabilitation costs have been included in the sustaining capital costs.

Currency

Costs are presented in United States Dollars (US$).

22.4 Financial Analysis / Sensitivity

Table 22.4.1 shows the key LOM performance measures for the Project at varying gold prices.

Table 22.4.1 Financial Analysis

Gold Price	Unit	US$1,100/oz	US$1,200/oz	U$S1,300/oz
Waste + Ore Mined	'000 tonnes	23,183	23,183	23,183
Ore processed	'000 tonnes	5,524	5,524	5,524
Head Grade	g/t gold	2.42	2.42	2.42
Weighted Average Recovery	%	89.72	89.72	89.72
Gold Produced	ozs	385,211	385,211	385,211
Development Capital	$'000	105,982	105,982	105,982
Sustaining Capital	$'000	5,249	5,249	5,249
Mining Costs	$'000	85,835	85,835	85,835
Processing Costs	$'000	91,461	91,461	91,461
Administration Costs	$'000	42,358	42,550	42,743
LOM Cash Operating Cost (C1)	$/oz	570	570	571
Year 1 & 2 Cash Operating Cost (C1)	$/oz	596	596	597
Total Site Cost	$/oz	628	632	643
Free cash	$'000	75,645	112,414	146,680
IRR	%	18.8	27.0	34.2
NPV 10%	$'000	26,382	52,515	76,794
Payback period	months	38	32	26
Tax paid	$'000	-	2	140
Royalties paid (State)	$'000	17,258	18,798	22,842

Applying a long term gold price of US$1,200/oz on a flat line basis from the commencement of production, estimated pre-tax cashflows generated are US$112.4M and the payback of the Project is estimated to occur after 2.7 years. The mine life is estimated at 5.3 years.

An internal rate of return of 27.0% and NPV of $52.5M, based upon real, post-tax cashflows, calculated using a 10% discount rate, are estimated.

Table 22.4.2 Key Technical Parameters

Parameter		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
PRODUCTION									
Ore mined	Mt	1.7	1.0	1.8	0.7	0.6	1.2	-	**5.5**
Waste mined	Mt	2.3	5.0	4.2	3.9	1.8	0.5	-	**17.7**
Total Material Mined	Mt	4.0	6.0	6.0	4.6	2.4	1.7	-	**23.2**
Strip ratio	t:t	1.4	4.8	2.3	5.3	2.9	0.5	-	**3.2**
Ore processed	Mt	-	1.1	1.1	1.1	1.1	1.0	0.2	**5.5**
Head grade	g/t gold	-	2.0	2.6	2.0	2.4	3.2	1.6	**2.4**
Recovery	%	-	91.7	90.3	89.2	90.7	88.6	77.7	**89.7**
Gold production	kozs	-	66.8	81.6	60.7	74.5	93.2	8.1	**385.2**
COSTS									
Unit mining costs	US$/t mined	-	3.18	3.31	4.07	6.35	7.61	-	**3.70**
Unit processing costs	US$/t milled	-	13.85	17.06	16.85	16.94	18.46	20.78	**16.75**
Annual G&A costs	US$ million	-	8.2	8.3	8.2	8.2	7.8	1.8	**42.6**
Production cash costs[1]	US$/oz	-	635	565	731	550	422	775	**569**
Royalties[2]	US$/oz	-	49	49	49	49	49	49	**49**
Sustaining capital costs	US$/oz	-	3	7	6	15	6	302	**14**
Total all-in site cash cost	**US$/oz**	-	**687**	**621**	**786**	**614**	**477**	**1,126**	**632**
Development Capital									
Construction Indirect Costs	US$ million	9.0	-	-	-	-	-	-	**9.0**
Treatment Plant Costs	US$ million	24.5	-	-	-	-	-	-	**24.5**
Reagents & Plant Services	US$ million	9.6	-	-	-	-	-	-	**9.6**
Infrastructure	US$ million	25.6	-	-	-	-	-	-	**25.6**
Mining	US$ million	11.9	-	-	-	-	-	-	**11.9**
Management Costs	US$ million	9.7	-	-	-	-	-	-	**9.7**
Owners project Costs	US$ million	12.9	-	-	-	-	-	-	**12.9**
Owners Operations Costs	US$ million	2.8	-	-	-	-	-	-	**2.8**
Total Development Capital	**US$ million**	**106.0**	**-**	**-**	**-**	**-**	**-**	**-**	**106.0**

Notes:

1. **Includes mining (incl. all waste stripping), processing, general and administration cash costs**
2. **Assumes US$1,200/oz gold price**

22.5 Sensitivity Analysis

A sensitivity analysis was performed on the post-tax NPV of the Project by varying the key variables of gold price, grade, recovery, operating costs and initial capital by 10%. Each assessment was done independent of the others. The results are shown below.

Figure 22.5.1 Sensitivity Analysis

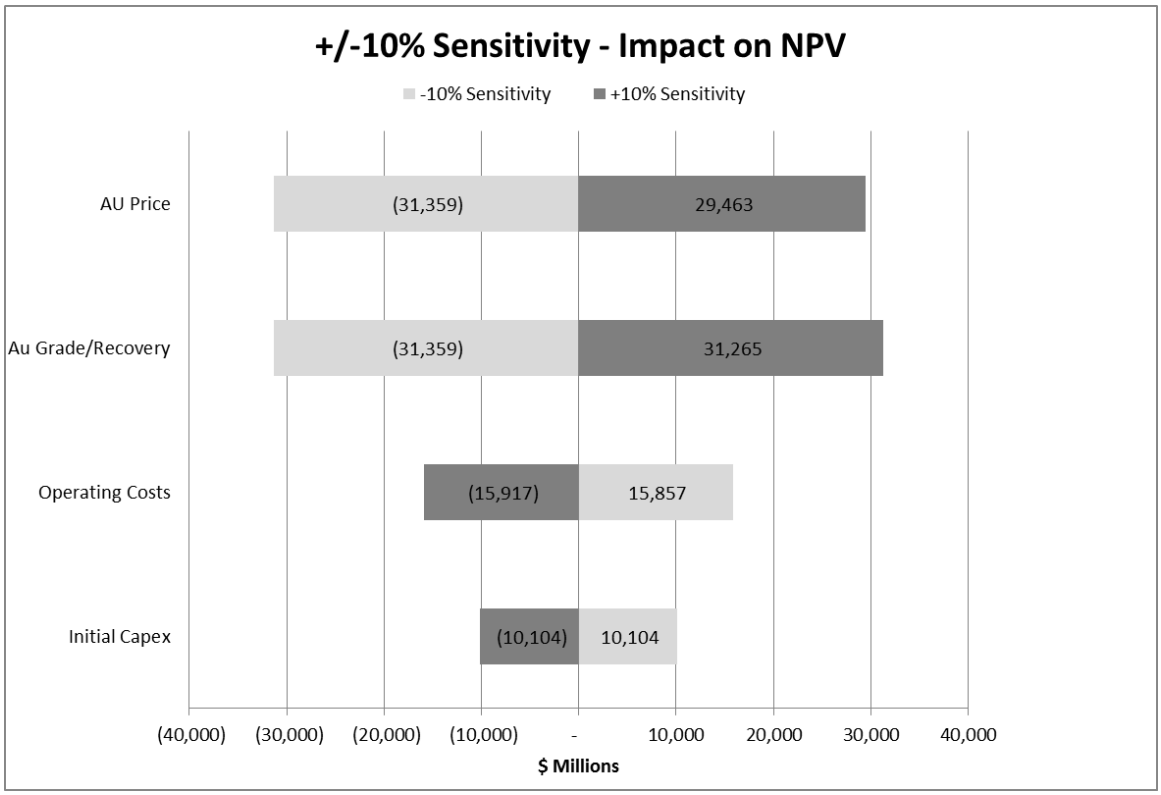


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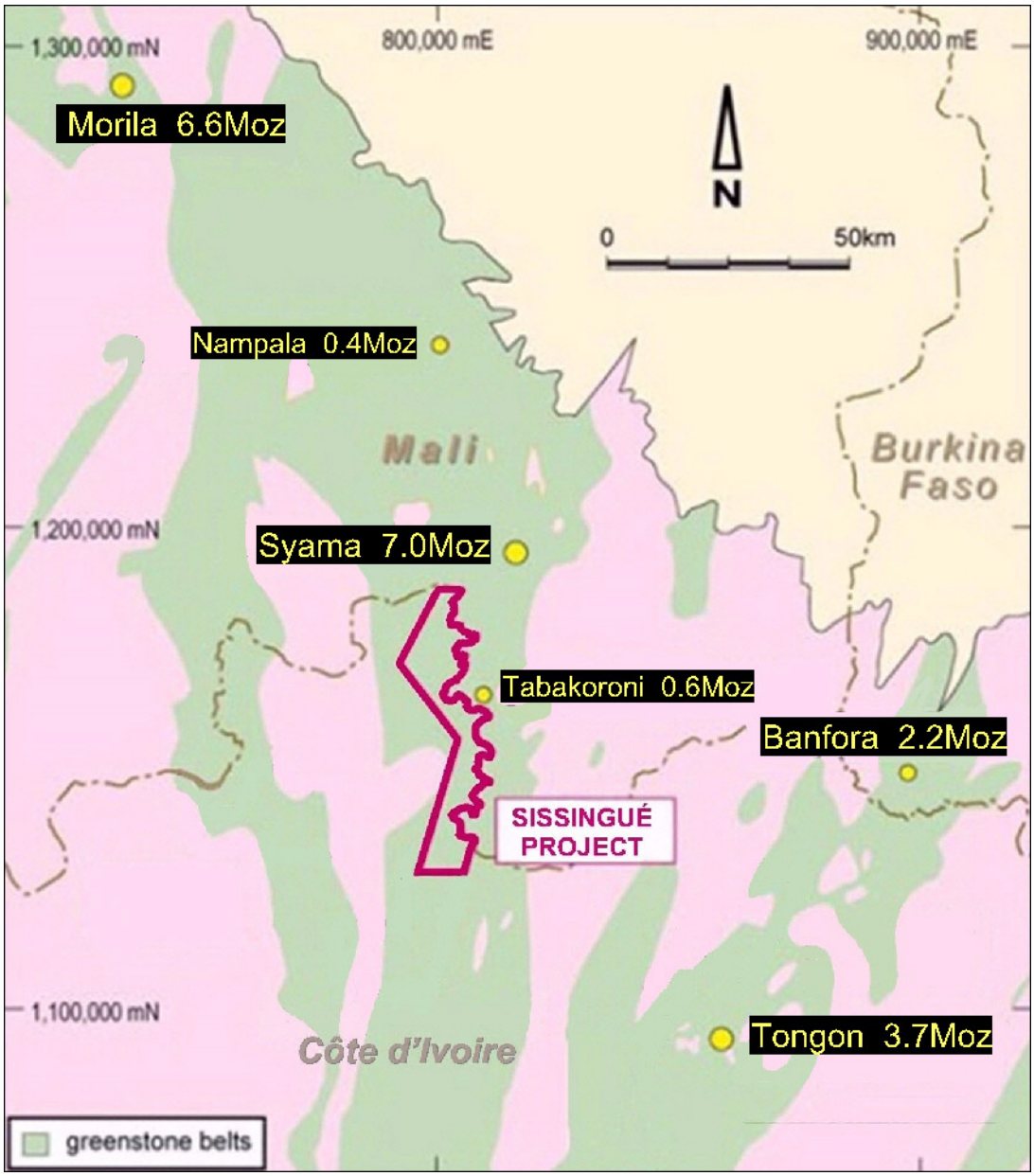
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23.0 ADJACENT PROPERTIES

23.1 Regional Gold Deposits

Figure 23.1.1 shows the location of known gold deposits in the region. The values for gold ounces shown were obtained from annual reports and/or the websites of Randgold, Resolute Mining, Robex Resources, Gryphon Minerals and Anglogold.

Figure 23.1.1 Regional Gold Deposits



23.2 Local Gold Occurrences

Figure 23.2.1 shows known local gold occurrences. The wider Tengrela leases are prospective with additional zones of mineralisation identified. Although this project is based on exploitation of the Sissingué deposit there is the opportunity that the operating life of the process plant will be extended.

Drilling to date at the Mahale / Bélé deposits, 40 km SW of Sissingué, shows potential for additional mill feed material. The deposit is within trucking distance of the Sissingué plant site. Additional work is required to develop resource and reserve estimates.

Development of the Project is based on exploitation of the Sissingué deposit only and no supplementary feed from any other deposit has been assumed in the economic evaluation of the Project.

Figure 23.2.1 Local Gold Occurrences

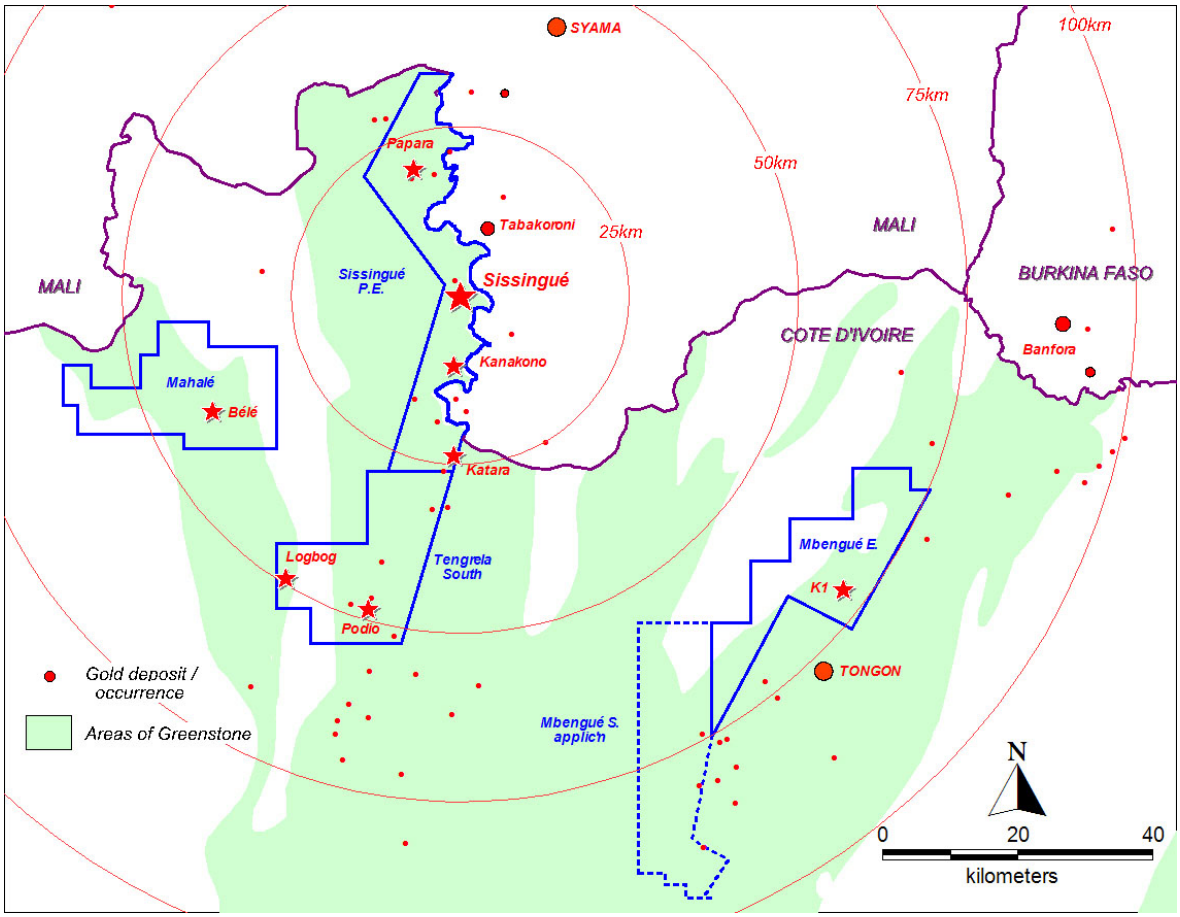


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24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

The approach to project implementation used as the basis for the capital cost and schedule was a standard EPCM approach. Perseus would engage a suitable Engineering, Procurement and Construction Management (EPCM) contactor for design and construction management of the process plant and infrastructure, which will then be handed over to an Owners operating team, and to engage an experienced mining contractor for the development of mine infrastructure and provision of ongoing drill and blast and mine operating services under an owners mine technical team.

The mine planning study is based on the owner engaging an experienced contractor (or contractors) to provide drill and blast and load and haul services. The contractor(s) will be managed and supervised by the owners mining / geology team. It is possible that the mining contractor will have the expertise and equipment to undertake bulk earthworks for the Project also.

A site will be provided for the contractor to set up support facilities and infrastructure such as offices, workshop, wash bay and workforce facilities.

The current mine plan anticipates mining commencing five months before first ore to the mill, with a total mine life of five and a half years and with the mill operating on material recovered from stockpiles for a further eight months after completion of mining.

The implementation strategy is structured into four broad stages:

- Detailed design of the process plant and infrastructure.
- Procurement.
- Construction.
- Commissioning and handover.

The estimated durations of key project activities are provided in Table 24.1.1

The durations shown are preliminary and are based on receipt of the relevant authorisations to proceed and availability of project funding.

Table 24.1.1 Key Project Milestones

Milestone Description	Planned Completion
EPCM Award Date	Week 0
Ball Mill Contract Award	Week 14
Earthworks and Civil Contractor Mobilisation	Week 19
Building Contractor Mobilisation	Week 35
SMP Contractor Mobilisation	Week 38
E&I Contractor Mobilisation	Week 60
Mills Mechanical Installation Commencement	Week 63
Power On	Week 60
First Gold Pour	Week 76

Several items and activities are potentially on or close to the schedule critical path for design and construction including procurement, manufacturing, transport and installation of long lead mechanical and electrical equipment items for the process plant. Their criticality will be investigated further during early engineering.

24.2 Operations

The current operating cost estimate is based on a preliminary organisational structure that is considered to be appropriate for an operation of this scale and type in the region.

The organisational structure will be reviewed and finalised during the initial stages of project implementation when senior operating personnel will take ownership of creation of the operating teams.

Allowance has been made to employ management and operating personnel in an orderly 'top down' method to meet orientation and training requirements prior to commencement of full operations.

Mine development and operations will be undertaken by an experienced mine contracting company under the direction of an owners technical and operations management team assisted, as required, by external consultants and service providers.

Several contract mining companies are already established in the region and it is expected that they will tender for the work and be in a position to mobilise the initial team and supplementary skilled personnel and equipment as required to ensure that productivity and progress meet the Project requirements.

Plant operations and administration roles will predominantly be filled by permanent staff. Exceptions to this include the maintenance and operation of the accommodation camp which will be contracted to a specialist service provider. Consideration will be given to contacting other non-core functions such as site security and, potentially, power generation should it benefit the Project.

24.3 Ramp -up Schedule

Mining and processing plant ramp up and production estimates have been prepared to facilitate the development of a financial cash flow model and thus more accurately determine the timing of likely expenditure and revenue streams.

The mining contractor will be mobilised during the construction phase to allow establishment of the mine services area, development of haul roads and other mining infrastructure and completion of the mine pre-strip and accumulation of an initial ROM stockpile.

The plant ramp up schedule has been developed to reflect the simple, robust flowsheet of the processing facility, using ramp up data from similar regional operations as a basis. It is estimated that nameplate plant ore throughput will be achieved in the third month after ore is first introduced.

24.4 Community Relations

Respect of the community and an ongoing 'social licence to operate' is paramount to the success of the Project and Perseus will maintain its consultation process where issues impacting all stakeholders can be discussed.

Perseus is committed to working with the community and will ensure that:

- consultation with all stakeholders will be undertaken to ensure rights are respected and opportunities offered
- community partnerships will be developed that are mutually beneficial and that contribute to social infrastructure within the regions where Perseus operates.

24.5 Risks and Opportunities

24.5.1 Identified Risks

Sovereign Risk

KPMG published a snapshot report covering sovereign risk in the first quarter of 2013. At that time none of the three major rating agencies rated sovereign risk in the country.

Following a period of hampered economic growth due to the political turmoil of the 2000’s, relative political stability and significant international debt relief has provided a level of stability. Foreign investment has recovered and the government is actively promoting investment in the energy and mining sectors.

KPMGs report states that 'Foreign interest in Ivory Coast has recovered since the stabilisation of the political environment, albeit that political risk remains relatively high'.

Sovereign risk is mitigated by the short payback period anticipated for the Project.

Cross Border Incursions

The project site is adjacent to the Côte d'Ivoire / Mali border opening up the possibility of security threats from people / groups crossing from Mali and retreating back across the border before Ivorian authorities can respond.

Non-violent attempts by individuals / groups targeting the theft of portable assets such as vehicles, tools, copper wire etc. will be discouraged by concentric lines of fixed security installations. Patrolled fencing around the entire site will deter casual access and opportunistic theft. A second fence line with more rigorous access control will ensure only authorised employees and visitors have access to core facilities and storage areas. A third, double fence, with camera monitoring and a further level of access control will surround the process plant. Finally, gold bullion will be processed and stored in a secure building with strict access control, intrusion alarms and camera monitoring.

The fixed security assets will be patrolled and monitored by a site security team that will develop a close working relationship with local Ivorian police and military personnel for the purposes of intelligence gathering and emergency response.

Gold bullion will be recovered on a weekly basis and promptly flown off site. If feasible, smelting days and the bullion flight schedule will be randomly varied to discourage attempts at theft.

24.5.2 Identified Opportunities

Project Timing

Prompt development of the Project at a time when construction activity in West Africa and elsewhere is well below recent peaks offers the opportunity to negotiate competitive prices for equipment supply and construction contracts. At the very least project timing reduces the risk of cost escalation and overruns, reducing the risk of exceeding project contingency allowances. It is likely, with the Project underway and 'money on the table' for equipment purchase and contract award, more favourable supplier prices and contractor rates could be negotiated.

Discussion with DTP has indicated there is potential to reduce the mining cost of the Project by 6 - 7% below their initial quote. Experience from the recent tendering process at Edikan indicates that the reduction could ultimately be larger due to the very competitive nature of the mining contractor sector at this point in time.

Extension to Mine Life

The wider Tengrela leases are prospective with additional zones of mineralisation identified. Although this project is based on exploitation of the Sissingué deposit there is the opportunity that the operating life of the process plant will be extended.

Drilling to date at the Mahale / Bélé deposits, 40 km SW of Sissingué, shows potential for additional mill feed material. The deposit is within trucking distance of the Sissingué plant site. Additional work is required to develop resource and reserve estimates.

Connection to Grid Power

Under the auspices of the Ministry of Petroleum and Energy Studies and Planning Directorate, CI-ENERGIES has produced a report providing a scope and cost for a connection to the 225 kV national grid at Boundiali via a 90 kV line to Tengrela and a 33 kV line on to Sissingué. The proposed route and overall concept is not the least expensive option as the intent is clearly to leverage off the development and supply grid power to Tengrela.

At the cost quoted, grid power supply as a standalone proposition at Sissingué is not the preferred option. However, there is a risk associated with the continuity of power supply to Perseus' Edikan operation in Ghana, and a potential mitigation method is to access power from Côte d'Ivoire through CI Energy. As part of a potential deal, Perseus could provide the additional capital to fund the construction of the power line from Boudiali to site and the overall outcome may be economically attractive to Perseus as a whole. The supply of grid power to site would potentially lower Sissingué operating costs.

This option is being pursued by Perseus.

Alternatively connection to grid power could become an attractive proposition in the future should additional discoveries, a rising gold price or other circumstances result in a significant extension to the mine life.

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25.0 INTERPRETATION AND CONCLUSIONS

25.1 Interpretation and Conclusions

The Sissingué Gold Project Feasibility Study has identified, defined and costed a low technical risk conventional open cut mine and carbon in leach processing facility.

Commercial gold mining has been established in West Africa for over 100 years and Côte d'Ivoire and surrounding countries have varying levels of established mining infrastructure.

Sovereign risk is understood and mitigated by the relatively short payback of the Project. Residual environmental and social issues are low and manageable.

The Sissingué Gold Project is an economically viable Project at current gold prices. Applying a long term gold price of US$1,200/oz on a flat line basis from the commencement of production, estimated pre-tax cashflows generated are US$112.4M and the payback of the Project is estimated to occur after 2.7 years. The mine life is estimated at 5.3 years.

25.2 Mineral Resource

The Mineral Resource model on which the Project is based is considered to be robust, with the key outcomes being:

- Resource classification is based on information and data provided from the Perseus database. The drilling techniques, survey, sampling, sample preparation, analytical techniques and database management / validation are well within industry standard. The database represents an accurate record of the drilling undertaken in the Project.
- Geological continuity is understood with reasonable confidence. The classification of the Mineral Resource reflects the level of confidence.
- The estimation technique and parameters are appropriate for the style of mineralisation.

25.3 Mining

The mine planning study has demonstrated the Project to be a viable and attractive prospect for development. The key outcomes of the Study are as follows:

- 3 phase mine development strategy from south to north.
- Total material movement and vertical rates of advance are readily achievable.

- Areas have further been identified that would likely improve the Project value.
  - Re-run the Whittle optimisation to reflect owner-operator mining operating costs.
  - Re-run the schedule optimising the output based on an owner-operator model.
  - Further studies into the potential for backfilling of the Phase 1 pit.

25.4 Mineral Reserve

The Mineral Reserve Estimate on which the Project is based is considered to be robust, with the key outcomes being:

- Total mineable ore of 5.5 Mt at 2.2 g/t Au and strip ratio of 3.2:1.
- Annual gold production is variable due to grade and the transition from oxide to sulphide, ranging from 61/koz to 94/koz per year.
- 5.5 year mining life.

25.5 Processing

The metallurgical and ore processing studies have demonstrated the Project to be viable and attractive for development. The key outcomes of the Project are:

- A simple crushing and grinding circuit with recovery via gravity and CIL is the preferred processing method.
- High recoveries are achievable from all major ore types that are planned to be processed.

Areas have further been identified that would likely improve the Project value, namely:

- Investigate methods for improving the recovery from sediment type ores.
- Investigate the availability of immediately available new or second-hand items of long lead equipment such as the crushing circuit and/or mill.

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26.0 RECOMMENDATIONS

26.1 Recommendation

It is recommended that Perseus Mining Limited commence implementation of the Project with a view to demonstrating commitment to commercial production before the expiry of the 8 August 2016 deadline currently in place as a condition of their Exploitation Permit. The project is expected to be developed for an initial capital cost of US$106.0M and a life of mine average operating cost (C1) of $39.75/t ore, or US$570/oz gold produced.

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APPENDIX 1

SISSINGUÉ RESOURCE REPORT



Perseus Mining Limited

Sissengue Resource Estimate

Project No. AU4520

October 2014

Final

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This report has been prepared by Snowden Mining Industry Consultants ('Snowden') on behalf of Perseus Mining Limited.

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1 Executive summary

This report has been prepared by Snowden Mining Industry Consultants Pty Ltd (Snowden) for Perseus Mining Limited (Perseus), in relation to the updated Mineral Resource for the Sissengue deposit which forms part of the Tengrela Gold Project.

The initial resource estimate was carried out in July 2013. Subsequently a trial optimisation was run to limit the base of potentially economic material in October 2014 after the receipt of additional information on mining and recovery. No additional drilling was carried out during this period.

Table 1.1 summarises the Sissengue Mineral Resource by category and area reported above a 0.6 g/t Au cut-off grade.

Table 1.1 October 2014 Sissengue Mineral Resource reported at a 0.6 g/t Au cut-off

Category	Area	Tonnage (kt)	Grade (g/t Au)	Contained Gold (oz)
Measured	Oxide	1,000	1.8	59,000
	Transitional	650	2.3	49,000
	Fresh	3,200	2.5	260,000
Total Measured		**4,800**	**2.4**	**370,000**
Indicated	Oxide	3,100	1.3	130,000
	Transitional	800	1.5	38,000
	Fresh	7,100	1.5	350,000
Total Indicated		**11,000**	**1.4**	**510,000**
Measured + Indicated	Oxide	4,100	1.4	190,000
	Transitional	1,400	1.9	87,000
	Fresh	10,000	1.8	600,000
Total Measured + Indicated		**16,000**	**1.7**	**880,000**
Inferred	Oxide	310	1.2	12,000
	Transitional	54	1.2	2,100
	Fresh	760	2.0	49,000
Total Inferred		**1,100**	**1.7**	**63,000**

Notes: Mineral Resources are inclusive of Mineral Reserves. Mineral Resources are reported to two significant figures. Rounding may cause minor discrepancies in the table. Oxide includes small portions of laterite (571 kt total)

The Mineral Resource was estimated using ordinary kriging and multiple indicator kriging using CAE Studio (Datamine) software.

Estimation was constrained within mineralisation envelopes (wireframes) based on geological logging and grade thresholds. The three main host lithologies are granite, porphyritic dykes and sediments. Where geological contacts were not clearly controlling the distribution of mineralisation, a grade cut-off of 0.3 g/t Au was used to construct Mineral Resource boundaries. Analysis of the global grade distribution shows that there is a natural change in grade population at around 0.3 g/t Au.

Due to the highly skewed nature and mixed populations evident in the granites and sediments, multiple indicator kriging (MIK) was used to estimate gold grades.

Ordinary kriging with top cuts was used to estimate the lower grade dyke domains. A dynamic anisotropy approach was used, whereby the true dip and azimuth of the mineralised lodes was estimated into each block in the model and the search and variogram orientations were locally adjusted to reflect the geological orientation. This method allows the estimate to better reflect the changing orientation and undulating nature of some of these dykes along strike.

Parent block dimensions of 10 mE by 10 mN by 5 mRL were used for estimation. All samples were composited to 2 m prior to estimation.

The Sissingue Mineral Resource has been classified in the Measured, Indicated and Inferred categories, in accordance with the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC Code) and the CIM Definition Standards (CIM, 2005). A range of criteria has been considered in determining this classification including geological and grade continuity, data quality and drillhole spacing. The key classification criteria are describes as follows:

- Geological continuity is understood with reasonable confidence. The classification reflects this level of confidence. Porphyry lithologies (dyke domains) were limited to Inferred and Indicated categories due to the mostly narrow nature of the mineralised bodies and consequently their sensitive geometry.
- Resource classification is based on information and data provided from the Perseus database. Descriptions of drilling techniques, survey, sampling, sample preparation, analytical techniques and database management/validation provided indicate to Snowden that data collection and management is well within industry standards. The database represents an accurate record of the drilling undertaken at the project.
- Drillhole location plots have been used to ensure that local drill spacing conforms to the minimum expected for the various resource classification categories.
  - The Measured category requires a drill spacing of 20 m by 20 m or closer.
  - The Indicated category is confined to areas where drill spacing is greater than 20 m by 20 m, but nominally up to 20 m (east-west) by 40 m (north-south) spacing. In the dyke domains, in addition to the drill spacing constraints, a minimum of two drillhole intercepts per drill section is required.
  - The Inferred category is assigned to all other estimated blocks within the potentially economic areas of the deposit.
  - Trial optimisation has been run at a USD2400 gold price to define the base of potentially economic material. All blocks outside this shell are unclassified.
- Snowden considers the estimation technique and parameters appropriate for this style of mineralisation.

The reporting cut-off is based on optimisation studies carried out by Perseus as part of the 2010 Feasibility Study, which have suggested that the deposit can be economically extracted at a gold cut-off in the range 0.4 g/t to 0.6 g/t. Metallurgical studies have confirmed that the Sissingue ore is amenable to processing by conventional CIL methods.

Snowden compared the updated Mineral Resource with the previously reported Mineral Resource (Widenbar, 2013). The comparison shows that the updated estimate is more selective, resulting in higher grades in the central, Measured portion of the estimate. This has resulted in an additional 60 koz in the Measured category.

The increased selectivity in the updated estimate is a result of using MIK estimation to honour the highly skewed, mixed distribution in the granite and sediment domains. The previous estimate used top-cut ordinary kriging in these domains, which tends to over smooth in these styles of strongly skewed grade distributions.

There is a decrease in Indicated and Inferred Resources as a result of the application of the pit shell to define the base of potentially mineable material.

2 Introduction

This report has been prepared by Snowden Mining Industry Consultants (Snowden) for Perseus Mining Limited (Perseus), in relation to the updated Mineral Resource for the Sissengue deposit which forms part of the Tengrela Gold project.

The Tengrela project is located in northern Ivory Coast, adjacent to the Mali border and approximately 700 km north of the commercial capital, Abidjan (Figure 2.1). The project area is centred at National Grid reference 1,141,666 mN and 804,166 mE.

Figure 2.1 General Location of the Tengrela project area (after Runge, 2010)



The estimates were prepared using the following steps:

- data validation – this was undertaken by Perseus and reviewed by Snowden
- data preparation
- geological interpretation and modelling
- establishment of block models
- compositing of assay intervals
- drillhole grade data analysis
- analysis of top cuts
- variogram analysis
- derivation of kriging plan and boundary conditions
- grade interpolation
- validation of grade estimates
- classification of estimates with respect to JORC (2012) guidelines
- resource tabulation and resource reporting.

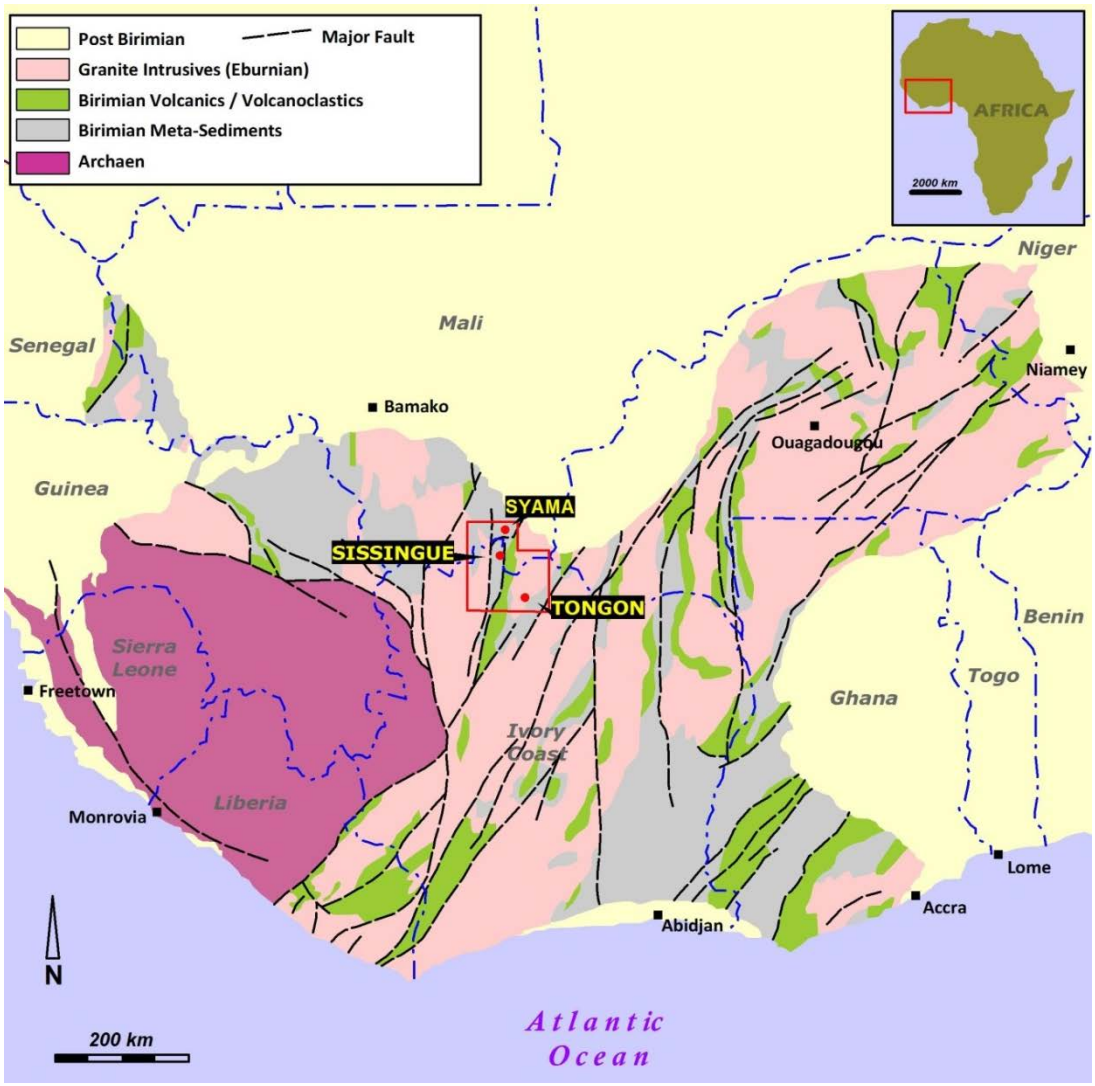
3 Geological setting and mineralisation

3.1 Regional geology

Most gold deposits in the Proterozoic Birimian Shield of West Africa occur in volcano-sedimentary greenstone belts. These belts generally consist of fine-grained sedimentary rocks and tholeiitic to calc-alkaline volcanic rocks. Granitic and gneissic rocks surround the greenstone belts. The metamorphic grade of the belts is usually lower greenschist but may increase to amphibolite grade adjacent to major intrusive bodies. The volcano-sedimentary sequences are in the most places exposed in arcuate monoclonal sequences.

A map of the regional geology is shown in Figure 3.1.

Figure 3.1 Regional geology of West Africa (image supplied by Perseus)



The southern part of the West African Craton (the Man Shield) comprises an Archean core in the south and west and the Proterozoic Baoulé-Mossi Domain, containing the Birimian Formations to the east. Ivory Coast is situated almost entirely over the later domain which has been involved in the Eburnean orogenic event (2,400 Ma to 2,100 Ma). It consists mainly of polyphase granite-gneiss complexes, sedimentary flysch-type deposits and volcanic and volcano-sedimentary complexes. Remnant Archean greenstone belts exist in Ivory Coast, mainly as relict narrow northeast-southwest striking belts that are situated in the gneissic complexes. They contain an assemblage of tholeiitic, volcanic, plutonic and volcano-sedimentary mafic rocks.

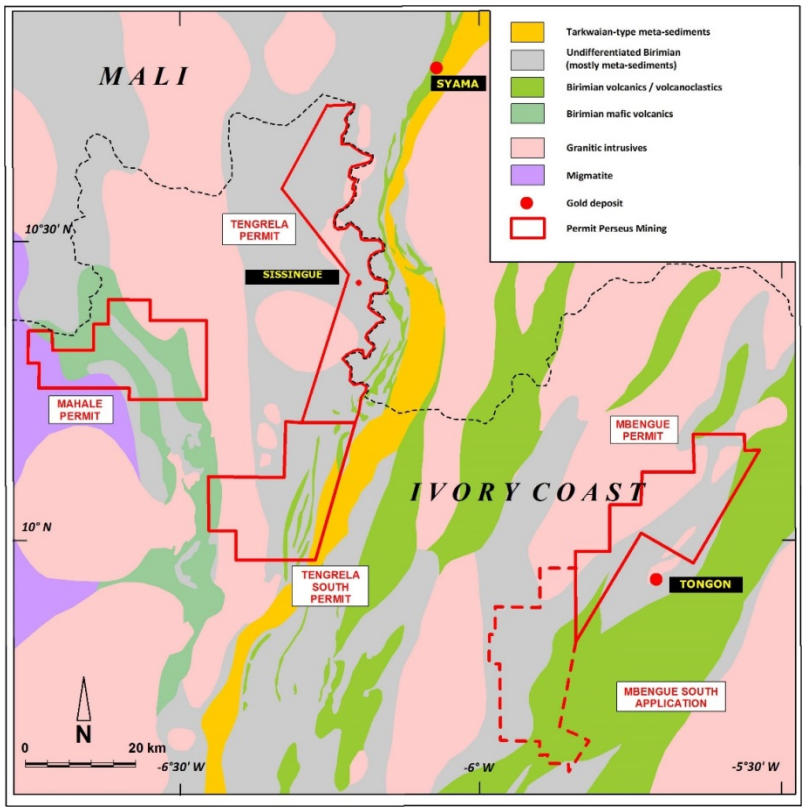
The volcano-sedimentary package appears laterally extensive and strikes variably north-south to north-east. Regionally, the most prominent feature is a north-south striking airborne magnetic feature that separates an interpreted mafic bearing rock package to the west and a siliceous sediment/granitic package of rocks to the east.

3.2 Local geology

The Sissengue deposit lies within the Tengrela project area is situated in the Syama-Boundiali Greenstone Belt. The geology of the belt has similarities to the better explored Ashanti Belt in neighbouring Ghana, where Birimian volcanoclastics dominate over competent volcanics, with the development of inner belt basins filled with Tarkwaian epiclastics. In the Bagoe River region near Tengrela, the terrain is comprised of undifferentiated granitoids, flysch sediments, intermediate volcanic and small occurrences of mafic intrusive and molasse sediments.

The permit area covers a strongly deformed Birimian greenstone belt intruded by granitoid bodies (Figure 3.2). Interpretation of both aeromagnetics and radiometric data indicates the inclusion of an extensive sequence of north trending Birimian volcanoclastics, intruded by a large felsic body (Kanakono Pluton) as well as several small uranium-rich felsic intrusions.

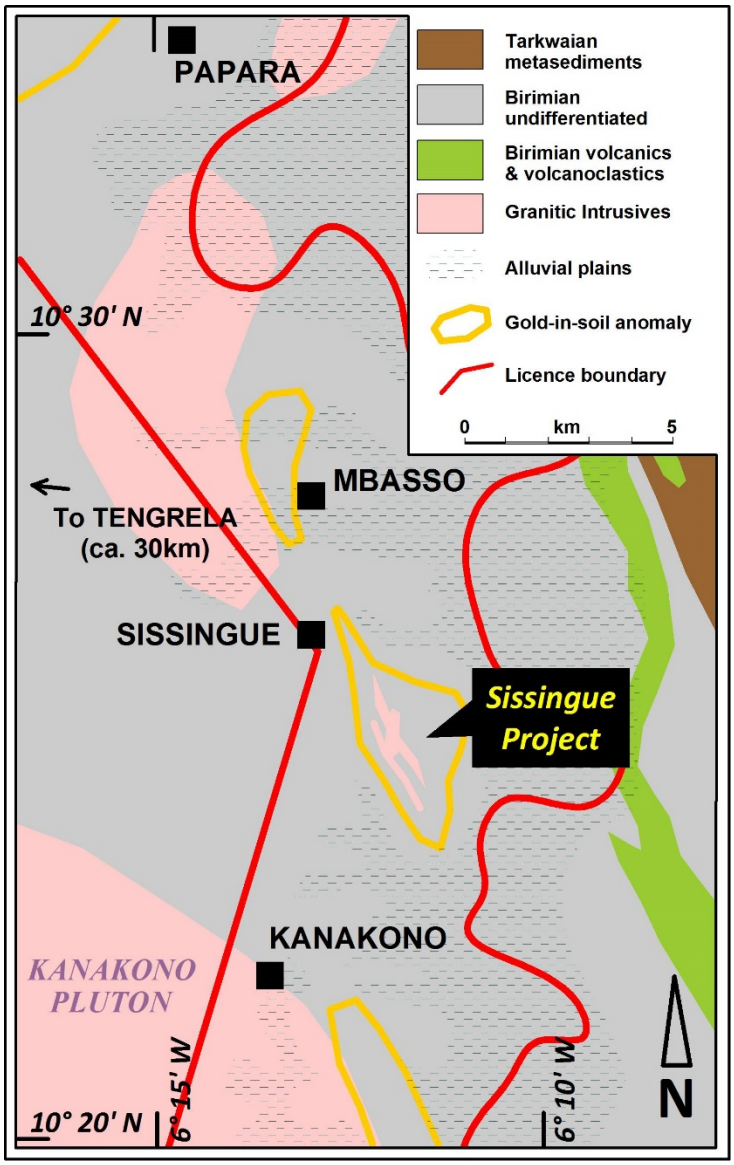
Figure 3.2 Geology of Northern Ivory Coast (image supplied by Perseus)



3.3 Property geology

The Sissingue deposit is defined by a 4 km long and up to 1.5 km wide gold-in-soil anomaly situated on the Syama-Boundiali Greenstone Belt (Figure 3.3). Rocks encountered in outcrops and drilling comprise predominantly north-north east striking, steeply west dipping and isoclinally folded sediments (sandstones, mudstones and subordinate conglomerates) of the Birimian Supergroup, interpreted as units of turbiditic flows. These sediments are cross cut by a swarm of narrow porphyritic dykes (sub-metre to several metres thick) which trend obliquely to the sedimentary package north-west with sub-vertical or steep to moderate dips towards the south-east. Some minor dykes appear to have exploited weak zones parallel to sedimentary bedding.

Figure 3.3 Tengrela property geology (image supplied by Perseus)



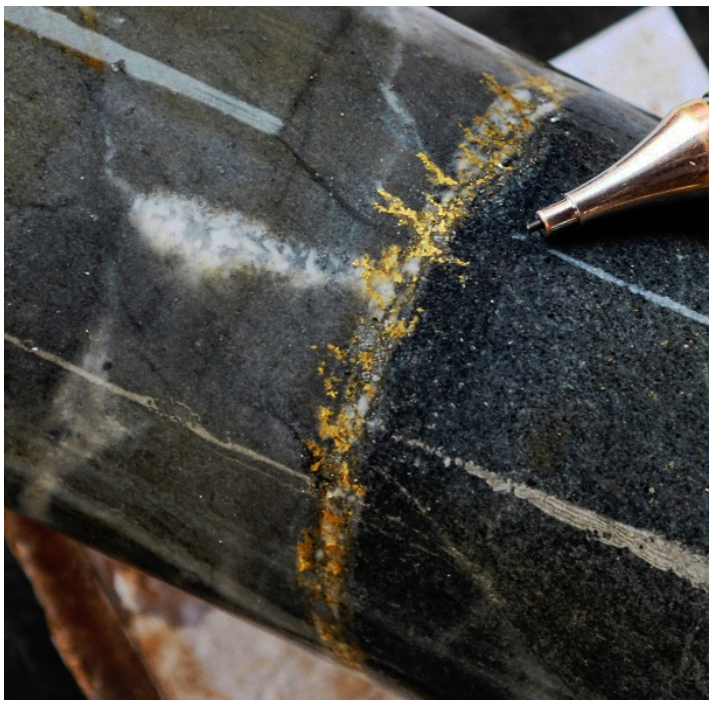
3.4 Mineralisation

Gold mineralisation at Sissingue is associated with the porphyritic dykes of tonalitic composition that cross cut the flysch sediments (turbidites). Subsequent metasomatism of the tonalite has led to:

- A sericite-carbonate alteration within the intrusives and the more permeable horizons (sandstones and conglomerates) of the turbidites.
- Low to moderate grade disseminated gold mineralisation.

Late stage high grade Au-As-quartz-carbonate veins exploited the altered and brittle portions of the intrusives and sediments with common occurrences of visible gold (Figure 3.4).

Figure 3.4 Quartz-carbonate vein with visible gold from drill core (SD223) (image supplied by Perseus)



Gold mineralisation is found as three distinct styles as summarized below:

- Disseminated or fracture related mineralisation in the felsic intrusive closely associated with sericite-carbonate alteration, silification and a pyrite and arsenopyrite mineralisation. The average gold tenor of this type of mineralisation is usually relatively high (2 g/t Au to 5 g/t Au).
- Significant low grade gold mineralisation (commonly below 2 g/t Au) is encountered within the sediments immediately adjacent to the intrusive bodies within their respective alteration halos. The gold tenor are perceived to be closely correlated to alteration intensity and hence with primary porosity of the rocks. Sandstones and conglomerate are usually better mineralised than fine grain mud- and siltstones.
- High grade mineralisation (with grades exceeding 1,000 g/t Au) is associated with quartz-carbonate veins which are encountered in the upper portion of the intrusive bodies and in the altered and brittle portion of sediments directly above. Visible gold is common.

Figure 3.5 shows a typical mineralised granite sediment contact with cross cutting quartz veins.

Figure 3.5 Mineralised granite in contact with sediments (after Runge, 2010)



It appears that weathering has an effect on the gold distribution. Boundary plots (Section 7) and sample histograms for the various weathering domains suggest that gold has been depleted in weathered (oxidised) material. Descending waters resulted in gold enrichment in the transition zone as well as in nugget growth where existing nuggets have been freed from country rock during weathering processes.

4 Informing data

4.1 Input data

The input dataset used for the Sissengue resource estimate contained 131,744 RC and DD samples for 199,269 m of which 36,113 m lie within the mineralised domains.

The total dataset within the Sissengue Mineral Resource area is summarised in Table 4.1 and illustrated in plan view Figure 4.1.

Table 4.1 Sissengue drillhole data summary

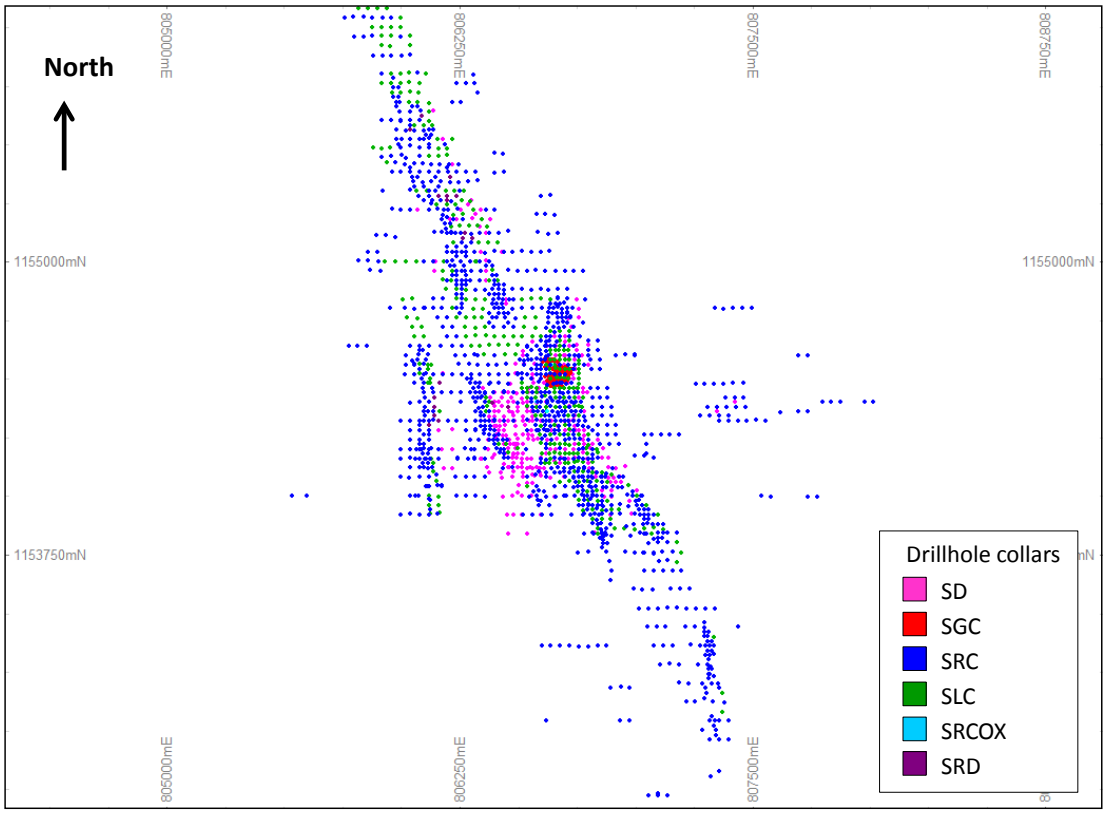
Hole prefix	Drill type	Number of holes	Number of meters
SD	DD	315	67,862.75
SGC	RC	58	3,230
SLC	RC	356	33,246
SRC	RC	1,271	94,551
SRCOX	DD	3	215,1
SRD	RC/DD	18	2,235.7
Total		**2,021**	**201,340.55**

The disseminated nature of the mineralisation and lack of discrete structures controlling grade distribution meant that selective sampling was not warranted and a sample length of 2 m was used throughout the project.

The steep nature of the mineralisation and sometimes limited drill access meant that holes were at a moderate to low angle to the mineralisation. True thickness of intersections is typically half the downhole thickness.

Perseus observed that core and RC samples showed very good recoveries and Perseus considers the samples to be representative of the mineralisation defined by the drilling.

Figure 4.1 Plan view of Sissengue drillhole data



4.2 Reverse circulation drilling

Since March 2007, a total of 1,685 RC holes and 131,027 m have been drilled at Sissingue within an area of 5.7 km strike and 2.5 km width, of which 931 holes have been used for the Mineral Resource estimate. Drilling was carried out on west-east aligned lines at nominal 80 m by 80 m spacing followed by 40 m by 40 m and 20 m by 20 m infill. A 120 m long zone between 1,154,460 mN and 1,154,580 mN has been drilled on a 10 m by 10 m grid (59 holes). All RC holes were aligned either 90° east or 270° west with inclinations between -50° and -60°.

1,326 RC holes were completed by contract drilling company FORAMIN Limited (formerly Drillex Limited), using four different drill rigs as follows:

- Two GK850 multi-purpose rigs, mounted on tracks with auxiliary Sullair compressors (900cfm/s50psi).
- DB750 multi-purpose rig, mounted on tracks and equipped with an Elgi compressor (900cfm/s50psi) and a separate Hydco V8 booster for additional 1,200cfm/650psi.
- DB900 track mounted RC rig with an Elgi compressor (900cfm/350psi) and a separate Hydco V8 booster for additional 2,300cfm/750psi.

The other 359 RC holes have been drilled by contract drilling company Boart Longyear using a CATMAX RC rig, equipped with a Sullair compressor (1,350cfm/500psi).

All RC rigs used at Sissingue were equipped with $5^1/_4$ inch diameter hammer. Most RC holes (1,072) were designed to stop at a nominal depth of 80 m or less. The deepest RC hole was 150 m.

RC sampling was generally conducted for the full length of the drillholes at regular 1 m intervals with 2 m composites submitted for analysis, with exception of the 59 RC holes drilled on the 10 m by 10 m grid where 1 m samples were submitted for analysis. Compositing of two consecutive samples was done using a scale to ensure equal representation of the individual samples. Nominally, composite samples consist of two 2.5 kg samples collected from the RC drilling are considered to be representative of the mineralisation.

RC drilling intersected sediments, granite and feldspar porphyry lithologies. The holes were sampled in their entirety. All RC samples were collected at the drill site at 1 m interval and split using a multi-stage riffle splitter. Each two consecutive samples were composited (where applicable) in one bag and given a pre-printed sample ticket.

The natural water table varies by season, but is typically at around 30 m to 40 m vertical in depth. With the use of a booster compressor during drilling, samples generally stay dry until approximately 80 m depth. Wet samples were taken from the drill bag at the cyclone using a spear. Approximately 4 % of all RC drill samples have been reported to be wet. There is no evidence of downhole smearing, depletion or enrichment during wet RC drilling.

A total of 69,728 RC samples were submitted to analytical laboratories for gold-only analysis.

4.3 Diamond core drilling

Diamond (DD) drilling commenced at Sissingue in January 2008. A total of 319 diamond drillholes for 68,148.35 m have been completed over the Sissingue prospect and an additional 18 RC holes that have been continued with diamond tails (for 898 m). All but one diamond drillhole were drilled either 90° east or 270° west with inclination between -50° and -60°. Drillhole SD80 was drilled at -50° towards 0° north.

260 diamond holes, and all 18 combined RC and diamond holes, have been completed by FORAMIN Limited with a track mounted YDXL/Coretec core rig, and the two multi-purpose rigs (see above). The other 59 diamond holes have been drilled by Boart Longyear with a platform-mounted Boart Longyear LY44 core rig. Drilling was typically completed at HQ size in weathered material and NQ size in fresh rock. All core was orientated by the “AceTool” orientation tool. The first 43 holes drilled by Boart Longyear were orientated using a spear.

Diamond core was geologically and structurally logged, then sawn in half using a motorised diamond blade saw. Half core samples of oxide and transitional material were taken accordingly to core run (1.5 m) and every meter in fresh rock where core recovery usually was close to 100 %. The material of the right side of the core was submitted for analysis, the left side stored in trays for reference.

Core sampling was conducted for the full length of the drillholes in regular intervals. No attempt was made to selectively sample portion of the holes or according to lithology. A total of 69,840 samples were dispatched to analytical laboratories for gold-only assaying. Samples collected from diamond drilling are considered to be representative of the mineralisation.

4.4 Collar location and survey

The collar locations of 1,225 of the RC and diamond core drillholes have been surveyed. Only 4 RC drillhole collars have not been surveyed, but estimated to be within 2 m to 3 m accuracy of the position established by handheld GPS.

All RC and diamond holes until September 2009 were surveyed using differential GPS, by a certified contract surveyor (SEMS Exploration Services Ltd, Ghana). Drillholes between September 2009 and October 2010 were surveyed by CBM Surveys Ltd of Ghana. All subsequent drillholes were surveyed by the company's surveyor.

All diamond core holes were downhole surveyed nominally every 30 m with a "Flexit SmartTool" digital downhole survey instrument. Most RC holes have not been surveyed downhole. Measurements of inclination have been taken at collar with a goniometer, azimuths taken with a precision compass.

4.5 Geological logging

RC drillholes have been logged from chip boards. Logged items include lithology, weathering, colour, grain size, vein type and estimated percentage, sulphide mineralisation and their estimated percentage, alteration and alteration intensity. While logging of fresh RC chip generally poses little challenges, strongly or entirely oxidized RC chips are often difficult to log. After commencement of diamond core drilling and a better understanding of the encountered rock types and structures, all chip boards have been re-logged.

Diamond core has been logged at the core yard on full core prior to cutting. Logged items include lithology, weathering, colour, grain size, vein type and vein volume percentage, sulphide mineralisation and their estimated percentage, alteration and alteration intensity. Structural measurements were taken regularly on all orientated core including fault, fold, cleavage and joint orientation, lithological contacts, vein orientation and bedding.

Geotechnical logging was carried out on all core and includes core recovery, RQD, rock strength and number and type of defects.

4.6 Sample preparation and assaying

Three analytical laboratories have been used to assay samples from the Sissingue project:

- ALS Chemex Laboratories (ALS), in Bamako/Mali with head office in Vancouver/Canada. This laboratory was formely operated by Abilabs Ltd (Abilabs) until taken over by ALS Global in March 2007.
- Intertek Minerals Ltd (Intertek) in Tarkwa/Ghana under the umbrella of Intertek Minerals / Genalysis Laboratory Services Pty Ltd with head office in Maddington/Australia. This facility was until August 2010 operated by Transworld Laboratories Ltd (TWL).
- Bureau Veritas Minerals Laboratory (BVML) in Bamako/Mali and Abidjan/Ivory Coast with head office in Paris/France.

A summary of the sample dispatch and assay methods are given in Table 4.2. All assaying has been carried out via fire assay (FA) or BLEG bottle roll (BR).

ALS Global Canada are covered by ISO 9001:2008 certification for the provision of assay and geochemical analytical services by QMI Quality Registrars (1 February 2011) and accredited under ISO 17025 for its analytical capabilities by SCC (31 August 2008).

ALS Global South Africa is accredited under ISO 17025 by SANAS (6 February 2013). The laboratory in Bamako is not accredited, but operates under the same quality protocol and the same work instructions as the accredited ones in Canada and South Africa.

Intertek /Genalysis Laboratory Services are accredited under ISO 17025 by NATA (No 3244 of 25 May 2011).

BVML Bamako, Mali and Abidjan, Ivory Coast have been accredited by Ultra Trace Pty Ltd in September 2010 but do not have a formal international accreditation. Ultra Trace accreditation covers technical competence in sorting and drying, weighing, crushing, splitting, pulverisation, sieving and gold analysis by fire assay with a detection limit of 0.01 ppm.

Table 4.2 Summary of assay methods

Laboratory	Drill Type	Number of samples			Period
		Dispatched	Method	Analysed	
ALS	DD	15,738	FA	15,584	since March 2007
			BR	154	
	RC	18,831	FA	18,831	
Abilabs	DD	0			until March 2007 (then acquired by ALS)
	RC	3,475	FA	307	
			BR	3,168	
Intertek	DD	23,600	FA	23,600	since August 2010
	RC	6,020	FA	6,020	
TWL	DD	22,373	FA	22,373	until August 2010 (then acquired by Intertek)
	RC	36,726	FA	36,726	
BVML	DD	1,312	FA	1,312	since December 2012
	RC	4,676	FA	4,676	
TOTAL	DD	63,023	FA	62,869	
			BR	154	
	RC	69,728	FA	66,560	
			BR	3,168	

Perseus staff have carried out regular unannounced laboratory visits to BVML, Ivory Coast; the latest during April 2013. No material issues were observed.

4.6.1 Sample preparation

RC samples are collected at the drill site and split through a multistage riffle splitter and each two consecutive samples are bagged into a 2 m composite and sent to the laboratory for preparation and analysis. Perseus employees carry out the sampling at the drill site.

Diamond drilling, core cutting and bagging of samples is conducted by Perseus employees and then sent to the laboratory for preparation and analysis. The right hand side of the core looking downhole was always submitted for analysis with the left side being stored in trays on site. After dispatch from the Project, Perseus staff had no involvement in the sample preparation or assay procedures of either RC or diamond samples.

Both core and RC samples followed a standard path of drying at 105° to 110° C, crushing and grinding. Samples were pulverized with a ring mill and thoroughly mixed on a rolling mat ("carpet roll"), and then a 200 g of sub-sample was split off. Internal laboratory checks required at least 90 % of the pulp passing -75 microns.

4.6.2 Sample analysis

Two types of analysis for gold were performed, a standard fire assay and BLEG bottle roll. Similar fire assay procedures were followed by ALS, Intertek and BVML. 50 g sub-samples (and at BVML 40 g sub-samples) were taken from the pulverised material and mixed with flux, then fired at 1,100°C. The prill is then dissolved in Aqua Regia solution and the resultant liquor read by AAS with a detection limit of 0.01 g/t Au.

For the bottle roll analysis, a 1 kg portion of the pulverised material is placed into the BLEG bottles with cyanide solution and water. The mix is then rolled for 24 hours with an additional 2 hours for setting. A 50 ml sub-sample of the clear liquor is filtered into a flask and extracted into 5 ml of DIBK which is then read by AAS with a detection limit of 0.01 g/t Au.

The first 26 RC holes were analysed by 1 kg bottle roll, however, analysis of the tails showed that, on average, 20 % of the gold was not recovered with this method. Subsequently, almost all samples were analysed by fire assay. In total, 3,168 RC samples and 154 diamond core samples (from 2 drillholes) were analysed by bottle roll.

4.7 Quality assurance and quality control

Perseus retains overall responsibility for the database quality including quality assurance (QA) and quality control (QC). QAQC analysis shown in the following sections is sourced from Widenbar, 2013 (Sissengue Resource Estimate).

Perseus implemented company-wide standard operating procedures (SOPs) for drilling, sampling and QAQC in early 2012, however, the majority of the Sissengue data was collected prior to the implementation of these procedures. Current procedures are to industry standard.

Extensive quality control (QC) programs have been used throughout the Perseus drilling programs to check the precision and accuracy of the laboratory analysis. The results of the QC analysis indicate that reasonable precision and accuracy has been achieved. Perseus considers that the sample preparation, security and analytical procedures are adequate for the project.

Initially, every 25th RC sample was duplicated and a blank inserted at a ratio of 1 in 25. Since April 2007, certified standard material was submitted in a rate of 1 in 50 samples and then at a rate of 1 in 20 since early 2008 for both RC and diamond core samples. Certified material was supplied by Gannet and Rocklabs until November 2008 and Geostats Pty Ltd (Geostats) since November 2008. QC samples were inserted by Perseus employees prior to sample dispatch.

4.7.1 Blanks

Blank reference material samples were inserted during RC and diamond sampling as part of the QC routine. Table 4.3 lists the blanks used since 2011. The analysis of the blank material indicates that generally there is good sample preparation with little evidence of contamination (Appendix A). Two isolated occurrences of failed blanks at ALS (Job No BK11116000 in 2011 and BK11224176 in 2012) remain unexplained. In both cases, the preceding samples were not or only very weakly mineralised and contamination is not likely.

Blank analysis prior to 2011 has been discussed in Runge's Technical Report (Runge, 2010) and indicates no material issues.

Table 4.3 Blanks used since 2011

CRM	Au Value ppb	Standard Deviation	Source
GLG302-4	3.23	1.27	Geostats
GLG307-1	2.86	1.70	Geostats
GLG307-3	2.83	1.80	Geostats
GLG907-1	3.58	2.61	Geostats
GLG910-5	2.78	1.58	Geostats
GLG911-3	3.74	2.84	Geostats

4.7.2 Standards

Certified reference material (CRM) sourced from Gannet and Rocklabs (until November 2008) and Geostats (since November 2008) has been used to test the accuracy of the sample analysis for both RC and diamond core samples. Since beginning of 2011, 29 different CRM have been used covering a grade range between 0.154 g/t Au and 7.24 g/t Au (Table 4.4).

Since early 2011, QAQC data has been reviewed for each of the assay batches upon return from the laboratory. QAQC samples which fall outside specified tolerance ranges are flagged and investigated.

Results of CRM assays were plotted over time for each sample batch and over grade range for each laboratory. An analysis of performance of CRMs by type and grade range shows that all three laboratories operate satisfactorily. 1 % of all standards submitted since 2011 reported values 3 standard deviation below or above the expected value, in all 3 laboratories (Table 4.5). This is considered by Snowden and Perseus to be acceptable and in line with industry standards.

Detailed quality control plots for all batches since 2011 are shown in Appendix B; earlier batches have been discussed in Runge's Technical Report (Runge, 2010) and indicate no material issues.

Table 4.4 CRM used since 2011

CRM	Au Value ppm	Standard Deviation	Source
GLG304-1	0.154	0.017	Geostats
G902-1	0.390	0.040	Geostats
G909-6	0.570	0.030	Geostats
G999-2	0.630	0.060	Geostats
G398-4	0.660	0.050	Geostats
G901-9	0.690	0.040	Geostats
G998-6	0.800	0.060	Geostats
G907-2	0.890	0.060	Geostats
G907-2	0.890	0.060	Geostats
G910-10	0.970	0.040	Geostats
G302-6	0.990	0.050	Geostats
G306-2	1.049	0.072	Geostats
G307-2	1.080	0.050	Geostats
G02	1.200	0.080	Geostats
G307-8	1.990	0.080	Geostats
G302-3	2.330	0.120	Geostats
G308-3	2.500	0.110	Geostats
G901-1	2.580	0.130	Geostats
G909-5	2.630	0.100	Geostats
G398-6	2.940	0.160	Geostats
G998-1	2.950	0.120	Geostats
G907-4	3.840	0.150	Geostats
G397-2	4.490	0.180	Geostats
G306-7	4.500	0.200	Geostats
G307-5	4.870	0.172	Geostats
G904-8	5.530	0.180	Geostats
G996-7	5.990	0.390	Geostats
G905-10	6.750	0.250	Geostats
G906-8	7.240	0.270	Geostats

Table 4.5 CRM results since 2011

CRM				ALS / Mali *			Intertek / Ghana **			BVML / Mali ***			All CRM		
ID	Source	Exp Value	Std Dev	#	> +-3 Std Dev	% Failed	#	> +-3 Std Dev	% Failed	#	> +-3 Std Dev	% Failed	#	> +-3 Std Dev	% Failed
ST16/5357	Gannet	0.52	0.03	7	0	0.0							7	0	0.0
GLG304-1	Geostats	0.154	0.017	70	0	0.0							70	0	0.0
G902-1	Geostats	0.39	0.04							77	0	0.0	77	0	0.0
G909-6	Geostats	0.57	0.03	129	2	1.6	48	5	10.4				177	7	4.0
G999-2	Geostats	0.63	0.06	190	0	0.0	136	0	0.0				326	0	0.0
G398-4	Geostats	0.66	0.05				136	0	0.0	11	0	0.0	147	0	0.0
G901-9	Geostats	0.69	0.04	30	1	3.3							30	1	3.3
G998-6	Geostats	0.8	0.06				19	0	0.0	28	1	3.6	47	1	2.1
G907-2	Geostats	0.89	0.06	155	0	0.0	154	1	0.6	6	0	0.0	315	1	0.3
G910-10	Geostats	0.97	0.04							33	1	3.0	33	1	3.0
G302-6	Geostats	0.99	0.05	24	2	8.3							24	2	8.3
G306-2	Geostats	1.049	0.072	18	0	0.0	116	0	0.0	19	0	0.0	153	0	0.0
G307-2	Geostats	1.08	0.05	26	2	7.7							26	2	7.7
G02	Geostats	1.2	0.08	14	0	0.0							14	0	0.0
G307-8	Geostats	1.99	0.08							7	1	14.3	7	1	14.3
G308-3	Geostats	2.5	0.11							23	0	0.0	23	0	0.0
G901-1	Geostats	2.58	0.13				78	1	1.3				78	1	1.3
G909-5	Geostats	2.63	0.1	21	0	0.0	19	0	0.0				40	0	0.0
G398-6	Geostats	2.94	0.16	8	1	12.5							8	1	12.5
G907-4	Geostats	3.84	0.15	170	0	0.0	63	1	1.6				233	1	0.4
G397-2	Geostats	4.49	0.018				4	1	25.0	48	0	0.0	52	1	1.9
G307-7	Geostats	4.5	0.2	13	0	0.0							13	0	0.0
G904-8	Geostats	5.53	0.18	6	0	0.0							6	0	0.0
G996-7	Geostats	5.99	0.39				126	0	0.0	6	0	0.0	132	0	0.0
G905-10	Geostats	6.75	0.25				10	0	0.0				10	0	0.0
G906-8	Geostats	7.24	0.27	148	1	0.7				189	2	1.1	337	3	0.9
Summary				1,029	9	0.9	909	9	1.0	447	5	1.1	2,385	23	1.0

** ALS/Mali June 2011 to June 2012, ** Intertek/Ghana July 2012 to January 2013, *** BVML/Mail December 2012 to January 2013*

4.7.3 Field duplicates

A total of 1,696 field duplicates were taken during RC drilling between February 2008 and the December 2012. Initially, nominally every 50th RC sample was duplicated. However, with increasing knowledge of the mineralisation, selective intervals were duplicated, usually one 2 m composite per drillhole, to achieve a meaningful data set and to avoid duplicating barren intervals.

No field duplicates have been submitted from diamond core drilling as Perseus considered the use of ¼ core as a field duplicate to be inadequate given the style of mineralisation, and the submission of the full remaining ½ core was considered to be undesirable. Snowden recommends that some of the remaining 50 % splits should be submitted to assess the precision of the sampling process, even though it will mean there will be some intervals with no residual core.

When outliers (both original and duplicate sample >5 g/t Au) and near detection limit samples (below 0.1 g/t Au = 10x detection limit) are removed, the Analysis of Variance (ANOVA) of the field duplicates shows that no systematic sample bias has been introduced during routine RC sampling: means and standard deviation of both populations are quasi-identical (Figure 4.2). However, the HARD plot (without outliers removed) indicates that only 50 % of duplicates are within 20 % of the value of the original samples. This suggests a nuggety mineralisation and supports the high nugget effects, particularly for the sediment domains.

4.7.4 Pulp duplicates

A total of 4,557 pulp duplicates were analysed at Intertek in Ghana for diamond core samples. These pulp duplicates are a mixture of random and preferential re-assays (internal laboratory duplicates) on selected mineralised samples.

The HARD plot shows that 90 % of the duplicates are within 10 % of the original samples which is acceptable precision for pulp duplicates (Figure 4.3).

Figure 4.2 Analysis of Sissingue RC drilling field duplicates

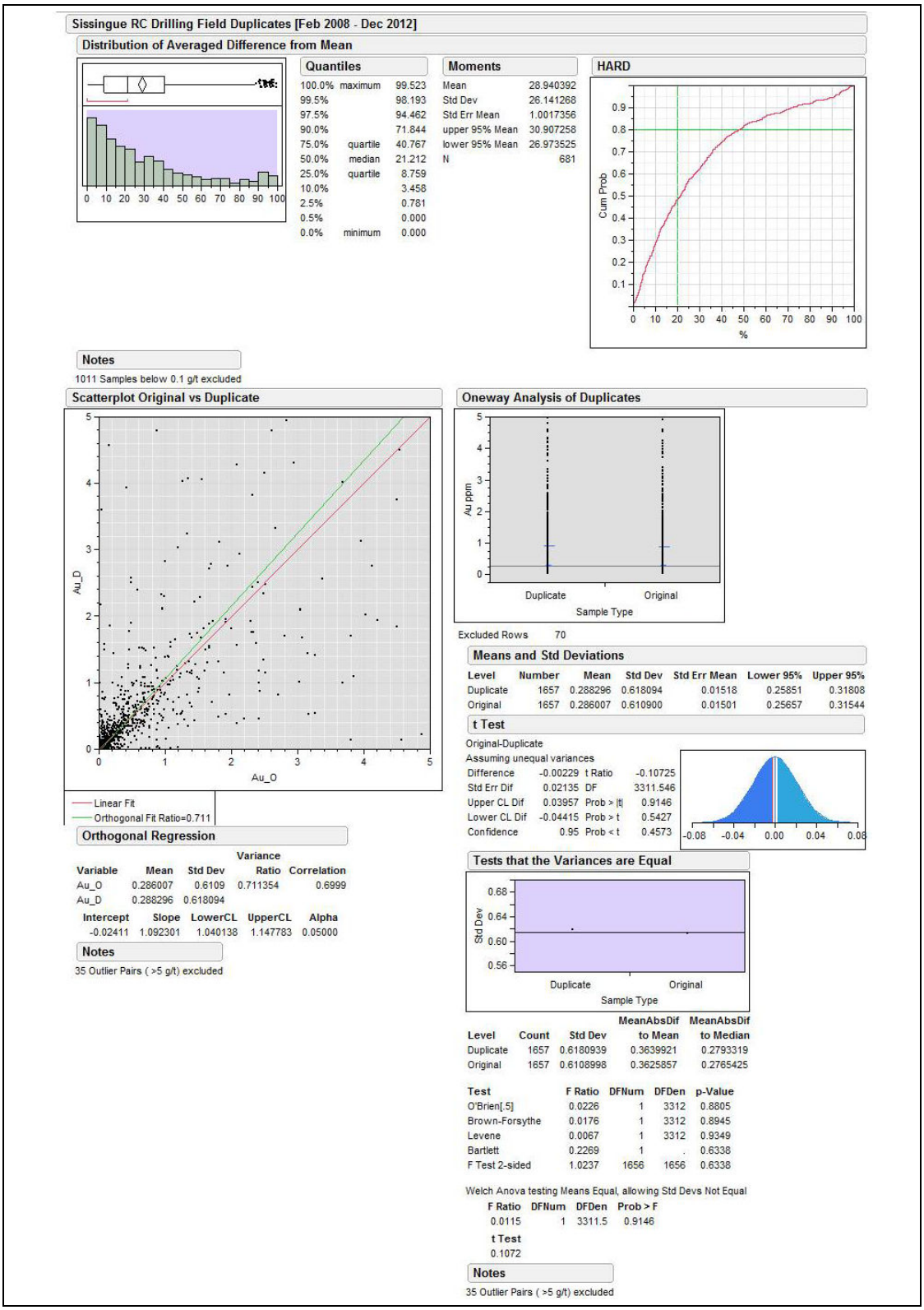
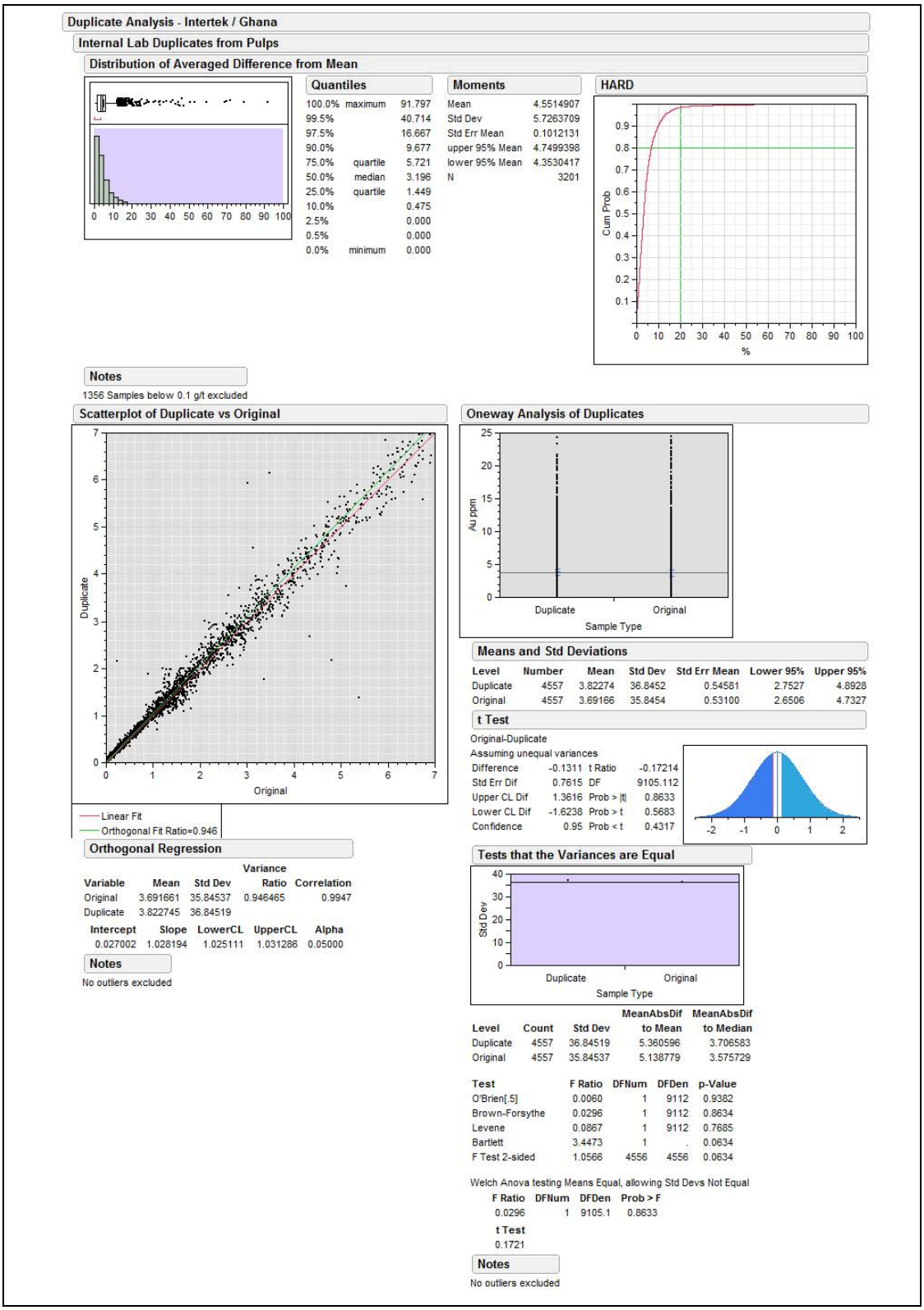


Figure 4.3 Analysis of Sissingue diamond pulp duplicates



4.7.5 Internal laboratory quality control

The following outlines the internal laboratory quality control procedures being carried out. Perseus considers these procedures to be adequate and in line with industry standard.

ALS internal quality control

The following procedure was forwarded by ALS for the quality assurance and quality control (QAQC) protocols:

- Preparation:
  - Samples are crushed to >70 % passing 2 mm and pulverised to better than 85 % passing 75 um.
  - Screen tests are done on 5 % of all samples to ensure that this is achieved.
  - A duplicate prep sample is taken after every 50th sample, prepared and subject to analyses.
- Analytical:
  - Work in batches of 24, consisting of 20 client samples, a CRM, a blank and 2 repeat samples.
  - Participate in round robins tests organized from ALS Vancouver on a monthly basis.
  - Participate in independent round robin analyses from Geostats, Australia on a six monthly basis.
  - Participate in six monthly tests from Rocklabs New Zealand in proficiency testing.

Intertek internal quality control

Intertek has several control procedures to monitor precision and accuracy of results. These are:

- Internal standards (1 in 20).
- Internal duplicates (1 in 20).
- Internal repeats (1 in 20).
- Reagent and sample blanks (1 in 20).
- Inter-laboratory cross checks.
- Sizing tests on pulverised material.
- Blank tests on jaw crusher, roll mills and pulverising mills.
- Fire assay copper maps and loss-lead maps.
- Repeat assays on unusual outliers (upon visual check of results).

BVML internal quality control

The following procedure was forwarded by BVML for the QAQC protocols:

- Internal standards (2 in 25 – one blank, one CRM).
- Internal duplicates (3 in 50).
- Sizing test on jaw crusher (2 % for 90 % passing 2 mm).
- Sizing tests on pulverised material (5 % of sample passing 85 % 75 um).

4.7.6 Umpire assaying

A total of 673 samples including 73 quality control samples (CRMs and blanks) where submitted in 10 batches for umpire assaying between January 2010 and June 2011. Pulps of samples previously assayed by Transworld/Intertek were randomly selected from mineralised intervals and submitted to ALS Chemex in Kumasi/Ghana and Genalysis in Perth/Australia.

An overview of comparison between principal and umpire laboratory is given in Table 4.6. Globally, the umpire assays were lower by about 8 %, with outliers >10 g/t Au removed.

A likely explanation of this discrepancy is that preferential re-sampling for umpire assaying introduced a bias into the procedure. This is supported when comparing only those samples where umpire assay results lie within ±30 % of the original value (Table 4.7), under the assumption that wider margins would not result from inaccurate analysis, but from inconsistency of the sample material and poor repeatability. Gold grains physically settling down in the sample bag during storage could also play a role when the re-assaying laboratory has scooped pulp material from the top of the bag: an issue that has been observed elsewhere during re-assaying exercises. The results in Table 4.7 suggest that Intertek, as principal analytical laboratory, performed satisfactorily over the time period observed.

Figure 4.4 shows quantile to quanitle (QQ) plots comparing the primary and umpire laboratory results for all assays (left) and assays within ±30 % (right).

Umpire assaying has been discontinued after June 2011 mainly due to continuous logistic difficulties (pulps from Intertek laboratory stored in Ghana) and political instability in the wake of general elections in the Ivory Coast in late 2011.

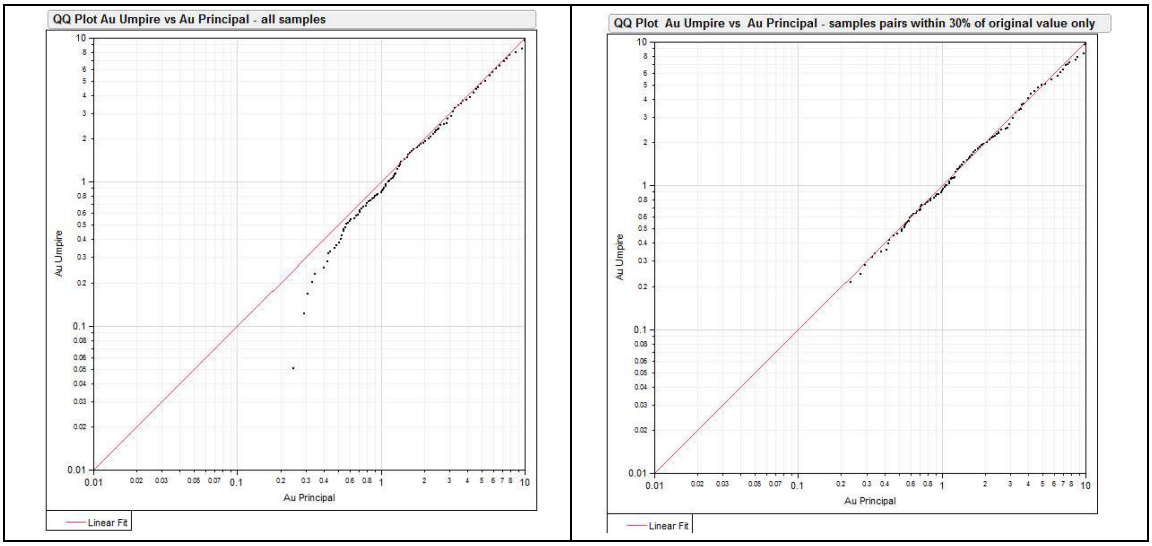
Table 4.6 Umpire laboratory results

Date	Umpire lab	# Samples	Original lab	All samples				Outliers (>10 g/t Au) excluded			
				Mean Au g/t umpire	Mean Au g/t original	Difference Au g/t	Difference %	Mean Au g/t umpire	Mean Au g/t original	Difference Au g/t	Difference %
January 2010	ALS Ghana	51	TWL	2.98	2.84	0.1	4.8	2.18	2.11	0.1	3.6
February 2010	ALS Ghana	50	TWL	4.67	5.08	-0.4	-8.1	2.34	2.66	-0.3	-12.1
March 2010	ALS Ghana	45	TWL	4.68	3.92	0.8	19.4	1.52	1.46	0.1	3.8
June 2010	Genalysis Australia	139	TWL	4.52	4.98	-0.5	-9.1	2.78	3.14	-0.4	-11.5
July 2010	ALS Ghana	93	TWL	11.06	36.92	-25.9	-70.0	2.13	2.31	-0.2	-7.9
August 2010	ALS Ghana	73	TWL	2.57	3.09	-0.5	-17.0	1.65	1.76	-0.1	-6.2
September 2010	ALS Ghana	46	TWL	1.61	1.76	-0.2	-8.6	1.49	1.51	0.0	-1.3
October 2010	ALS Ghana	46	Intertek	0.89	1.37	-0.5	-35.3	0.89	1.37	-0.5	-35.3
November 2010	ALS Ghana	41	Intertek	2.19	1.55	0.6	41.4	1.59	1.44	0.2	10.8
June 2011	ALS Ghana	16	Intertek	0.92	1.07	-0.1	-13.8	0.92	1.07	-0.1	-13.8
Total		**600**		**4.43**	**8.59**	**-4.16**	**-48.4**	**1.96**	**2.12**	**-0.16**	**-7.5**

Table 4.7 Umpire laboratory results – only samples pairs within ±30 %

Date	Umpire Lab	# Samples	Original lab	Mean Au g/t umpire	Mean Au g/t original
January 2010	ALS Ghana	35	TWL	3.23	3.22
February 2010	ALS Ghana	35	TWL	5.86	6.18
March 2010	ALS Ghana	30	TWL	6.25	5.17
June 2010	Genalysis Australia	72	TWL	3.80	4.02
July 2010	ALS Ghana	58	TWL	2.97	3.01
August 2010	ALS Ghana	46	TWL	1.80	1.88
September 2010	ALS Ghana	31	TWL	1.39	1.51
October 2010	ALS Ghana	27	Intertek	1.11	1.14
November 2010	ALS Ghana	29	Intertek	1.65	1.61
June 2011	ALS Ghana	9	Intertek	0.74	0.74
Total		**372**		**3.12**	**3.13**

Figure 4.4 Umpire laboratory versus primary laboratory QQ plots – all assays (left) and assays within ±30 % (right)



4.7.7 Sample security

Samples from RC drilling are collected and bagged at drill site during the drilling operation. Core samples are cut in a central facility in Tengrela and samples placed into sample bags as they are cut.

All samples are then catalogued and placed in large woven bags and sealed (Figure 4.5) prior to dispatch to ALS, Intertek or BVML for preparation and analysis. Dispatch from site to Korhogo (Intertek) is by Perseus staff and vehicles. Samples dispatched to ALS and BVML are collected from Tengrela by staff and vehicles of the respective laboratories.

Sample numbering is limited to non-descriptive number. All aspects of the process are supervised by Perseus personnel and limited opportunity exists for tampering with samples.

Figure 4.5 Samples at Tengrela office facility ready for dispatch (after Runge, 2010)



4.7.8 Conclusions

Perseus retains overall responsibility for the database quality including quality assurance (QA) and quality control (QC).

Perseus implemented company-wide standard operating procedures (SOPs) for drilling, sampling and QAQC in early 2012, however, the majority of the Sissengue data was collected prior to the implementation of these procedures. Current procedures are to industry standard.

Perseus considers that the sample preparation, security and analytical procedures are adequate for the project.

Extensive quality control (QC) programs have been used throughout the Perseus drilling programs to check the precision and accuracy of the laboratory analysis. The results of the QC analysis suggests reasonable precision and accuracy has been achieved.

Overall the QC data provided for the project does not indicate any bias and supports the suitability of the assay data for resource estimation purposes.

4.8 Data validation

Perseus carried out detailed validation of the dataset and retains overall responsibility for the database quality. All drillhole data was validated during data entry by Perseus including:

- Checks for duplicate collars (LogChief, Datashed).
- Checks for missing samples (Datashed).
- Checks for downhole from-to interval consistency (LogChief, Datashed).
- Checks for overlapping samples (LogChief, Datashed).
- Checks for samples beyond hole depth (LogChief, Datashed).
- Checks for inexistent or misspelt log items (LogChief).
- Check for missing assays (Datashed).
- Check for downhole information beyond hole depth (Datashed).

Snowden carried out a basic statistical and visual validation prior to estimation including:

- Drillholes with overlapping sample intervals.
- Sample intervals with no assay data.
- Duplicate records.
- Assay grade ranges.
- Collar coordinate ranges.
- Valid hole orientation data.

No issues were found with the data and Snowden considers the data to be appropriate for estimation.

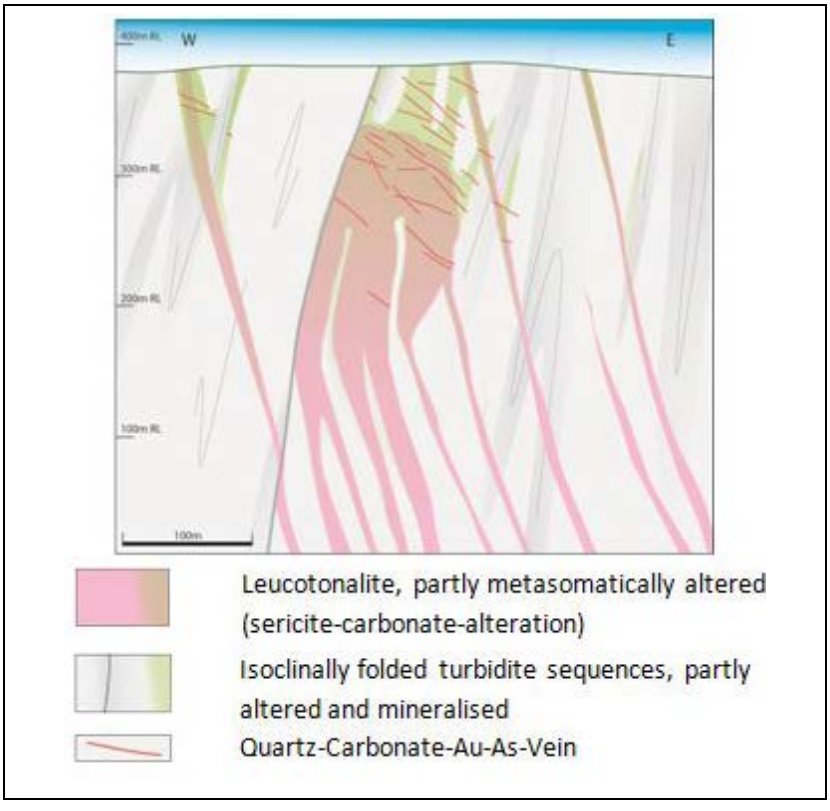
5 Geological interpretation and modelling

Perseus carried out the geological interpretation for Sissengue. Snowden reviewed the interpretation prior to estimation and considers it to be an accurate reflection of the geology of the orebody. The following section is taken from Widenbar (2013) and summarises the geological interpretation provided by Perseus.

In the central part of the Sissingue prospect ("Sissingue Main"), the major host rock is a medium to coarse grained, veined, altered and sulphidised granite. Chemically identical to the porphyritic dykes, it is formed by several of these dykes that have been attenuated by a structural feature, possibly a fold plane, to flow into each other and to form a pear-shaped, approximately 50 m thick body (Figure 5.1).

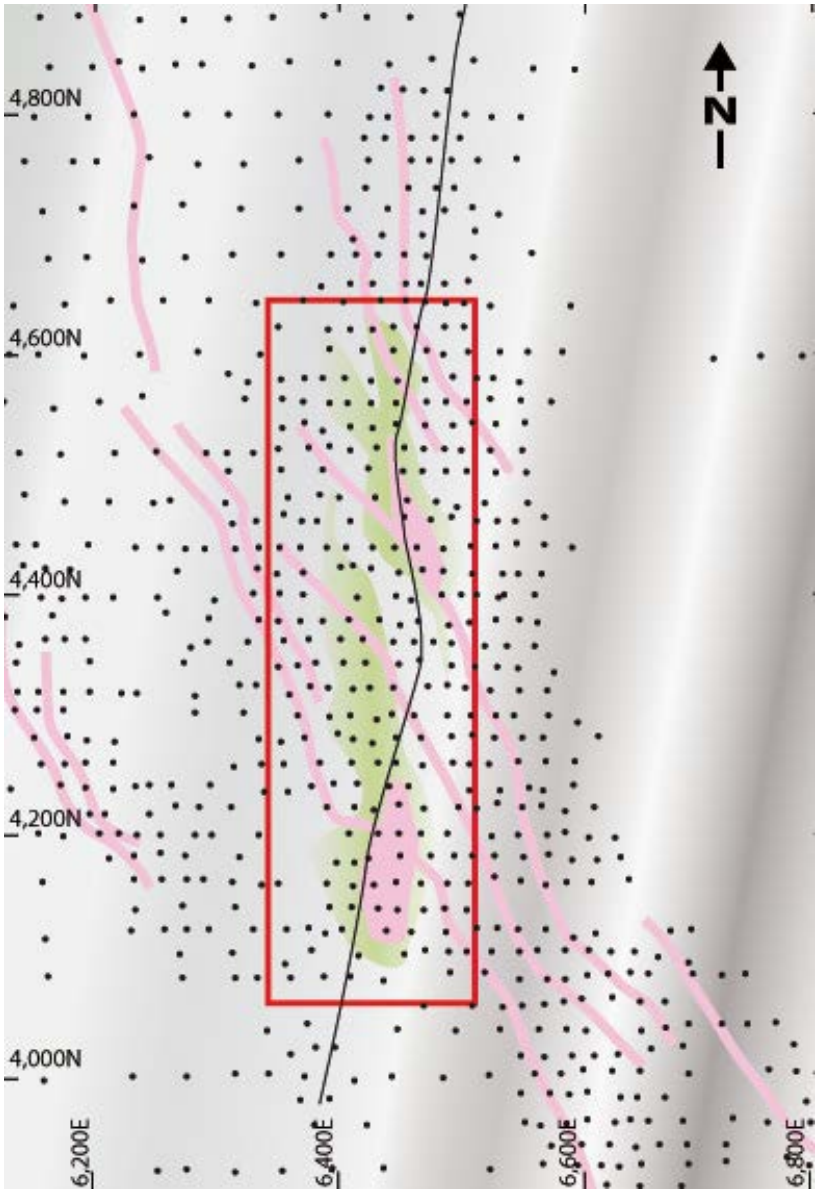
Alteration is most intense and gold grades, on average, are highest in the roof of this intrusive body and in the coarse grained sediments above. Emplacement of the dykes is believed to have occurred in several stages or over an extended period as several dykes cross cut the granitic body, some with fuzzy contacts (interpreted as emplaced during cooling period of the granitic body), others with sharp contacts (emplaced after cooling of granite). Gold mineralisation is relatively homogeneous in the upper portion the granite. At depth, with decreasing intensity of alteration, the tenor of the mineralisation decreases.

Figure 5.1 Schematic cross section (around, 1,154,200 mN) (after Widenbar, 2013)



Elsewhere, mineralisation is interpreted to be associated with isolated dykes and altered sediments in their respective vicinities. This distribution leads to a bi-directional ore body: Granite and associated altered sediments in north-northeastern extent and with sub-vertical or steep westerly dips, and north-west striking dykes with associated altered sediments, obliquely cross cutting and steep to moderate easterly dips. The most significant zone of gold mineralisation is the Main prospect with its granitic bodies, various dykes and a pervasive alteration halo due to the larger mass of the intrusives (Figure 5.2).

Figure 5.2 Surface geology of Sissingue Main prospect (red) with drill collar layout (after Widenbar, 2013)



5.1 Geology and mineralisation domains

The mineralisation wireframes were supplied by Perseus as dxf format files. A plan view of the major modelling domains and a list of the files is shown in Figure 5.3 and Table 5.1 respectively.

Figure 5.3 Plan view of mineralisation domains with drillholes and project areas

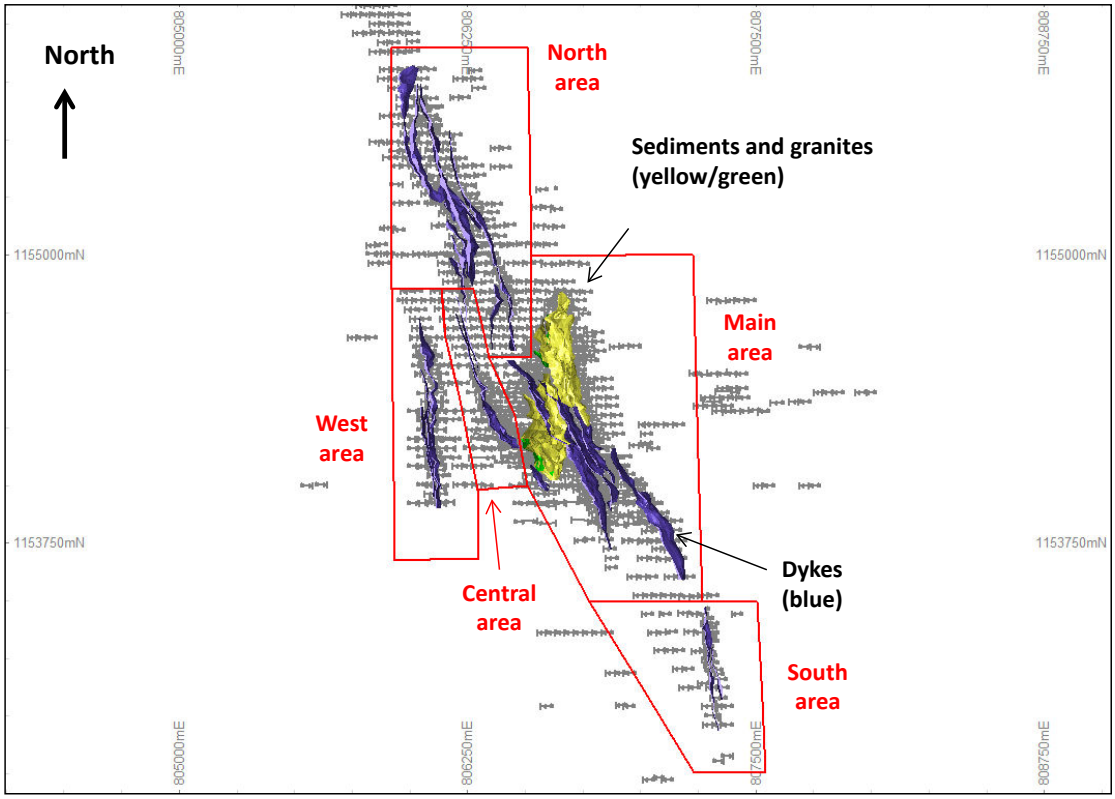


Table 5.1 Summary of mineralisation domains

Prospect	Domain	Wireframe
Main	Granite South	granite1.dxf
	Granite North	granite2.dxf
	Sediments South	sed_env1.dxf
	Sediments North	sed_snv2.dxf
	Dyke 1	dyke_1.dxf
	Dyke 2	dyke_2.dxf
	Dyke 3	dyke_3.dxf
	Dyke 4	dyke_4.dxf
	Dyke 5	dyke_5.dxf
	Dyke 6	dyke_6.dxf
	Dyke 7	dyke_7.dxf
	Dyke 8	dyke_8.dxf
Central	Dyke 11	dyke_c1.dxf
	Dyke 12	dyke_c2.dxf
	Dyke 13	dyke_c3.dxf
	Dyke 14	dyke_4c.dxf
North	Dyke21	dyke_n1.dxf
	Dyke22	dyke_n2.dxf
	Dyke23	dyke_n3.dxf
West	Dyke31	dyke_w1.dxf
	Dyke32	dyke_w2.dxf
	Dyke33	dyke_w3.dxf
South	Dyke41	dyke_s1.dxf
	Dyke42	dyke_s2.dxf

Mineralisation was domained by its host lithology for modelling (granite, porphyritic dykes and sediments). Within the dyke domain mineralised sediments in the alteration halo of the dykes were included in the dyke domain to maintain a minimum width of the wireframes (see below) and to maintain continuity along strike. Where geological contacts were not clearly controlling the distribution of mineralisation, a grade cut-off of 0.3 g/t Au was used to construct Mineral Resource boundaries and to provide overall geometry to mineralised zones. A minimum of 4 m width was used for the wireframes and samples of grades below the nominal cut-off of 0.3 g/t Au were included where the wireframe would otherwise be less than 4 m wide. Analysis of the global grade distribution shows that there is a natural change in grade population at around 0.3 g/t Au.

The interpreted geology and grade boundaries were digitised on 10 m and 20 m spaced, west-east orientated vertical sections and manually triangulated to form wireframe solids. To form ends to the wireframes along strike, either:

- The end sections strings were copied to a position midway to the next section and adjusted to match dip, strike and plunge of the zone.
- A single point was digitised midway to the next section. Vertically, strings were closed at a depth that matched that of the adjacent sections.

The wireframe objects were validated prior to compiling the volume model. In total, 23 solids were created.

5.2 Topography and weathering domains

A topographic surface and three surfaces of the weathering profile were created by Perseus who supplied the wireframes to Snowden as dxf files as detailed in the following table (Table 5.2).

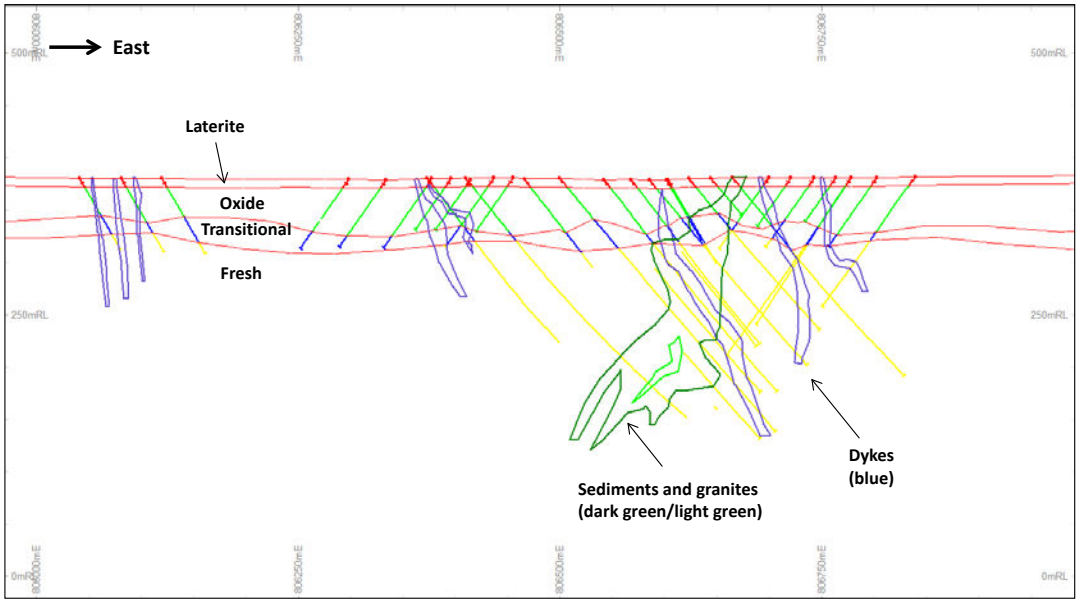
Table 5.2 Summary of topography and weathering wireframes

Description	Wireframe
Topography	topo1.dxf
Base of laterite	bolat1.dxf
Base of total oxidation	botox.dxf
Base of transitional	bottr1.dxf

The topography covering the extent of the Sissingue Mineral Resource model was created as a digital terrain model (DTM) in Surpac using surveyed drillhole collars and an additional 639 survey points across the prospect.

Weathering interfaces were digitised on 20 m spaced, west-east orientated drill sections and automatically triangulated to form DTMs. A cross section illustrating the topography and the four weathering surfaces is shown in Figure 5.4.

Figure 5.4 Cross section view of weathering domains – 1154240 mN



6 Coding and compositing of data

The validated desurveyed drillhole data was coded with fields to describe the mineralisation domains and weathering domains as detailed in Table 6.1. Alpha numeric and numeric fields were coded for consistency with previous models.

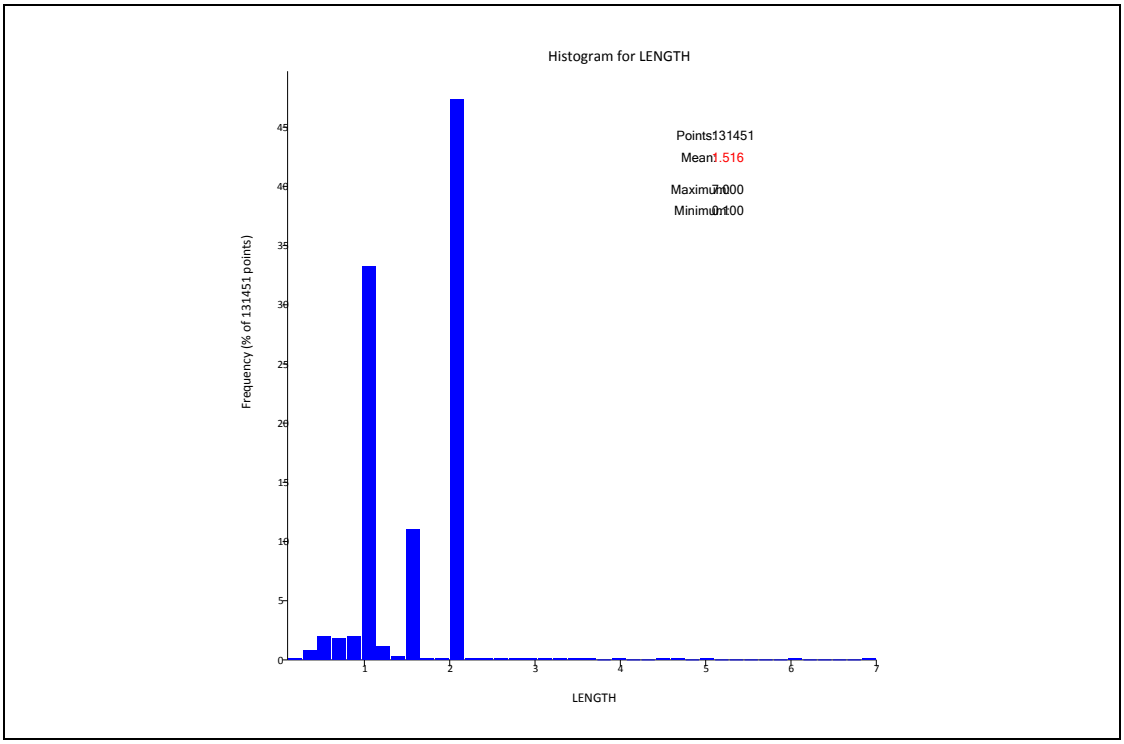
Table 6.1 Drillhole coding

Field	Value	Description
dom	1-9	Main dykes
	11-14	Central dukes
	21-23	Northern dykes
	31-33	Western dykes
	41-42	Southern dykes
	100	Southern granite mineralised domain
	200	Northern granite mineralised domain
	300	Northern sediments mineralised domain
	400	Southern sediments mineralised domain
domain	WASTE	Background
	GR	Granites
	SED	Sediments
	FSP	Dykes
weath_n	1	Laterite
	2	Oxide
	3	Transitional
	4	Sulphide
weatheri	OX	Laterite and oxide
	TR	Transitional
	SUL	Sulphide

The coded dataset was composited to 2 m downhole with the composite lengths adjusted to include all intervals and avoid the loss of residual samples (MODE=1 option in Datamine). Over 99 % of the raw samples are 2 m or less in length as illustrated in the histogram in Figure 6.1 which Snowden considers to be a good outcome.

The mineralisation domains (dom) and weathering domains (weath_n) were used as hard boundaries for compositing purposes.

Figure 6.1 Histogram of raw sample lengths



7 Domain boundary analysis

A statistical and geological analysis was carried out to determine the boundary conditions between the various geology, mineralisation and the weathering domains.

This analysis was used to determine which data to combine for statistical analysis, variography and estimation.

7.1 Geology and mineralisation domains

The granite, sediments and dykes were kept separate for statistical analysis, variography and estimation as they are different geological units with mostly hard boundaries between them. The southern and northern granite and sediment domains were also kept separate as there is a physical separation and a difference in grade tenor between the two areas, particularly for the granite (higher in the southern area, Figure 7.1).

Review of the individual dykes showed that they are statistically similar and, given the relatively small amount of data in the individual dykes, the dykes were combined for statistical analysis and variography (Figure 7.2). The dykes were kept separate for estimation as there are hard boundaries between them.

Figure 7.1 Log-probability plots for southern (100) and northern (200) granites

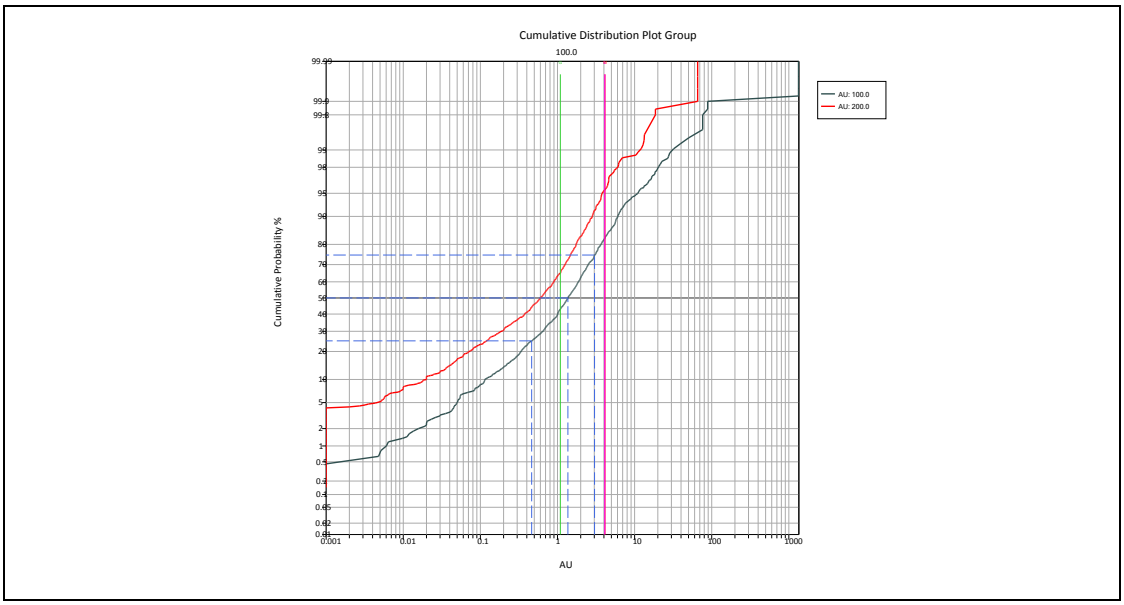
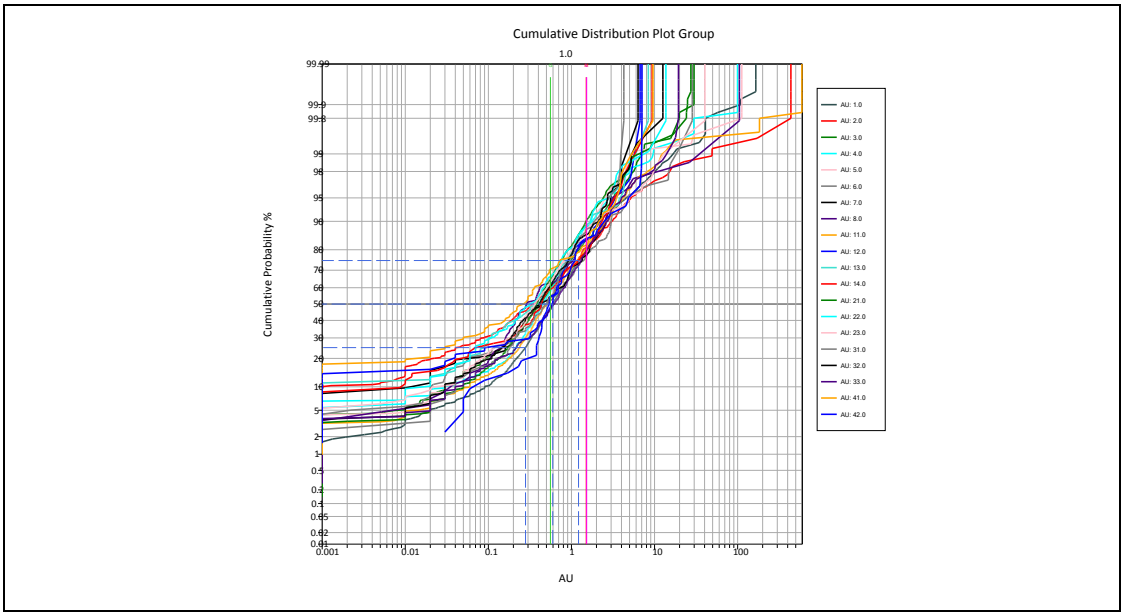


Figure 7.2 Log-probability plots for all dykes



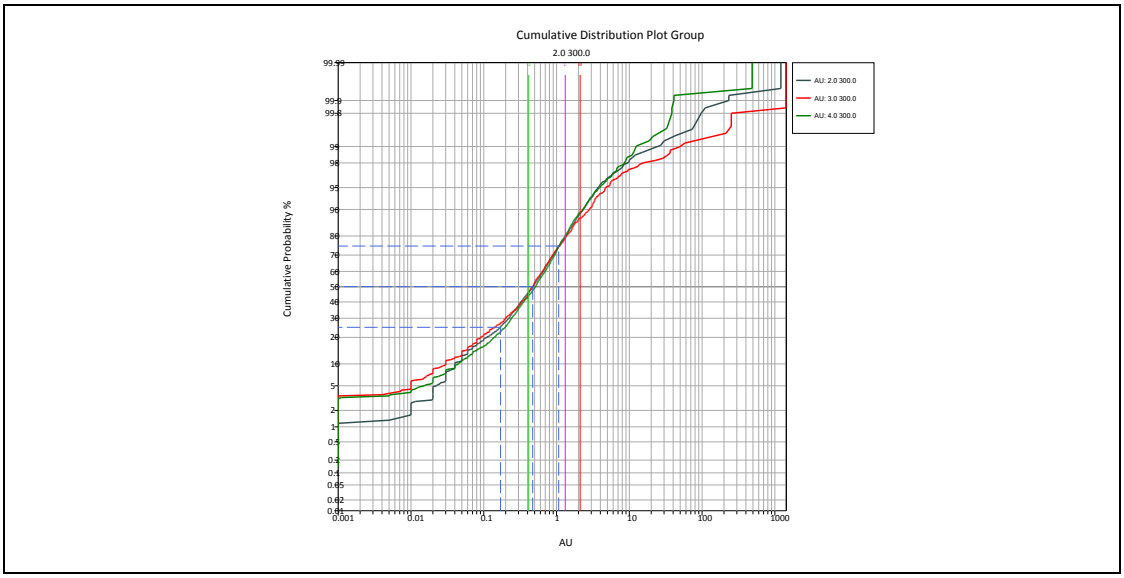
7.2 Weathering domains

Review of the oxide, transitional and fresh weathering domains within the granites, sediments and dykes indicates that the grade distributions have similar shapes, with typically higher grades in the transitional domain. Figure 7.3 shows an example of the oxide, transitional and fresh log probability plots for the northern granite.

This pattern of grade distribution fits with the geological model discussed in Section 3.4, which suggests that gold has been depleted in weathered (oxidised) material and descending waters resulted in gold enrichment in the transitional zone.

The transitional domain is typically narrow and contains limited data compared to the oxide and fresh domains.

Figure 7.3 Log-probability plots for oxide (2), transitional (3) and fresh (4) domains in the northern granite (300)



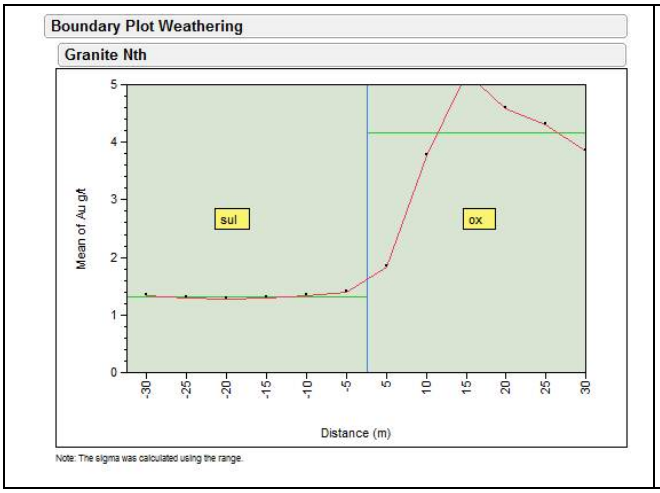
Perseus generated boundary plots to assess the change in grade across for the weathering domains for each of the mineralisation domains, except for the southern granite which contains little data outside of the fresh domain (Figure 7.4). Except for the southern sediments where the transition zone is clearly enriched, the boundary plots of all domains show strong differences between the oxide and fresh domains.

Based on the statistical analysis and boundary conditions, the oxide, transitional and fresh domains were kept separate for statistical analysis, however, they were combined for variography. The oxide and transitional data were used to estimate the oxide, while the transitional and fresh data were used to estimate the transitional and fresh domains.

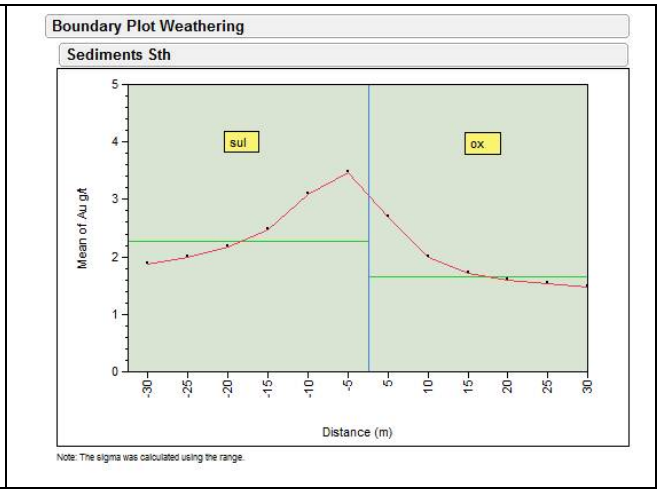
The laterite domain has very little data in it. Visual inspection of the drillhole grades between the laterite and the oxide does not show evidence of a depleted zone. As a result of the small dataset, the laterite was combined with the oxide for estimation. The laterite data was not used for variography.

Figure 7.4 Boundary plots showing change in grade across weathering domains

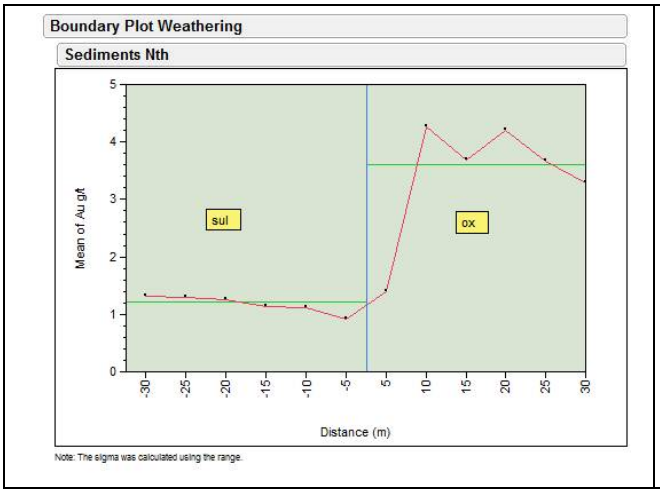
Northern granite



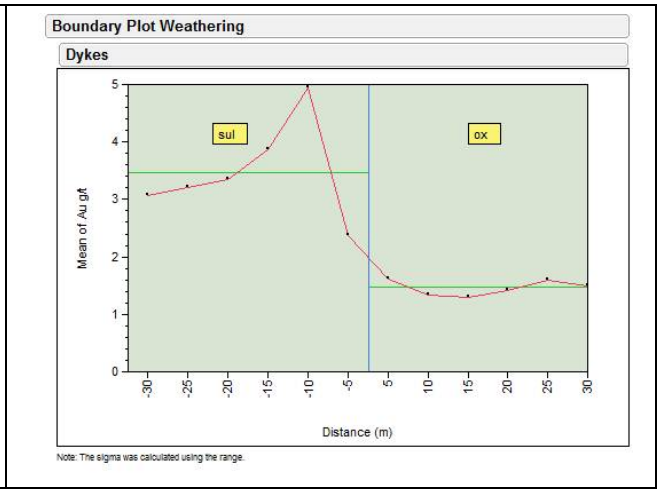
Southern sediments



Northern sediments



Dykes



8 Exploratory data analysis and top cuts

Statistical analysis was carried out on the gold composite data to determine the grade characteristics and to ensure that the domains are adequate for estimation. Based on the boundary analysis carried out initially, all dykes were combined for statistical analysis.

The data exhibits a bias as a result of data clustering. As a result, all statistics were declustered using cell weighted declustering with a cell size of 40 mE by 40 mN by 2 mRL.

Summary statistics (Table 8.1 to Table 8.3), histograms and log-probability plots (Appendix A) were generated for each mineralisation and weathering domain.

All gold grade distributions are positively skewed with high coefficient of variation (CV). The granite and sediment domains in particular contain significant outliers with three grades above 1000 g/t Au. Review of the histograms and log-probability plots for each domain indicates that there is some mixing of grade populations. As a result of the mixed populations, together with the high CV values and outliers evident in the granite and sediment domains, multiple indicator kriging was selected as an appropriate estimation method. As result, no top cuts were required for these domains.

For the lower grade dykes, ordinary kriging was selected as appropriate, with top cuts to limit the influence of the higher grade composites. A top cut of 30 g/t Au was selected for the dykes in all weathering domains, based on review of the histograms and natural breaks in the populations in the log-probability plots. Top cut statistics are presented at the bottom of Table 8.3.

Table 8.1 Summary statistics by mineralisation domain and weathering domain - granites

Statistic		Southern granite					Northern granite				
		Total	**Laterite**	**Oxide**	**Transitional**	**Fresh**	**Total**	**Laterite**	**Oxide**	**Transitional**	**Fresh**
Samples		1,095	0	0	37	1,058	690	2	57	73	558
Minimum		0.00	-	-	0.02	0.00	0.00	0.05	0.00	0.00	0.00
Maximum		1,336.00	-	-	51.78	1,336.00	65.82	0.09	5.13	65.82	18.65
Mean		**4.33**	**-**	**-**	**3.33**	**4.38**	**1.03**	**0.07**	**1.23**	**2.59**	**0.85**
Std Dev.		49.38	-	-	7.09	50.47	2.57	0.02	1.28	7.00	1.54
CV		11.40	-	-	2.13	11.53	2.49	0.35	1.04	2.70	1.81
Variance		2,438.00	-	-	50.30	2,547.00	6.59	0.00	1.63	49.00	2.39
Skewness		26.65	-	-	5.68	26.09	16.08	-	1.47	8.18	5.61
Percentile	10 %	0.10	-	-	0.35	0.10	0.01	0.05	0.02	0.00	0.01
	20 %	0.27	-	-	0.58	0.26	0.05	0.05	0.07	0.12	0.05
	30 %	0.47	-	-	0.95	0.44	0.13	0.05	0.37	0.63	0.12
	40 %	0.77	-	-	1.24	0.75	0.26	0.05	0.82	0.76	0.22
	50 %	1.15	-	-	1.56	1.11	0.45	0.05	0.91	1.24	0.38
	60 %	1.59	-	-	1.78	1.55	0.69	0.05	1.00	1.71	0.58
	70 %	2.13	-	-	2.14	2.12	1.00	0.05	1.32	2.23	0.88
	80 %	3.18	-	-	3.59	3.17	1.46	0.09	2.03	2.82	1.30
	90 %	5.48	-	-	4.89	5.48	2.39	0.09	2.94	4.54	2.00
	95 %	7.88	-	-	8.86	7.72	3.60	0.09	3.63	6.93	3.15
	97.50 %	16.28	-	-	18.34	15.14	4.64	0.09	5.05	11.13	4.35
	99 %	26.97	-	-	18.34	26.97	10.42	0.09	5.13	12.21	6.21

Table 8.2 Summary statistics by mineralisation domain and weathering domain - sediments

Statistic		Southern sediment					Northern sediment				
		Total	Laterite	Oxide	Transitional	Fresh	Total	Laterite	Oxide	Transitional	Fresh
Samples		5,087	101	987	454	3,545	4,091	209	1,796	629	1,457
Minimum		0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00
Maximum		286.00	3.75	60.48	137.70	286.00	1,420.55	23.22	1,209.20	1,420.55	487.31
Mean		1.29	0.80	1.34	1.46	1.28	1.70	0.88	1.33	2.34	1.88
Std Dev.		5.93	0.84	3.75	5.85	6.37	21.36	1.91	18.83	35.10	18.98
CV		4.59	1.04	2.81	4.00	4.97	12.55	2.16	14.13	15.00	10.11
Variance		35.14	0.70	14.08	34.21	40.58	456.10	3.64	354.40	1,232.00	360.30
Skewness		31.59	1.96	9.13	18.71	32.12	44.73	6.94	60.36	37.82	24.98
Percentile	10 %	0.02	0.16	0.03	0.03	0.02	0.04	0.07	0.04	0.03	0.05
	20 %	0.07	0.21	0.07	0.08	0.07	0.13	0.15	0.11	0.09	0.14
	30 %	0.16	0.28	0.13	0.24	0.16	0.23	0.24	0.21	0.19	0.24
	40 %	0.28	0.43	0.26	0.40	0.27	0.33	0.32	0.31	0.30	0.34
	50 %	0.43	0.50	0.43	0.55	0.42	0.46	0.38	0.46	0.43	0.51
	60 %	0.64	0.65	0.67	0.74	0.62	0.63	0.43	0.63	0.53	0.68
	70 %	0.95	0.88	1.00	1.02	0.93	0.84	0.64	0.82	0.72	0.90
	80 %	1.48	1.08	1.57	1.59	1.46	1.22	1.14	1.22	1.04	1.28
	90 %	2.59	2.02	2.90	3.05	2.49	2.19	1.98	2.03	2.30	2.28
	95 %	4.06	2.93	4.81	4.74	3.96	3.30	2.40	3.00	4.38	3.58
	97.50 %	6.32	3.18	7.72	8.16	6.14	5.94	3.78	4.01	5.87	6.72
	99 %	12.34	3.72	15.79	9.18	12.34	11.64	9.69	8.82	16.05	13.62

Table 8.3 Summary statistics by mineralisation domain and weathering domain - dykes

Statistic		Dykes				
		Total	Laterite	Oxide	Transitional	Fresh
Samples		7,288	459	3,591	1,009	2,229
Minimum		0.00	0.00	0.00	0.00	0.00
Maximum		593.80	105.03	437.60	99.72	593.80
Mean		1.27	1.15	1.18	1.17	1.48
Std Dev.		9.93	6.29	9.48	4.09	12.56
CV		7.79	5.46	8.02	3.50	8.48
Variance		98.56	39.51	89.90	16.69	157.70
Skewness		43.08	13.54	39.76	18.62	40.10
Percentile	10 %	0.04	0.03	0.03	0.05	0.05
	20 %	0.13	0.08	0.11	0.16	0.16
	30 %	0.23	0.17	0.22	0.25	0.25
	40 %	0.33	0.23	0.32	0.35	0.36
	50 %	0.44	0.33	0.43	0.46	0.47
	60 %	0.59	0.44	0.58	0.61	0.64
	70 %	0.83	0.60	0.81	0.90	0.87
	80 %	1.21	0.86	1.19	1.32	1.29
	90 %	2.15	1.57	2.07	2.26	2.33
	95 %	3.39	2.42	3.07	3.68	3.88
	97.50 %	5.25	3.72	4.70	5.38	6.20
	99 %	10.10	10.19	9.36	10.86	11.96
Top cut		30	30	30	30	30
Top cut count		17	2	5	2	8
Top cut mean		1.02	0.89	0.96	1.08	1.12
Top cut Std Dev.		2.38	2.84	2.22	2.28	2.54
Top cut CV		2.33	3.21	2.32	2.12	2.27
Top cut variance		5.68	8.08	4.94	5.22	6.45

9 Variogram analysis

9.1 Indicator variography – granites and sediments

Indicator variograms were generated and modelled for the southern and northern sediments using the combined oxide, transitional and fresh data. The southern and northern granites contained insufficient data for full indicator variography and as such the variogram models for the sediments were also used for the granite estimates.

Three extreme outliers were removed from the datasets prior to variography as they appear to be part of a very small, extremely high grade population which is not representative of the main population. These values are 1420.55 g/t Au and 1209.2 g/t Au in the northern sediments (next highest grade in domain is 487) and 1336.00 g/t Au in the southern granite (next highest grade in domain is 88).

Twelve indicator thresholds were defined based on a combination of ensuring an even spread of sample numbers and metal within each bin. Linear models between each bin were reviewed on the cumulative probability plots to ensure that the linear models honoured the statistical distribution within each bin.

Thresholds were defined as gold cut-off grades and the same cut-off grades were used for all domains, as there appears to be a consistent change in population at certain grades in all domains. These changes in population occur at around 0.3 g/t Au and 10.0 g/t Au.

Figure 9.1 and Figure 9.2 show the log-probability plots with the indicator thresholds highlighted and the cumulative distribution plots showing the linear model fit between the indicator thresholds, for the southern and northern sediments respectively.

Figure 9.1 Log-probability plot showing indicator thresholds and cumulative distribution plot showing linear model fit between indicator thresholds - southern sediments

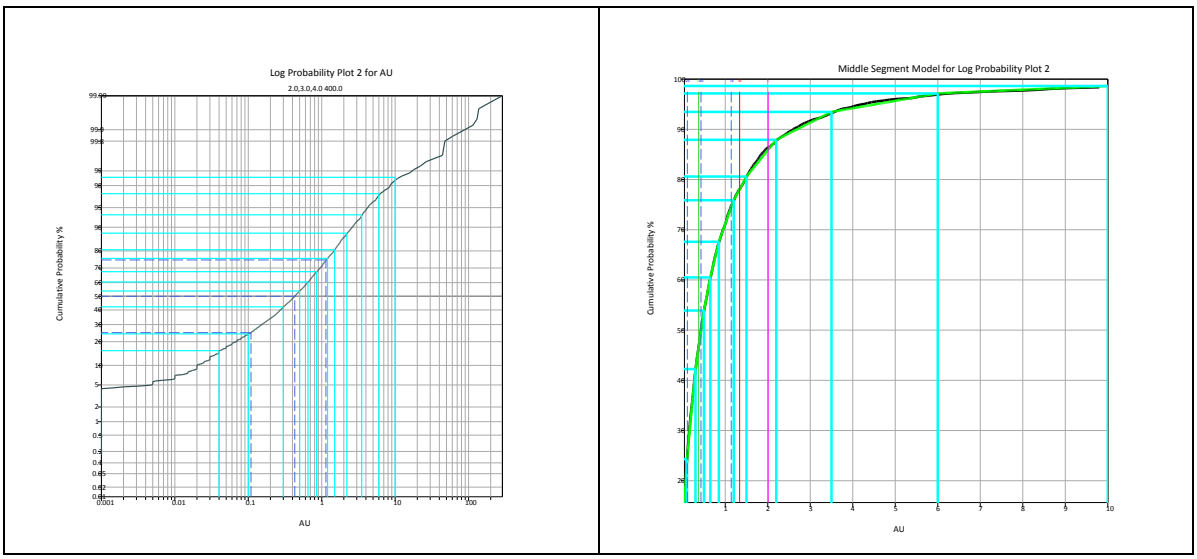
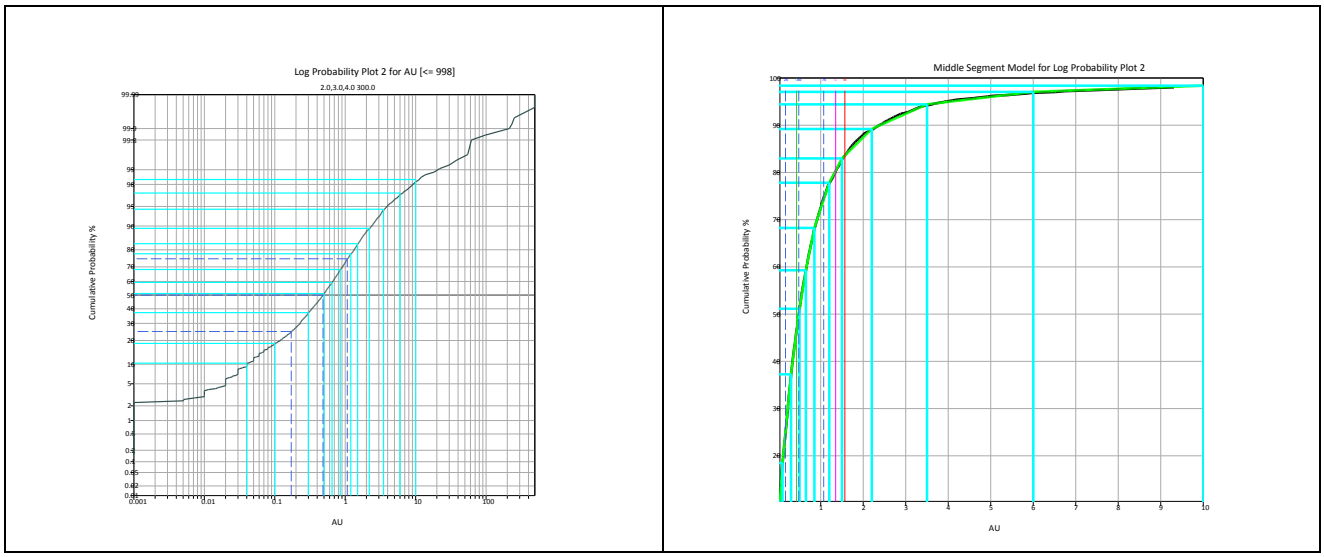


Figure 9.2 Log-probability plot showing indicator thresholds and cumulative distribution plot showing linear model fit between indicator thresholds - northern sediments



Variogram fans were assessed to determine the dominant orientation of the mineralisation at each threshold. Results of the analysis for both southern and northern sediments indicate that the lower grade mineralisation, below 0.3 g/t Au, trends to north-northwest, dipping sub vertically to the west, in line with the main geological trend. At the higher grade above 0.3 g/t Au, the mineralisation is plunging to the east-southeast, in line with expectation based on the orientation of high grade vein sets in these areas.

A downhole variogram was modelled for each threshold to determine the nugget, followed by directional variography. The variograms for the upper two or three thresholds were poorly structured and the patterns seen in the previous thresholds were used to predict the variogram models at these higher grades.

Variogram models for each threshold are detailed in Table 9.1 and Table 9.2. Variogram plots are included in Appendix D. Figure 9.3 and Figure 9.4 illustrate the grade continuity ellipse at low grade thresholds (0.1 g/t Au) and higher grade thresholds (0.5 g/t Au) respectively.

Table 9.1 Indicator variogram models for gold – southern sediments

Domain	Percentile	Cut-off	Orientation	Nugget	Structure 1		Structure 2	
					Sill	Range	Sill	Range
Southern sediments	15.65	0.04	00→350	0.49	0.24	25	0.27	145
			-70→260			25		145
			20→260			10		70
	24.30	0.10	00→350	0.49	0.28	10	0.23	70
			-70→260			10		70
			20→260			10		40
	42.25	0.30	00→350	0.55	0.32	10	0.13	60
			-70→260			10		60
			20→260			6		30
	53.90	0.50	-42→119	0.55	0.32	7	0.13	30
			34→173			7		30
			-30→240			5		15
	60.50	0.65	-42→119	0.55	0.31	6	0.14	25
			34→173			6		25
			-30→240			5		10
	67.60	0.85	-42→119	0.56	0.28	5	0.16	20
			34→173			5		20
			-30→240			5		10
	75.88	1.20	-42→119	0.59	0.20	5	0.21	20
			34→173			5		20
			-30→240			5		10
	80.60	1.50	-42→119	0.66	0.13	5	0.21	20
			34→173			5		20
			-30→240			3		5
	87.92	2.20	-42→119	0.67	0.12	5	0.21	20
			34→173			5		20
			-30→240			3		5
	93.47	3.50	-42→119	0.67	0.12	5	0.21	15
			34→173			5		15
			-30→240			3		5
	97.15	6.00	-42→119	0.67	0.12	4	0.21	10
			34→173			4		10
			-30→240			3		5
	98.64	10.00	-42→119	0.67	0.12	3	0.21	5
			34→173			3		5
			-30→240			3		5

Table 9.2 Indicator variogram models for gold – northern sediments

Domain	Percentile	Cut-off	Orientation	Nugget	Structure 1		Structure 2	
					Sill	Range	Sill	Range
Northern sediments	10.35	0.04	00→350	0.54	0.19	30	0.27	100
			-70→260			30		100
			20→260			10		70
	18.52	0.10	00→350	0.54	0.19	20	0.27	80
			-70→260			20		80
			20→260			6		60
	37.28	0.30	00→350	0.59	0.21	20	0.20	80
			-70→260			20		80
			20→260			6		60
	51.17	0.50	-42→119	0.59	0.21	15	0.20	60
			34→173			15		60
			-30→240			6		30
	59.32	0.65	-42→119	0.61	0.21	10	0.18	40
			34→173			15		40
			-30→240			6		25
	68.29	0.85	-42→119	0.61	0.21	10	0.18	20
			34→173			10		20
			-30→240			6		20
	77.85	1.20	-42→119	0.58	0.26	5	0.16	10
			34→173			5		10
			-30→240			5		10
	83.00	1.50	-42→119	0.58	0.26	5	0.16	10
			34→173			5		10
			-30→240			5		10
	89.23	2.20	-42→119	0.57	0.27	4	0.16	8
			34→173			4		8
			-30→240			4		8
	94.45	3.50	-42→119	0.57	0.27	3	0.16	5
			34→173			3		5
			-30→240			3		5
	97.09	6.00	-42→119	0.57	0.27	3	0.16	5
			34→173			3		5
			-30→240			3		5
	98.40	10.00	-42→119	0.57	0.27	3	0.16	5
			34→173			3		5
			-30→240			3		5

Figure 9.3 Orthogonal view showing grade continuity ellipse at 0.1 g/t Au

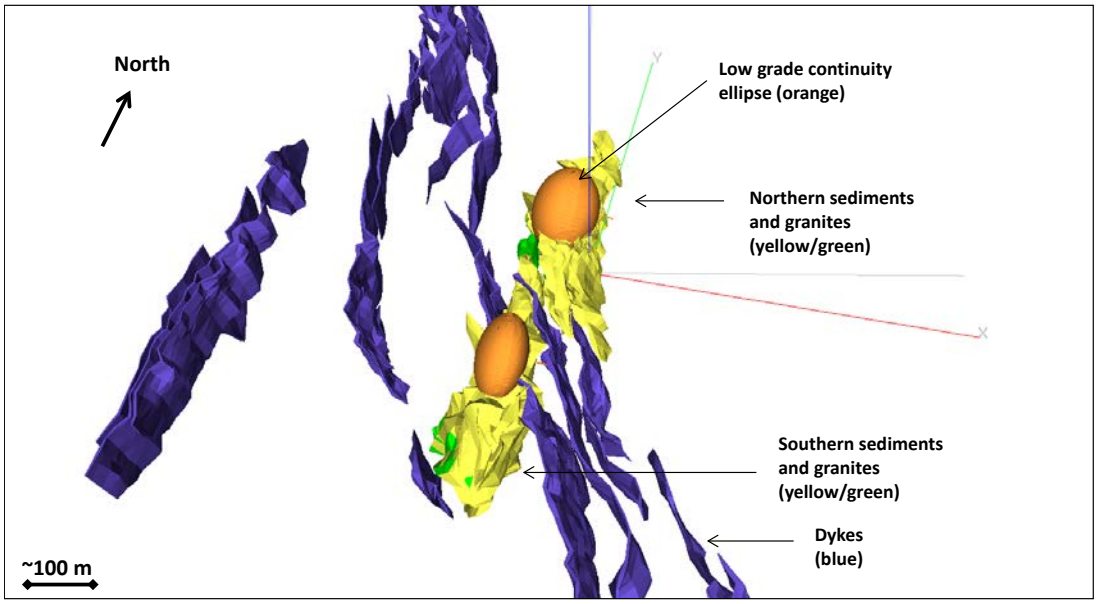
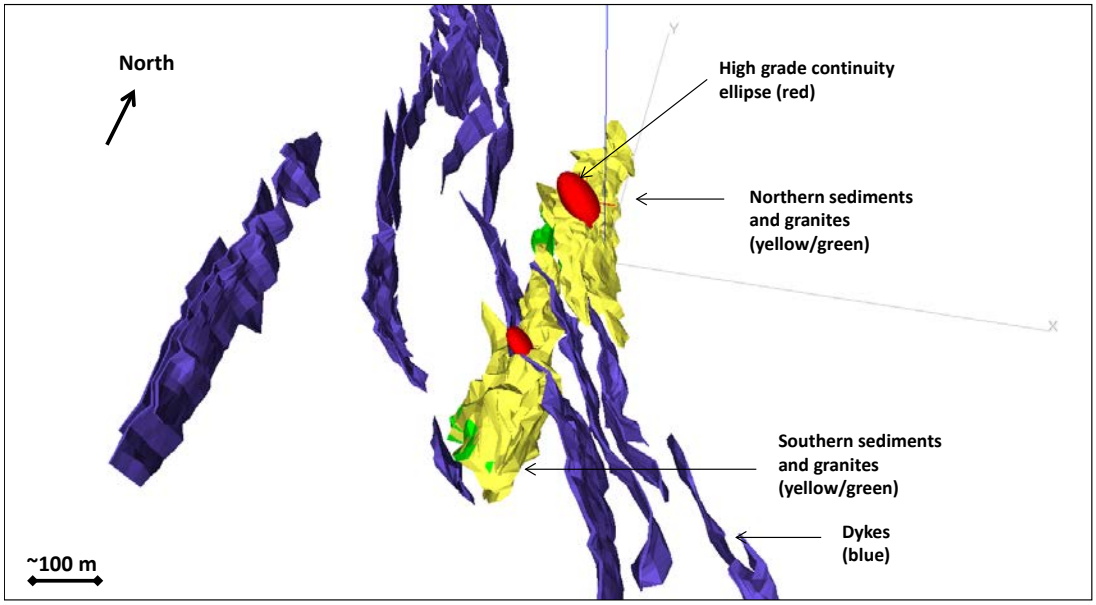


Figure 9.4 Orthogonal view showing grade continuity ellipse at 0.5 g/t Au



9.2 Normal scores variography – dykes

Variogram analysis was carried out on the gold data for the dykes using the oxide, transitional and fresh data combined.

Variogram fans were assessed to determine the dominant orientation of the mineralisation. Results of the analysis indicate that the mineralisation not surprisingly follows the main north-northwest trend of the dykes, dipping sub vertically to the east. The orientation of the dykes changes along the strike length of the deposit. As a result of this Snowden elected to generate and model variograms in the dominant orientation. The orientation of the model was then adjusted in different parts of the deposit using the Dynamic Anisotropy method in Datamine to match the local trends. Dynamic anisotropy involves estimating a local dip and dip direction into each block and using these values to locally orientate the variogram and search ellipses.

Due to the positively skewed nature of the grade distributions, normal scores variograms were modelled. A downhole variogram was modelled to determine the nugget, followed by directional variography. All normal scores variogram models were back-transformed prior to estimation.

Normal scores and back-transformed variogram models are tabulated in Table 9.3 and Table 9.4 respectively. The variogram models are illustrated in Appendix E.

Table 9.3 Normal scores variogram models for dykes

Domain	Orientation	Nugget	Structure 1		Structure 2	
			Sill	Range	Sill	Range
Dykes	00→350	0.42	0.35	35	0.23	92
	-80→080			8		17
	-10→260			5		6

Table 9.4 Back-transformed variogram models for dykes

Domain	Orientation	Nugget	Structure 1		Structure 2	
			Sill	Range	Sill	Range
Dykes	00→350	0.85	0.13	35	0.02	92
	-80→080			8		17
	-10→260			5		6

10 Kriging neighbourhood analysis

Snowden carried out a kriging neighbourhood analysis (KNA) on several test areas within the granite and sediments to determine the optimal parent block size and number of informing samples for estimation. Test estimates were run in Snowden's Supervisor software using the 0.5 g/t Au threshold indicator variograms. Kriging efficiency, slope of regression, and number and sum of negative weights were calculated and reviewed.

10.1 Block size

Block sizes were varied from 5 mE by 5 mN by 2.5 mRL up to 20 mE by 20 mN by 10 mRL. Search parameters for the block size tests were based on the orientation and ranges of the variograms. Graphs of the kriging efficiency and slope of regression for the northern and southern sediments are illustrated in Figure 10.1 and Figure 10.2.

Based on this analysis, a parent block size of 10 mE by 10 mN by 5 mRL was selected as appropriate. In the southern sediments, a smaller block size gives a marginally better result, while in the northern sediments a larger block size gives a marginally better result. The selected block size appears to give reasonable results in both areas.

Figure 10.1 Kriging neighbourhood analysis – northern sediment block size

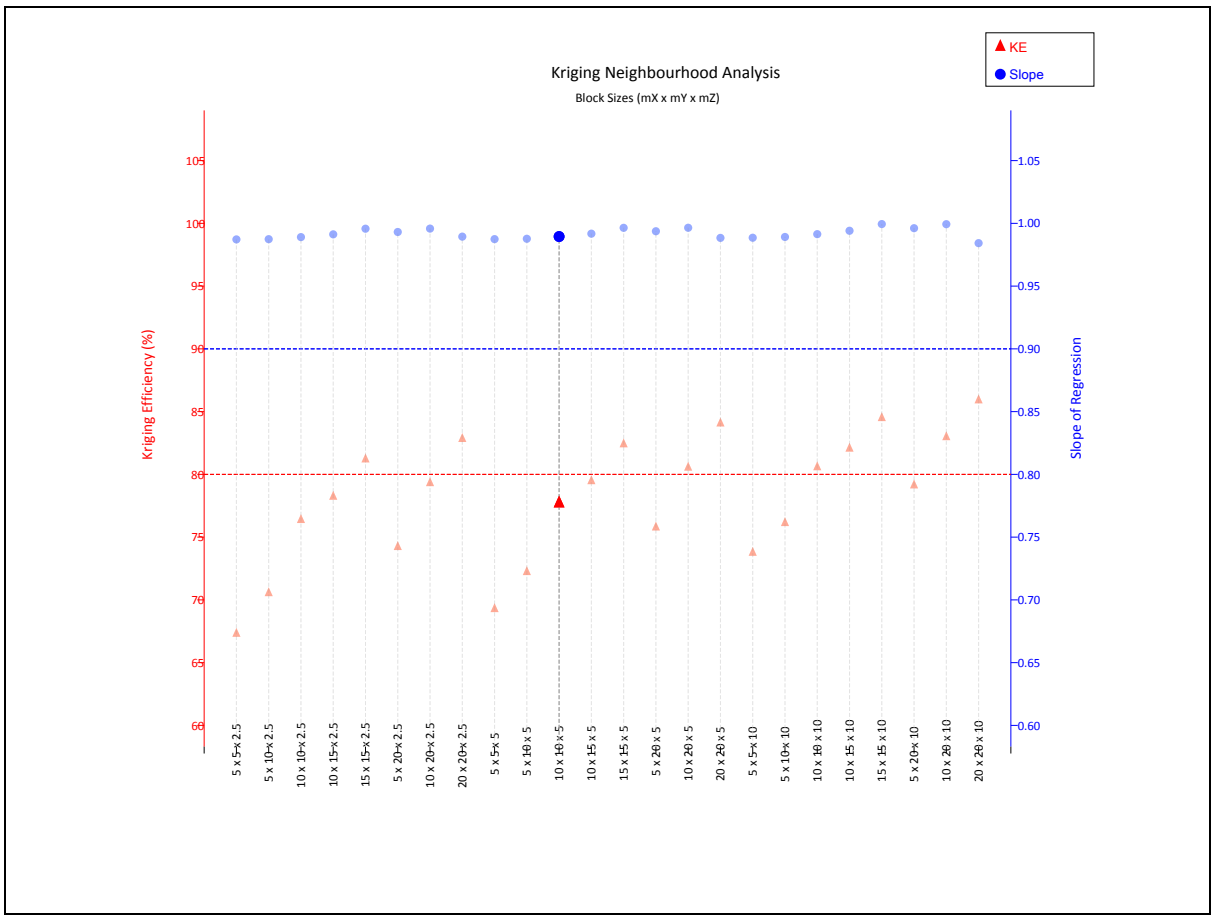
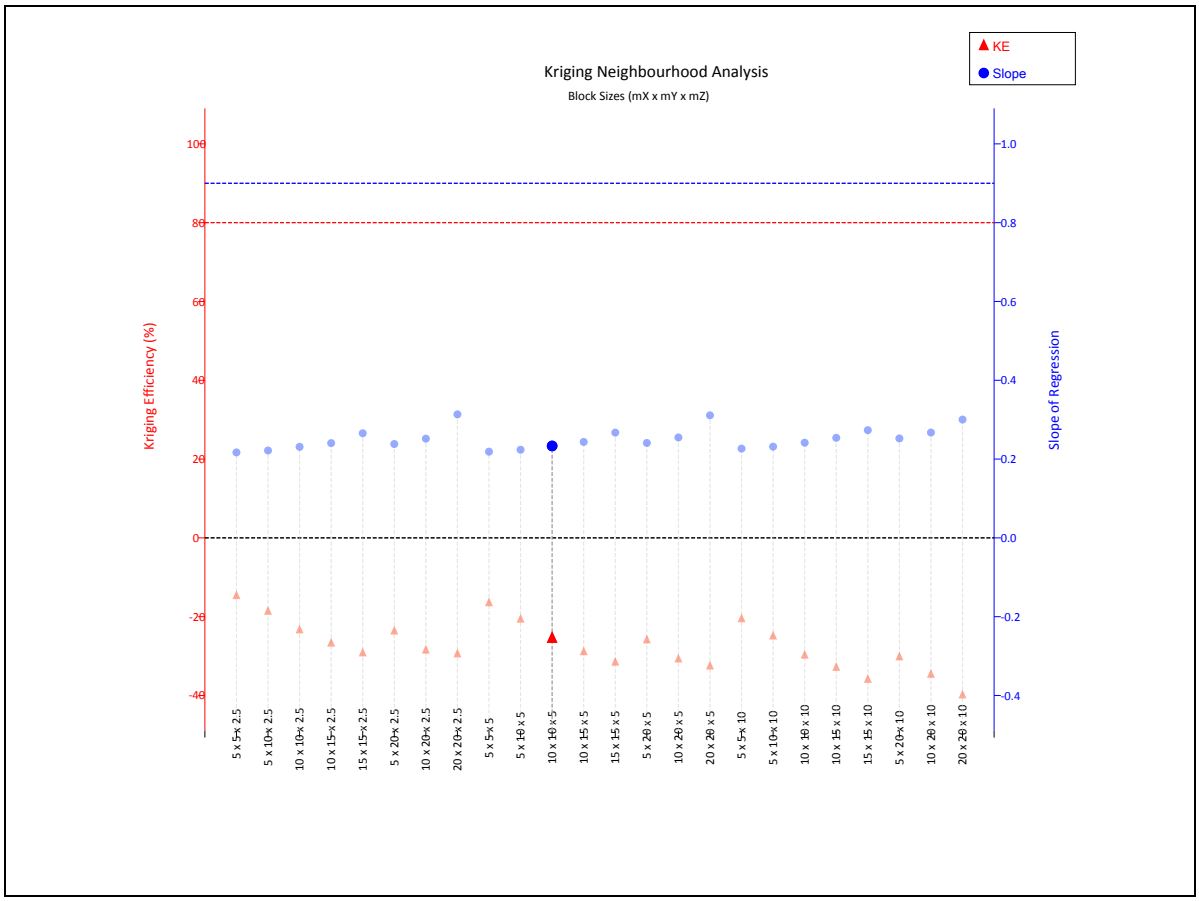


Figure 10.2 Kriging neighbourhood analysis – southern sediment block size



10.2 Number of informing samples

Using the 10 mE by 10 mN by 5 mRL parent cell, a second test was run to look at the impact of changing the number of informing samples. The results are graphed for the northern and southern sediments in Figure 10.3 and Figure 10.4. Based on this analysis, a minimum of 8 and a maximum of 24 samples were selected for estimation.

Figure 10.3 Kriging neighbourhood analysis – northern sediment number of informing samples

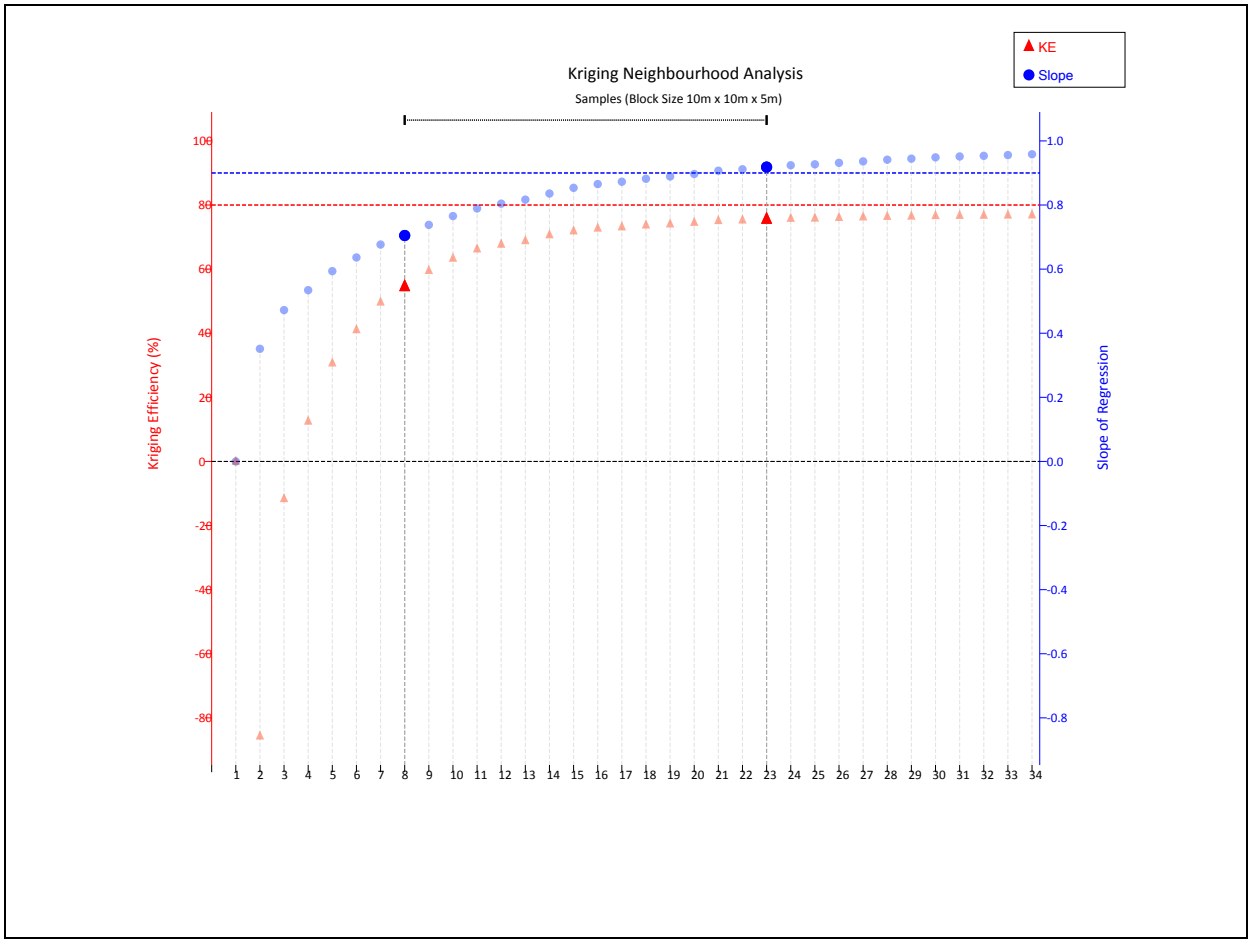
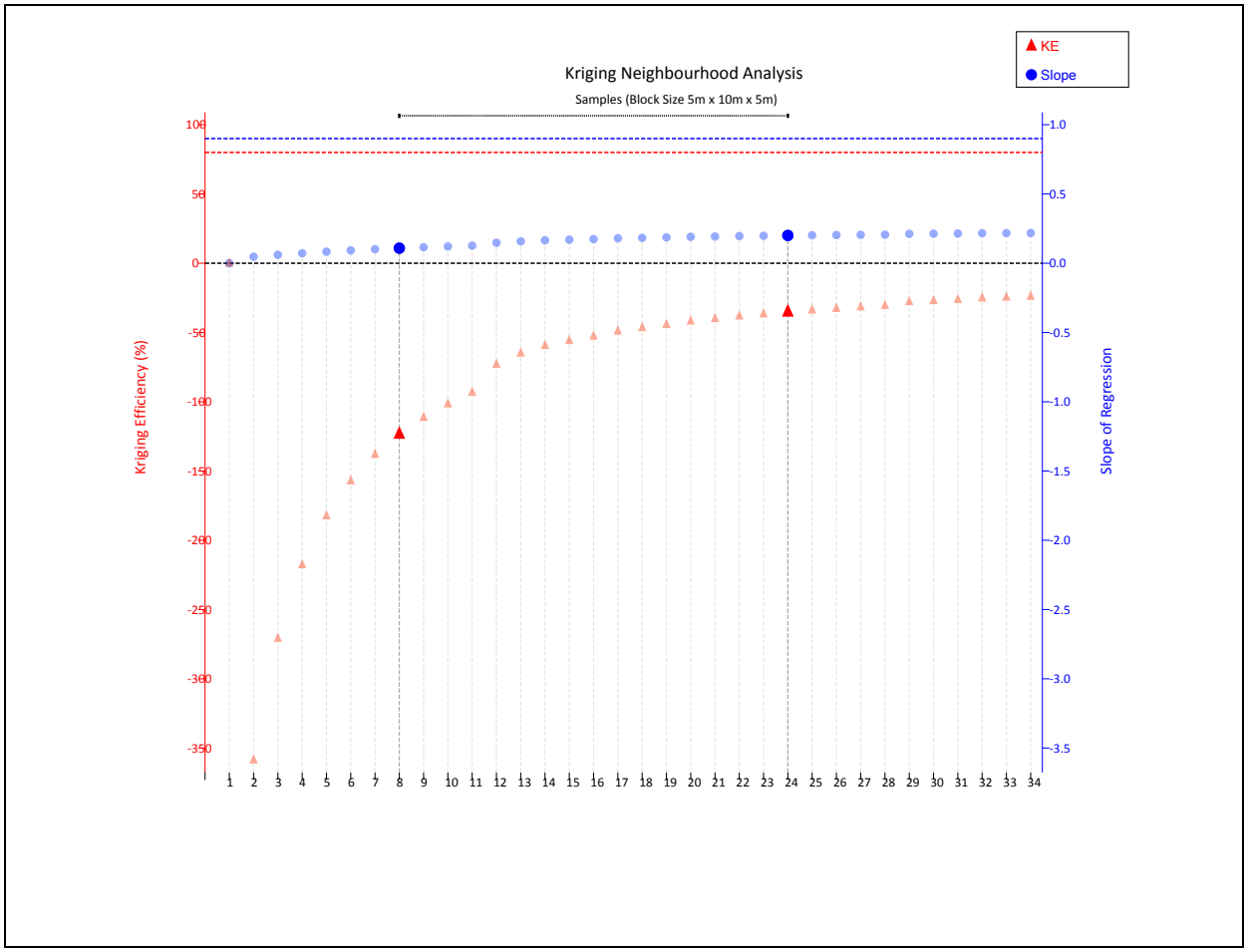


Figure 10.4 Kriging neighbourhood analysis – southern sediment number of informing samples



11 Block modelling

11.1 Model prototype

Table 11.1 summarises the prototype used for the block model. The block model was coded by mineralisation domain (dom and domain), weathering domain (weath_n and weatheri) and area (area) as described in Table 14.1.

A waste model was created outside of the mineralisation area to provide sufficient area around the mineralisation for pit optimisation. Subcelling was used to more accurately reflect the wireframe volumes.

Table 11.1 Block model parameters

	Easting (X)	Northing (Y)	Elevation (Z)
Origin	805847	1152819	0
Limit	807547	1155849	400
Parent block size (m)	10	10	5
Number of blocks	170	303	80
Minimum sub-cell size (m)	2.5	2.5	1.25

11.2 Grade interpolation and boundary conditions

11.2.1 Multiple indicator kriging – granites and sediments

Due to the highly skewed nature and mixed populations evident in the granites and sediments, multiple indicator kriging (MIK) was used to estimate gold grades. Twelve indicator thresholds were selected as discussed in Section 9. Datamine software was used to estimate the probability of grade being above or below each threshold into parent blocks.

As discussed in Section 7, the laterite, oxide and transitional data were used to estimate the laterite and oxide, while the transitional and fresh data were used to estimate the transitional and fresh domains. Hard boundaries were used between the granite and sediments and the southern and northern areas.

The initial search pass used ranges equivalent to the ranges of continuity seen in the variograms at around 90 % to 100 % of the variance in the 0.3 g/t Au to 0.5 g/t Au threshold variograms, with the search ellipse orientated as per the higher grade thresholds (0.5 g/t Au and above) as detailed in Table 11.2.

Blocks were estimated using a minimum and maximum number of samples of 8 and 24 respectively based on the KNA work discussed in Section 10. A maximum of 6 composites was allowed per drillhole for estimation. This restriction was set to prevent over smoothing in the downhole direction and help retain the grade trends seen in the drilling data.

If the initial search failed to find the minimum number of samples required, then a second search was conducted using double the search radii. Where required a third search using eight times the search radii was used with the minimum number of samples reduced to 2 and the maximum reduced to 8.

Table 11.2 Search parameters – multiple indicator kriging

Domain	Search Orientation	Search Dimensions (m)		
		Pass 1	Pass 2	Pass 3
Dykes	-42→119	60	120	480
	34→173	60	120	480
	-30→240	30	60	240

Post processing of the MIK probability estimates was carried out in GSLIB software. Post processing was used to carry out order relation corrections using an averaging approach, and to calculate the e-type grade for each block. The e-type grades were calculated using a linear model between indicator thresholds and a hyperbolic or power model for the upper and lower tails (above the top threshold and below the bottom threshold) (Appendix F). A minimum of 0.001 was used and a maximum based on the distribution of the input data used to estimate each domain. Table 11.3 summarises the lower and upper tail models for each domain.

Table 11.3 Lower and upper tail modelling

Domain	Lower Tail			Upper Tail		
	Minimum	Model Type	Parameter	Maximum	Model Type	Parameter
Southern granites Laterite, oxide	0.001	power	0.60	51.78	power	0.49
Southern granites Transitional, fresh	0.001	power	0.60	88.63	hyperbolic	1.70
Northern granites Laterite, oxide	0.001	power	0.45	66.00	power	0.24
Northern granites Transitional, fresh	0.001	power	0.45	66.00	power	0.21
Southern sediments Laterite, oxide	0.001	power	1.07	137.70	hyperbolic	1.36
Southern sediments Transitional, fresh	0.001	power	0.60	286.00	hyperbolic	1.14
Northern sediments Laterite, oxide	0.001	power	1.13	257.00	hyperbolic	1
Northern sediments Transitional, fresh	0.001	power	0.65	487.40	hyperbolic	1

Note the three extreme grades which were removed for variography, were included in the probability estimates but were removed for the upper tail modelling. A sensitivity model was run, changing the maximum value in the two affected domains (northern sediments and southern granite) to 500 g/t Au. There was no significant change (0.01 g/t Au change) in the average grade and as a result the original estimate was accepted.

11.2.2 Ordinary kriging – dykes

Ordinary kriging into parent cells was used to estimate gold grades for the dykes using a discretisation of 5 east by 5 north by 3 elevation. As discussed in Section 7, the laterite, oxide and transitional data were used to estimate the laterite and oxide, while the transitional and fresh data were used to estimate the transitional and fresh domains. Hard boundaries were used between the dykes.

A dynamic anisotropy approach was used, whereby the true dip and azimuth of the mineralised lodes was estimated into each block in the model and the search and variogram orientations were locally adjusted to reflect the geological orientation. This method allows the estimate to better reflect the changing orientation and undulating nature of some of these dykes along strike.

The initial search pass used ranges equivalent to the ranges of continuity seen in the variograms at around 90 % to 100 % of the variance as detailed in Table 11.4.

Blocks were estimated using a minimum and maximum number of samples of 8 and 24 respectively based on the KNA work discussed in Section 10. A maximum of 6 composites was allowed per drillhole for estimation. This restriction was set to prevent over smoothing in the downhole direction and help retain the grade trends seen in the drilling data.

If the initial search failed to find the minimum number of samples required, then a second search was conducted using double the search radii. Where required a third search using eight times the search radii was used with the minimum number of samples reduced to 2 and the maximum reduced to 8.

Table 11.4 Search parameters – ordinary kriging

Domain	Search Orientation	Search Dimensions (m)		
		Pass 1	Pass 2	Pass 3
Dykes	Strike	40	80	240
	Dip	40	80	240
	Cross-strike	20	40	120

11.3 Density

A total of 770 bulk density measurements were taken from HQ and NQ drill core: 380 from oxide material, 132 from transitional material and 258 from fresh material. Sample length was nominally 10 cm. Bulk densities were determined by the water immersion method using a scale with a precision of ±0.02 g. Oxide and transitional material was wrapped in foil to avoid intake of water and sample disintegration. The wrapper was afterwards weighted independently and deducted from the measured total weight.

Calculated means for the transition and fresh weathering domains as well as the lateritic horizon within the oxide domain (i.e. in the top 8 to 10 vertical metres) were assigned to the block model.

The oxide domain below the lateritic horizon, however, shows a gradient to higher densities with increasing depth (Figure 11.1) and it is considered that a single value for the oxide domain would not be adequate. Instead, the bulk density for oxide blocks has been estimated using vertically orientated, inverse distance squared interpolation.

Of the 770 bulk density measurements, 625 were taken from sediments as the predominant domain, 39 from granite, 28 from porphyritic dykes and 78 from the lateritic horizon. Differences between the respective means exist across the various lithologies. However, as the differences across domains are smaller than the differences within individual domains (for the respective weathering zone), bulk density assignment has been identical for all lithologies (other than the lateritic horizon).

No variation in bulk density between mineralised and non-mineralised material was assumed. The raw measurements and assigned bulk densities used in the resource estimate are listed in Table 11.5 and Table 11.6 respectively.

Figure 11.1 Bulk density versus depth in oxide

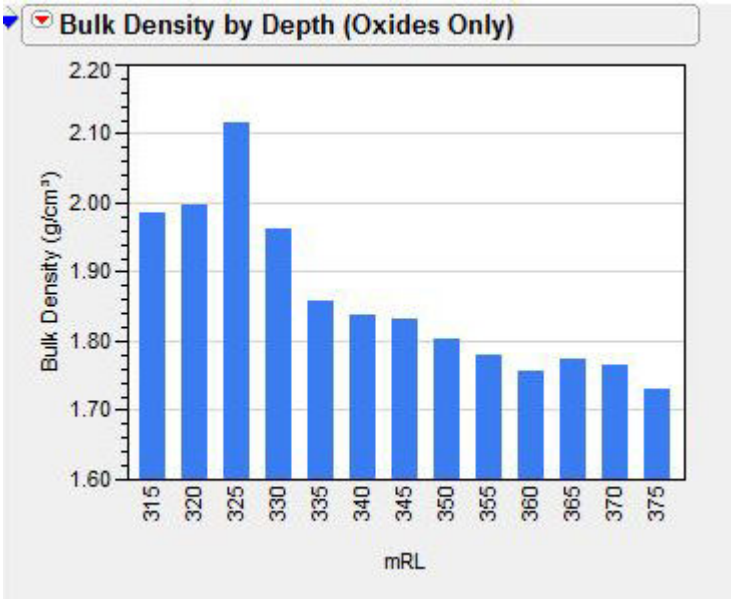


Table 11.5 Bulk density measurements

Weathering	Bulk Density g/cm^3			
	# samples	Mean	Minimum	Maximum
Completely oxidised	89	1.91	1.49	2.53
Strongly oxidised	293	1.78	1.42	2.26
Transition (moderately oxidised)	74	2.02	1.62	2.57
Transition (weakly oxidised)	56	2.31	1.71	2.75
Fresh, weak oxidation along fractures	42	2.68	2.47	2.81
Fresh	216	2.74	2.36	2.96

Table 11.6 Bulk density values assigned to resource estimate

Field	Value	Description	Bulk Density g/cm^3
weath_n	1	Laterite	1.90
	2	Oxide	1.70-2.20
	3	Transitional	2.30
	4	Sulphide	2.73

11.4 Model validation

Snowden validated by the Sissengue model by:

- Comparison of top cut input grades with tonnage weighted output grades.
- Visual inspection of the model against the input composites.
- Comparison of moving window input and output statistics.
- Global change of support to check level of selectivity in model.

11.4.1 Global statistical comparison

The final grade estimates were validated statistically against the input drillhole composites. Table 11.7 provides a comparison of the estimated grades compared to the input declustered, top cut grades for the global estimate. This statistical comparison shows that all domains validate reasonably well globally although the southern granite and northern sediments appear to be underestimated as a result of the three extreme grades in these domains. These three grades have been purposefully not honoured in the estimate as they are not indicative of the surrounding mineralisation. To provide a more meaningful comparison in the domains estimated by MIK, a top cut of 500 g/t Au has been applied to these statistics.

Table 11.7 Statistical comparison between estimated grades and input data

	Southern Granites	Northern Granites	Northern Sediments	Southern Sediments	Dykes
Samples	1,095	690	4,091	5,087	7,288
Top cut data	3.20	1.03	1.58	1.29	1.02
Estimate	2.72	0.96	1.24	1.26	0.92
Difference	-15%	-7%	-22%	-2%	-10%

11.4.2 Visual validation

The final grade estimates were visually compared to the input drillhole data on section. This comparison shows that the estimate honours the input drillhole data. Example cross sections are illustrated in Figure 11.2 and Figure 11.3.

Figure 11.2 Example cross section showing estimated gold grades versus input drillhole data – 1154240 mN (±10 m)

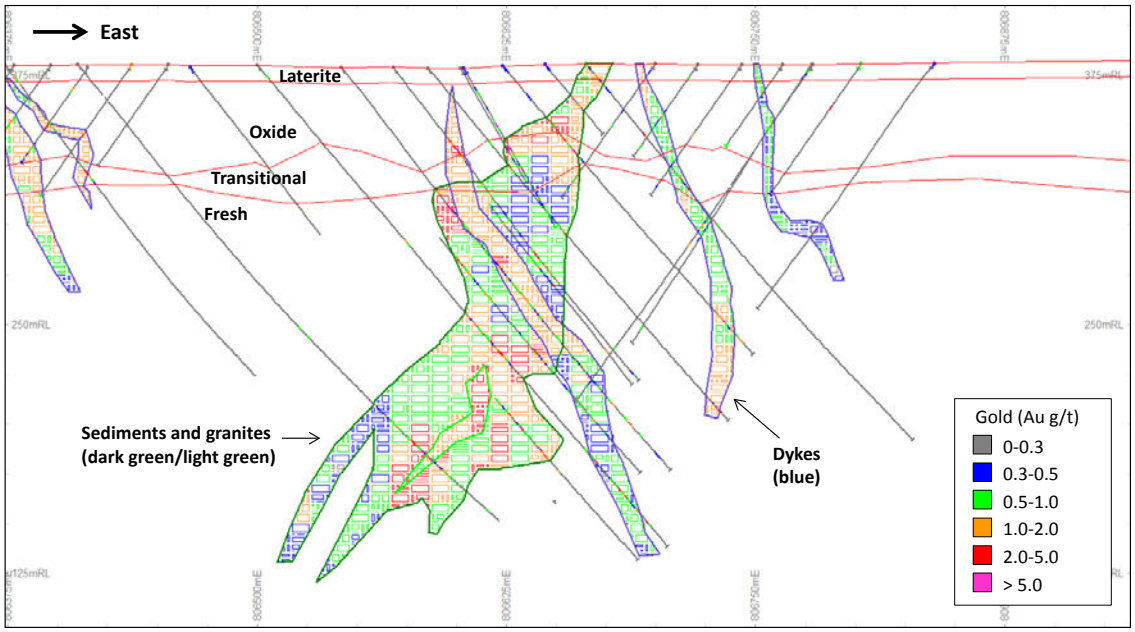
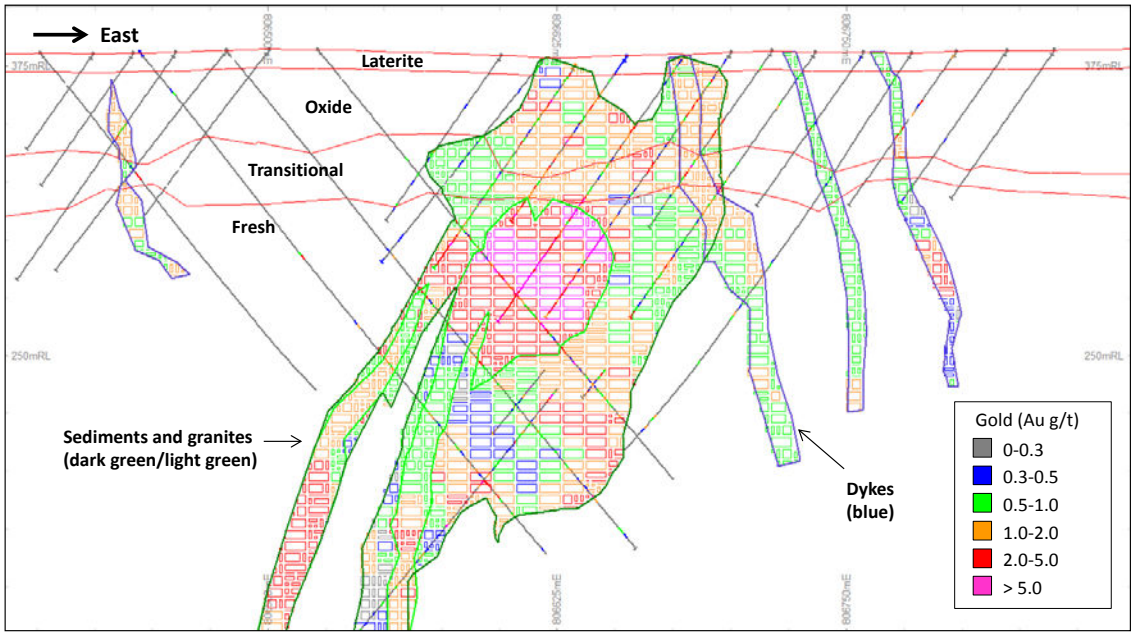


Figure 11.3 Example cross section showing estimated gold grades versus input drillhole data – 11542180 mN (±10 m)



11.4.3 Moving window validation

Sectional validation graphs were created for each mineralised domain to assess the reproduction of local means and to validate the grade trends in the model. These graphs compare the mean of the estimated grades to the mean of the input grades (declustered, not cut) within model slices (bins) for gold. The graphs also show the number of input samples on the right axis, to give an indication of the support for each bin.

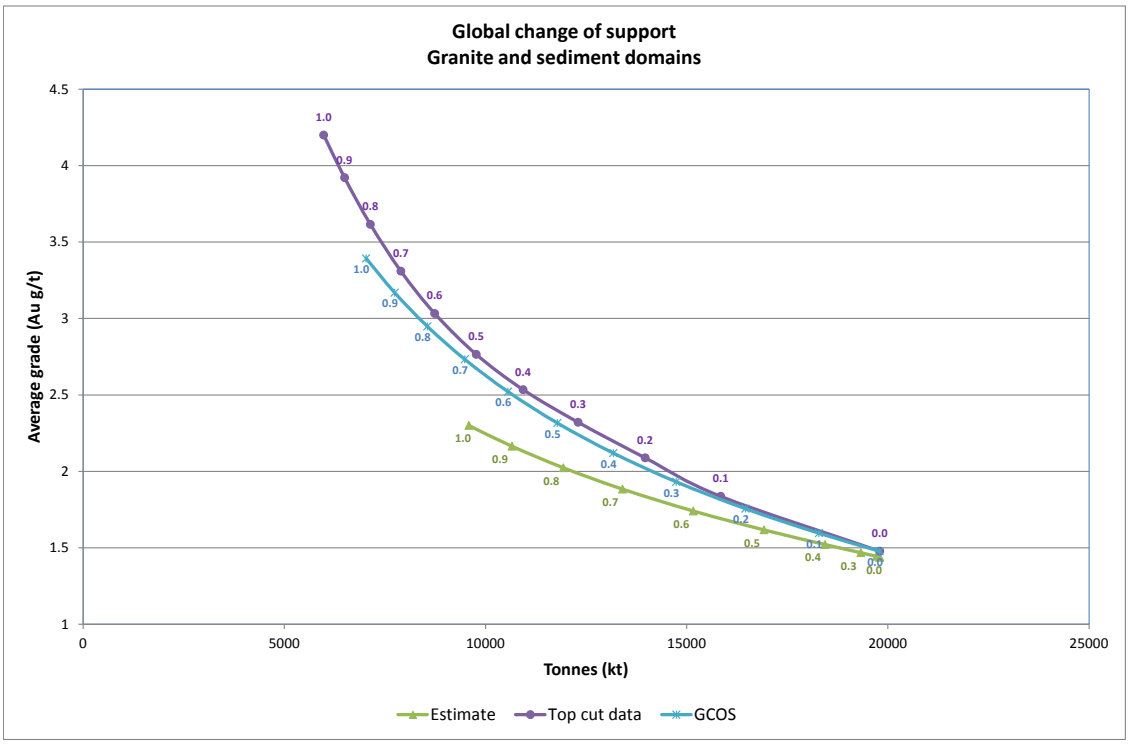
The graphs are located in Appendix G. These graphs indicate that there is good local reproduction of the input grades in both the horizontal and vertical directions.

11.4.4 Global change of support

Snowden carried out a global change of support (GCOS) on the declustered input data to assess the selectivity in the MIK estimate. This was carried out on the granite and sediment data only. The three extreme grades were removed from the dataset prior to running the global change of support to provide a more realistic result.

The theoretical grade tonnage curve at the parent block size was compared to the estimated grade tonnage curve (Figure 11.4). The results show that the MIK estimate is over smoothed compared to the theoretical result, however Snowden considers the result to be acceptable.

Figure 11.4 Global change of support



12 Mineral Resource classification

Snowden reviewed the classification applied to the previous estimate (Widenbar, 2013) and concluded that the classification is appropriate with respect to confidence in tonnes and grade estimates; however recent information on recovery and mining factors has resulted in a trial optimisation being run to determine the potential for economic extraction, in particular for the lower grade and peripheral mineralisation.

The Sissingue Mineral Resource has been classified in the Measured, Indicated and Inferred categories, in accordance with the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC Code – 2012 edition). A range of criteria has been considered in determining this classification including:

- Geological continuity.
- Data quality.
- Drillhole spacing.
- Estimation properties including kriging neighbourhood analysis to determine appropriate block size and search strategy.
- Potential for economic extraction.

12.1 Geological continuity

Geological and grade continuity is understood with reasonable confidence. The classification reflects this level of confidence. Granite and sediment domains were able to be assigned to Measured, Indicated or Inferred categories according to other parameters such as data distribution. Porphyry lithologies (dyke domains) were limited to Inferred and Indicated categories due to the mostly narrow nature of the mineralised bodies and consequently their sensitive geometry.

12.2 Data quality

Resource classification is based on information and data provided from the Perseus database. Descriptions of drilling techniques, survey, sampling/sample preparation, analytical techniques and database management/validation provided indicate that data collection and management is well within industry standards. The database represents an accurate record of the drilling undertaken at the project.

12.3 Drilling spacing

Drillhole location plots have been used to ensure that local drill spacing conforms to the minimum expected for the resource classification.

- The Measured category requires a drill spacing of 20 m by 20 m or closer.
- The Indicated category is confined to areas where drill spacing is greater than 20 m by 20 m, but nominally up to 20 (east-west) by 40 m (north-south) spacing. In the dyke domains, in additional to drill spacing constraints, a minimum of two drillhole intercepts per drill section is required.
- The Inferred category is assigned to all other estimated blocks within the potentially economic areas of the deposit.

12.4 Estimation

The resource model was generated using an ordinary kriging and MIK interpolation method with a minimum number of 8 samples and a maximum number of 24 samples used based on a kriging neighbourhood analysis. The number of samples used and the kriging variance were all stored in the block model. Snowden considers the estimation technique and parameters appropriate for this style of mineralisation.

12.5 Potential for economic extraction

A Whittle optimisation was carried out to determine the limits of potentially economically mineable material for the purposes of classifying the open pit Mineral Resource estimate.

It is usual in this case to use an elevated metal price that could be achieved in a 10 to 20 year timeframe. In this case the optimisation was based on a USD2400/oz gold price (the spot price at the time was USD1200/oz).

The mining inventory derived from the optimisation was based on the preliminary coding of the Measured, Indicated and Inferred Mineral Resource classification.

Key assumptions used for the optimisation were provided by Perseus and are included in Appendix H. Overall these assumptions are considered fair for the purpose of determining the Mineral Resource potential of the project.

12.5.1 Summary of results

Figure 12.1 presents the Whittle shell versus operating surplus and operating surplus present value (PV). The PV assumes a production rate of 1 Mtpa of ore feed and a 10 % discounted cash-flow factor. The PV does not include processing plant or infrastructure capital cost repayment. The schedule assumes a fixed stripping ratio for the life of the project. The points on the graph (i.e. shell 6) are for the USD1200/oz gold price shell. Shell 16 is the USD2400/oz shell. At USD1200/oz, the USD2400/oz shell is near breakeven.

Figure 12.1 Whittle shell versus operating surplus and operating surplus PV (at 10 %)

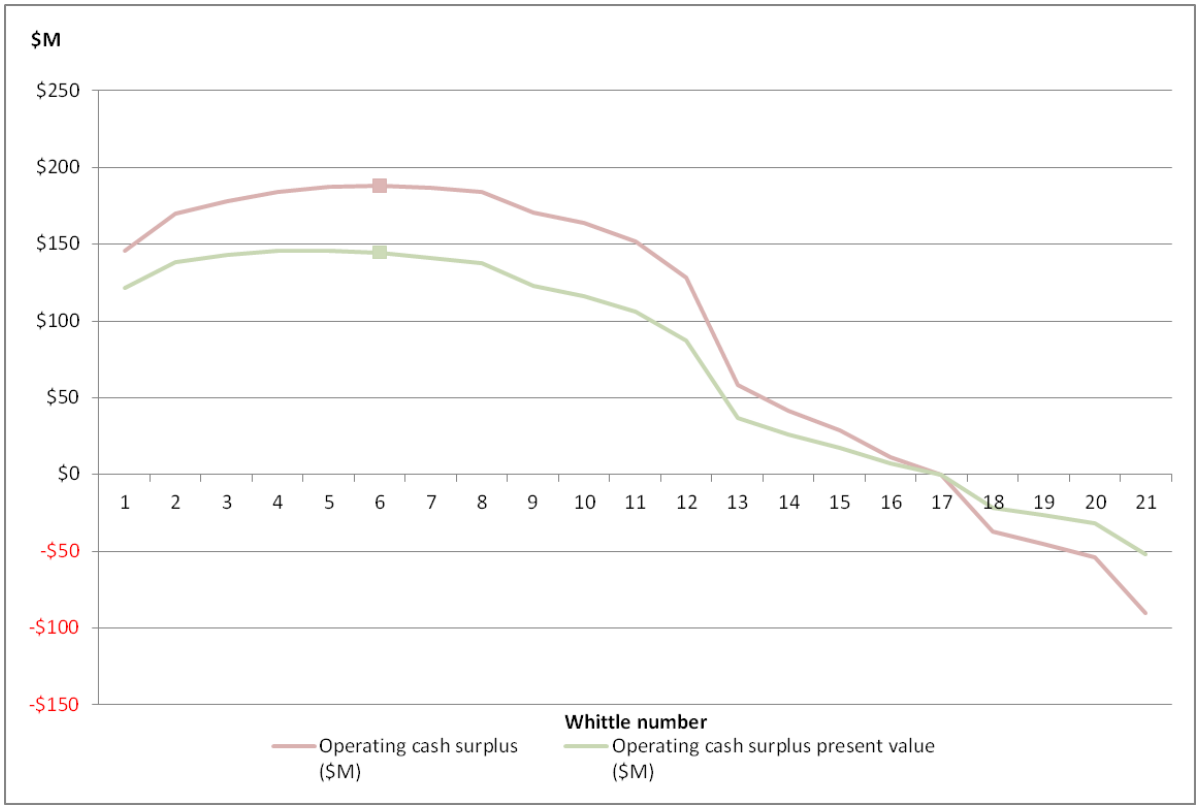


Figure 12.2 shows the Whittle optimisation for the nominal USD1200/oz shell and the USD2400/oz shell.

Figure 12.2 Whittle shell 6 (USD1200/oz) and shell 16 (USD2400/oz)

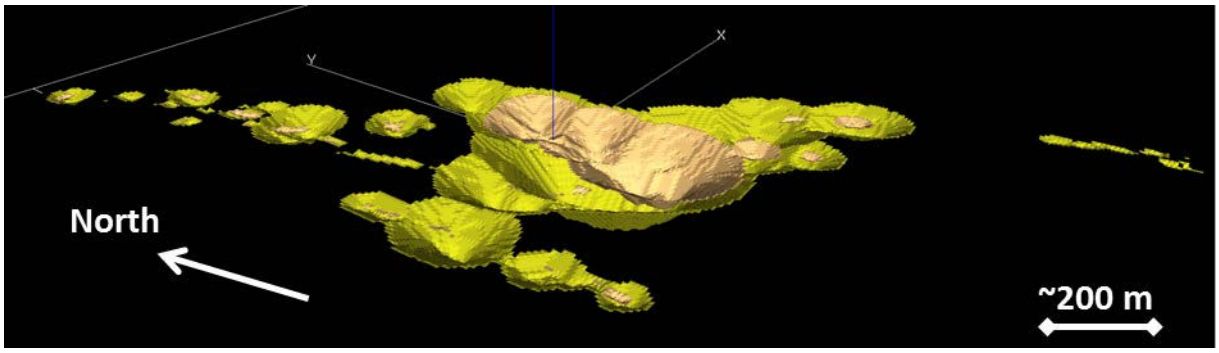


Table 12.1 presents the Whittle optimisation results.

Table 12.1 Whittle optimisation results

Whittle shell	Metal Reduction Factor	Gold Price (USD/oz)	Total tonnes	Total volume	Unclassified	Measured	Indicated	Inferred	Ore tonnes	Au grade (g/t)	Mining cost (%/t)	Whittle processing cost ($/t)	Recovered grade (g/t)	Revenue ($/t)	Life (years)	Operating revenue ($M)	Operating cost ($M)	Operating cash surplus ($M)	Operating cash surplus present value ($M)
1	0.50	600	11,237,795	5,470,297	8,295,629	2,134,759	785,235	22,172	2,858,156	2.89	4.30	32.13	2.59	100.10	2.9	$286	$140	$146	$122
2	0.60	720	14,330,132	6,904,578	10,830,874	2,402,538	1,069,436	27,284	3,402,157	2.93	4.31	32.49	2.61	100.53	3.4	$342	$172	$170	$138
3	0.70	840	15,746,729	7,559,078	11,946,940	2,564,016	1,205,183	30,590	3,698,618	2.89	4.32	32.59	2.57	99.06	3.7	$366	$189	$178	$143
4	0.80	960	17,842,788	8,549,156	13,655,830	2,796,810	1,353,521	36,628	4,078,397	2.83	4.32	32.76	2.51	96.86	4.1	$395	$211	$184	$146
5	0.90	1,080	19,307,463	9,227,125	14,803,682	2,939,565	1,518,712	45,504	4,385,893	2.76	4.32	32.84	2.45	94.55	4.4	$415	$227	$187	$146
6	1.00	1,200	20,669,993	9,842,023	15,890,585	3,067,039	1,660,116	52,253	4,655,238	2.71	4.32	32.93	2.40	92.50	4.7	$431	$243	$188	$145
7	1.10	1,320	22,838,735	10,806,094	17,630,988	3,263,502	1,880,860	63,385	5,061,723	2.62	4.33	33.12	2.32	89.57	5.1	$453	$267	$187	$141
8	1.20	1,440	24,847,447	11,739,453	19,325,201	3,317,176	2,125,708	79,362	5,342,880	2.58	4.33	33.21	2.28	87.86	5.3	$469	$285	$184	$138
9	1.30	1,560	31,994,515	14,870,461	25,553,683	3,429,913	2,923,086	87,833	6,193,144	2.46	4.36	33.56	2.17	83.64	6.2	$518	$347	$171	$123
10	1.40	1,680	35,394,320	16,535,906	28,572,558	3,447,054	3,267,299	107,409	6,537,024	2.42	4.36	33.48	2.13	82.10	6.5	$537	$373	$164	$116
11	1.50	1,800	40,832,827	19,150,820	33,577,382	3,462,282	3,671,038	122,125	6,956,816	2.38	4.35	33.50	2.10	80.86	7.0	$563	$411	$152	$106
12	1.60	1,920	50,450,807	23,651,867	42,466,179	3,465,055	4,359,100	160,473	7,652,898	2.33	4.36	33.49	2.05	78.99	7.7	$604	$476	$128	$87
13	1.70	2,040	74,491,128	33,293,820	64,657,307	3,516,285	5,767,885	549,651	9,420,476	2.25	4.44	34.26	1.96	75.57	9.4	$712	$654	$58	$37
14	1.80	2,160	81,165,211	36,572,938	71,006,085	3,517,650	6,009,921	631,555	9,716,783	2.25	4.43	34.20	1.96	75.49	9.7	$734	$692	$42	$26
15	1.90	2,280	85,231,314	38,331,023	74,787,670	3,520,380	6,228,975	694,289	9,986,465	2.23	4.43	34.26	1.94	74.95	10.0	$748	$720	$29	$18
16	2.00	2,400	91,008,907	41,204,023	80,235,655	3,520,380	6,426,982	825,890	10,302,753	2.22	4.42	34.19	1.93	74.37	10.3	$766	$755	$11	$7
17	2.10	2,520	94,430,496	42,653,227	83,451,350	3,520,380	6,553,599	905,167	10,497,888	2.21	4.43	34.23	1.92	74.03	10.5	$777	$777	-$0	-$0
18	2.20	2,640	105,527,636	47,504,172	94,004,215	3,520,380	6,915,952	1,087,088	11,015,617	2.19	4.44	34.25	1.90	73.39	11.0	$808	$845	-$37	-$22
19	2.30	2,760	108,004,329	48,585,086	96,365,375	3,520,380	6,954,374	1,164,201	11,125,898	2.18	4.44	34.25	1.90	73.25	11.1	$815	$860	-$45	-$27
20	2.40	2,880	110,526,601	49,748,172	98,759,431	3,520,380	7,036,513	1,210,277	11,246,766	2.18	4.44	34.25	1.89	73.03	11.2	$821	$876	-$54	-$32
21	2.50	3,000	120,674,338	54,663,664	108,379,830	3,520,380	7,482,907	1,291,221	11,723,423	2.14	4.43	34.14	1.87	72.04	11.7	$845	$935	-$91	-$52
22	Infinite	Infinite	267,025,877	117,977,422	252,331,459	3,520,380	7,992,386	3,181,652	13,994,979	2.07	4.48	34.15	1.80	69.55	14.0	$973	$1,676	-$702	-$370

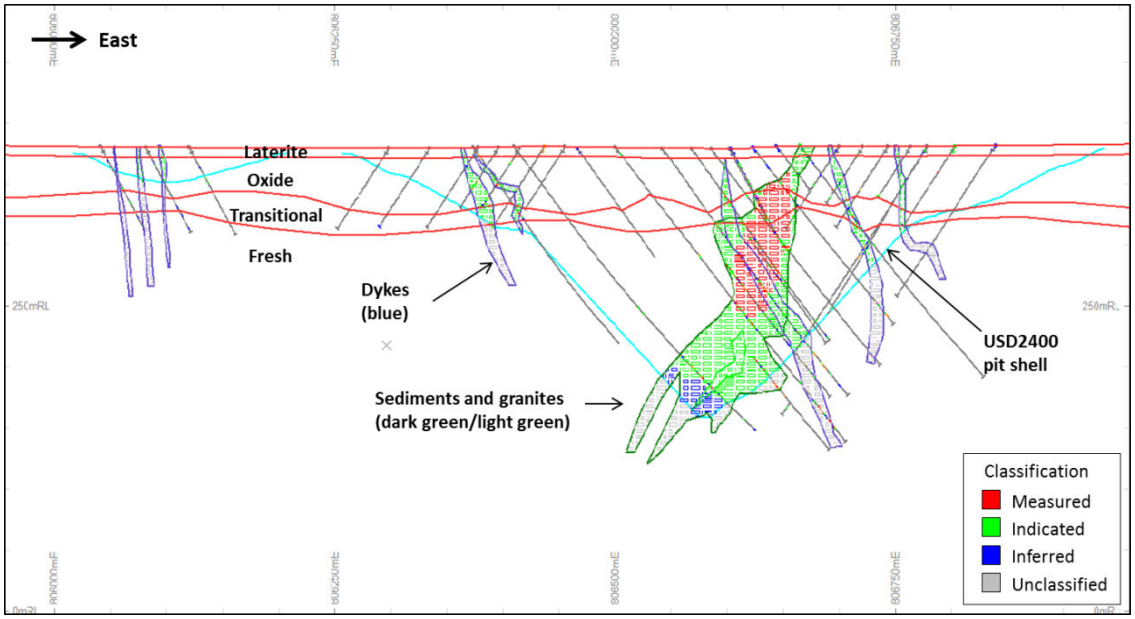
12.6 Final classification

The above parameters were used in combination to arrive at an initial resource classification. Two wireframes were created: one for the Indicated category, and the other for the Measured category. The appropriate category was then assigned to the block model cells. All other cells were initially set to Inferred.

Subsequent to this the USD2400 pit shell was used to constrain the base of potentially economic material. All blocks outside this shell are unclassified.

A typical cross-section view is provided in Figure 12.3. Volumetrically, approximately 28 % of the Sissingue Mineral Resource has been classified as Measured, 65 % as Indicated and 7 % as Inferred.

Figure 12.3 Cross section view of resource classification (±10 m)



12.7 JORC 2012 table 1

Table 12.2 to Table 12.4 list the criteria pertinent to Table 1 of the JORC Code 2012 that have been used to derive the resource classification.

Table 12.2 JORC 2012 Table 1 – Section 1 sampling techniques and data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as downhole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.	• Sissengue data used for Mineral Resource includes: – Total of 131,027 m of RC drilling. – Total of 68,148 m of diamond drilling. – Total 2,236 m of combined RC/diamond drilling. • Reverse Circulation drilling (5¼"" diameter) was used to collect 1 m samples. Majority composited to 2 m samples (by weighing); close spaced infill submitted as 1 m samples. Sample weight nominally of 2.5 kg and 5 kg respectively. • Half-core from Diamond drillholes ('right' side systematically taken; 1.5 m in oxide and transition, 1 m in fresh. • 50 gm charge produced for Fire subsequent Assay.
Drilling techniques	• Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• Reverse Circulation drilling (5¼"" diameter), usually 80 m or less in depth. Generally RC holes have collar azimuth and inclination only measured. • Diamond drilling, HQ in weathered rock, NQ in fresh rock. All diamond holes downhole surveyed at 30 m intervals. • 43 holes oriented by core spear; 217 holes oriented by "AceTool" device.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• 1 m RC samples weighed and composited to 2 m. • Diamond core recovery measured; close to 100 % recovery for virtually all core in Fresh rock. • No apparent relation between sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• RC drill chip boards were prepared and the chips logged geologically, including rock type, alteration type and intensity (where recognisable), vein quartz content in estimated percentage, sulphide mineralisation and estimated content and weathering domain. • Diamond drill core was geologically and structurally logged and photographed, before being sawn in half, including fault, fold, cleavage and joint orientation, lithological contacts, vein orientation and bedding. Logged items are lithology, weathering, colour, grain size, vein type and vein volume percentage, sulphide mineralisation and their estimated percentage, alteration and alteration intensity. • All sample intervals in both RC and DD holes were sampled and assayed. • Logging is considered appropriate and reliable.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• All RC samples were collected at the drill site at 1 m interval and split using a multi-stage riffle splitter. Each two consecutive samples were composited (where applicable) in one bag. • DD core sawn in half using a motorized diamond blade saw; right half sent for assaying, left half stored in core trays for reference. • Both core and RC samples followed a standard path of drying, crushing and grinding. Samples were pulverized with a ring mill and thoroughly mixed on a rolling mat ("carpet roll"), and then 200 g of sub-sample was collected. Internal laboratory checks required at least 90 % of the pulp passing -75 microns. • Sub-sampling is considered appropriate and representative.

Criteria	JORC Code explanation	Commentary
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.	• A small number of initial holes were assayed by bottle roll, which was found to be partial and inaccurate. • All subsequent assaying by standard Fire Assay. • Field duplicates (RC only) inserted at 1 in 25. • No field duplicates for DD as ¼ core considered as inadequate sample, and submission remaining ½ core considered undesirable. • Blanks inserted at 1 in 25. • Certified standards at 1 in 50 up to 2008; thereafter at 1 in 20. • Internal laboratory standards, duplicates and repeats and various other tests have been carried out throughout the drilling programs. • QAQC shows no bias, but only moderate reproducibility, particularly at high grades. This is as expected with the nugget mineralisation. • Overall assaying quality is considered acceptable.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• There are no twinned holes; lack of these is not considered significant given the close drill spacing. • Drillhole information for both RC and diamond core holes is captured at the drill site on paper. • All hard copies are handed over to the database administrator in Tengrela site office and the information provided on paper is entered into the computer. • All hard copies are kept in Tengrela site office. • Downhole survey data and collar survey data are provided by the drilling contractors and surveyors respectively in digital format. • Perseus maintains a centralised database for its various operations in Ghana and Ivory Coast. Database administration is based in Perseus' head office in Accra/Ghana and under the supervision of the company's Resource Geologist. • No adjustments are made to raw assay data. Top cutting is only applied after database compositing and statistical analysis and prior to resource estimation.
Location of data points	• Accuracy and quality of surveys used to locate drillholes (collar and downhole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. • Specification of the grid system used. • Quality and adequacy of topographic control.	• All RC and DD holes were surveyed using differential GPS, until September 2009 by a certified contract surveyor (SEMS Exploration Services Ltd, Ghana). Drillholes between September 2009 and October 2010 were surveyed by CBM Surveys Ltd of Ghana. All subsequent drill holes were surveyed by the company's surveyor. • Grid system used is WGS84 UTM 29N. • The topography covering the extent of the Sissingue Mineral Resource model was created as a digital terrain model (DTM) in Surpac using the accurately-surveyed drillhole collar data and an additional 639 survey points across the prospect.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• Data spacing for resource estimation varies from 10 m x 10 m to 20 m x 20 m for most areas of the deposit. • Where data spacing is wider (to a maximum of 40 m x 40 m), an Inferred classification is used. • Data spacing is sufficient to establish grade and geological continuity appropriate to the resource estimation procedures and classifications applied. • Samples have been composited (by computer) to 2 m, honouring geological divisions.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Orientation of drilled section lines is at right angles to the strike of the geology and mineralisation domains. • Drill holes are angled to cross the sub-vertical dip of the geological domains. • Disseminated mineralised veins have developed within the overall geological domains; the estimation method is considered to account for this.
Sample security	• The measures taken to ensure sample security.	• Samples from RC drilling are collected and bagged at drill site during the drilling operation. Core samples are cut in a central facility in Tengrela and samples placed into sample bags as they are cut. • All samples are then catalogued and placed in large woven bags and sealed prior to dispatch to ALS, Intertek or BVML for preparation and analysis. • Dispatch from site to Korhogo (Intertek) is undertaken by Perseus staff and vehicles. • Samples dispatched to ALS and BVML are collected from Tengrela by staff and vehicles of the respective laboratories. • All aspects of the process are supervised by Perseus personnel and limited opportunity exists for tampering with samples.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• Steffen Brammer and Kevin Thomson of Perseus have reviewed sample techniques and data during regular site visits between 2008 and 2013, and consider them adequate. • Reviews were also carried out by personnel from consulting company Runge Limited during 2009 and 2010, and Mr L Widenbar of Widenbar & Associates in October 2012 with acceptable conclusions.

Table 12.3 JORC 2012 table 1 – section 2 reporting of exploration results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. • The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	• The Sissingue Mineral Resource lies within Occidental Gold's mining permit PE39 (Permit d'Exploitation Tengrela Est), tenement number PE39. • The mining permit is valid for 6 years until August 2018 and is renewable. • The Government of Ivory Coast is entitled to a 3 % royalty on production. • The Sissingue Project area has no known environmental liabilities.
Exploration done by other parties	• Acknowledgment and appraisal of exploration by other parties.	• All exploration was by Perseus using soil geochemical sampling, with follow-up drilling in areas with anomalous gold mineralisation, which led to the discovery of the Sissingue deposit.
Geology	• Deposit type, geological setting and style of mineralisation.	• The Sissingue Deposit occurs in a strongly deformed Birimian greenstone belt intruded by granitoid bodies. • Gold mineralisation at Sissingue is associated with the porphyritic dykes of tonalitic chemistry that cross cut the flysch sediments (turbidites). • Subsequent hydrothermal activities and metasomatism of the tonalite has led to a sericite-carbonate alteration within the intrusives and the more permeable horizons (sandstones and conglomerates) of the turbidites, and a low to moderate grade disseminated gold mineralisation. • Late stage high grade Au-As-quartz-carbonate veins exploited the altered and brittle portions of the intrusives and sediments with common occurrences of visible gold.
Drill hole Information	• A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drillholes: • Easting and northing of the drillhole collar. • Elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar. • Dip and azimuth of the hole. • Downhole length and interception depth. • Hole length. • If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	• Exploration results are not being reported for the Mineral Resource area.
Data aggregation methods	• In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated. • Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. • The assumptions used for any reporting of metal equivalent values should be clearly stated.	• Exploration results are not being reported for the Mineral Resource area.
Relationship between mineralisation widths and intercept lengths	• These relationships are particularly important in the reporting of Exploration Results. • If the geometry of the mineralisation with respect to the drillhole angle is known, its nature should be reported. • If it is not known and only the downhole lengths are reported, there should be a clear statement to this effect (e.g. 'downhole length, true width not known').	• Exploration results are not being reported for the Mineral Resource area. • Orientation of drilled section lines is at right angles to the strike of the geology and mineralisation domains. • Drillholes are angled to cross the sub-vertical dip of the geological domains.
Diagrams	• Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	• Location plans and example sections are included in the Mineral Resource technical documentation.
Balanced reporting	• Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	• Exploration results are not being reported for the Mineral Resource area.
Other substantive exploration data	• Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	• Exploration results are not being reported for the Mineral Resource area.

Criteria	JORC Code explanation	Commentary
Further work	• The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). • Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	• Exploration results are not being reported for the Mineral Resource area. • Further work will be undertaken during mining, if economics and the environment supports potential expansions. Exploration over possible satellite deposits is currently on-going.

Table 12.4 JORC 2012 Table 1 – Section 3 estimation and reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• Perseus maintains a centralised database for its various operations in Ghana and Ivory Coast. Database administration is based in Perseus' head office in Accra/Ghana and under the supervision of the company's Resource Geologist. • Perseus carried out detailed validation of the dataset and retain overall responsibility for the database quality. All drillhole data was validated during data entry by Perseus including: – Checks for duplicate collars (LogChief, Datashed). – Checks for missing samples (Datashed). – Checks for down hole from-to interval consistency (LogChief, Datashed). – Checks for overlapping samples (LogChief, Datashed). – Checks for samples beyond hole depth (LogChief, Datashed). – Checks for inexistent or misspelt log items (LogChief). – Check for missing assays (Datashed). – Check for down-hole information beyond hole depth (Datashed). • Snowden carried out a basic statistical and visual validation prior to estimation including: – Drillholes with overlapping sample intervals. – Sample intervals with no assay data. – Duplicate records. – Assay grade ranges. – Collar coordinate ranges. – Valid hole orientation data. • No issues were found with the data and Snowden considers the data to be appropriate for estimation.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• Steffen Brammer and Kevin Thomson of Perseus have undertaken regular site visits between 2008 and 2013. Steffen and Kevin have reviewed the geology and data collection processes during this time, • Perseus is taking responsibility for the geological model and database; hence it was not required for Snowden to undertake a site visit at this stage.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• Mineralisation was domained by its host lithology for modelling. The three main host lithologies are granite, porphyritic dykes and sediments. Mineralised sediments in the alteration halo of the dykes were included in the dyke domain to maintain a minimum width of the wireframes and to maintain continuity along strike. • Where geological contacts were not clearly controlling the distribution of mineralisation, a grade cut-off of 0.3 g/t Au was used to construct Mineral Resource boundaries and to provide overall geometry to mineralised zones. A minimum of 4 m width was used for the wireframes and samples of grades below the nominal cut-off of 0.3 g/t Au were included where the wireframe would otherwise be less than 4 m wide. Analysis of the global grade distribution shows that there is a natural change in grade population at around 0.3 g/t Au. • Geological continuity of the granite and sediment domains is understood with reasonable confidence. The classification reflects this level of confidence. • Porphyry lithologies (dyke domains) were limited to Inferred and Indicated categories due to the mostly narrow nature of the mineralised bodies and consequently the decreased confidence in their geometry. • Continuity and variability is also influenced by weathering and these have been interpreted and incorporated into the oxide, transitional and fresh domains.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Sissingue deposit extends for 3.15 km along strike and to a depth of typically 140 m to 160 m below surface, with a maximum depth of 290 m. • Thickness across strike is typically 50 m to 80 m for the granite and sediment domains, but limited to 10 m to 20 m for the porphyry dykes.

Criteria	JORC Code explanation	Commentary
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	Boundary conditions: • The granite, sediments and dykes were kept separate for statistical analysis, variography and estimation as they are different geological units with mostly hard boundaries between them. The southern and northern granite and sediment domains were also kept separate as there is a physical separation and a difference in grade tenor between the two areas, particularly for the granite (higher in the southern area. • Based on statistical analysis and boundary conditions, the oxide, transitional and fresh domains were kept separate for statistical analysis, however, they were combined for variography. The oxide and transitional data were used to estimate the oxide, while the transitional and fresh data were used to estimate the transitional and fresh domains. The laterite domain has very little data in it. Visual inspection of the drillhole grades between the laterite and the oxide does not show evidence of a depleted zone. As a result of the small dataset, the laterite was combined with the oxide for estimation. The laterite data was not used for variography. • Review of the individual dykes showed that they are statistically similar and, given the relatively small amount of data in the individual dykes, the dykes were combined for statistical analysis and variography. The dykes were kept separate for estimation as there are hard boundaries between them. Estimation – granites and sediments: • Due to the highly skewed nature and presence of mixed populations the granites and sediments domains, multiple indicator kriging (MIK) was used to estimate gold grades. CAE Studio (Datamine) software was used to estimate the probability of grade being above or below each threshold into parent blocks of 10 mE by 10 mN by 5 mRL. • Kriging neighbourhood analysis (KNA) was used to define an appropriate block size for estimation and number of informing samples. Blocks were estimated using a minimum and maximum number of samples of 8 and 24 respectively. A maximum of 6 composites was allowed per drillhole for estimation. • The initial search pass used ranges equivalent to the ranges of continuity seen in the variograms at around 90 % to 100 % of the variance in the 0.3 g/ Au to 0.5 g/t Au threshold variograms, with the search ellipse orientated as per the higher grade thresholds (0.5 g/t Au and above). • Post processing of the MIK probability estimates was carried out in GSLIB software. Post processing was used to carry out order relation corrections using an averaging approach, and to calculate the e-type grade for each block. The e-type grades were calculated using a linear model between indicator thresholds and a hyperbolic or power model for the upper and lower tails. Estimation – dykes: • For the lower grade dykes, ordinary kriging was selected as appropriate, with top cuts to limit the influence of the higher grade composites. A top cut of 30 g/t Au was selected for the dykes in all weathering domains, based on review of the histograms and natural breaks in the populations in the log-probability plots. • Ordinary kriging (parent cell estimates) was used to estimate gold grades for the dykes using a discretisation of 5 east by 5 north by 3 elevation. • Due to the positively skewed nature of the grade distributions, normal scores variograms were modelled for the dykes. • A dynamic anisotropy approach was used, whereby the true dip and azimuth of the mineralised lodes was estimated into each block in the model and the search and variogram orientations were locally adjusted to reflect the geological orientation. This method allows the estimate to better reflect the changing orientation and undulating nature of some of these dykes along strike. • Blocks were estimated using a minimum and maximum number of samples of 8 and 24 respectively based on the KNA work. A maximum of 6 composites was allowed per drillhole for estimation. • The initial search pass used ranges equivalent to the ranges of continuity seen in the variograms at around 90 % to 100% of the variance. Validation: • Snowden validated the Sissengue model by: – Comparison of top cut input grades with tonnage weighted output grades. – Visual inspection of the model against the input composites. – Comparison of moving window input and output statistics. – Global change of support to check level of selectivity in model. – Snowden considers the estimate to be a reasonable reflection of the input data. – There has been no mining to date; no reconciliation data is available.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages are reported on a dry basis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The reporting cut-off is based on optimisation studies carried out as part of the 2010 Feasibility Study, which have suggested that the deposit can be economically extracted at a gold cut-off in the range 0.4 to 0.6 g/t.

Criteria	JORC Code explanation	Commentary
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Trial optimisation has been run in Whittle at a USD2400 gold price (double the current spot price) to define the base of potentially economic material. • Assumptions for the optimisation were based on studies carried out by Perseus as follows. • Cut-off grades: o Oxide 0.8 g/t Au o Transition 1.0 g/t Au o Granite/Porphyry 1.1 g/t Au o Sediments 1.3 g/t Au • Whittle processing cost (treatment + general administration) were: o Oxide ($21.79/t+$7.19/t) $28.98/t o Transition ($25.04/t+$8.14/t) $33.18/t o Granite/Porphyry ($26.29/t+$8.32/t) $34.61/t o Sediments ($30.29/t+$9.87/t) $40.13/t • Dilution: o Mining ore recovery (before diluted) 97% o Mining ore dilution (at 0 g/t Au dilutant grade) 3% • Geotechnical parameters: o Oxide (approximately from 390 mRL to 325 mRL) 31 degrees o Transition (approximately from 325 mRL to 300mRL) 41 degrees o Fresh (approximately from 300 mRL and below) 45 degrees • Mining cost estimate: o Contract miner $3.61/t o Fuel ($1.40/L) $0.59/t o Day-works (3%) $0.13/t o Total mining cost $4.33/t • Whittle mining cost calculations: o Base cost at surface $4.10/t o Incremental trucking cost $0.005/t/vm • The Whittle optimisation assumes that processing and infrastructure capital costs have been sunk • Where key information is absent or deficient, Snowden has provided considered assumptions. Overall these assumptions are considered fair for the purpose of determining the Mineral Resource potential of the project.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• There has been a considerable amount of metallurgical test work to date. • All indications are that gold can be satisfactorily recovered from Sissingue ore using conventional extraction techniques. • The metallurgical work is considered sufficient to determine that the Sissingue resource represents a deposit capable of economic extraction. • Recoveries used for the trial optimisation to determine the base of potentially economic material were based on studies carried out by Perseus as follows: o Oxide 92% o Transition 91% o Granite/Porphyry 90% o Sediments 78%
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• There are other gold mines operating within Mali and Ivory Coast in the general region where Sissingue is located. • The Sissingue Project area has no known environmental liabilities.

Criteria	JORC Code explanation	Commentary
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• A total of 770 bulk density measurements were taken from HQ and NQ drill core. • 380 results are from oxide material, 132 from transitional material and 258 from fresh material. • Calculated means for the transitional (2.30 g/cm^3) and fresh (2.73 g/cm^3) weathering domains as well as the lateritic horizon (1.90 g/cm^3) within the oxide domain were assigned to the block model. • The oxide domain, however, shows a gradient to higher densities with increasing depth and it is considered that a single value for the oxide domain would not be adequate. Instead, the bulk density for oxide blocks has been estimated using vertically orientated, inverse distance squared interpolation. The oxide bulk density ranges from 1.70 to 2.20 g/cm^3.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• Snowden reviewed the classification applied to the previous estimate (Widenbar, 2013) and concluded that the classification is appropriate with respect to confidence in tonnes and grade estimates; however recent information on recovery and mining factors has resulted in a trial optimisation being run to determine the potential for economic extraction, in particular for the lower grade and peripheral mineralisation. • The Sissingue Mineral Resource has been classified in the Measured, Indicated and Inferred categories, in accordance with the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC Code) and the CIM Definition Standards (CIM, 2005). A range of criteria has been considered in determining this classification including: – Geological continuity. – Data quality. – Drillhole spacing. – Estimation properties including kriging neighbourhood analysis to determine appropriate block size and search strategy. – Potential for economic extraction • The above parameters were used in combination arrive to guide the manual digitising of strings on drill sections to control classification. • Trial optimisation has been run at a USD2400 gold price to define the base of potentially mineable material. • The Competent Person endorses the final results and classification.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• The Mineral Resource estimation procedure and results has been internally peer reviewed by Snowden. • As yet, there are no external audits or reviews of the Mineral Resource estimate.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• No studies of relative accuracy have been carried out. The classification applied reflects the confidence in the Mineral Resource.

13 Mineral Resource reporting

Table 13.1 summarises the resources by category and area reported above a 0.6 g/t Au cut-off grade. The classification categories of Inferred, Indicated and Measured under the JORC Code (2012). They are equivalent to the CIM categories of the same name (CIM, 2010).

The reporting cut-off is based on optimisation studies carried out as part of the 2010 Feasibility Study, which have suggested that the deposit can be economically extracted at a gold cut-off in the range 0.4 g/t to 0.6 g/t. Trial optimisation has been run at a USD2400 gold price to define the base of potentially economic material (Section 12.5).

Metallurgical studies have confirmed that the Sissingue ore is amenable to processing by conventional CIL methods.

Table 13.1 October 2014 Sissengue Mineral Resource reported at a 0.6 g/t Au cut-off

Category	Area	Tonnage (kt)	Grade (g/t Au)	Contained Gold (oz)
Measured	Oxide	1,000	1.8	59,000
	Transitional	650	2.3	49,000
	Fresh	3,200	2.5	260,000
Total Measured		**4,800**	**2.4**	**370,000**
Indicated	Oxide	3,100	1.3	130,000
	Transitional	800	1.5	38,000
	Fresh	7,100	1.5	350,000
Total Indicated		**11,000**	**1.4**	**510,000**
Measured + Indicated	Oxide	4,100	1.4	190,000
	Transitional	1,400	1.9	87,000
	Fresh	10,000	1.8	600,000
Total Measured + Indicated		**16,000**	**1.7**	**880,000**
Inferred	Oxide	310	1.2	12,000
	Transitional	54	1.2	2,100
	Fresh	760	2.0	49,000
Total Inferred		**1,100**	**1.7**	**63,000**

Notes: Mineral Resources are inclusive of Mineral Reserves. Mineral Resources are reported to two significant figures. Rounding may cause minor discrepancies in the table. Oxide includes small portions of laterite (571 kt total).

13.1 Comparison to previous model

Snowden compared the updated Mineral Resource with the previously reported Mineral Resource (Widenbar, 2013) (Table 13.2). The comparison shows that the updated estimate is more selective, resulting in higher grades in the central, Measured portion of the estimate. This has resulted in an additional 60 koz in the Measured category.

The increased selectivity in the updated estimate is a result of using MIK estimation to honour the highly skewed, mixed distribution in the granite and sediment domains. The previous estimate used top cut ordinary kriging in these domains, which tends to over smooth in these styles of strongly skewed grade distributions.

There is a decrease in Indicated and Inferred Resources as a result of the application of the pit shell to define the base of potentially mineable material.

Table 13.2 Widenbar 2013 Sissengue Mineral Resource reported at a 0.6 g/t Au cut-off

Category	Area	Tonnage (kt)	Grade (g/t Au)	Contained Gold (oz)
Measured	Oxide	925	1.6	48,000
	Transitional	600	2.0	39,000
	Fresh	2,700	2.5	217,000
Total Measured		**4,225**	**2.2**	**305,000**
Indicated	Oxide	4,600	1.2	171,000
	Transitional	1,300	1.3	56,000
	Fresh	8,850	1.4	394,000
Total Indicated		**14,750**	**1.3**	**620,000**
Measured + Indicated	Oxide	5,520	1.2	218,000
	Transitional	1,900	1.6	95,000
	Fresh	11,560	1.6	612,000
Total Measured + Indicated		**18,980**	**1.5**	**925,000**
Inferred	Oxide	950	1.0	31,000
	Transitional	650	1.0	21,000
	Fresh	5,400	1.4	239,000
Total Inferred		**7,000**	**1.3**	**291,000**

14 Model fields

The final Datamine model (ssest1306.dm) was converted into a Surpac model (ssest1306.mdl). Non-essential fields generated during the estimation process were removed prior to the conversion for ease of use by the engineers. The final models fields are detailed in Table 14.1.

Table 14.1 Final model fields

Field	Value	Description
area	main	Main area
	central	Central area
	north	Northern area
	south	Southern area
	west	Western area
	nil	Outside of area wireframes
dom	1-9	Main dykes
	11-14	Central dukes
	21-23	Northern dykes
	30-33	Western dykes
	41-42	Southern dykes
	100	Southern granite mineralised domain
	200	Northern Granite mineralised domain
	300	Northern sediments mineralised domain
	400	Southern sediments mineralised domain
domain*	WASTE	Background
	GR	Granites
	SED	Sediments
	FSP	Dykes
weath_n	1	Laterite
	2	Oxide
	3	Transitional
	4	Sulphide
weatheri*	OX	Laterite and oxide
	TR	Transitional
	SUL	Sulphide
rescat	1	Measured
	2	Indicated
	3	Inferred
	4	Unclassified
bd	1.7 to 2.73	Bulk density g/cm^3
au		Final gold grade estimate
mikau*		MIK gold grade estimate in granite and sediment
au_ok*		OK gold grade estimate in dykes
numsamik*		Number of informing samples – MIK
numsamok*		Number of informing samples – OK
svolik*		Search pass – MIK
svolok*		Search pass – OK
varik*		Kriging variance – MIK
varok*		Kriging variance – OK
ke*		Kriging efficiency – OK
slope*		Slope of regression – OK
trdip*		Dip estimate for dykes
trdipdir*		Dip direction estimate for dykes

** fields not contained in the Surpac version of the model*

15 References

Author	Title
CIM, 2003	CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines , adopted by CIM Council on November 23, 2003
CIM, 2010	CIM DEFINITION STANDARDS - For Mineral Resources and Mineral Reserves. Prepared by the CIM Standing Committee on Reserve Definitions. Adopted by CIM Council on 27 November, 2010
JORC, 2012	The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Prepared by the Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).
Runge, 2010	Technical Report, Tengrela Gold Project, Ivory Coast. NI43-101 Technical Report compiled by Runge Limited for Perseus Mining, December 2010.
Widenbar, 2013	Sissengue Resource Estimate. Internal report written by Lynn Widenbar and Associates Pty Ltd for Perseus Mining, March 2013.

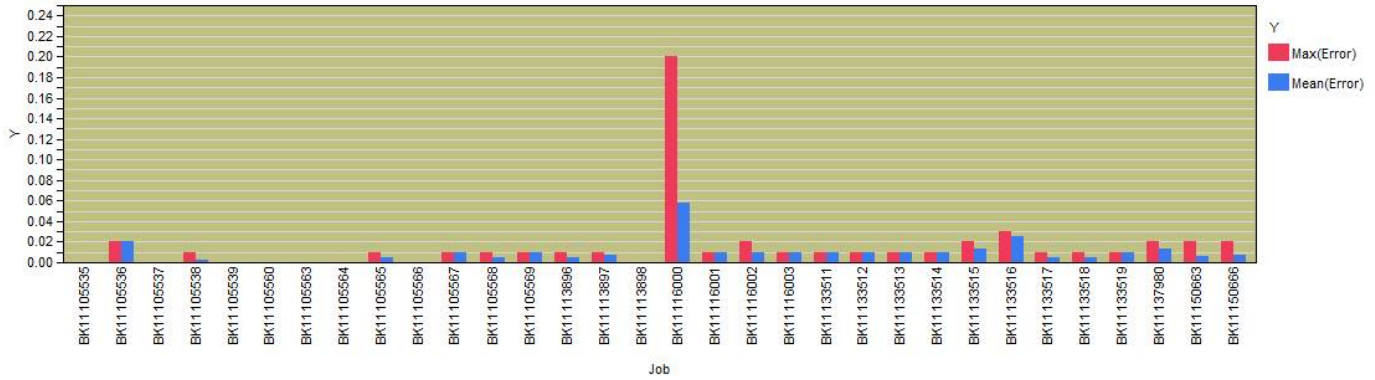
Appendix A QAQC plots – blanks

Lab Performance ALS Bamako - Blanks in Au g/t

Statistics

Job	# Blanks in Batch	Max Error	Mean Error
BK11105535	2	0	0
BK11105536	1	0.02	0.02
BK11105537	2	0	0
BK11105538	4	0.01	0
BK11105539	2	0	0
BK11105560	2	0	0
BK11105563	2	0	0
BK11105564	2	0	0
BK11105565	2	0.01	0.01
BK11105566	2	0	0
BK11105567	1	0.01	0.01
BK11105568	2	0.01	0.01
BK11105569	1	0.01	0.01
BK11113896	4	0.01	0.01
BK11113897	3	0.01	0.01
BK11113898	2	0	0
BK11116000	4	0.2	0.06
BK11116001	3	0.01	0.01
BK11116002	3	0.02	0.01
BK11116003	1	0.01	0.01
BK11133511	4	0.01	0.01
BK11133512	2	0.01	0.01
BK11133513	1	0.01	0.01
BK11133514	4	0.01	0.01
BK11133515	3	0.02	0.01
BK11133516	2	0.03	0.03
BK11133517	2	0.01	0.01
BK11133518	2	0.01	0.01
BK11133519	4	0.01	0.01
BK11137980	3	0.02	0.01
BK11150663	5	0.02	0.01
BK11150666	4	0.02	0.01

Max & Mean Error by Batch (in Au g/t) [Period between Jun 29, 2011 and Aug 30, 2011]

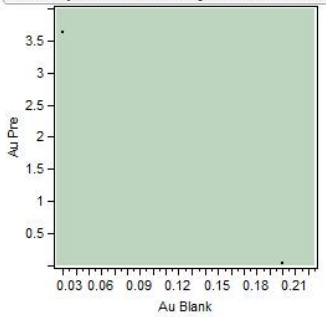


Failed Blanks

Job	Failed Blanks	Max Error
BK11116000	1	0.2
BK11133516	1	0.03

Failed if max error > 0.025Au g/t

Scatterplot - Blank Assay Value vs. Preceeding Assay



CRM Blanks in Au g/t

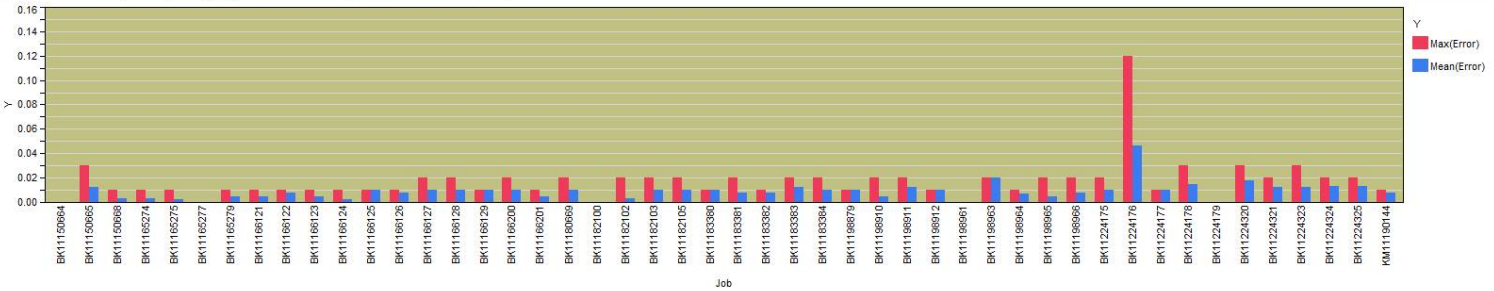
CRM	#	Expected Value	Reported Mean	Avg Error
GLG302-4	50	0	0.01	0.01
GLG307-1	7	0	0.01	0.01
GLG307-3	22	0	0.01	0.01
GLG907-1	2	0	0.11	0.11

Lab Performance ALS Bamako - Blanks in Au g/t

Statistics

Job	# Blanks in Batch	Max Error	Mean Error
BK11150664	1	0	0
BK11150665	4	0.03	0.01
BK11150668	3	0.01	0
BK11165274	3	0.01	0
BK11165275	4	0.01	0
BK11165277	1	0	0
BK11165279	4	0.01	0.01
BK11166121	4	0.01	0.01
BK11166122	4	0.01	0.01
BK11166123	4	0.01	0.01
BK11166124	5	0.01	0
BK11166125	3	0.01	0.01
BK11166126	4	0.01	0.01
BK11166127	4	0.02	0.01
BK11166128	4	0.02	0.01
BK11166129	4	0.01	0.01
BK11166200	3	0.02	0.01
BK11166201	4	0.01	0.01
BK11180669	4	0.02	0.01
BK11182100	1	0	0
BK11182102	6	0.02	0
BK11182103	2	0.02	0.01
BK11182105	4	0.02	0.01
BK11183380	4	0.01	0.01
BK11183381	4	0.02	0.01
BK11183382	4	0.01	0.01
BK11183383	4	0.02	0.01
BK11183384	3	0.02	0.01
BK11198879	4	0.01	0.01
BK11198910	4	0.02	0.01
BK11198911	4	0.02	0.01
BK11198912	4	0.01	0.01
BK11198961	2	0	0
BK11198963	1	0.02	0.02
BK11198964	3	0.01	0.01
BK11198965	4	0.02	0.01
BK11198966	4	0.02	0.01
BK11224175	5	0.02	0.01
BK11224176	3	0.12	0.05
BK11224177	1	0.01	0.01
BK11224178	4	0.03	0.02
BK11224179	2	0	0
BK11224320	4	0.03	0.02
BK11224321	4	0.02	0.01
BK11224323	4	0.03	0.01
BK11224324	3	0.02	0.01
BK11224325	3	0.02	0.01
KM11190144	5	0.01	0.01

Max & Mean Error by Batch (in Au g/t) [Period between Sep 5, 2011 and Dec 12, 2011]

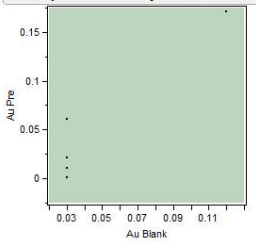


Failed Blanks

Job	Failed Blanks	Max Error
BK11150665	1	0.03
BK11224176	1	0.12
BK11224178	1	0.03
BK11224320	2	0.03
BK11224323	1	0.03

Failed if max error > 0.025Au g/t

Scatterplot - Blank Assay Value vs. Preceeding Assay



CRM Blanks in Au g/t

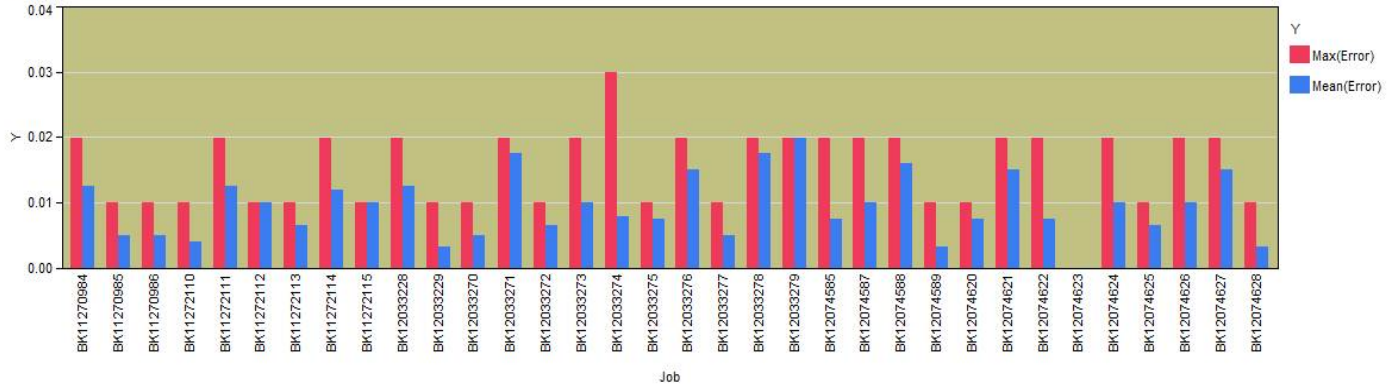
CRM	#	Expected Value	Reported Mean	Avg Error
GLG302-4	70	0	0.01	0.01
GLG307-1	38	0	0.01	0.01
GLG307-3	53	0	0.01	0.01
GLG907-1	5	0	0	0
GLG910-5	1	0	0	0

Lab Performance ALS Bamako - Blanks in Au g/t

Statistics

Job	# Blanks in Batch	Max Error	Mean Error
BK11270984	4	0.02	0.01
BK11270985	4	0.01	0.01
BK11270986	2	0.01	0.01
BK11272110	5	0.01	0
BK11272111	4	0.02	0.01
BK11272112	3	0.01	0.01
BK11272113	3	0.01	0.01
BK11272114	5	0.02	0.01
BK11272115	1	0.01	0.01
BK12033228	4	0.02	0.01
BK12033229	3	0.01	0
BK12033270	4	0.01	0.01
BK12033271	4	0.02	0.02
BK12033272	3	0.01	0.01
BK12033273	4	0.02	0.01
BK12033274	5	0.03	0.01
BK12033275	4	0.01	0.01
BK12033276	4	0.02	0.02
BK12033277	4	0.01	0.01
BK12033278	4	0.02	0.02
BK12033279	3	0.02	0.02
BK12074585	4	0.02	0.01
BK12074587	4	0.02	0.01
BK12074588	5	0.02	0.02
BK12074589	3	0.01	0
BK12074620	4	0.01	0.01
BK12074621	4	0.02	0.02
BK12074622	4	0.02	0.01
BK12074623	4	0	0
BK12074624	4	0.02	0.01
BK12074625	3	0.01	0.01
BK12074626	4	0.02	0.01
BK12074627	4	0.02	0.02
BK12074628	3	0.01	0

Max & Mean Error by Batch (in Au g/t) [Period between Feb 15, 2012 and Jun 19, 2012]



Detailed Investigation of Failed Blanks

Less than 2% of Blanks failed (Error > 0.025Au g/t). No further investigation required

CRM Blanks in Au g/t

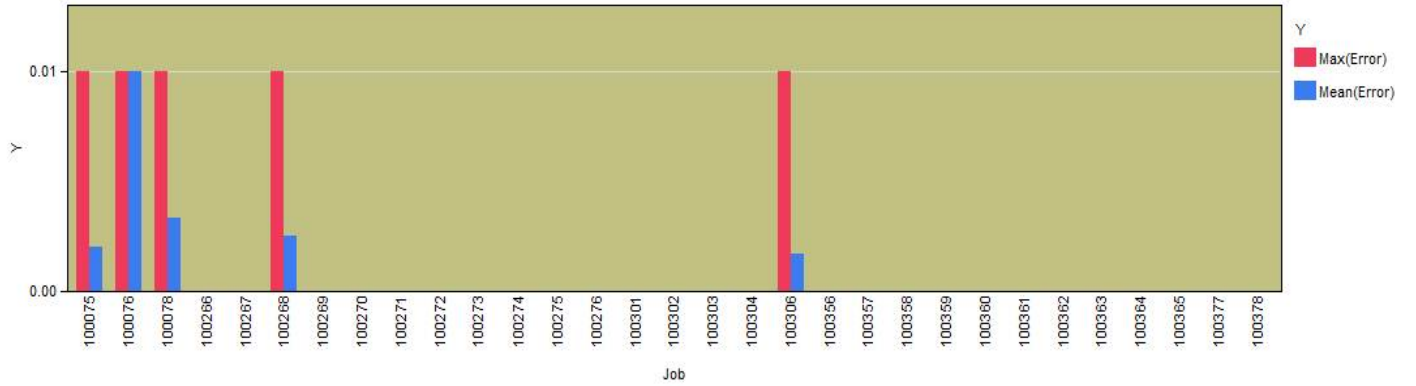
CRM	#	Expected Value	Reported Mean	Avg Error
GLG302-4	63	0	0.01	0.01
GLG307-3	64	0	0.01	0.01

Lab Performance Bureau Veritas Bamako- Blanks in Au g/t

Statistics

Job	# Blanks in Batch	Max Error	Mean Error
100075	5	0.01	0
100076	2	0.01	0.01
100078	3	0.01	0
100266	4	0	0
100267	4	0	0
100268	4	0.01	0
100269	4	0	0
100270	4	0	0
100271	4	0	0
100272	3	0	0
100273	3	0	0
100274	4	0	0
100275	4	0	0
100276	5	0	0
100301	4	0	0
100302	4	0	0
100303	5	0	0
100304	7	0	0
100306	12	0.01	0
100356	4	0	0
100357	3	0	0
100358	4	0	0
100359	4	0	0
100360	4	0	0
100361	4	0	0
100362	4	0	0
100363	4	0	0
100364	4	0	0
100365	5	0	0
100377	6	0	0
100378	3	0	0

Max & Mean Error by Batch (in Au g/t) [Period between Oct 3, 2012 and Jan 18, 2013]



Detailed Investigation of Failed Blanks

No detailed investigation of failed blanks has been carried out

CRM Blanks in Au g/t

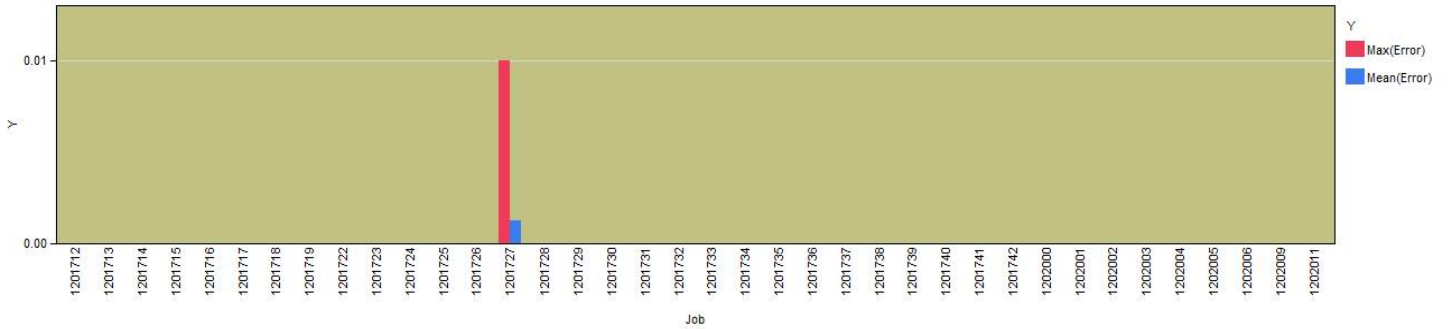
CRM	#	Expected Value	Reported Mean	Avg Error
GLG307-1	2	0	0	0
GLG307-3	38	0	0	0
GLG911-3	94	0	0	0

Lab Performance Intertek Tarkwa - Blanks in Au g/t

Statistics

Job	# Blanks in Batch	Max Error	Mean Error
1201712	6	0	0
1201713	6	0	0
1201714	6	0	0
1201715	6	0	0
1201716	6	0	0
1201717	6	0	0
1201718	6	0	0
1201719	6	0	0
1201722	6	0	0
1201723	2	0	0
1201724	6	0	0
1201725	6	0	0
1201726	6	0	0
1201727	8	0.01	0
1201728	7	0	0
1201729	6	0	0
1201730	7	0	0
1201731	6	0	0
1201732	6	0	0
1201733	8	0	0
1201734	6	0	0
1201735	5	0	0
1201736	5	0	0
1201737	6	0	0
1201738	6	0	0
1201739	6	0	0
1201740	8	0	0
1201741	6	0	0
1201742	5	0	0
1202000	6	0	0
1202001	5	0	0
1202002	5	0	0
1202003	6	0	0
1202004	6	0	0
1202005	7	0	0
1202006	5	0	0
1202009	6	0	0
1202011	1	0	0

Max & Mean Error by Batch (in Au g/t) [Period between Jul 10, 2012 and Aug 31, 2012]



Detailed Investigation of Failed Blanks

No detailed investigation of failed blanks has been carried out

CRM Blanks in Au g/t

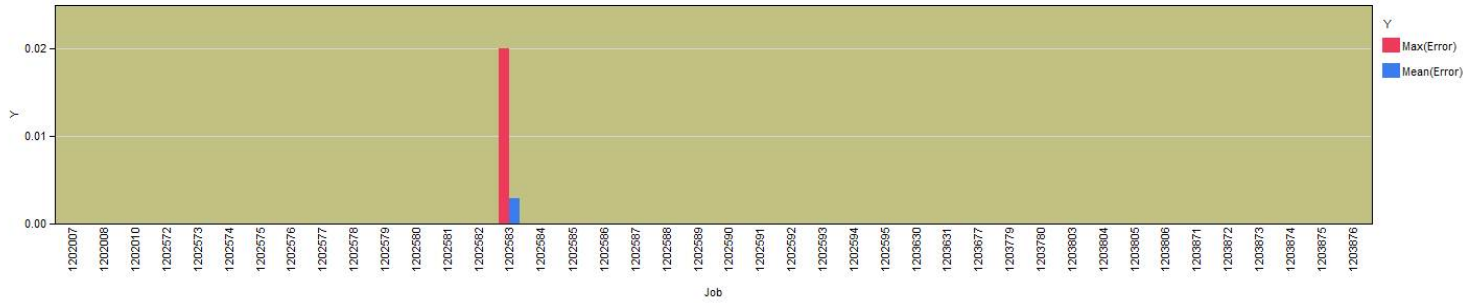
CRM	#	Expected Value	Reported Mean	Avg Error
GLG302-4	24	0	0	0
GLG307-1	26	0	0	0
GLG307-3	111	0	0	0
GLG911-3	61	0	0	0

Lab Performance Intertek Tarkwa - Blanks in Au g/t

Statistics

Job	# Blanks in Batch	Max Error	Mean Error
1202007	6	0	0
1202008	5	0	0
1202010	4	0	0
1202572	6	0	0
1202573	7	0	0
1202574	6	0	0
1202575	6	0	0
1202576	6	0	0
1202577	6	0	0
1202578	7	0	0
1202579	6	0	0
1202580	6	0	0
1202581	6	0	0
1202582	5	0	0
1202583	7	0.02	0
1202584	6	0	0
1202585	6	0	0
1202586	6	0	0
1202587	5	0	0
1202588	6	0	0
1202589	6	0	0
1202590	8	0	0
1202591	6	0	0
1202592	6	0	0
1202593	6	0	0
1202594	6	0	0
1202595	4	0	0
1203630	5	0	0
1203631	5	0	0
1203677	4	0	0
1203779	6	0	0
1203780	7	0	0
1203803	7	0	0
1203804	7	0	0
1203805	8	0	0
1203806	2	0	0
1203871	6	0	0
1203872	5	0	0
1203873	6	0	0
1203874	6	0	0
1203875	5	0	0
1203876	1	0	0

Max & Mean Error by Batch (in Au g/t) [Period between Sep 3, 2012 and Jan 1, 2013]



Detailed Investigation of Failed Blanks

No detailed investigation of failed blanks has been carried out

CRM Blanks in Au g/t

CRM	#	Expected Value	Reported Mean	Avg Error
GLG307-1	39	0	0	0
GLG307-3	87	0	0	0
GLG910-5	2	0	0	0
GLG911-3	112	0	0	0

Appendix B QAQC plots - standards

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G02	1.2	0.08

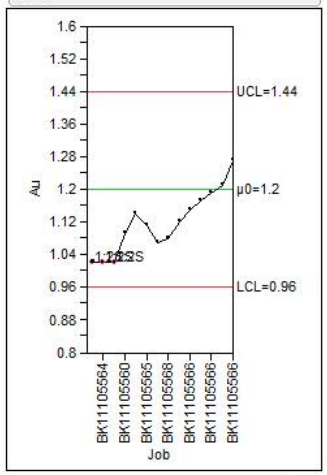
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G02	14	1.2	1.119	0.076	7.74	13.52	0.093	3	21.4	0	0

Levey Jennings Control Chart

G02



Notes

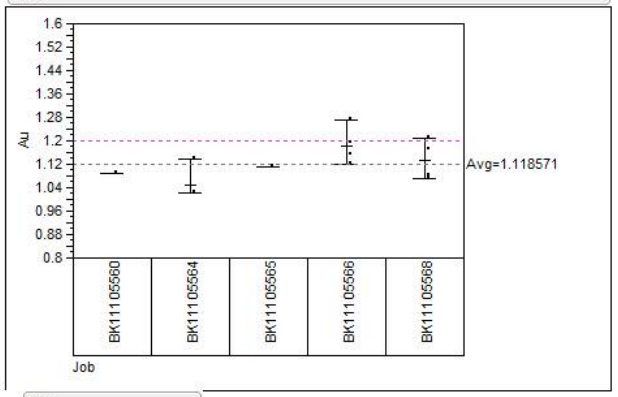
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105560	1	1.2	1.09	.	9.17	.	0.11	0	0	0	0
BK11105564	4	1.2	1.05	0.06	12.5	11.43	0.15	3	75	0	0
BK11105565	1	1.2	1.11	.	7.5	.	0.09	0	0	0	0
BK11105566	4	1.2	1.183	0.065	4.38	10.99	0.053	0	0	0	0
BK11105568	4	1.2	1.133	0.068	6.04	12.1	0.072	0	0	0	0

Variability Gauge

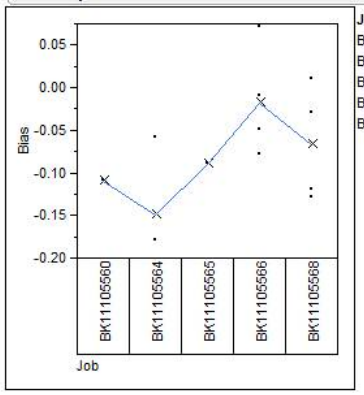
G02



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105560	-0.11000
BK11105564	-0.15000
BK11105565	-0.09000
BK11105566	-0.01750
BK11105568	-0.06750

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G302-6	0.99	0.05

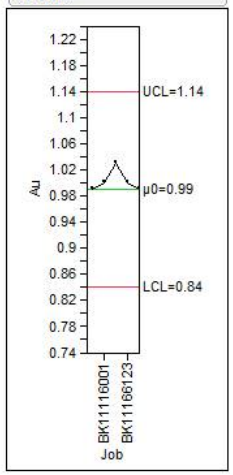
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G302-6	5	0.99	1.002	0.016	1.21	3.28	0.012	0	0	0	0

Levey Jennings Control Chart

G302-6

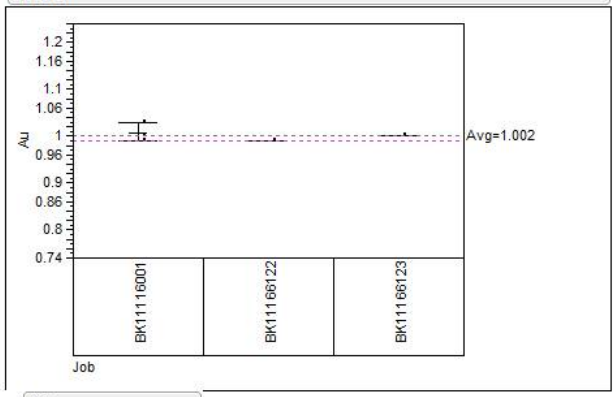


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G302-6



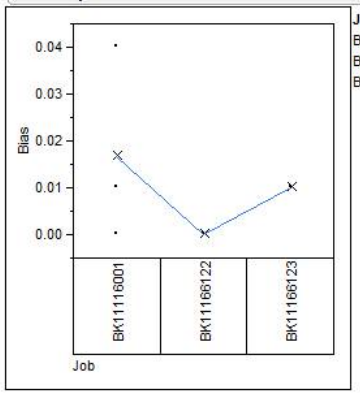
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11116001	3	0.99	1.007	0.021	1.68	4.14	0.017	0	0	0	0
BK11166122	1	0.99	0.99	.	0	.	0	0	0	0	0
BK11166123	1	0.99	1	.	1.01	.	0.01	0	0	0	0

Bias Report for Job



Job	Avg Bias
BK11116001	0.016667
BK11166122	0.000000
BK11166123	0.010000

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G306-2	1.049	0.072

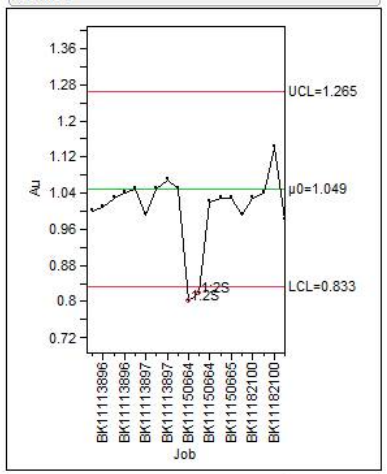
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G306-2	19	1.049	1.009	0.078	4.97	15.55	0.052	2	10.5	2	10.5

Levey Jennings Control Chart

G306-2

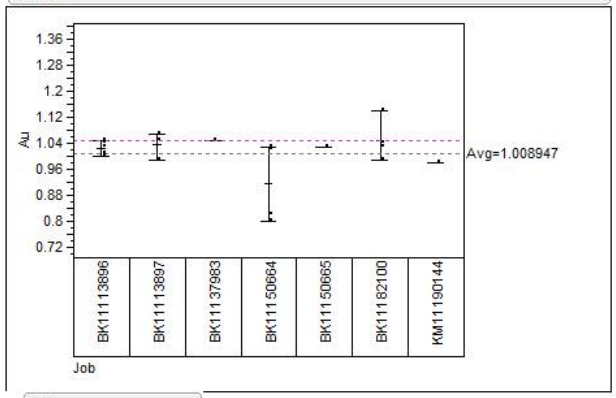


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G306-2



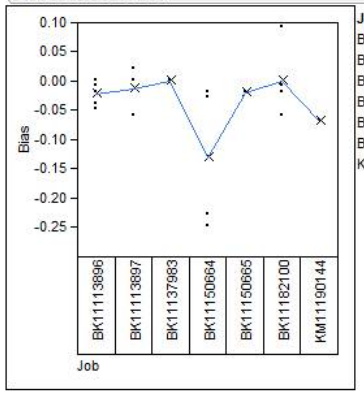
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11113896	5	1.049	1.026	0.021	2.23	4.04	0.023	0	0	0	0
BK11113897	3	1.049	1.037	0.042	2.57	8.03	0.027	0	0	0	0
BK11137983	1	1.049	1.05	.	0.1	.	0.001	0	0	0	0
BK11150664	4	1.049	0.918	0.124	12.54	27.13	0.131	2	50	2	50
BK11150665	1	1.049	1.03	.	1.81	.	0.019	0	0	0	0
BK11182100	4	1.049	1.05	0.064	4.24	12.15	0.044	0	0	0	0
KM11190144	1	1.049	0.98	.	6.58	.	0.069	0	0	0	0

Bias Report for Job



Job	Avg Bias
BK11113896	-0.02300
BK11113897	-0.01233
BK11137983	0.00100
BK11150664	-0.13150
BK11150665	-0.01900
BK11182100	0.00100
KM11190144	-0.06900

QAQC Analysis for Period From Feb 15, 2012 - Jun 19, 2012 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G306-2	1.049	0.072

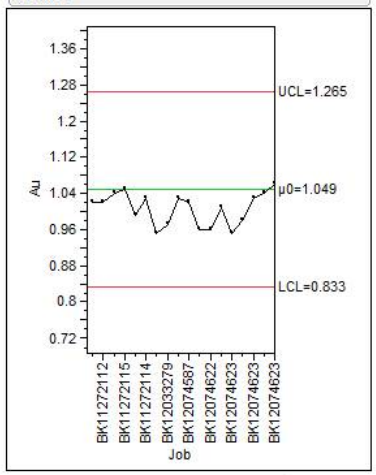
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G306-2	18	1.049	1.006	0.036	4.22	7.19	0.044	0	0	0	0

Levey Jennings Control Chart

G306-2



Notes

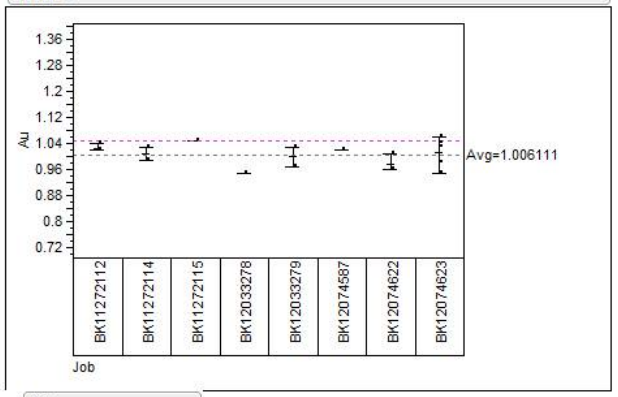
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11272112	3	1.049	1.027	0.012	2.13	2.25	0.022	0	0	0	0
BK11272114	2	1.049	1.01	0.028	3.72	5.6	0.039	0	0	0	0
BK11272115	1	1.049	1.05	.	0.1	.	0.001	0	0	0	0
BK12033278	1	1.049	0.95	.	9.44	.	0.099	0	0	0	0
BK12033279	2	1.049	1	0.042	4.67	8.49	0.049	0	0	0	0
BK12074587	1	1.049	1.02	.	2.76	.	0.029	0	0	0	0
BK12074622	3	1.049	0.977	0.029	6.9	5.91	0.072	0	0	0	0
BK12074623	5	1.049	1.012	0.045	3.95	8.99	0.041	0	0	0	0

Variability Gauge

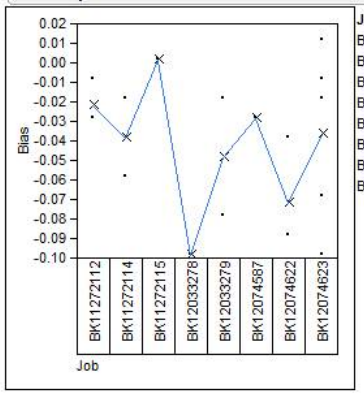
G306-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11272112	-0.02233
BK11272114	-0.03900
BK11272115	0.00100
BK12033278	-0.09900
BK12033279	-0.04900
BK12074587	-0.02900
BK12074622	-0.07233
BK12074623	-0.03700

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G306-2	1.049	0.072

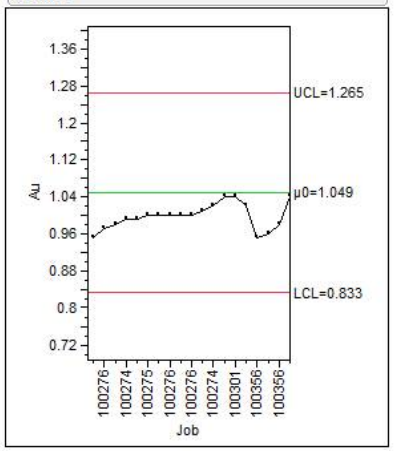
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G306-2	19	1.049	0.997	0.028	4.97	5.6	0.052	0	0	0	0

Levey Jennings Control Chart

G306-2



Notes

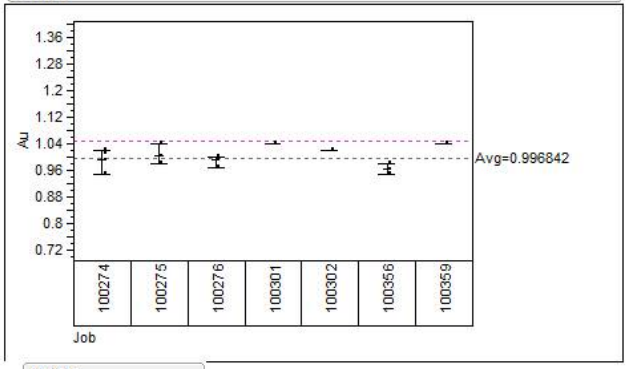
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100274	4	1.049	0.993	0.031	5.39	6.24	0.056	0	0	0	0
100275	4	1.049	1.005	0.025	4.19	5.01	0.044	0	0	0	0
100276	5	1.049	0.992	0.013	5.43	2.63	0.057	0	0	0	0
100301	1	1.049	1.04	.	0.86	.	0.009	0	0	0	0
100302	1	1.049	1.02	.	2.76	.	0.029	0	0	0	0
100356	3	1.049	0.963	0.015	8.17	3.17	0.086	0	0	0	0
100359	1	1.049	1.04	.	0.86	.	0.009	0	0	0	0

Variability Gauge

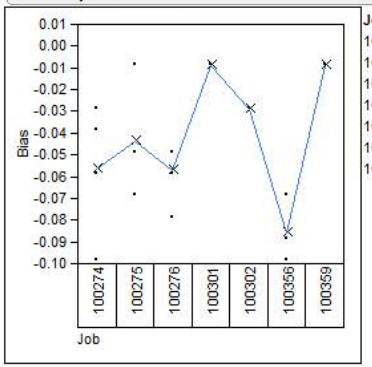
G306-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
100274	-0.05650
100275	-0.04400
100276	-0.05700
100301	-0.00900
100302	-0.02900
100356	-0.08567
100359	-0.00900

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 --- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G306-2	1.049	0.072

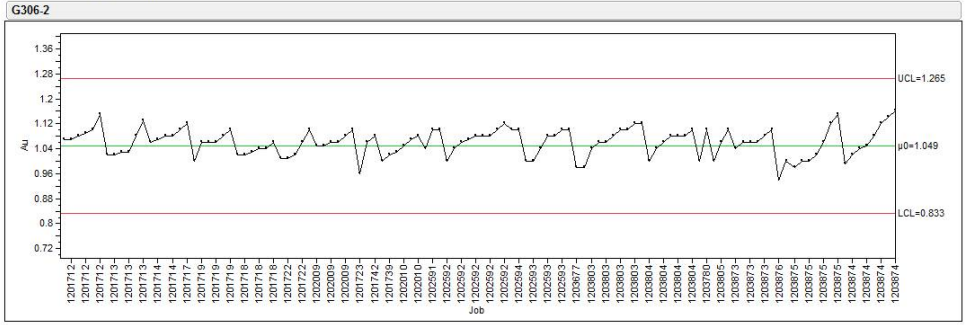
Manufactured By
Geostats

Summary Lab Performance for Period From Jul 10, 2012 - Jan 1, 2013

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G306-2	116	1.049	1.06	0.043	3.48	8.07	0.037	0	0	0	0

Levey Jennings Control Chart

G306-2



Notes

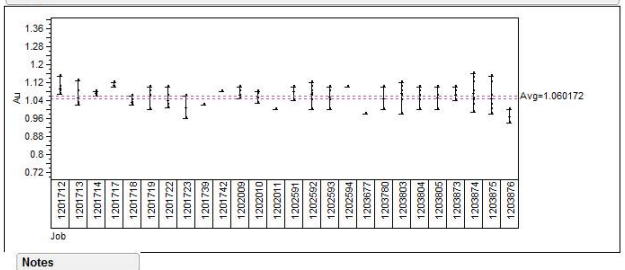
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201712	6	1.049	1.093	0.03	4.23	5.51	0.044	0	0	0	0
1201713	6	1.049	1.052	0.044	3.3	8.46	0.035	0	0	0	0
1201714	4	1.049	1.073	0.01	2.24	1.79	0.024	0	0	0	0
1201717	2	1.049	1.11	0.014	5.82	2.55	0.061	0	0	0	0
1201718	6	1.049	1.035	0.015	1.68	2.93	0.018	0	0	0	0
1201719	6	1.049	1.06	0.033	2.61	6.31	0.027	0	0	0	0
1201722	5	1.049	1.04	0.039	3.22	7.57	0.034	0	0	0	0
1201723	2	1.049	1.01	0.071	4.77	14	0.05	0	0	0	0
1201739	1	1.049	1.02	.	2.76	.	0.029	0	0	0	0
1201742	1	1.049	1.08	.	2.96	.	0.031	0	0	0	0
1202009	6	1.049	1.067	0.02	1.68	3.69	0.018	0	0	0	0
1202010	4	1.049	1.058	0.022	1.72	4.19	0.018	0	0	0	0
1202011	1	1.049	1	.	4.67	.	0.049	0	0	0	0
1202591	3	1.049	1.08	0.035	3.53	6.42	0.037	0	0	0	0
1202592	9	1.049	1.07	0.035	3.23	6.47	0.034	0	0	0	0
1202593	7	1.049	1.057	0.044	3.69	8.3	0.039	0	0	0	0
1202594	2	1.049	1.1	0	4.86	0	0.051	0	0	0	0
1203677	1	1.049	0.98	.	6.58	.	0.069	0	0	0	0
1203780	2	1.049	1.05	0.071	4.77	13.47	0.05	0	0	0	0
1203803	9	1.049	1.073	0.045	3.97	8.33	0.042	0	0	0	0
1203804	7	1.049	1.063	0.034	2.9	6.31	0.03	0	0	0	0
1203805	3	1.049	1.053	0.05	3.53	9.56	0.037	0	0	0	0
1203873	6	1.049	1.067	0.021	1.97	3.87	0.021	0	0	0	0
1203874	8	1.049	1.075	0.06	4.79	11.25	0.05	0	0	0	0
1203875	7	1.049	1.047	0.066	5.16	12.51	0.054	0	0	0	0
1203876	2	1.049	0.97	0.042	7.53	8.75	0.079	0	0	0	0

Variability Gauge

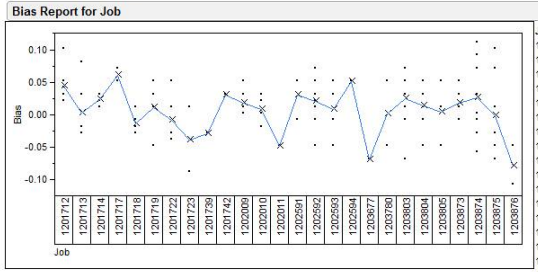
G306-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1201712	0.04433
1201713	0.00267
1201714	0.02350
1201717	0.06100
1201718	-0.01400
1201719	0.01100
1201722	-0.00900
1201723	-0.03900
1201739	-0.02900
1201742	0.03100
1202009	0.01767
1202010	0.00850
1202011	-0.04900
1202591	0.03100
1202592	0.02100
1202593	0.00814
1202594	0.05100
1203677	-0.06900
1203780	0.00100
1203803	0.02433
1203804	0.01386
1203805	0.00433
1203873	0.01767
1203874	0.02600
1203875	-0.00186
1203876	-0.07900

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G306-7	4.5	0.2

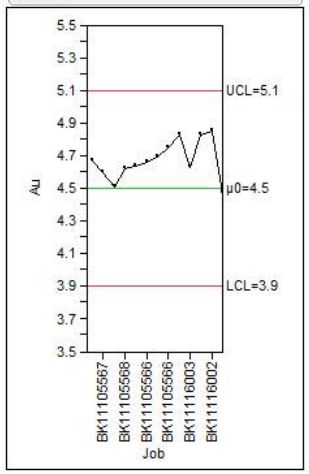
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G306-7	13	4.5	4.673	0.118	3.95	5.03	0.178	0	0	0	0

Levey Jennings Control Chart

G306-7



Notes

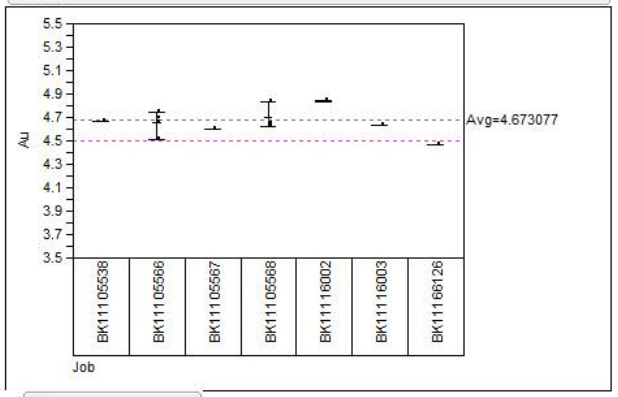
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105538	1	4.5	4.67	.	3.78	.	0.17	0	0	0	0
BK11105566	4	4.5	4.653	0.102	3.39	4.39	0.153	0	0	0	0
BK11105567	1	4.5	4.6	.	2.22	.	0.1	0	0	0	0
BK11105568	3	4.5	4.697	0.116	4.37	4.94	0.197	0	0	0	0
BK11116002	2	4.5	4.84	0.014	7.56	0.58	0.34	0	0	0	0
BK11116003	1	4.5	4.63	.	2.89	.	0.13	0	0	0	0
BK11166126	1	4.5	4.47	.	0.67	.	0.03	0	0	0	0

Variability Gauge

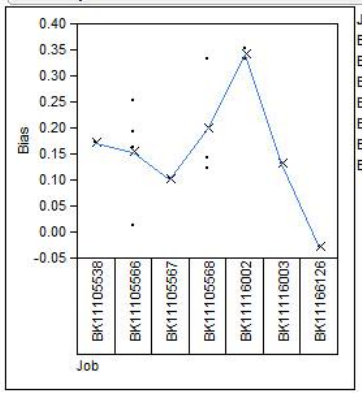
G306-7



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105538	0.17000
BK11105566	0.15250
BK11105567	0.10000
BK11105568	0.19667
BK11116002	0.34000
BK11116003	0.13000
BK11166126	-0.03000

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G307-2	1.08	0.05

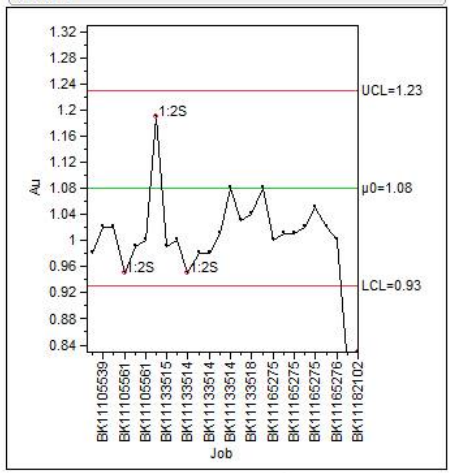
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G307-2	26	1.08	1.001	0.072	8.08	14.46	0.087	8	30.8	2	7.7

Levey Jennings Control Chart

G307-2



Notes

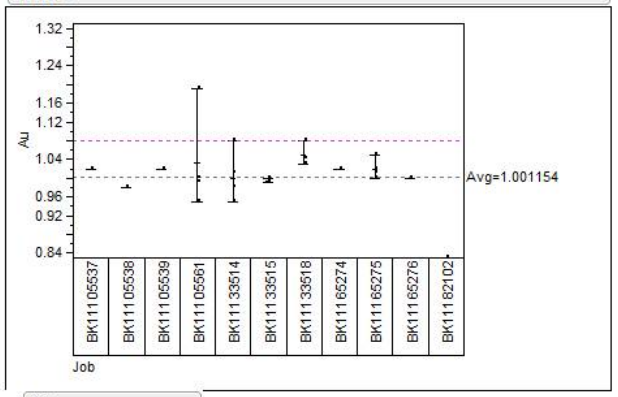
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105537	1	1.08	1.02	.	5.56	.	0.06	0	0	0	0
BK11105538	1	1.08	0.98	.	9.26	.	0.1	1	100	0	0
BK11105539	1	1.08	1.02	.	5.56	.	0.06	0	0	0	0
BK11105561	4	1.08	1.033	0.107	9.49	20.76	0.103	2	50	0	0
BK11133514	5	1.08	1	0.049	7.41	9.9	0.08	3	60	0	0
BK11133515	2	1.08	0.995	0.007	7.87	1.42	0.085	0	0	0	0
BK11133518	3	1.08	1.05	0.026	2.78	5.04	0.03	0	0	0	0
BK11165274	1	1.08	1.02	.	5.56	.	0.06	0	0	0	0
BK11165275	5	1.08	1.018	0.019	5.74	3.78	0.062	0	0	0	0
BK11165276	1	1.08	1	.	7.41	.	0.08	0	0	0	0
BK11182102	2	1.08	0.815	0.021	24.54	5.21	0.265	2	100	2	100

Variability Gauge

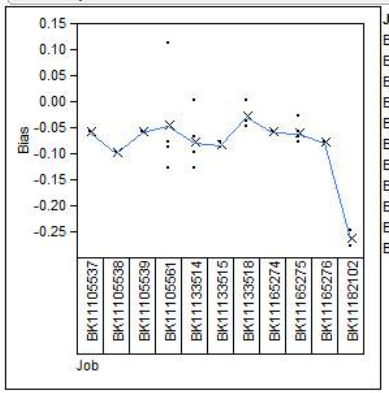
G307-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105537	-0.06000
BK11105538	-0.10000
BK11105539	-0.06000
BK11105561	-0.04750
BK11133514	-0.08000
BK11133515	-0.08500
BK11133518	-0.03000
BK11165274	-0.06000
BK11165275	-0.06200
BK11165276	-0.08000
BK11182102	-0.26500

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G307-8	1.99	0.08

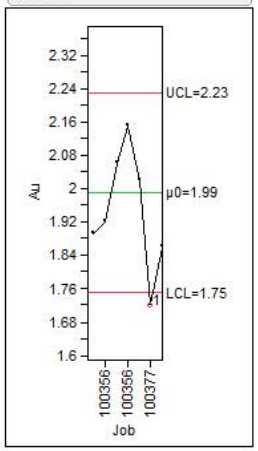
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G307-8	7	1.99	1.946	0.143	5.96	14.68	0.119	2	28.6	1	14.3

Levey Jennings Control Chart

G307-8



Notes

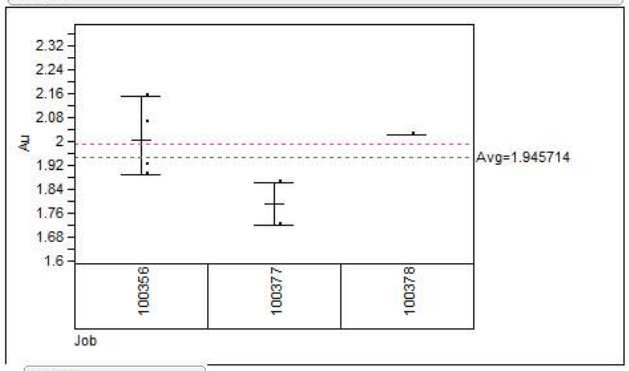
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100356	4	1.99	2.005	0.122	5.03	12.15	0.1	1	25	0	0
100377	2	1.99	1.79	0.099	10.05	11.06	0.2	1	50	1	50
100378	1	1.99	2.02	.	1.51	.	0.03	0	0	0	0

Variability Gauge

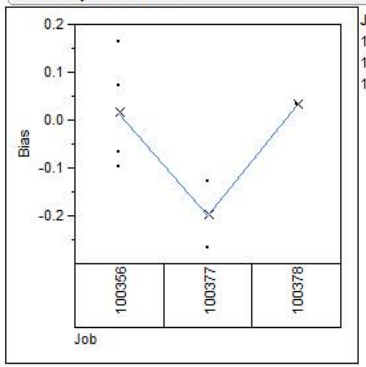
G307-8



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
100356	0.01500
100377	-0.20000
100378	0.03000

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G308-3	2.5	0.11

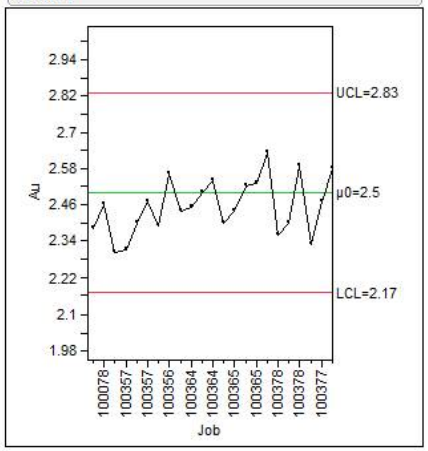
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G308-3	23	2.5	2.454	0.091	3.39	7.46	0.085	0	0	0	0

Levey Jennings Control Chart

G308-3

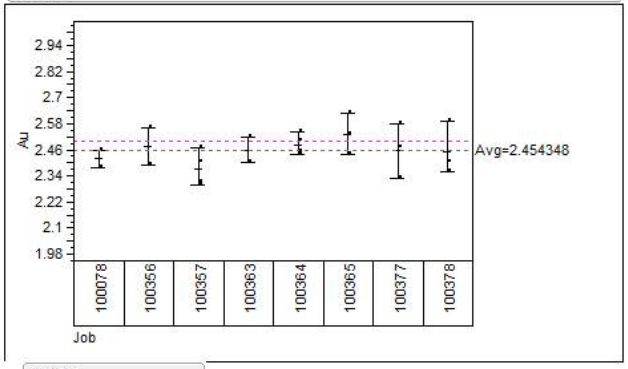


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

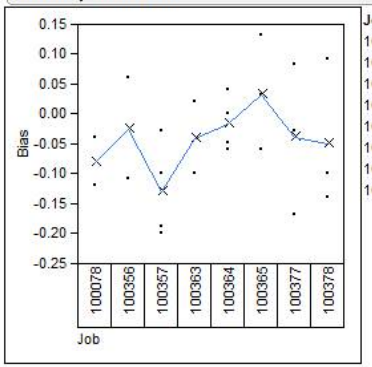
G308-3



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
100078	-0.08000
100356	-0.02500
100357	-0.13000
100363	-0.04000
100364	-0.01750
100365	0.03333
100377	-0.04000
100378	-0.05000

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100078	2	2.5	2.42	0.057	3.2	4.68	0.08	0	0	0	0
100356	2	2.5	2.475	0.12	3.4	9.71	0.085	0	0	0	0
100357	4	2.5	2.37	0.08	5.2	6.79	0.13	0	0	0	0
100363	2	2.5	2.46	0.085	2.4	6.9	0.06	0	0	0	0
100364	4	2.5	2.483	0.046	1.5	3.74	0.037	0	0	0	0
100365	3	2.5	2.533	0.095	2.93	7.5	0.073	0	0	0	0
100377	3	2.5	2.46	0.125	3.73	10.19	0.093	0	0	0	0
100378	3	2.5	2.45	0.123	4.4	10.03	0.11	0	0	0	0

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G397-2	4.49	0.18

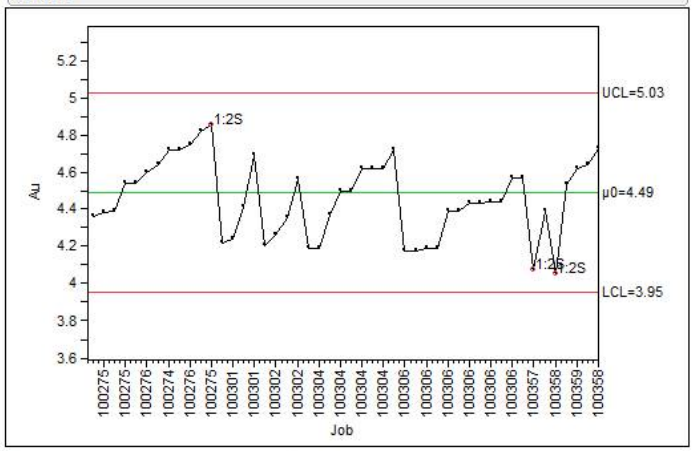
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G397-2	48	4.49	4.458	0.206	3.87	9.25	0.174	3	6.3	0	0

Levey Jennings Control Chart

G397-2

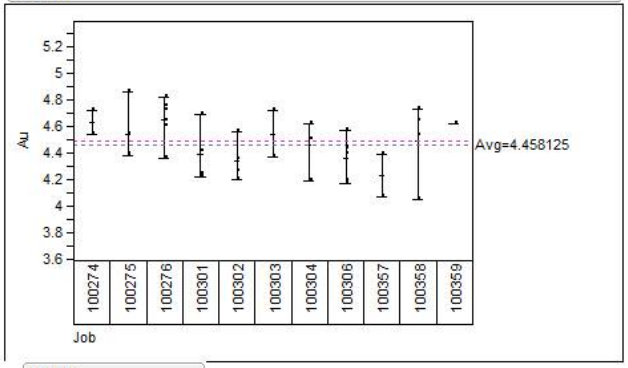


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G397-2



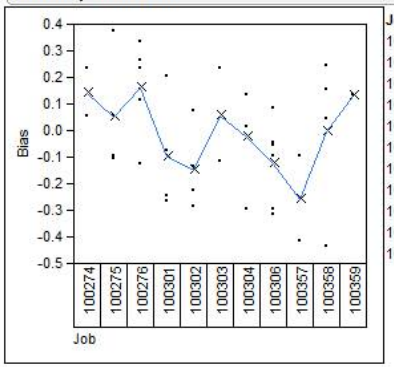
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100274	2	4.49	4.63	0.127	3.12	5.5	0.14	0	0	0	0
100275	4	4.49	4.543	0.224	3.51	9.86	0.158	1	25	0	0
100276	6	4.49	4.648	0.162	4.49	6.95	0.202	0	0	0	0
100301	4	4.49	4.39	0.217	4.45	9.9	0.2	0	0	0	0
100302	4	4.49	4.343	0.158	4.06	7.26	0.183	0	0	0	0
100303	2	4.49	4.545	0.247	3.9	10.89	0.175	0	0	0	0
100304	7	4.49	4.463	0.194	3.21	8.69	0.144	0	0	0	0
100306	12	4.49	4.365	0.148	3.38	6.8	0.152	0	0	0	0
100357	2	4.49	4.23	0.226	5.79	10.7	0.26	1	50	0	0
100358	4	4.49	4.488	0.303	4.84	13.5	0.218	1	25	0	0
100359	1	4.49	4.62	.	2.9	.	0.13	0	0	0	0

Bias Report for Job



Job	Avg Bias
100274	0.14000
100275	0.05250
100276	0.15833
100301	-0.10000
100302	-0.14750
100303	0.05500
100304	-0.02714
100306	-0.12500
100357	-0.26000
100358	-0.00250
100359	0.13000

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 --- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G397-2	4.49	0.18

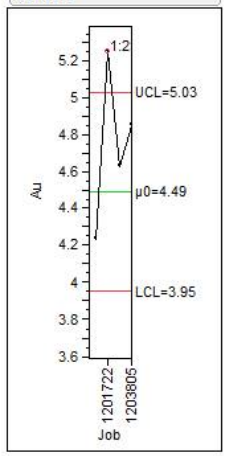
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G397-2	4	4.49	4.74	0.426	8.46	17.97	0.38	2	50	1	25

Levey Jennings Control Chart

G397-2

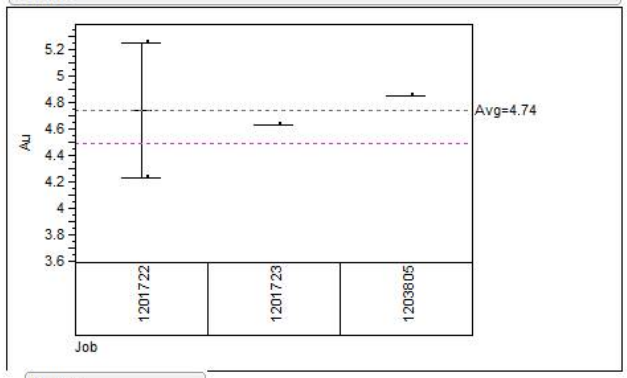


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G397-2



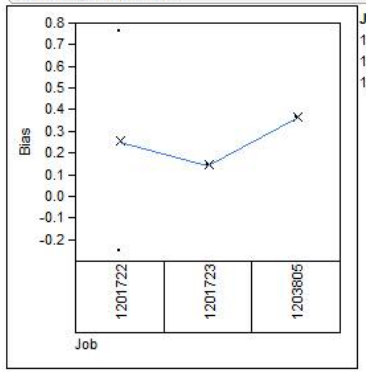
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201722	2	4.49	4.74	0.721	11.36	30.43	0.51	1	50	1	50
1201723	1	4.49	4.63	.	3.12	.	0.14	0	0	0	0
1203805	1	4.49	4.85	.	8.02	.	0.36	1	100	0	0

Bias Report for Job



Job	Avg Bias
1201722	0.250000
1201723	0.140000
1203805	0.360000

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G398-4	0.66	0.05

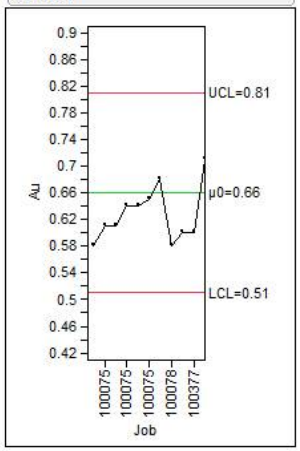
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G398-4	11	0.66	0.627	0.041	6.89	13.15	0.045	0	0	0	0

Levey Jennings Control Chart

G398-4



Notes

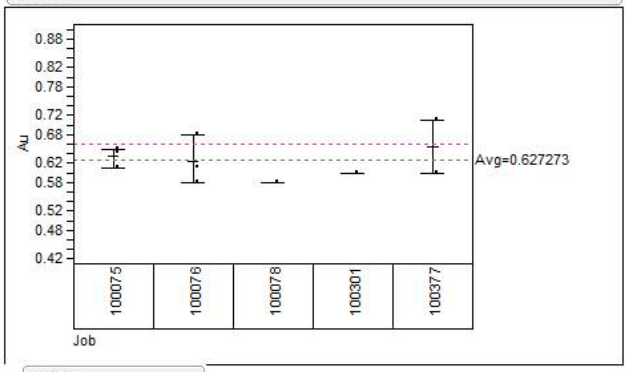
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100075	4	0.66	0.635	0.017	3.79	5.46	0.025	0	0	0	0
100076	3	0.66	0.623	0.051	7.58	16.47	0.05	0	0	0	0
100078	1	0.66	0.58	.	12.12	.	0.08	0	0	0	0
100301	1	0.66	0.6	.	9.09	.	0.06	0	0	0	0
100377	2	0.66	0.655	0.078	8.33	23.75	0.055	0	0	0	0

Variability Gauge

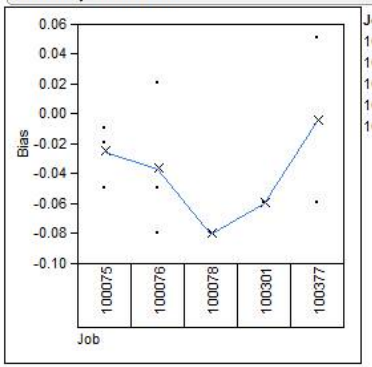
G398-4



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
100075	-0.02500
100076	-0.03667
100078	-0.08000
100301	-0.06000
100377	-0.00500

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 --- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G398-4	0.66	0.05

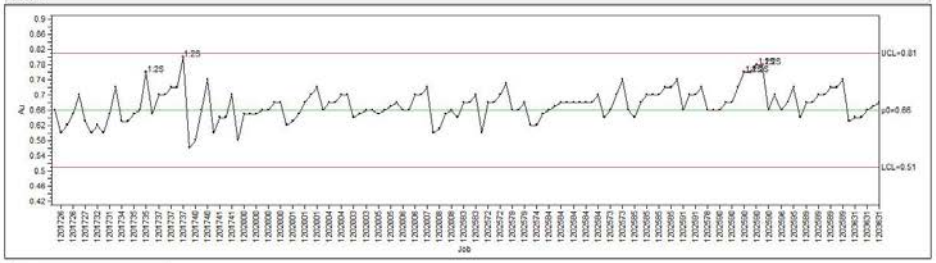
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G398-4	156	0.66	0.673	0.042	4.97	12.4	0.033	7	5.1	0	0

Levey Jennings Control Chart

G398-4



Notes

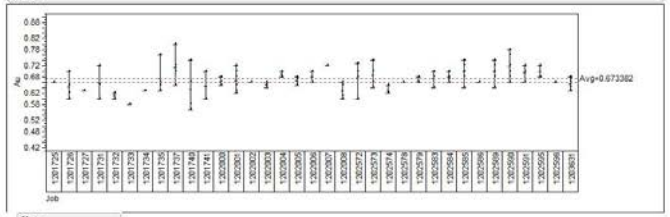
Red Line - 3 Std Dev
Westgard Rule 1.2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201725	1	0.66	0.66	.	0	.	0	0	0	0	0
1201726	4	0.66	0.643	0.043	5.68	13.54	0.038	0	0	0	0
1201727	1	0.66	0.63	.	4.55	.	0.03	0	0	0	0
1201731	3	0.66	0.657	0.06	6.57	18.36	0.043	0	0	0	0
1201732	2	0.66	0.61	0.014	7.58	4.64	0.05	0	0	0	0
1201733	1	0.66	0.58	.	12.12	.	0.08	0	0	0	0
1201734	1	0.66	0.63	.	4.55	.	0.03	0	0	0	0
1201735	4	0.66	0.675	0.058	5.3	17.19	0.035	1	25	0	0
1201737	6	0.66	0.715	0.049	8.84	13.67	0.058	1	16.7	0	0
1201740	4	0.66	0.635	0.082	9.85	25.91	0.065	1	25	0	0
1201741	4	0.66	0.645	0.041	5.3	12.78	0.035	0	0	0	0
1202000	7	0.66	0.661	0.013	1.52	4.07	0.01	0	0	0	0
1202001	6	0.66	0.667	0.04	5.05	11.95	0.033	0	0	0	0
1202002	1	0.66	0.66	.	0	.	0	0	0	0	0
1202003	4	0.66	0.653	0.01	1.14	2.93	0.008	0	0	0	0
1202004	4	0.66	0.69	0.012	4.55	3.35	0.03	0	0	0	0
1202005	4	0.66	0.665	0.013	1.52	3.88	0.01	0	0	0	0
1202006	4	0.66	0.68	0.023	3.03	6.79	0.02	0	0	0	0
1202007	1	0.66	0.72	.	9.09	.	0.06	0	0	0	0
1202008	4	0.66	0.63	0.029	4.55	9.35	0.03	0	0	0	0
1202572	5	0.66	0.678	0.048	6.36	14.21	0.042	0	0	0	0
1202573	4	0.66	0.685	0.044	5.3	12.95	0.035	0	0	0	0
1202574	3	0.66	0.63	0.017	4.55	5.5	0.03	0	0	0	0
1202578	2	0.66	0.66	0	0	0	0	0	0	0	0
1202579	3	0.66	0.667	0.012	1.01	3.46	0.007	0	0	0	0
1202583	4	0.66	0.675	0.025	3.79	7.46	0.025	0	0	0	0
1202584	9	0.66	0.679	0.011	2.86	3.11	0.019	0	0	0	0
1202585	8	0.66	0.7	0.03	6.82	8.64	0.045	0	0	0	0
1202586	1	0.66	0.66	.	0	.	0	0	0	0	0
1202589	8	0.66	0.698	0.031	6.44	8.9	0.042	0	0	0	0
1202590	10	0.66	0.718	0.048	8.79	13.51	0.058	4	40	0	0
1202591	4	0.66	0.695	0.025	5.3	7.24	0.035	0	0	0	0
1202595	2	0.66	0.7	0.028	6.06	8.08	0.04	0	0	0	0
1202596	1	0.66	0.66	.	0	.	0	0	0	0	0
1203631	6	0.66	0.653	0.02	2.53	6.02	0.017	0	0	0	0

Variability Gauge

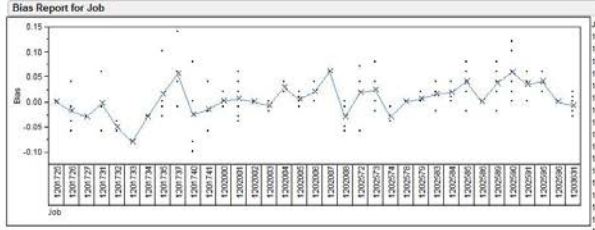
G398-4



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1201725	0.00000
1201726	-0.01750
1201727	-0.03000
1201731	-0.00333
1201732	-0.05000
1201733	-0.08000
1201734	-0.03000
1201735	0.01500
1201737	0.05500
1201740	-0.02500
1201741	-0.01500
1202000	0.00143
1202001	0.00667
1202002	0.00000
1202003	-0.00750
1202004	0.03000
1202005	0.00500
1202006	0.02000
1202007	0.06000
1202008	-0.03000
1202572	0.01800
1202573	0.02500
1202574	-0.03000
1202578	0.00000
1202579	0.00667
1202583	0.01500
1202584	0.01889
1202585	0.04000
1202586	0.00000
1202589	0.03750
1202590	0.05800
1202591	0.03500
1202595	0.04000
1202596	0.00000
1203631	-0.00667

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G398-6	2.94	0.16

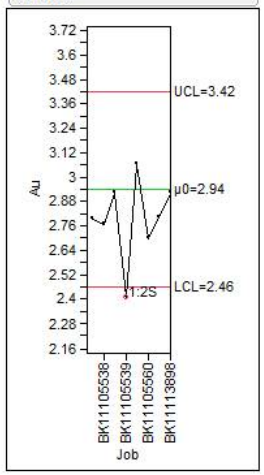
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G398-6	8	2.94	2.796	0.192	5.91	13.77	0.174	1	12.5	1	12.5

Levey Jennings Control Chart

G398-6



Notes

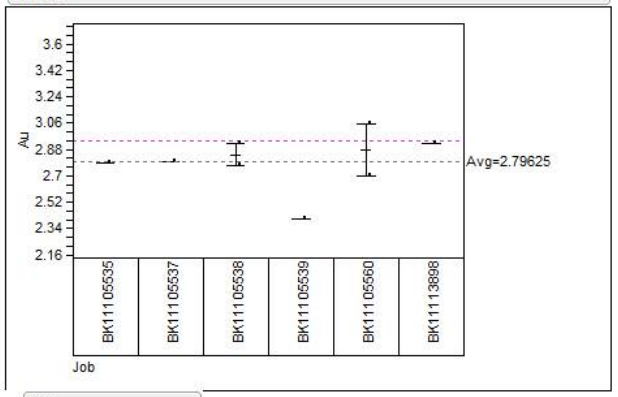
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105535	1	2.94	2.79	.	5.1	.	0.15	0	0	0	0
BK11105537	1	2.94	2.8	.	4.76	.	0.14	0	0	0	0
BK11105538	2	2.94	2.845	0.106	3.23	7.46	0.095	0	0	0	0
BK11105539	1	2.94	2.41	.	18.03	.	0.53	1	100	1	100
BK11105560	2	2.94	2.88	0.255	6.12	17.68	0.18	0	0	0	0
BK11113898	1	2.94	2.92	.	0.68	.	0.02	0	0	0	0

Variability Gauge

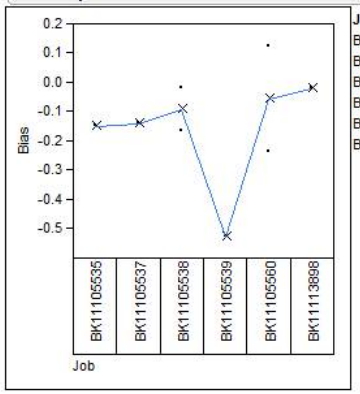
G398-6



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105535	-0.15000
BK11105537	-0.14000
BK11105538	-0.09500
BK11105539	-0.53000
BK11105560	-0.06000
BK11113898	-0.02000

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 --- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G901-1	2.58	0.13

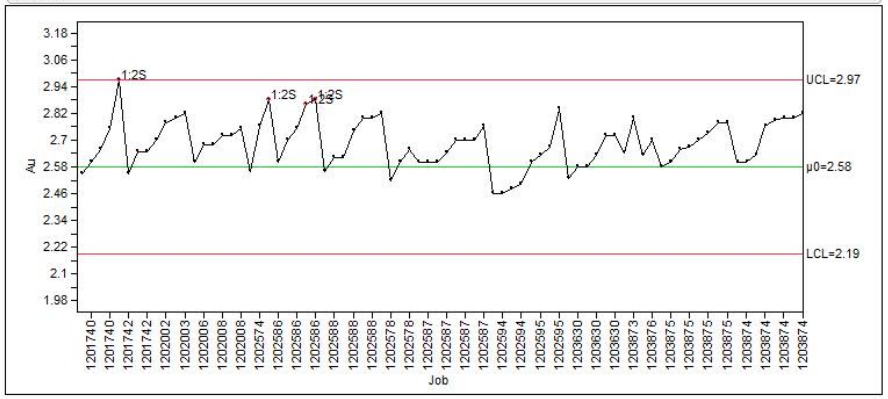
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G901-1	78	2.58	2.681	0.106	4.55	7.89	0.117	5	6.4	1	1.3

Levey Jennings Control Chart

G901-1



Notes

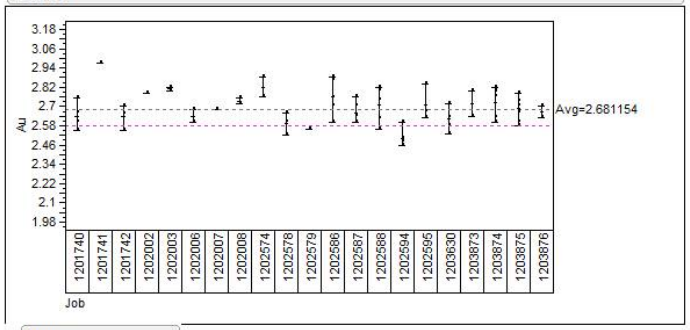
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201740	4	2.58	2.64	0.086	2.91	6.52	0.075	0	0	0	0
1201741	1	2.58	2.97	.	15.12	.	0.39	1	100	1	100
1201742	4	2.58	2.638	0.063	2.81	4.77	0.073	0	0	0	0
1202002	1	2.58	2.78	.	7.75	.	0.2	0	0	0	0
1202003	2	2.58	2.81	0.014	8.91	1.01	0.23	0	0	0	0
1202006	2	2.58	2.64	0.057	2.33	4.29	0.06	0	0	0	0
1202007	1	2.58	2.68	.	3.88	.	0.1	0	0	0	0
1202008	3	2.58	2.73	0.017	5.81	1.27	0.15	0	0	0	0
1202574	2	2.58	2.82	0.085	9.3	6.02	0.24	1	50	0	0
1202578	3	2.58	2.593	0.07	2.07	5.42	0.053	0	0	0	0
1202579	1	2.58	2.56	.	0.78	.	0.02	0	0	0	0
1202586	5	2.58	2.758	0.116	6.9	8.4	0.178	2	40	0	0
1202587	8	2.58	2.663	0.061	3.2	4.57	0.083	0	0	0	0
1202588	7	2.58	2.709	0.106	5.2	7.85	0.134	0	0	0	0
1202594	5	2.58	2.5	0.058	3.41	4.66	0.088	0	0	0	0
1202595	3	2.58	2.713	0.112	5.17	8.22	0.133	1	33.3	0	0
1203630	6	2.58	2.627	0.079	2.45	6.01	0.063	0	0	0	0
1203873	2	2.58	2.72	0.113	5.43	8.32	0.14	0	0	0	0
1203874	8	2.58	2.725	0.097	5.62	7.13	0.145	0	0	0	0
1203875	8	2.58	2.688	0.075	4.17	5.58	0.107	0	0	0	0
1203876	2	2.58	2.665	0.049	3.29	3.71	0.085	0	0	0	0

Variability Gauge

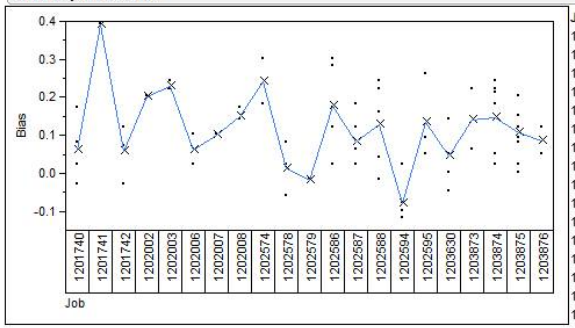
G901-1



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1201740	0.06000
1201741	0.39000
1201742	0.05750
1202002	0.20000
1202003	0.23000
1202006	0.06000
1202007	0.10000
1202008	0.15000
1202574	0.24000
1202578	0.01333
1202579	-0.02000
1202586	0.17800
1202587	0.08250
1202588	0.12857
1202594	-0.08000
1202595	0.13333
1203630	0.04667
1203873	0.14000
1203874	0.14500
1203875	0.10750
1203876	0.08500

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G901-9	0.69	0.04

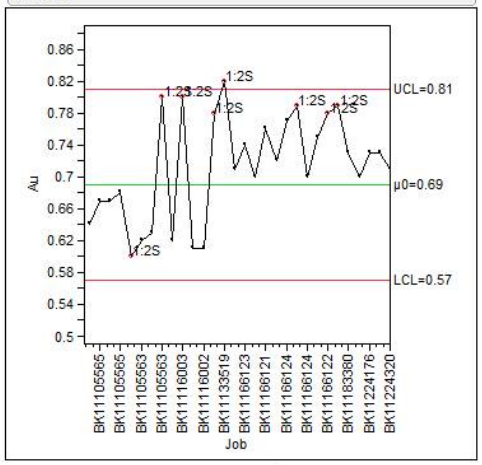
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G901-9	30	0.69	0.712	0.066	8.5	18.42	0.059	11	36.7	1	3.3

Levey Jennings Control Chart

G901-9



Notes

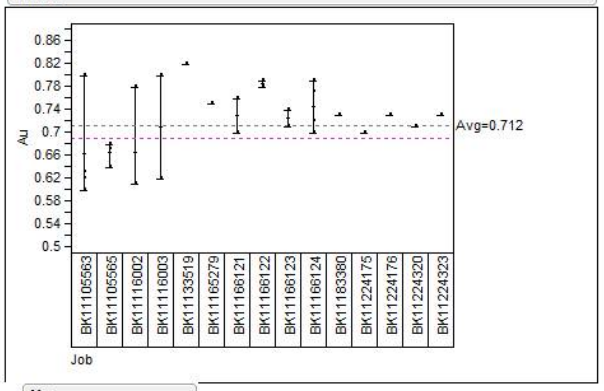
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105563	4	0.69	0.663	0.093	11.96	27.93	0.083	2	50	0	0
BK11105565	4	0.69	0.665	0.017	3.62	5.21	0.025	0	0	0	0
BK11116002	3	0.69	0.667	0.098	12.08	29.44	0.083	3	100	0	0
BK11116003	2	0.69	0.71	0.127	13.04	35.85	0.09	1	50	0	0
BK11133519	1	0.69	0.82	.	18.84	.	0.13	1	100	1	100
BK11165279	1	0.69	0.75	.	8.7	.	0.06	0	0	0	0
BK11166121	2	0.69	0.73	0.042	5.8	11.62	0.04	0	0	0	0
BK11166122	2	0.69	0.785	0.007	13.77	1.8	0.095	2	100	0	0
BK11166123	2	0.69	0.725	0.021	5.07	5.85	0.035	0	0	0	0
BK11166124	4	0.69	0.745	0.042	7.97	11.28	0.055	2	50	0	0
BK11183380	1	0.69	0.73	.	5.8	.	0.04	0	0	0	0
BK11224175	1	0.69	0.7	.	1.45	.	0.01	0	0	0	0
BK11224176	1	0.69	0.73	.	5.8	.	0.04	0	0	0	0
BK11224320	1	0.69	0.71	.	2.9	.	0.02	0	0	0	0
BK11224323	1	0.69	0.73	.	5.8	.	0.04	0	0	0	0

Variability Gauge

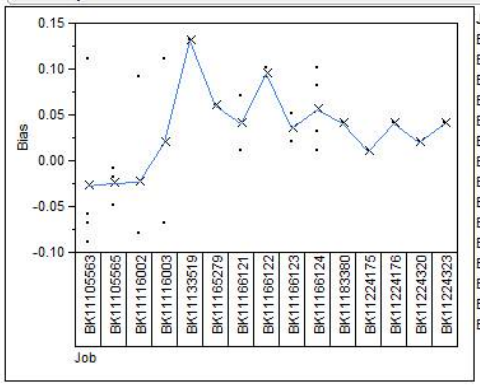
G901-9



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105563	-0.02750
BK11105565	-0.02500
BK11116002	-0.02333
BK11116003	0.02000
BK11133519	0.13000
BK11165279	0.06000
BK11166121	0.04000
BK11166122	0.09500
BK11166123	0.03500
BK11166124	0.05500
BK11183380	0.04000
BK11224175	0.01000
BK11224176	0.04000
BK11224320	0.02000
BK11224323	0.04000

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G902-1	0.39	0.04

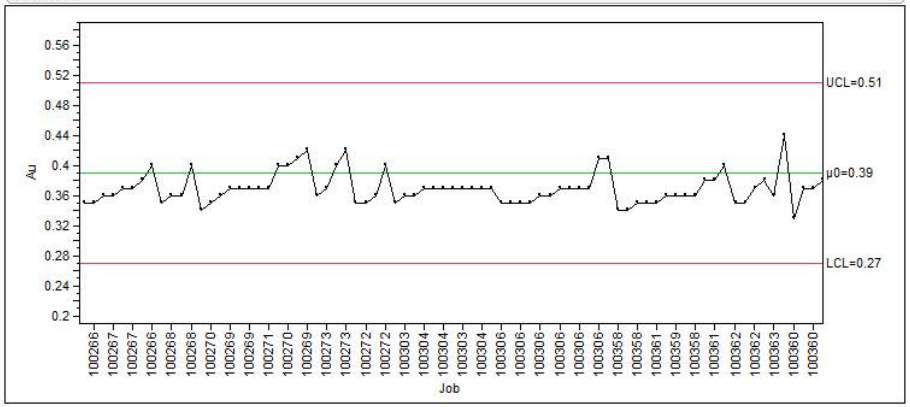
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G902-1	77	0.39	0.369	0.021	6.93	11.42	0.027	0	0	0	0

Levey Jennings Control Chart

G902-1



Notes

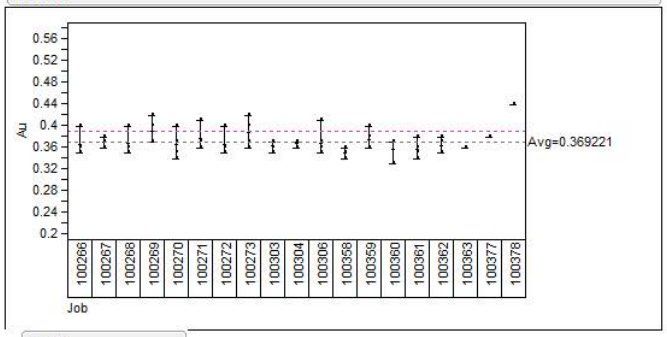
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100266	4	0.39	0.365	0.024	7.69	13.04	0.03	0	0	0	0
100267	4	0.39	0.37	0.008	5.13	4.41	0.02	0	0	0	0
100268	4	0.39	0.368	0.022	7.05	12.07	0.028	0	0	0	0
100269	4	0.39	0.39	0.024	5.13	12.56	0.02	0	0	0	0
100270	4	0.39	0.365	0.026	7.69	14.5	0.03	0	0	0	0
100271	4	0.39	0.378	0.022	5.77	11.75	0.023	0	0	0	0
100272	4	0.39	0.365	0.024	7.69	13.04	0.03	0	0	0	0
100273	4	0.39	0.388	0.028	5.77	14.21	0.023	0	0	0	0
100303	4	0.39	0.363	0.01	7.05	5.28	0.028	0	0	0	0
100304	7	0.39	0.369	0.004	5.49	2.05	0.021	0	0	0	0
100306	12	0.39	0.368	0.021	7.26	11.54	0.028	0	0	0	0
100358	4	0.39	0.35	0.008	10.26	4.67	0.04	0	0	0	0
100359	4	0.39	0.375	0.019	5.13	10.21	0.02	0	0	0	0
100360	3	0.39	0.357	0.023	8.55	12.95	0.033	0	0	0	0
100361	4	0.39	0.358	0.017	8.33	9.55	0.033	0	0	0	0
100362	4	0.39	0.363	0.015	7.05	8.28	0.028	0	0	0	0
100363	1	0.39	0.36	.	7.69	.	0.03	0	0	0	0
100377	1	0.39	0.38	.	2.56	.	0.01	0	0	0	0
100378	1	0.39	0.44	.	12.82	.	0.05	0	0	0	0

Variability Gauge

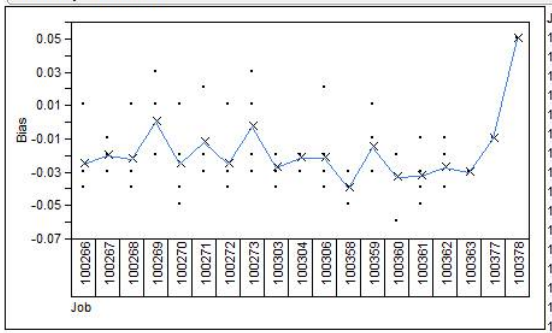
G902-1



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
100266	-0.02500
100267	-0.02000
100268	-0.02250
100269	-1.4e-17
100270	-0.02500
100271	-0.01250
100272	-0.02500
100273	-0.00250
100303	-0.02750
100304	-0.02143
100306	-0.02167
100358	-0.04000
100359	-0.01500
100360	-0.03333
100361	-0.03250
100362	-0.02750
100363	-0.03000
100377	-0.01000
100378	0.05000

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G904-8	5.53	0.18

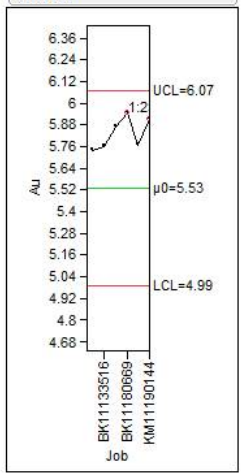
Manufactured By

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G904-8	6	5.53	5.833	0.088	5.49	3.03	0.303	2	33.3	0	0

Levey Jennings Control Chart

G904-8



Notes

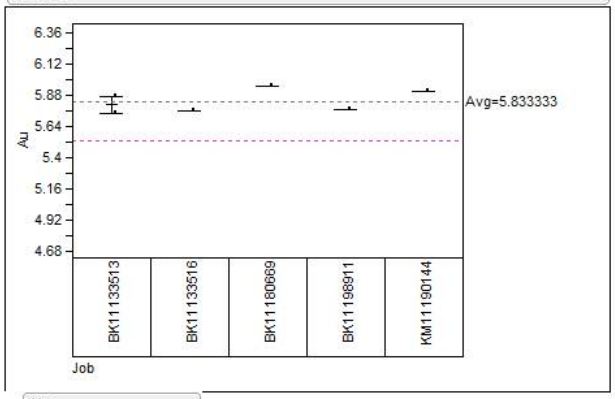
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11133513	2	5.53	5.805	0.092	4.97	3.17	0.275	0	0	0	0
BK11133516	1	5.53	5.76	.	4.16	.	0.23	0	0	0	0
BK11180669	1	5.53	5.95	.	7.59	.	0.42	1	100	0	0
BK11198911	1	5.53	5.77	.	4.34	.	0.24	0	0	0	0
KM11190144	1	5.53	5.91	.	6.87	.	0.38	1	100	0	0

Variability Gauge

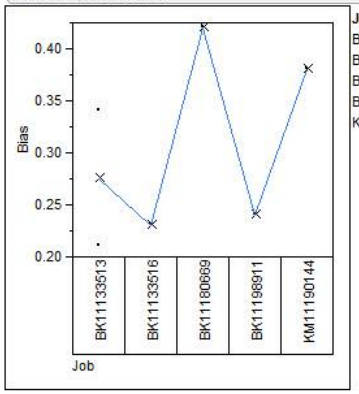
G904-8



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11133513	0.275000
BK11133516	0.230000
BK11180669	0.420000
BK11198911	0.240000
KM11190144	0.380000

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 ---- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G905-10	6.75	0.25

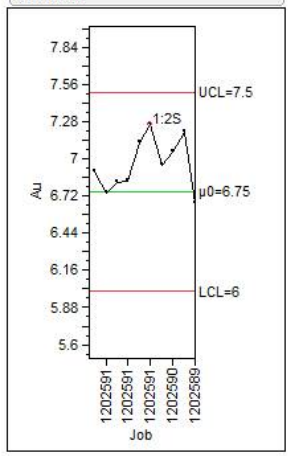
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G905-10	10	6.75	6.953	0.2	3.3	5.75	0.223	1	10	0	0

Levey Jennings Control Chart

G905-10

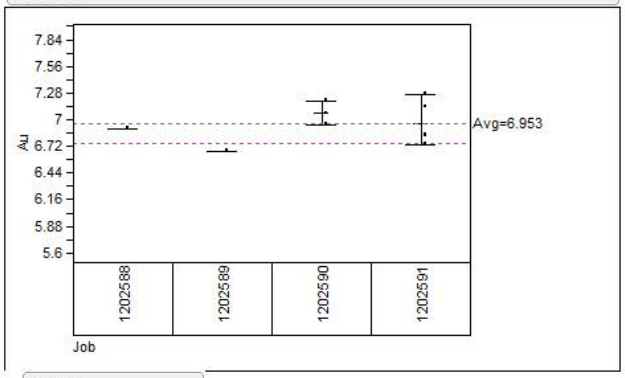


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

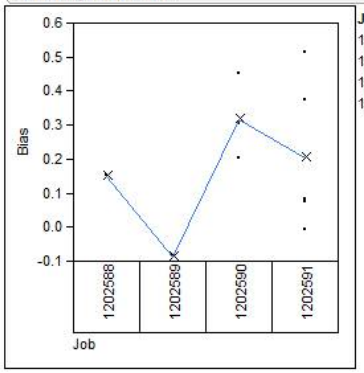
G905-10



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1202588	0.15000
1202589	-0.09000
1202590	0.31667
1202591	0.20400

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1202588	1	6.75	6.9	.	2.22	.	0.15	0	0	0	0
1202589	1	6.75	6.66	.	1.33	.	0.09	0	0	0	0
1202590	3	6.75	7.067	0.126	4.69	3.56	0.317	0	0	0	0
1202591	5	6.75	6.954	0.224	3.08	6.44	0.208	1	20	0	0

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G906-8	7.24	0.27

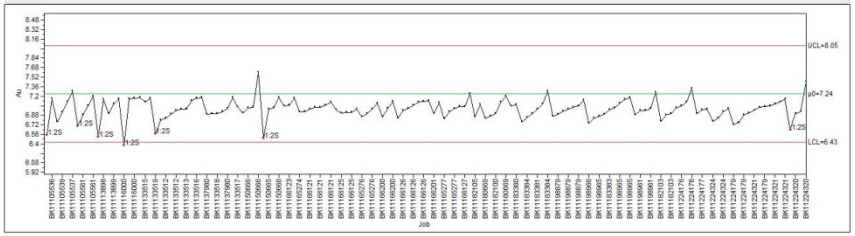
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G906-8	148	7.24	6.981	0.167	3.71	4.78	0.269	7	4.7	1	0.7

Levey Jennings Control Chart

G906-8



Notes

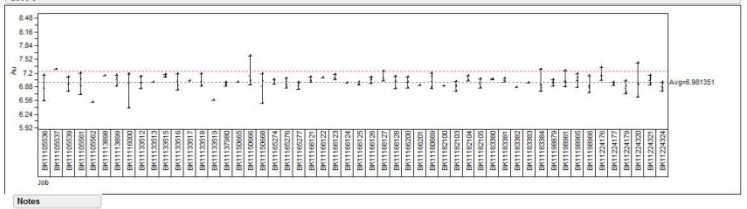
Red Line - 3 Std Dev
Weetgard Rule 1 2a - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105536	2	7.24	6.85	0.424	5.39	12.39	0.39	1	50	0	0
BK11105537	1	7.24	7.28	.	0.55	.	0.04	0	0	0	0
BK11105539	3	7.24	6.933	0.165	4.24	4.76	0.307	0	0	0	0
BK11105561	4	7.24	6.955	0.214	3.94	6.17	0.285	1	25	0	0
BK11105562	1	7.24	6.52	.	9.94	.	0.72	1	100	0	0
BK11113898	1	7.24	7.14	.	1.38	.	0.1	0	0	0	0
BK11113899	3	7.24	7.04	0.128	2.76	3.63	0.2	0	0	0	0
BK11116000	4	7.24	6.968	0.392	3.76	11.25	0.273	1	25	1	25
BK11133512	3	7.24	6.97	0.145	3.73	4.17	0.27	0	0	0	0
BK11133513	2	7.24	6.98	0	3.59	0	0.26	0	0	0	0
BK11133515	2	7.24	7.13	0.042	1.52	1.19	0.11	0	0	0	0
BK11133516	4	7.24	7.008	0.191	3.21	5.46	0.233	0	0	0	0
BK11133517	1	7.24	7.01	.	3.18	.	0.23	0	0	0	0
BK11133518	3	7.24	6.997	0.159	3.36	4.54	0.243	0	0	0	0
BK11133519	1	7.24	6.57	.	9.25	.	0.67	1	100	0	0
BK11137980	3	7.24	6.94	0.05	4.14	1.44	0.3	0	0	0	0
BK11150665	1	7.24	6.98	.	3.59	.	0.26	0	0	0	0
BK11150666	4	7.24	7.133	0.314	3.97	8.81	0.288	0	0	0	0
BK11150668	3	7.24	6.89	0.358	4.83	10.39	0.35	1	33.3	0	0
BK11165274	4	7.24	6.985	0.054	3.52	1.56	0.255	0	0	0	0
BK11165276	4	7.24	6.953	0.1	3.97	2.89	0.288	0	0	0	0
BK11165277	3	7.24	6.917	0.087	4.47	2.53	0.323	0	0	0	0
BK11166121	3	7.24	7.03	0.062	2.9	1.76	0.21	0	0	0	0
BK11166122	1	7.24	7.09	.	2.07	.	0.15	0	0	0	0
BK11166123	3	7.24	7.083	0.067	2.16	1.88	0.157	0	0	0	0
BK11166124	1	7.24	6.97	.	3.73	.	0.27	0	0	0	0
BK11166125	4	7.24	6.94	0.027	4.14	0.78	0.3	0	0	0	0
BK11166126	3	7.24	7.04	0.075	2.76	2.14	0.2	0	0	0	0
BK11166127	3	7.24	7.093	0.127	2.03	3.58	0.147	0	0	0	0
BK11166128	4	7.24	7.01	0.133	3.18	3.79	0.23	0	0	0	0
BK11166200	3	7.24	6.983	0.13	3.55	3.73	0.257	0	0	0	0
BK11166201	1	7.24	6.91	.	4.56	.	0.33	0	0	0	0
BK11180669	4	7.24	6.998	0.181	3.35	5.18	0.243	0	0	0	0
BK11182100	1	7.24	6.9	.	4.7	.	0.34	0	0	0	0
BK11182103	4	7.24	6.89	0.09	4.83	2.62	0.35	0	0	0	0
BK11182104	2	7.24	7.07	0.085	2.35	2.4	0.17	0	0	0	0
BK11182105	2	7.24	6.955	0.148	3.94	4.27	0.285	0	0	0	0
BK11183380	2	7.24	7.05	0.014	2.62	0.4	0.19	0	0	0	0
BK11183381	2	7.24	7.025	0.064	2.97	1.81	0.215	0	0	0	0
BK11183382	1	7.24	6.86	.	5.25	.	0.38	0	0	0	0
BK11183383	1	7.24	6.97	.	3.73	.	0.27	0	0	0	0
BK11183384	4	7.24	6.948	0.228	4.32	6.56	0.313	0	0	0	0
BK11198879	4	7.24	6.965	0.062	3.8	1.79	0.275	0	0	0	0
BK11198961	5	7.24	7.01	0.146	3.29	4.15	0.238	0	0	0	0
BK11198965	4	7.24	7.03	0.135	2.9	3.85	0.21	0	0	0	0
BK11198966	4	7.24	6.9	0.171	4.7	4.97	0.34	0	0	0	0
BK11224176	3	7.24	7.153	0.157	2.03	4.39	0.147	0	0	0	0
BK11224177	3	7.24	6.953	0.038	3.96	1.09	0.287	0	0	0	0
BK11224179	4	7.24	6.843	0.137	5.49	4.01	0.398	0	0	0	0
BK11224320	4	7.24	6.98	0.329	4.9	9.43	0.355	1	25	0	0
BK11224321	7	7.24	7.036	0.08	2.82	2.27	0.204	0	0	0	0
BK11224324	4	7.24	6.885	0.097	4.9	2.81	0.355	0	0	0	0

Variability Gauge

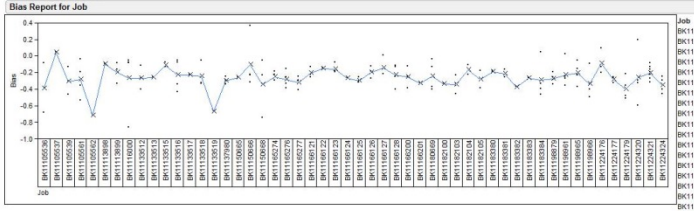
G906-8



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105536	-0.39000
BK11105537	0.04000
BK11105539	-0.30667
BK11105561	-0.28500
BK11105562	-0.72000
BK11113898	-0.10000
BK11113899	-0.20000
BK11116000	-0.27250
BK11133512	-0.27000
BK11133513	-0.26000
BK11133515	-0.11000
BK11133516	-0.23250
BK11133517	-0.23000
BK11133518	-0.24333
BK11133519	-0.67000
BK11137980	-0.30000
BK11150665	-0.26000
BK11150666	-0.10750
BK11150668	-0.35000
BK11165274	-0.25500
BK11165276	-0.26750
BK11165277	-0.32333
BK11166121	-0.21000
BK11166122	-0.15000
BK11166123	-0.15667
BK11166124	-0.27000
BK11166125	-0.30000
BK11166126	-0.20000
BK11166127	-0.14667
BK11166128	-0.23000
BK11166200	-0.25667
BK11166201	-0.33000
BK11180669	-0.24250
BK11182100	-0.34000
BK11182103	-0.35000
BK11182104	-0.17000
BK11182105	-0.28500
BK11183380	-0.19000
BK11183381	-0.21500
BK11183382	-0.38000
BK11183383	-0.27000
BK11183384	-0.29250
BK11198879	-0.27500
BK11198961	-0.23000
BK11198965	-0.21000
BK11198966	-0.34000
BK11224176	-0.08667
BK11224177	-0.28667
BK11224179	-0.39750
BK11224320	-0.26000
BK11224321	-0.20429
BK11224324	-0.35500

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 — Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G906-8	7.24	0.27

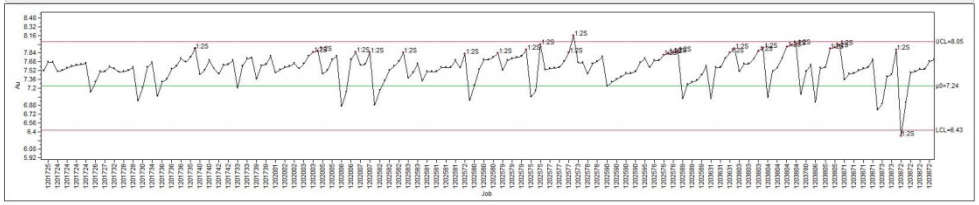
Manufactured By Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G906-8	189	7.24	7.551	0.265	5.08	7.03	0.368	32	16.9	2	1.1

Levey Jennings Control Chart

G906-8



Notes

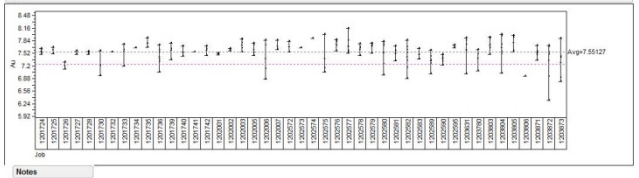
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201724	7	7.24	7.583	0.057	4.74	1.51	0.343	0	0	0	0
1201725	3	7.24	7.617	0.092	5.2	2.43	0.377	0	0	0	0
1201726	2	7.24	7.21	0.127	1.24	3.53	0.09	0	0	0	0
1201727	3	7.24	7.53	0.052	4.01	1.38	0.29	0	0	0	0
1201728	4	7.24	7.52	0.036	3.87	0.95	0.28	0	0	0	0
1201730	3	7.24	7.247	0.313	3.04	8.63	0.22	0	0	0	0
1201732	1	7.24	7.55	.	4.28	.	0.31	0	0	0	0
1201733	4	7.24	7.573	0.258	4.87	6.8	0.352	0	0	0	0
1201734	1	7.24	7.66	.	5.8	.	0.42	0	0	0	0
1201735	3	7.24	7.783	0.127	7.5	3.25	0.543	1	33.3	0	0
1201736	6	7.24	7.432	0.246	3.52	6.63	0.255	0	0	0	0
1201739	4	7.24	7.595	0.174	4.9	4.58	0.355	1	25	0	0
1201740	3	7.24	7.557	0.129	4.37	3.41	0.317	0	0	0	0
1201741	1	7.24	7.55	.	4.28	.	0.31	0	0	0	0
1201742	4	7.24	7.603	0.101	5.01	2.67	0.362	0	0	0	0
1202001	2	7.24	7.505	0.035	3.66	0.94	0.265	0	0	0	0
1202002	3	7.24	7.61	0.036	5.11	0.95	0.37	0	0	0	0
1202003	5	7.24	7.742	0.143	6.93	3.69	0.502	3	60	0	0
1202005	4	7.24	7.62	0.154	5.25	4.04	0.38	1	25	0	0
1202006	4	7.24	7.39	0.477	5.52	12.92	0.4	1	25	0	0
1202007	3	7.24	7.7	0.13	6.35	3.38	0.46	1	33.3	0	0
1202572	2	7.24	7.69	0.184	6.22	4.79	0.45	1	50	0	0
1202573	2	7.24	7.66	0	5.8	0	0.42	0	0	0	0
1202574	1	7.24	7.9	.	9.12	.	0.66	1	100	0	0
1202575	3	7.24	7.393	0.528	4.88	14.28	0.353	1	33.3	0	0
1202576	7	7.24	7.761	0.195	7.2	2.71	0.521	4	57.1	0	0
1202577	7	7.24	7.7	0.23	6.35	5.97	0.46	2	28.6	1	14.3
1202578	4	7.24	7.638	0.13	5.49	3.41	0.397	0	0	0	0
1202579	5	7.24	7.7	0.195	6.35	2.72	0.46	1	20	0	0
1202580	7	7.24	7.537	0.319	5.17	8.45	0.374	1	14.3	0	0
1202581	8	7.24	7.531	0.105	4.02	2.79	0.291	0	0	0	0
1202582	7	7.24	7.43	0.333	4.4	8.97	0.319	1	14.3	0	0
1202583	3	7.24	7.5	0.131	3.59	3.5	0.26	0	0	0	0
1202589	6	7.24	7.323	0.198	2.26	5.42	0.163	0	0	0	0
1202590	7	7.24	7.39	0.1	2.11	2.7	0.153	0	0	0	0
1202595	2	7.24	7.7	0.057	6.35	1.47	0.46	0	0	0	0
1203631	6	7.24	7.61	0.329	6.22	8.64	0.45	2	33.3	0	0
1203780	3	7.24	7.4	0.284	3.68	7.66	0.267	0	0	0	0
1203803	6	7.24	7.72	0.162	6.63	4.19	0.48	2	33.3	0	0
1203804	7	7.24	7.679	0.354	6.93	9.22	0.501	3	42.9	0	0
1203805	5	7.24	7.796	0.208	7.68	5.32	0.556	3	60	0	0
1203806	1	7.24	6.94	.	4.14	.	0.3	0	0	0	0
1203871	7	7.24	7.52	0.114	3.87	3.04	0.28	0	0	0	0
1203872	8	7.24	7.339	0.477	5.61	13	0.406	1	12.5	1	12.5
1203873	5	7.24	7.288	0.447	4.97	12.26	0.36	1	20	0	0

Variability Gauge

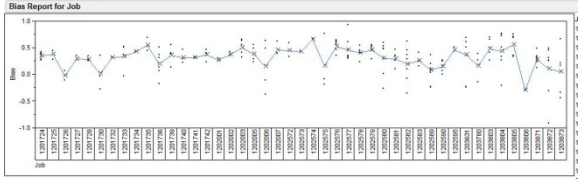
G906-8



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1201724	0.34286
1201725	0.37667
1201726	-0.03000
1201727	0.29000
1201728	0.28000
1201730	0.00867
1201732	0.31000
1201733	0.33250
1201734	0.42000
1201735	0.54333
1201736	0.19167
1201739	0.35500
1201740	0.31667
1201741	0.31000
1201742	0.36250
1202001	0.26500
1202002	0.37000
1202003	0.50200
1202005	0.38000
1202006	0.15000
1202007	0.46000
1202572	0.45000
1202573	0.42000
1202574	0.66000
1202575	0.15333
1202576	0.52143
1202577	0.46000
1202578	0.39750
1202579	0.46000
1202580	0.29714
1202581	0.29125
1202582	0.19000
1202583	0.26000
1202589	0.08333
1202590	0.15000
1202595	0.46000
1203631	0.37000
1203780	0.16000
1203803	0.48000
1203804	0.43857
1203805	0.55600
1203806	-0.30000
1203871	0.28000
1203872	0.09875
1203873	0.04800

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G907-2	0.89	0.06

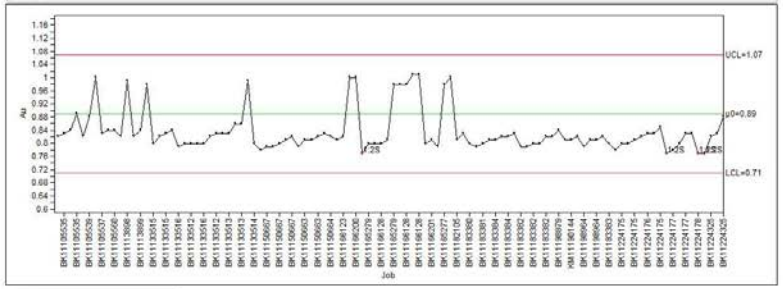
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G907-2	106	0.89	0.836	0.063	6.09	15	0.079	6	5.7	0	0

Levey Jennings Control Chart

G907-2



Notes

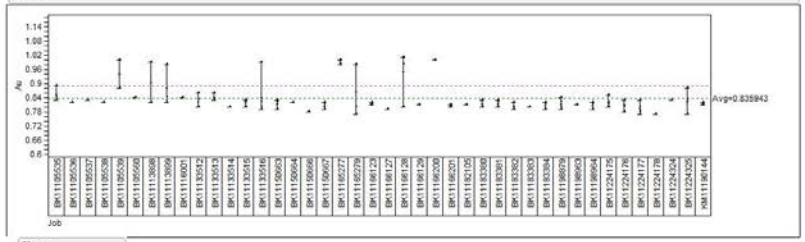
Red Line - 3 Std Dev
Westgard Rule 1.2k - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105535	3	0.89	0.853	0.032	4.12	7.53	0.037	0	0	0	0
BK11105536	1	0.89	0.82	.	7.87	.	0.07	0	0	0	0
BK11105537	1	0.89	0.83	.	6.74	.	0.06	0	0	0	0
BK11105538	1	0.89	0.82	.	7.87	.	0.07	0	0	0	0
BK11105539	2	0.89	0.94	0.085	6.74	18.05	0.06	0	0	0	0
BK11105560	2	0.89	0.84	0	5.62	0	0.05	0	0	0	0
BK11113090	2	0.89	0.905	0.12	9.55	26.57	0.085	0	0	0	0
BK11113899	3	0.89	0.88	0.087	7.87	19.81	0.07	0	0	0	0
BK11116001	1	0.89	0.84	.	5.62	.	0.05	0	0	0	0
BK11133512	4	0.89	0.823	0.029	7.58	6.98	0.068	0	0	0	0
BK11133513	3	0.89	0.84	0.017	5.62	4.13	0.05	0	0	0	0
BK11133514	1	0.89	0.8	.	10.11	.	0.09	0	0	0	0
BK11133515	3	0.89	0.817	0.015	8.24	3.74	0.073	0	0	0	0
BK11133516	5	0.89	0.84	0.085	10.11	20.13	0.09	0	0	0	0
BK11150663	5	0.89	0.812	0.015	8.76	3.65	0.078	0	0	0	0
BK11150664	1	0.89	0.82	.	7.87	.	0.07	0	0	0	0
BK11150666	1	0.89	0.78	.	12.36	.	0.11	0	0	0	0
BK11150667	5	0.89	0.802	0.013	9.89	3.25	0.088	0	0	0	0
BK11165277	2	0.89	0.99	0.014	11.24	2.86	0.1	0	0	0	0
BK11165279	5	0.89	0.866	0.105	10.79	24.2	0.096	1	20	0	0
BK11166123	2	0.89	0.815	0.007	8.43	1.74	0.075	0	0	0	0
BK11166127	1	0.89	0.79	.	11.24	.	0.1	0	0	0	0
BK11166128	4	0.89	0.95	0.101	11.8	21.26	0.105	2	50	0	0
BK11166129	1	0.89	0.81	.	8.99	.	0.08	0	0	0	0
BK11166200	2	0.89	1	0	12.36	0	0.11	0	0	0	0
BK11166201	2	0.89	0.805	0.007	9.55	1.76	0.085	0	0	0	0
BK11182105	1	0.89	0.81	.	8.99	.	0.08	0	0	0	0
BK11183380	2	0.89	0.815	0.021	8.43	5.21	0.075	0	0	0	0
BK11183381	4	0.89	0.815	0.013	8.43	3.17	0.075	0	0	0	0
BK11183382	3	0.89	0.803	0.015	9.74	3.8	0.087	0	0	0	0
BK11183383	1	0.89	0.8	.	10.11	.	0.09	0	0	0	0
BK11183384	3	0.89	0.807	0.015	9.36	3.79	0.083	0	0	0	0
BK11198879	4	0.89	0.813	0.022	8.71	5.46	0.078	0	0	0	0
BK11198963	1	0.89	0.81	.	8.99	.	0.08	0	0	0	0
BK11198964	3	0.89	0.807	0.015	9.36	3.79	0.083	0	0	0	0
BK11224175	4	0.89	0.82	0.022	7.87	5.27	0.07	0	0	0	0
BK11224176	4	0.89	0.81	0.024	8.99	6.05	0.08	0	0	0	0
BK11224177	4	0.89	0.795	0.028	10.67	6.66	0.095	1	25	0	0
BK11224178	1	0.89	0.77	.	13.48	.	0.12	1	100	0	0
BK11224324	1	0.89	0.83	.	6.74	.	0.06	0	0	0	0
BK11224325	4	0.89	0.825	0.045	7.3	10.93	0.065	1	25	0	0
KM11190144	3	0.89	0.813	0.006	8.61	1.42	0.077	0	0	0	0

Variability Gauge

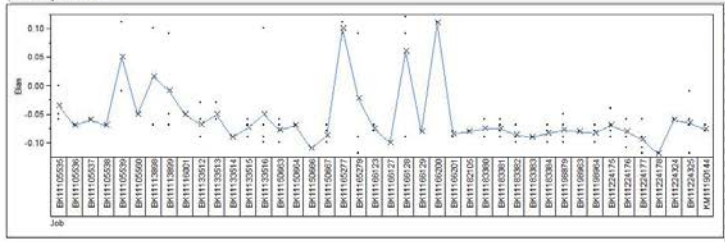
G907-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105535	-0.03667
BK11105536	-0.07000
BK11105537	-0.06000
BK11105538	-0.07000
BK11105539	0.05000
BK11105560	-0.05000
BK11113090	0.01500
BK11113899	-0.01000
BK11116001	-0.05000
BK11133512	-0.06750
BK11133513	-0.05000
BK11133514	-0.09000
BK11133515	-0.07333
BK11133516	-0.05000
BK11150663	-0.07000
BK11150664	-0.07000
BK11150666	-0.11000
BK11150667	-0.08550
BK11165277	0.10000
BK11165279	-0.02400
BK11166123	-0.07500
BK11166127	-0.10000
BK11166128	0.06000
BK11166129	-0.08000
BK11166200	0.11000
BK11166201	-0.08500
BK11182105	-0.08000
BK11183380	-0.07500
BK11183381	-0.07500
BK11183382	-0.08667
BK11183383	-0.09000
BK11183384	-0.08333
BK11198879	-0.07750
BK11198963	-0.08000
BK11198964	-0.08333
BK11224175	-0.07000
BK11224176	-0.08000
BK11224177	-0.09500
BK11224178	-0.12000
BK11224324	-0.06000
BK11224325	-0.06500
KM11190144	-0.07667

QAQC Analysis for Period From Feb 15, 2012 - Jun 19, 2012 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G907-2	0.89	0.06

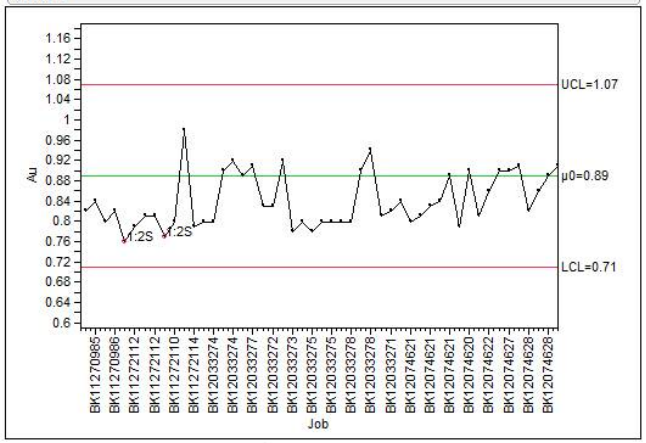
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G907-2	49	0.89	0.84	0.052	6.99	12.47	0.062	2	4.1	0	0

Levey Jennings Control Chart

G907-2



Notes

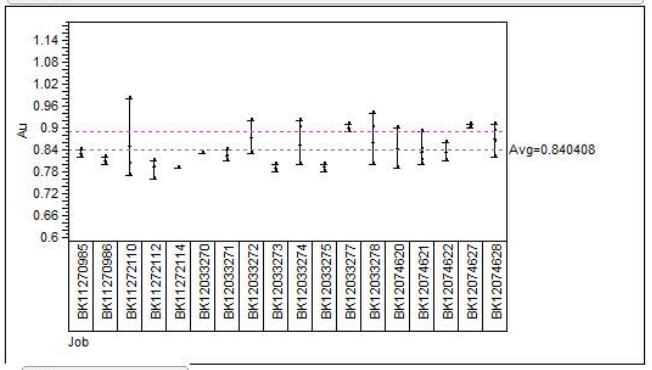
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11270985	2	0.89	0.83	0.014	6.74	3.41	0.06	0	0	0	0
BK11270986	2	0.89	0.81	0.014	8.99	3.49	0.08	0	0	0	0
BK11272110	3	0.89	0.85	0.114	11.24	26.72	0.1	1	33.3	0	0
BK11272112	4	0.89	0.793	0.024	10.96	5.96	0.097	1	25	0	0
BK11272114	1	0.89	0.79	.	11.24	.	0.1	0	0	0	0
BK12033270	1	0.89	0.83	.	6.74	.	0.06	0	0	0	0
BK12033271	3	0.89	0.823	0.015	7.49	3.71	0.067	0	0	0	0
BK12033272	2	0.89	0.875	0.064	5.06	14.55	0.045	0	0	0	0
BK12033273	2	0.89	0.79	0.014	11.24	3.58	0.1	0	0	0	0
BK12033274	4	0.89	0.855	0.064	6.18	14.98	0.055	0	0	0	0
BK12033275	3	0.89	0.793	0.012	10.86	2.91	0.097	0	0	0	0
BK12033277	2	0.89	0.9	0.014	1.12	3.14	0.01	0	0	0	0
BK12033278	4	0.89	0.86	0.071	6.74	16.55	0.06	0	0	0	0
BK12074620	2	0.89	0.845	0.078	6.18	18.41	0.055	0	0	0	0
BK12074621	5	0.89	0.834	0.035	6.29	8.41	0.056	0	0	0	0
BK12074622	2	0.89	0.835	0.035	6.18	8.47	0.055	0	0	0	0
BK12074627	3	0.89	0.903	0.006	1.5	1.28	0.013	0	0	0	0
BK12074628	4	0.89	0.87	0.039	3.37	9	0.03	0	0	0	0

Variability Gauge

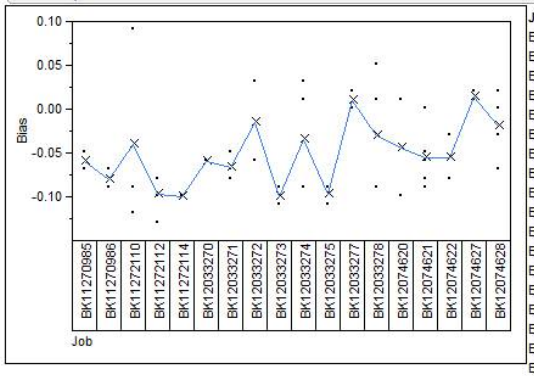
G907-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11270985	-0.06000
BK11270986	-0.08000
BK11272110	-0.04000
BK11272112	-0.09750
BK11272114	-0.10000
BK12033270	-0.06000
BK12033271	-0.06667
BK12033272	-0.01500
BK12033273	-0.10000
BK12033274	-0.03500
BK12033275	-0.09667
BK12033277	0.01000
BK12033278	-0.03000
BK12074620	-0.04500
BK12074621	-0.05600
BK12074622	-0.05500
BK12074627	0.01333
BK12074628	-0.02000

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G907-2	0.89	0.06

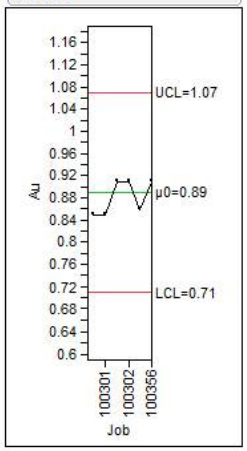
Manufactured By

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G907-2	6	0.89	0.882	0.031	3.18	7.09	0.028	0	0	0	0

Levey Jennings Control Chart

G907-2

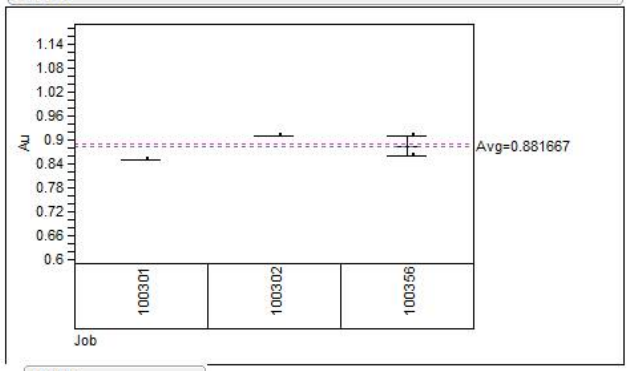


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

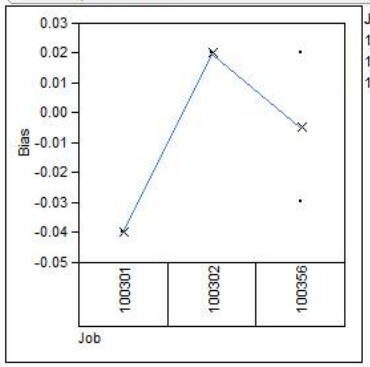
G907-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
100301	-0.04000
100302	0.02000
100356	-0.00500

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100301	2	0.89	0.85	0	4.49	0	0.04	0	0	0	0
100302	2	0.89	0.91	0	2.25	0	0.02	0	0	0	0
100356	2	0.89	0.885	0.035	2.81	7.99	0.025	0	0	0	0

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 --- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G907-2	0.89	0.06

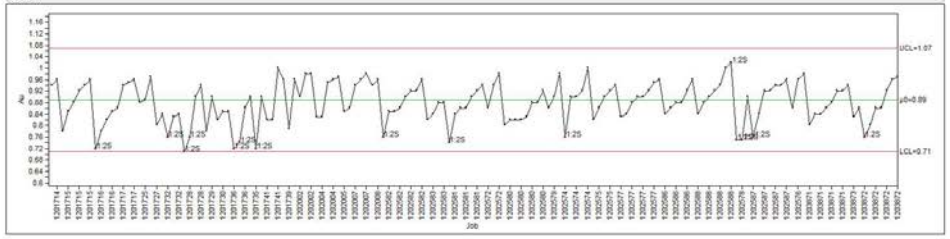
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G907-2	154	0.89	0.88	0.069	6.44	15.73	0.057	15	9.7	1	0.6

Levey Jennings Control Chart

G907-2

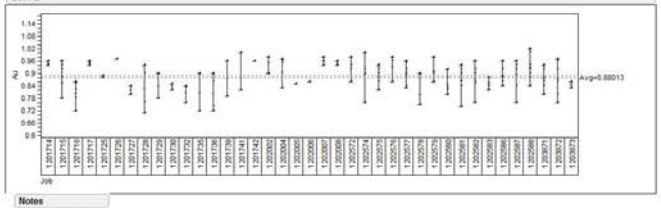


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G907-2



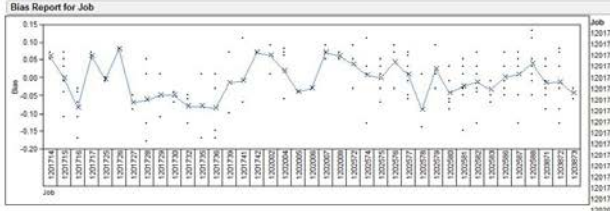
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201714	2	0.89	0.95	0.014	6.74	2.96	0.06	0	0	0	0
1201715	6	0.89	0.888	0.066	5.81	14.96	0.052	0	0	0	0
1201716	5	0.89	0.856	0.057	9.44	14.21	0.084	1	20	0	0
1201717	3	0.89	0.95	0.01	6.74	2.11	0.06	0	0	0	0
1201725	2	0.89	0.885	0.007	0.56	1.6	0.005	0	0	0	0
1201726	1	0.89	0.97	.	8.99	.	0.08	0	0	0	0
1201727	2	0.89	0.82	0.028	7.87	6.9	0.07	0	0	0	0
1201728	4	0.89	0.828	0.11	10.39	26.58	0.093	2	50	1	25
1201729	2	0.89	0.84	0.085	6.74	20.2	0.06	0	0	0	0
1201730	3	0.89	0.84	0.017	5.62	4.12	0.05	0	0	0	0
1201732	3	0.89	0.81	0.044	8.99	10.76	0.08	1	33.3	0	0
1201735	2	0.89	0.81	0.127	10.11	31.43	0.09	1	50	0	0
1201736	4	0.89	0.805	0.089	10.11	21.99	0.09	2	50	0	0
1201739	2	0.89	0.875	0.12	9.55	27.48	0.085	0	0	0	0
1201741	3	0.89	0.88	0.104	9.36	23.62	0.083	0	0	0	0
1201742	1	0.89	0.96	.	7.87	.	0.07	0	0	0	0
1202002	3	0.89	0.953	0.046	7.12	9.69	0.063	0	0	0	0
1202004	5	0.89	0.908	0.072	7.42	15.76	0.066	0	0	0	0
1202005	1	0.89	0.85	.	4.49	.	0.04	0	0	0	0
1202006	1	0.89	0.86	.	3.37	.	0.03	0	0	0	0
1202007	3	0.89	0.96	0.02	7.87	4.17	0.07	0	0	0	0
1202008	2	0.89	0.95	0.014	6.74	2.96	0.06	0	0	0	0
1202572	3	0.89	0.927	0.061	6.37	13.19	0.057	0	0	0	0
1202574	5	0.89	0.896	0.086	6.52	19.31	0.058	1	20	0	0
1202575	5	0.89	0.868	0.048	4.27	10.85	0.038	0	0	0	0
1202576	3	0.89	0.933	0.064	7.12	13.78	0.063	0	0	0	0
1202577	8	0.89	0.898	0.047	4.21	10.44	0.038	0	0	0	0
1202578	3	0.89	0.8	0.097	10.86	21.65	0.097	2	66.7	0	0
1202579	3	0.89	0.913	0.061	4.87	13.38	0.043	0	0	0	0
1202580	9	0.89	0.846	0.042	5.76	9.86	0.051	0	0	0	0
1202581	7	0.89	0.866	0.066	5.62	15.26	0.05	1	14.3	0	0
1202582	8	0.89	0.876	0.062	5.34	14.04	0.048	1	12.5	0	0
1202583	4	0.89	0.855	0.03	3.93	7.02	0.035	0	0	0	0
1202586	6	0.89	0.89	0.043	3.75	9.74	0.033	0	0	0	0
1202587	7	0.89	0.897	0.072	6.56	15.96	0.059	1	14.3	0	0
1202588	7	0.89	0.929	0.064	6.26	13.82	0.056	1	14.3	0	0
1203871	8	0.89	0.875	0.049	4.78	11.13	0.043	0	0	0	0
1203872	7	0.89	0.876	0.079	7.38	18.07	0.066	1	14.3	0	0
1203873	2	0.89	0.845	0.021	5.06	5.02	0.045	0	0	0	0

Bias Report for Job



Job	Avg Bias
1201714	0.06000
1201715	-0.00167
1201716	-0.08400
1201717	0.06000
1201725	-0.00500
1201726	0.08000
1201727	-0.07000
1201728	-0.06250
1201729	-0.05000
1201730	-0.05000
1201732	-0.08000
1201735	-0.09000
1201736	-0.09500
1201739	-0.01500
1201741	-0.01000
1201742	0.07000
1202002	0.06333
1202004	0.01800
1202005	-0.04000
1202006	-0.03000
1202007	0.07000
1202008	0.06000
1202572	0.03667
1202574	0.00600
1202575	-0.00200
1202576	0.04333
1202577	0.00750
1202578	-0.09000
1202579	0.02333
1202580	-0.04375
1202581	-0.02429
1202582	-0.01250
1202583	-0.03500
1202586	-1.9e-17
1202587	0.00714
1202588	0.03857
1203871	-0.01500
1203872	-0.01429
1203873	-0.04500

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G907-4	3.84	0.15

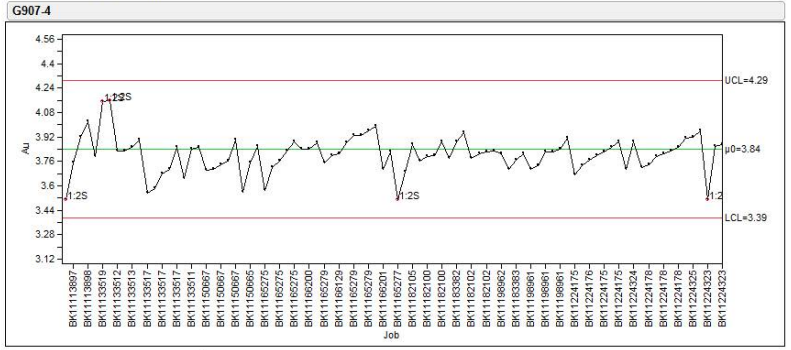
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G907-4	90	3.84	3.803	0.119	2.34	6.25	0.09	5	5.6	0	0

Levey Jennings Control Chart

G907-4

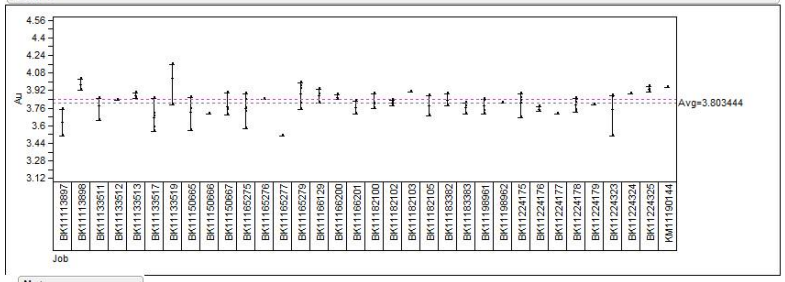


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G907-4



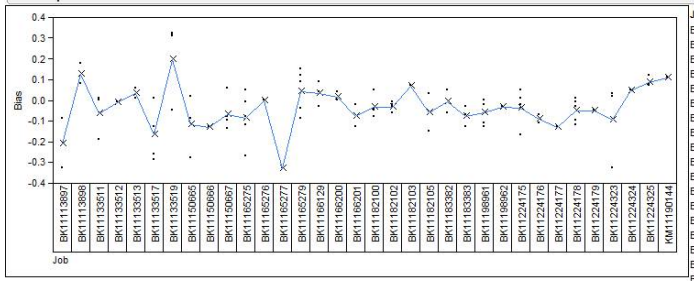
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11113897	2	3.84	3.63	0.17	5.47	9.35	0.21	1	50	0	0
BK11113898	2	3.84	3.97	0.071	3.39	3.56	0.13	0	0	0	0
BK11133511	3	3.84	3.76	0.113	1.74	5.96	0.067	0	0	0	0
BK11133512	2	3.84	3.83	0	0.26	0	0.01	0	0	0	0
BK11133513	2	3.84	3.875	0.035	0.91	1.82	0.035	0	0	0	0
BK11133517	5	3.84	3.674	0.119	4.43	6.47	0.17	0	0	0	0
BK11133519	3	3.84	4.033	0.211	5.9	10.45	0.227	2	66.7	0	0
BK11150665	3	3.84	3.723	0.152	3.39	8.15	0.13	0	0	0	0
BK11150666	1	3.84	3.71	.	3.39	.	0.13	0	0	0	0
BK11150667	4	3.84	3.775	0.087	2.47	4.61	0.095	0	0	0	0
BK11165275	5	3.84	3.754	0.122	2.76	6.49	0.106	0	0	0	0
BK11165276	1	3.84	3.84	.	0	.	0	0	0	0	0
BK11165277	1	3.84	3.51	.	8.59	.	0.33	1	100	0	0
BK11165279	5	3.84	3.886	0.105	2.55	5.41	0.098	0	0	0	0
BK11166129	3	3.84	3.873	0.06	1.39	3.11	0.053	0	0	0	0
BK11166200	2	3.84	3.86	0.028	0.52	1.47	0.02	0	0	0	0
BK11166201	2	3.84	3.765	0.078	1.95	4.13	0.075	0	0	0	0
BK11182100	4	3.84	3.81	0.056	1.43	2.94	0.055	0	0	0	0
BK11182102	4	3.84	3.81	0.022	0.78	1.13	0.03	0	0	0	0
BK11182103	1	3.84	3.91	.	1.82	.	0.07	0	0	0	0
BK11182105	2	3.84	3.78	0.127	2.34	6.73	0.09	0	0	0	0
BK11183382	2	3.84	3.835	0.078	1.43	4.06	0.055	0	0	0	0
BK11183383	3	3.84	3.763	0.05	2	2.67	0.077	0	0	0	0
BK11198961	5	3.84	3.784	0.059	1.46	3.14	0.056	0	0	0	0
BK11198962	1	3.84	3.81	.	0.78	.	0.03	0	0	0	0
BK11224175	5	3.84	3.806	0.083	1.51	4.37	0.058	0	0	0	0
BK11224176	2	3.84	3.75	0.028	2.34	1.51	0.09	0	0	0	0
BK11224177	1	3.84	3.71	.	3.39	.	0.13	0	0	0	0
BK11224178	5	3.84	3.79	0.057	1.41	3.01	0.054	0	0	0	0
BK11224179	1	3.84	3.79	.	1.3	.	0.05	0	0	0	0
BK11224323	3	3.84	3.747	0.205	3.3	10.94	0.127	1	33.3	0	0
BK11224324	1	3.84	3.89	.	1.3	.	0.05	0	0	0	0
BK11224325	3	3.84	3.93	0.026	2.34	1.35	0.09	0	0	0	0
KM11190144	1	3.84	3.95	.	2.86	.	0.11	0	0	0	0

Bias Report for Job



Job	Avg Bias
BK11113897	-0.21000
BK11113898	0.13000
BK11133511	-0.06000
BK11133512	-0.01000
BK11133513	0.03500
BK11133517	-0.16600
BK11133519	0.19333
BK11150665	-0.11667
BK11150666	-0.13000
BK11150667	-0.06500
BK11165275	-0.08600
BK11165276	0.00000
BK11165277	-0.33000
BK11165279	0.04600
BK11166129	0.03333
BK11166200	0.02000
BK11166201	-0.07500
BK11182100	-0.03000
BK11182102	-0.03000
BK11182103	0.07000
BK11182105	-0.06000
BK11183382	-0.00500
BK11183383	-0.07667
BK11198961	-0.05600
BK11198962	-0.03000
BK11224175	-0.03400
BK11224176	-0.09000
BK11224177	-0.13000
BK11224178	-0.05000
BK11224179	-0.05000
BK11224323	-0.09333
BK11224324	0.05000
BK11224325	0.09000
KM11190144	0.11000

QAQC Analysis for Period From Feb 15, 2012 - Jun 19, 2012 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G907-4	3.84	0.15

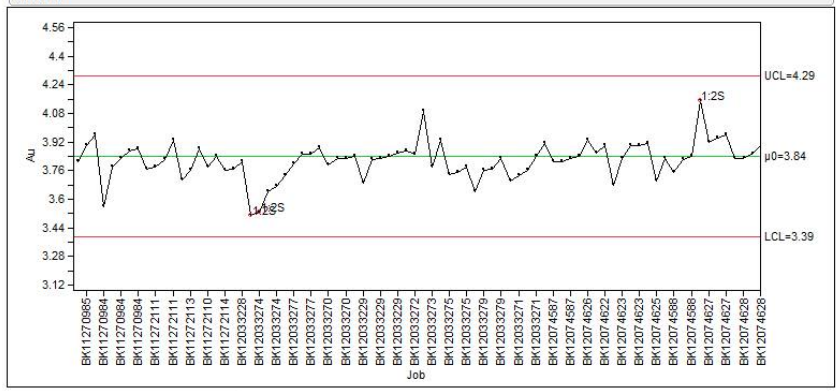
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G907-4	80	3.84	3.816	0.103	1.91	5.38	0.073	3	3.8	0	0

Levey Jennings Control Chart

G907-4

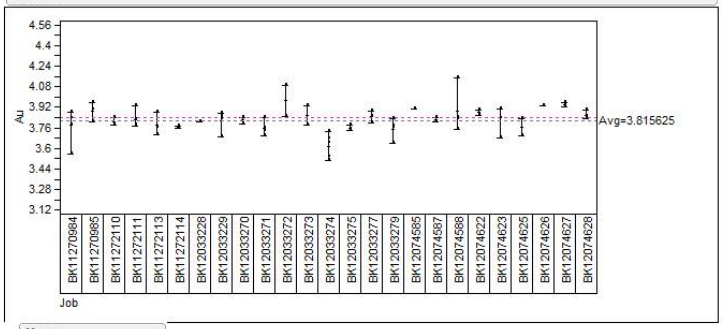


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G907-4



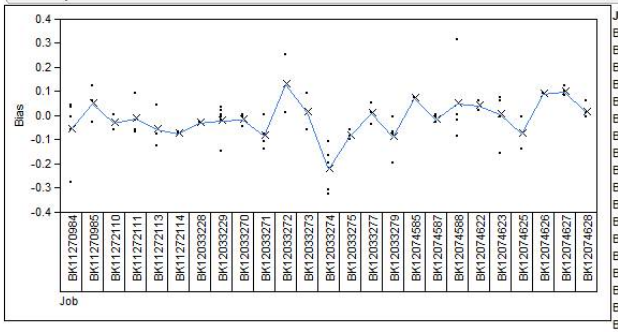
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11270984	5	3.84	3.784	0.131	2.19	6.94	0.084	0	0	0	0
BK11270985	3	3.84	3.89	0.075	1.82	3.88	0.07	0	0	0	0
BK11272110	2	3.84	3.81	0.042	0.78	2.23	0.03	0	0	0	0
BK11272111	4	3.84	3.825	0.073	1.56	3.83	0.06	0	0	0	0
BK11272113	3	3.84	3.783	0.087	2.17	4.62	0.083	0	0	0	0
BK11272114	2	3.84	3.765	0.007	1.95	0.38	0.075	0	0	0	0
BK12033228	1	3.84	3.81	.	0.78	.	0.03	0	0	0	0
BK12033229	6	3.84	3.818	0.066	1	3.43	0.038	0	0	0	0
BK12033270	4	3.84	3.823	0.022	0.46	1.16	0.017	0	0	0	0
BK12033271	4	3.84	3.758	0.06	2.15	3.2	0.082	0	0	0	0
BK12033272	2	3.84	3.97	0.17	3.39	8.55	0.13	0	0	0	0
BK12033273	2	3.84	3.855	0.106	1.95	5.5	0.075	0	0	0	0
BK12033274	5	3.84	3.616	0.094	5.83	5.18	0.224	2	40	0	0
BK12033275	3	3.84	3.757	0.021	2.17	1.11	0.083	0	0	0	0
BK12033277	4	3.84	3.848	0.037	0.72	1.92	0.028	0	0	0	0
BK12033279	4	3.84	3.75	0.08	2.34	4.24	0.09	0	0	0	0
BK12074585	1	3.84	3.91	.	1.82	.	0.07	0	0	0	0
BK12074587	4	3.84	3.823	0.015	0.46	0.78	0.017	0	0	0	0
BK12074588	4	3.84	3.89	0.178	2.73	9.13	0.105	1	25	0	0
BK12074622	2	3.84	3.88	0.028	1.04	1.46	0.04	0	0	0	0
BK12074623	5	3.84	3.844	0.097	1.87	5.05	0.072	0	0	0	0
BK12074625	2	3.84	3.765	0.092	1.95	4.88	0.075	0	0	0	0
BK12074626	1	3.84	3.93	.	2.34	.	0.09	0	0	0	0
BK12074627	3	3.84	3.94	0.02	2.6	1.02	0.1	0	0	0	0
BK12074628	4	3.84	3.853	0.033	0.59	1.72	0.022	0	0	0	0

Bias Report for Job



Job	Avg Bias
BK11270984	-0.05600
BK11270985	0.05000
BK11272110	-0.03000
BK11272111	-0.01500
BK11272113	-0.05667
BK11272114	-0.07500
BK12033228	-0.03000
BK12033229	-0.02167
BK12033270	-0.01750
BK12033271	-0.08250
BK12033272	0.13000
BK12033273	0.01500
BK12033274	-0.22400
BK12033275	-0.08333
BK12033277	0.00750
BK12033279	-0.09000
BK12074585	0.07000
BK12074587	-0.01750
BK12074588	0.05000
BK12074622	0.04000
BK12074623	0.00400
BK12074625	-0.07500
BK12074626	0.09000
BK12074627	0.10000
BK12074628	0.01250

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 --- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G907-4	3.84	0.15

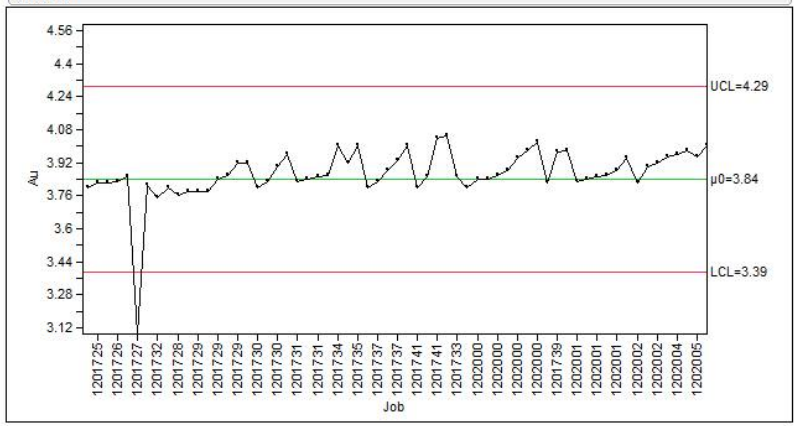
Manufactured By
Geostats

Summary Lab Performance for Period From Jul 10, 2012 - Jan 1, 2013

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G907-4	63	3.84	3.867	0.126	1.96	6.52	0.075	1	1.6	1	1.6

Levey Jennings Control Chart

G907-4

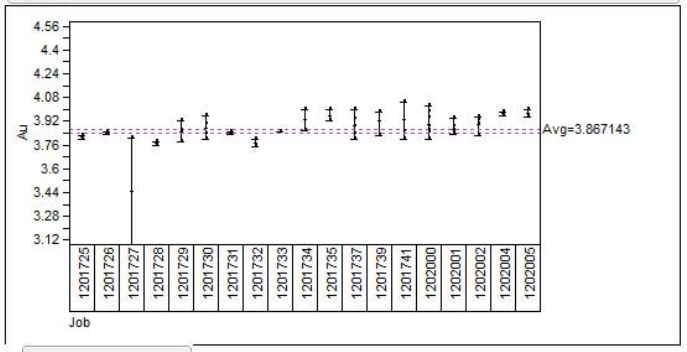


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

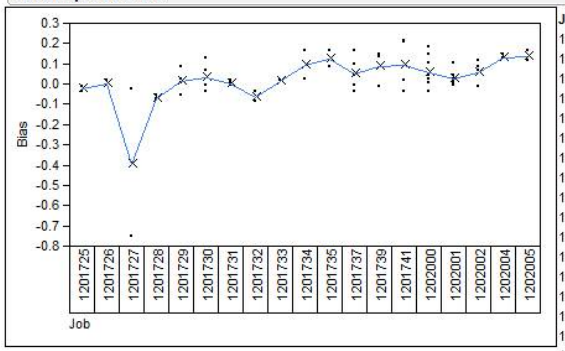
G907-4



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1201725	-0.02667
1201726	2.22e-16
1201727	-0.39500
1201728	-0.07000
1201729	0.01000
1201730	0.03250
1201731	1.48e-16
1201732	-0.06500
1201733	0.01000
1201734	0.09000
1201735	0.12000
1201737	0.04800
1201739	0.08333
1201741	0.09500
1202000	0.05500
1202001	0.02667
1202002	0.05750
1202004	0.13000
1202005	0.13500

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201725	3	3.84	3.813	0.012	0.69	0.61	0.027	0	0	0	0
1201726	2	3.84	3.84	0.014	0.26	0.74	0.01	0	0	0	0
1201727	2	3.84	3.445	0.516	10.29	29.97	0.395	1	50	1	50
1201728	2	3.84	3.77	0.014	1.82	0.75	0.07	0	0	0	0
1201729	6	3.84	3.85	0.063	1.3	3.27	0.05	0	0	0	0
1201730	4	3.84	3.873	0.072	1.5	3.71	0.058	0	0	0	0
1201731	3	3.84	3.84	0.01	0.17	0.52	0.007	0	0	0	0
1201732	2	3.84	3.775	0.035	1.69	1.87	0.065	0	0	0	0
1201733	1	3.84	3.85	.	0.26	.	0.01	0	0	0	0
1201734	2	3.84	3.93	0.099	2.34	5.04	0.09	0	0	0	0
1201735	2	3.84	3.96	0.057	3.13	2.86	0.12	0	0	0	0
1201737	5	3.84	3.888	0.08	1.77	4.11	0.068	0	0	0	0
1201739	3	3.84	3.923	0.09	2.52	4.57	0.097	0	0	0	0
1201741	4	3.84	3.935	0.129	2.99	6.54	0.115	0	0	0	0
1202000	8	3.84	3.895	0.077	1.69	3.95	0.065	0	0	0	0
1202001	6	3.84	3.867	0.04	0.78	2.06	0.03	0	0	0	0
1202002	4	3.84	3.898	0.056	1.76	2.85	0.068	0	0	0	0
1202004	2	3.84	3.97	0.014	3.39	0.71	0.13	0	0	0	0
1202005	2	3.84	3.975	0.035	3.52	1.78	0.135	0	0	0	0

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G909-5	2.63	0.1

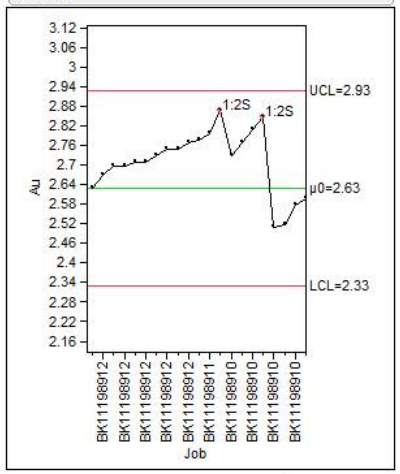
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G909-5	21	2.63	2.711	0.098	4.22	7.24	0.111	2	9.5	0	0

Levey Jennings Control Chart

G909-5



Notes

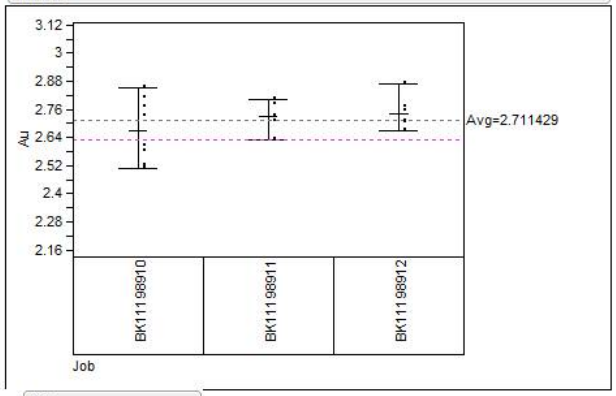
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11198910	8	2.63	2.671	0.135	4.52	10.07	0.119	1	12.5	0	0
BK11198911	5	2.63	2.73	0.067	3.8	4.89	0.1	0	0	0	0
BK11198912	8	2.63	2.74	0.062	4.18	4.53	0.11	1	12.5	0	0

Variability Gauge

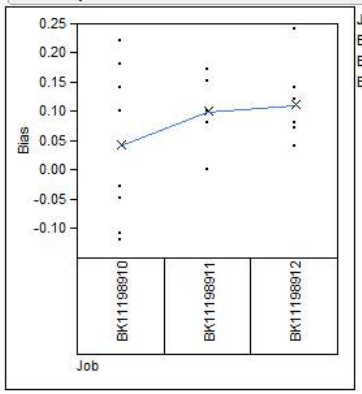
G909-5



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11198910	0.041250
BK11198911	0.100000
BK11198912	0.110000

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 ---- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G909-5	2.63	0.1

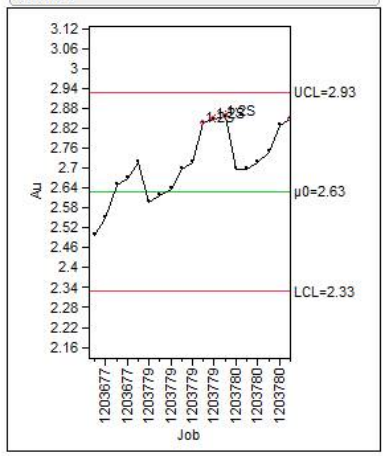
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G909-5	19	2.63	2.709	0.104	4	7.69	0.105	5	26.3	0	0

Levey Jennings Control Chart

G909-5

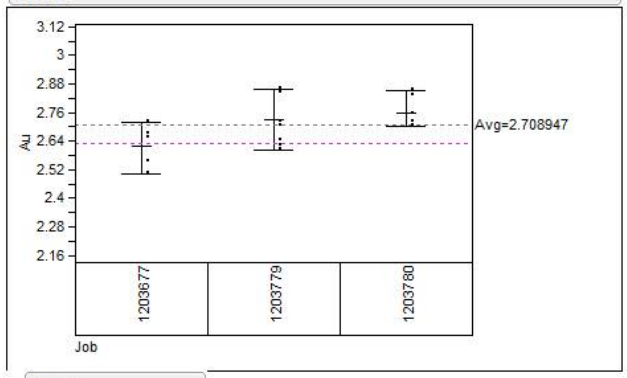


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

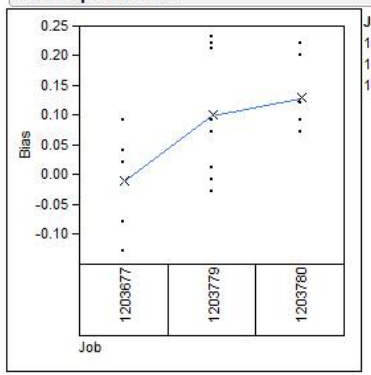
G909-5



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1203677	-0.01200
1203779	0.09875
1203780	0.12833

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1203677	5	2.63	2.618	0.09	2.74	6.91	0.072	0	0	0	0
1203779	8	2.63	2.729	0.108	4.13	7.91	0.109	3	37.5	0	0
1203780	6	2.63	2.758	0.066	4.88	4.8	0.128	2	33.3	0	0

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G909-6	0.57	0.03

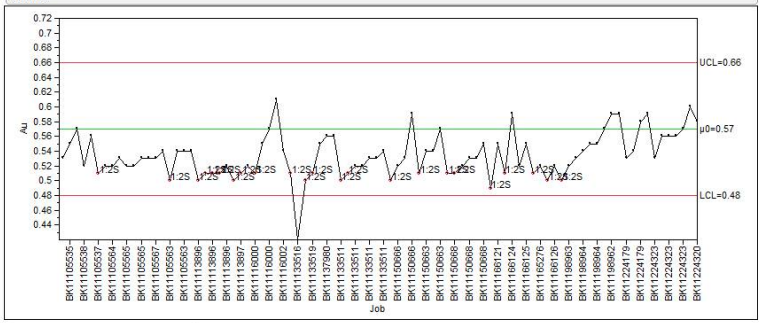
Manufactured By Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G909-6	90	0.57	0.534	0.03	7.04	11.11	0.04	24	26.7	1	1.1

Levey Jennings Control Chart

G909-6



Notes

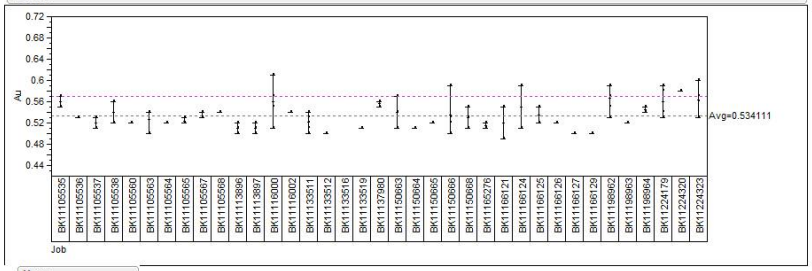
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105535	2	0.57	0.56	0.014	1.75	5.05	0.01	0	0	0	0
BK11105536	1	0.57	0.53	.	7.02	.	0.04	0	0	0	0
BK11105537	2	0.57	0.52	0.014	8.77	5.44	0.05	1	50	0	0
BK11105538	2	0.57	0.54	0.028	5.26	10.48	0.03	0	0	0	0
BK11105560	1	0.57	0.52	.	8.77	.	0.05	0	0	0	0
BK11105563	3	0.57	0.527	0.023	7.6	8.77	0.043	1	33.3	0	0
BK11105564	1	0.57	0.52	.	8.77	.	0.05	0	0	0	0
BK11105565	3	0.57	0.523	0.006	8.19	2.21	0.047	0	0	0	0
BK11105567	3	0.57	0.533	0.006	6.43	2.17	0.037	0	0	0	0
BK11105568	1	0.57	0.54	.	5.26	.	0.03	0	0	0	0
BK11113896	5	0.57	0.51	0.007	10.53	2.77	0.06	4	80	0	0
BK11113897	3	0.57	0.51	0.01	10.53	3.92	0.06	2	66.7	0	0
BK11116000	4	0.57	0.56	0.042	5.26	14.87	0.03	1	25	0	0
BK11116002	1	0.57	0.54	.	5.26	.	0.03	0	0	0	0
BK11133511	7	0.57	0.521	0.013	8.52	5.16	0.049	2	28.6	0	0
BK11133512	1	0.57	0.5	.	12.28	.	0.07	1	100	0	0
BK11133516	1	0.57	0.42	.	26.32	.	0.15	1	100	1	100
BK11133519	2	0.57	0.51	0	10.53	0	0.06	2	100	0	0
BK11137980	3	0.57	0.557	0.006	2.34	2.07	0.013	0	0	0	0
BK11150663	4	0.57	0.54	0.024	5.26	9.07	0.03	1	25	0	0
BK11150664	1	0.57	0.51	.	10.53	.	0.06	1	100	0	0
BK11150665	1	0.57	0.52	.	8.77	.	0.05	0	0	0	0
BK11150666	4	0.57	0.535	0.039	7.89	14.48	0.045	1	25	0	0
BK11150668	4	0.57	0.53	0.016	7.02	6.16	0.04	1	25	0	0
BK11165276	2	0.57	0.515	0.007	9.65	2.75	0.055	1	50	0	0
BK11166121	2	0.57	0.52	0.042	8.77	16.32	0.05	1	50	0	0
BK11166124	2	0.57	0.55	0.057	7.02	20.57	0.04	1	50	0	0
BK11166125	2	0.57	0.535	0.021	6.14	7.93	0.035	0	0	0	0
BK11166126	1	0.57	0.52	.	8.77	.	0.05	0	0	0	0
BK11166127	1	0.57	0.5	.	12.28	.	0.07	1	100	0	0
BK11166129	1	0.57	0.5	.	12.28	.	0.07	1	100	0	0
BK11198962	5	0.57	0.566	0.026	3.51	9.21	0.02	0	0	0	0
BK11198963	1	0.57	0.52	.	8.77	.	0.05	0	0	0	0
BK11198964	2	0.57	0.545	0.007	4.39	2.59	0.025	0	0	0	0
BK11224179	4	0.57	0.56	0.029	4.39	10.51	0.025	0	0	0	0
BK11224320	1	0.57	0.58	.	1.75	.	0.01	0	0	0	0
BK11224323	6	0.57	0.563	0.023	2.92	7.99	0.017	0	0	0	0

Variability Gauge

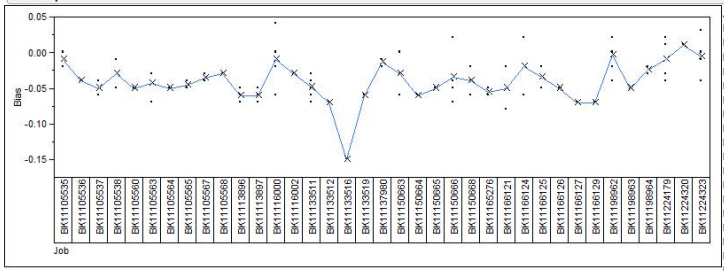
G909-6



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105535	-0.01000
BK11105536	-0.04000
BK11105537	-0.05000
BK11105538	-0.03000
BK11105560	-0.05000
BK11105563	-0.04333
BK11105564	-0.05000
BK11105565	-0.04667
BK11105567	-0.03667
BK11105568	-0.03000
BK11113896	-0.06000
BK11113897	-0.06000
BK11116000	-0.01000
BK11116002	-0.03000
BK11133511	-0.04857
BK11133512	-0.07000
BK11133516	-0.15000
BK11133519	-0.06000
BK11137980	-0.01333
BK11150663	-0.03000
BK11150664	-0.06000
BK11150665	-0.05000
BK11150666	-0.03500
BK11150668	-0.04000
BK11165276	-0.05500
BK11166121	-0.05000
BK11166124	-0.02000
BK11166125	-0.03500
BK11166126	-0.05000
BK11166127	-0.07000
BK11166129	-0.07000
BK11198962	-0.00400
BK11198963	-0.05000
BK11198964	-0.02500
BK11224179	-0.01000
BK11224320	0.01000
BK11224323	-0.00667

QAQC Analysis for Period From Feb 15, 2012 - Jun 19, 2012 ---- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G909-6	0.57	0.03

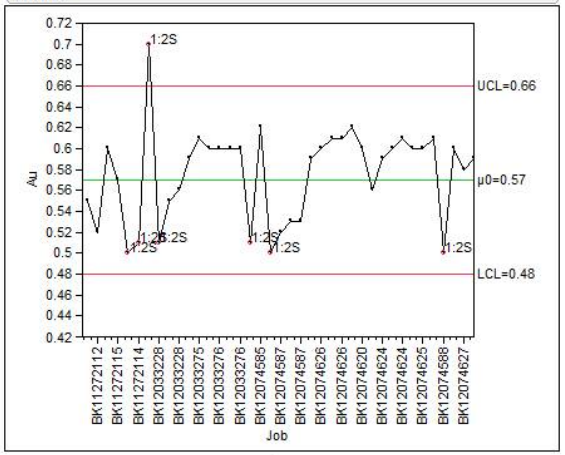
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G909-6	39	0.57	0.576	0.044	6.66	15.43	0.038	7	17.9	1	2.6

Levey Jennings Control Chart

G909-6

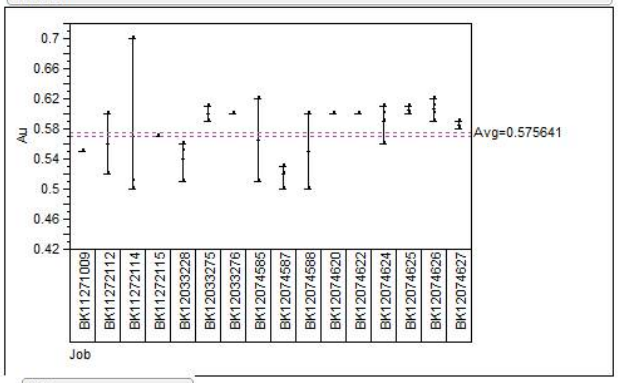


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G909-6



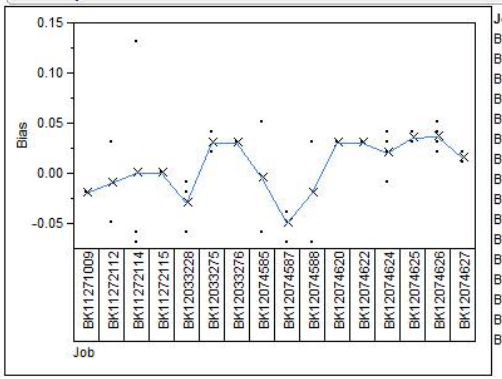
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11271009	1	0.57	0.55	.	3.51	.	0.02	0	0	0	0
BK11272112	2	0.57	0.56	0.057	7.02	20.2	0.04	0	0	0	0
BK11272114	3	0.57	0.57	0.113	15.2	39.54	0.087	3	100	1	33.3
BK11272115	1	0.57	0.57	.	0	.	0	0	0	0	0
BK12033228	3	0.57	0.54	0.026	5.26	9.8	0.03	1	33.3	0	0
BK12033275	2	0.57	0.6	0.014	5.26	4.71	0.03	0	0	0	0
BK12033276	4	0.57	0.6	0	5.26	0	0.03	0	0	0	0
BK12074585	2	0.57	0.565	0.078	9.65	27.53	0.055	1	50	0	0
BK12074587	4	0.57	0.52	0.014	8.77	5.44	0.05	1	25	0	0
BK12074588	2	0.57	0.55	0.071	8.77	25.71	0.05	1	50	0	0
BK12074620	1	0.57	0.6	.	5.26	.	0.03	0	0	0	0
BK12074622	1	0.57	0.6	.	5.26	.	0.03	0	0	0	0
BK12074624	4	0.57	0.59	0.022	4.39	7.32	0.025	0	0	0	0
BK12074625	2	0.57	0.605	0.007	6.14	2.34	0.035	0	0	0	0
BK12074626	5	0.57	0.606	0.011	6.32	3.76	0.036	0	0	0	0
BK12074627	2	0.57	0.585	0.007	2.63	2.42	0.015	0	0	0	0

Bias Report for Job



Job	Avg Bias
BK11271009	-0.02000
BK11272112	-0.01000
BK11272114	3.7e-17
BK11272115	0.00000
BK12033228	-0.03000
BK12033275	0.03000
BK12033276	0.03000
BK12074585	-0.00500
BK12074587	-0.05000
BK12074588	-0.02000
BK12074620	0.03000
BK12074622	0.03000
BK12074624	0.02000
BK12074625	0.03500
BK12074626	0.03600
BK12074627	0.01500

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 ---- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G909-6	0.57	0.03

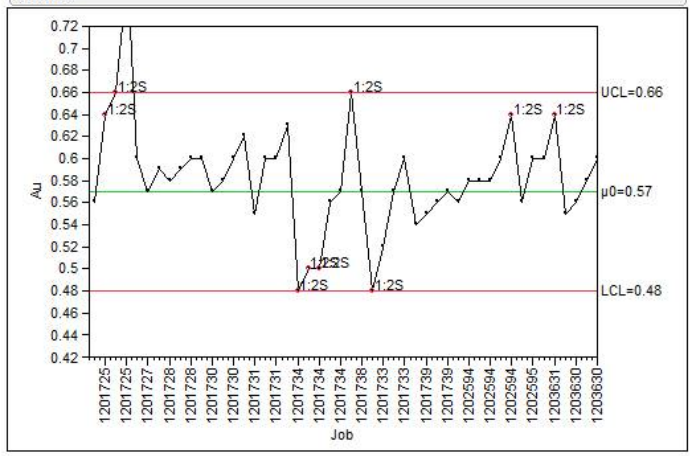
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G909-6	48	0.57	0.583	0.049	6.03	16.74	0.034	11	22.9	5	10.4

Levey Jennings Control Chart

G909-6

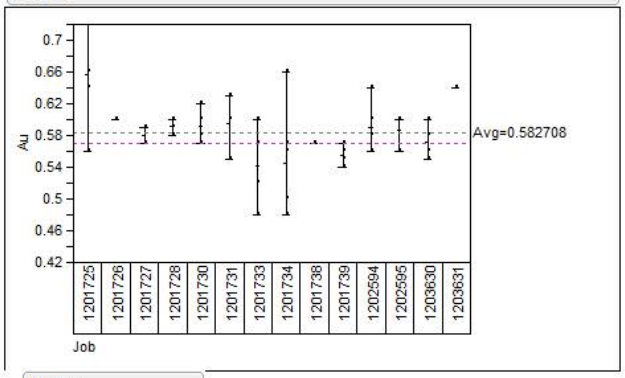


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

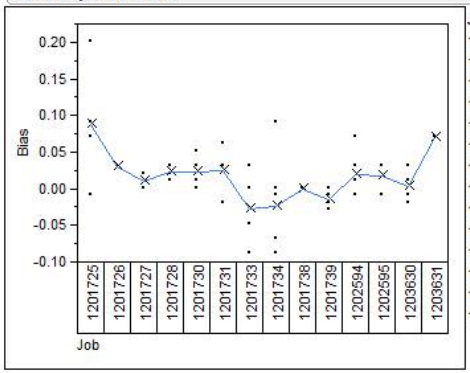
G909-6



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1201725	0.08750
1201726	0.03000
1201727	0.01000
1201728	0.02250
1201730	0.02250
1201731	0.02500
1201733	-0.02750
1201734	-0.02500
1201738	0.00000
1201739	-0.01500
1202594	0.02000
1202595	0.01667
1203630	0.00250
1203631	0.07000

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201725	4	0.57	0.658	0.087	16.23	26.33	0.093	3	75	2	50
1201726	1	0.57	0.6	.	5.26	.	0.03	0	0	0	0
1201727	2	0.57	0.58	0.014	1.75	4.88	0.01	0	0	0	0
1201728	4	0.57	0.593	0.01	3.95	3.23	0.023	0	0	0	0
1201730	4	0.57	0.593	0.022	3.95	7.48	0.023	0	0	0	0
1201731	4	0.57	0.595	0.033	6.14	11.15	0.035	1	25	0	0
1201733	4	0.57	0.543	0.053	7.46	19.59	0.042	1	25	1	25
1201734	6	0.57	0.545	0.067	9.65	24.54	0.055	4	66.7	2	33.3
1201738	1	0.57	0.57	.	0	.	0	0	0	0	0
1201739	4	0.57	0.555	0.013	2.63	4.65	0.015	0	0	0	0
1202594	6	0.57	0.59	0.028	4.09	9.35	0.023	1	16.7	0	0
1202595	3	0.57	0.587	0.023	4.09	7.87	0.023	0	0	0	0
1203630	4	0.57	0.573	0.022	3.07	7.75	0.017	0	0	0	0
1203631	1	0.57	0.64	.	12.28	.	0.07	1	100	0	0

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G910-10	0.97	0.04

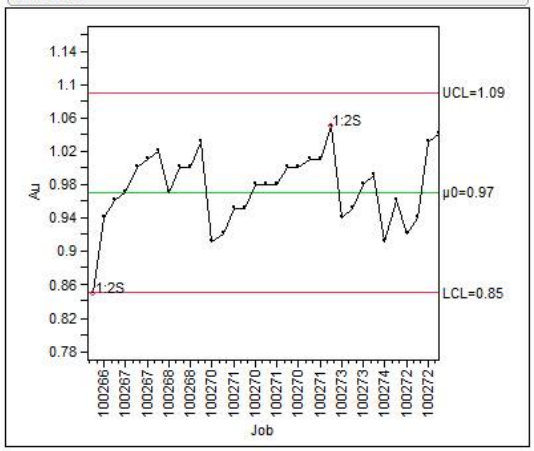
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G910-10	33	0.97	0.974	0.044	3.62	8.98	0.035	2	6.1	1	3

Levey Jennings Control Chart

G910-10

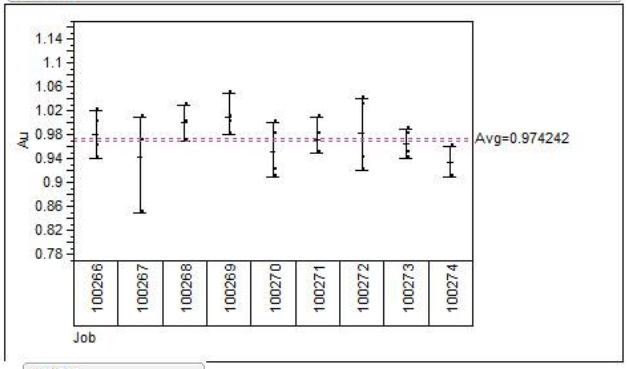


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

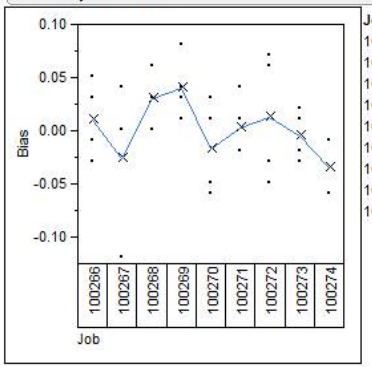
G910-10



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
100266	0.01000
100267	-0.02667
100268	0.03000
100269	0.04000
100270	-0.01750
100271	0.00250
100272	0.01250
100273	-0.00500
100274	-0.03500

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100266	4	0.97	0.98	0.037	3.09	7.45	0.03	0	0	0	0
100267	3	0.97	0.943	0.083	5.5	17.65	0.053	1	33.3	1	33.3
100268	4	0.97	1	0.024	3.09	4.9	0.03	0	0	0	0
100269	4	0.97	1.01	0.029	4.12	5.83	0.04	1	25	0	0
100270	4	0.97	0.953	0.044	3.87	9.29	0.037	0	0	0	0
100271	4	0.97	0.973	0.029	2.32	5.91	0.023	0	0	0	0
100272	4	0.97	0.983	0.061	5.41	12.48	0.053	0	0	0	0
100273	4	0.97	0.965	0.024	2.06	4.93	0.02	0	0	0	0
100274	2	0.97	0.935	0.035	3.61	7.56	0.035	0	0	0	0

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G996-7	5.99	0.39

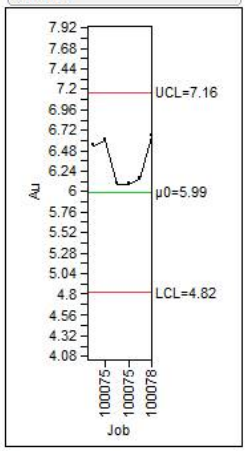
Manufactured By

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G996-7	6	5.99	6.35	0.27	6.01	8.49	0.36	0	0	0	0

Levey Jennings Control Chart

G996-7

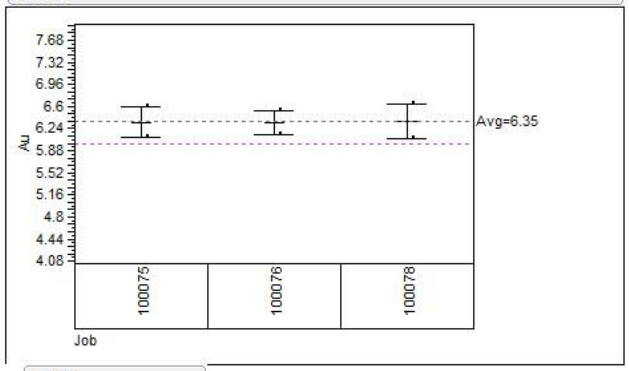


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G996-7



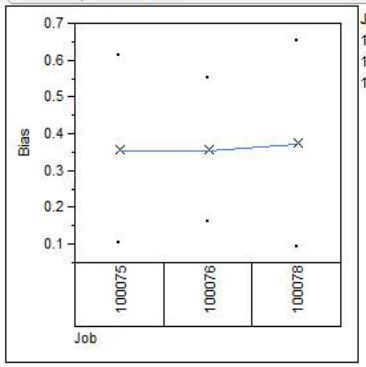
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100075	2	5.99	6.345	0.361	5.93	11.37	0.355	0	0	0	0
100076	2	5.99	6.345	0.276	5.93	8.69	0.355	0	0	0	0
100078	2	5.99	6.36	0.396	6.18	12.45	0.37	0	0	0	0

Bias Report for Job



Job	Avg Bias
100075	0.355000
100076	0.355000
100078	0.370000

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 .... Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G996-7	5.99	0.39

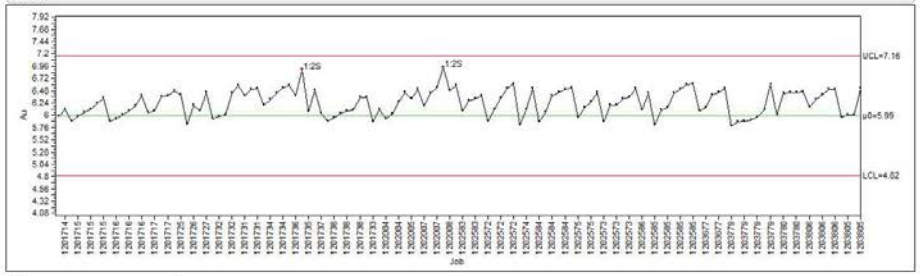
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G996-7	126	5.99	6.242	0.25	4.81	8.01	0.268	2	1.6	0	0

Levey Jennings Control Chart

G996-7



Notes

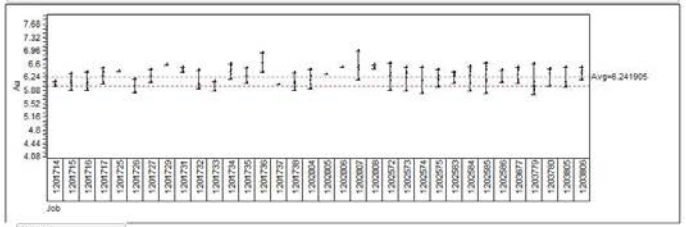
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201714	2	5.99	6.05	0.099	1.17	3.27	0.07	0	0	0	0
1201715	6	5.99	6.102	0.167	2.53	5.47	0.152	0	0	0	0
1201716	6	5.99	6.072	0.184	2.37	6.07	0.142	0	0	0	0
1201717	5	5.99	6.272	0.19	4.71	6.06	0.282	0	0	0	0
1201725	1	5.99	6.4	.	6.84	.	0.41	0	0	0	0
1201726	2	5.99	6.005	0.262	2.09	8.71	0.105	0	0	0	0
1201727	2	5.99	6.27	0.255	4.67	8.12	0.28	0	0	0	0
1201729	1	5.99	6.58	.	9.85	.	0.59	0	0	0	0
1201731	3	5.99	6.467	0.076	7.96	2.34	0.477	0	0	0	0
1201732	4	5.99	6.083	0.228	2.21	7.5	0.132	0	0	0	0
1201733	2	5.99	5.995	0.177	2.09	5.9	0.125	0	0	0	0
1201734	5	5.99	6.408	0.166	6.98	5.17	0.418	0	0	0	0
1201735	2	5.99	6.28	0.297	4.84	9.46	0.29	0	0	0	0
1201736	2	5.99	6.64	0.368	10.85	11.08	0.65	1	50	0	0
1201737	1	5.99	6.04	.	0.83	.	0.05	0	0	0	0
1201738	7	5.99	6.106	0.184	2.69	6.02	0.161	0	0	0	0
1202004	4	5.99	6.158	0.234	3.38	7.59	0.202	0	0	0	0
1202005	1	5.99	6.32	.	5.51	.	0.33	0	0	0	0
1202006	1	5.99	6.5	.	8.51	.	0.51	0	0	0	0
1202007	4	5.99	6.518	0.321	8.81	9.85	0.528	1	25	0	0
1202008	2	5.99	6.525	0.078	8.93	2.38	0.535	0	0	0	0
1202572	5	5.99	6.296	0.3	5.84	9.54	0.35	0	0	0	0
1202573	6	5.99	6.242	0.221	4.92	7.09	0.295	0	0	0	0
1202574	3	5.99	6.147	0.361	4.73	11.74	0.283	0	0	0	0
1202575	4	5.99	6.198	0.201	3.71	6.49	0.222	0	0	0	0
1202583	4	5.99	6.265	0.13	4.59	4.15	0.275	0	0	0	0
1202584	6	5.99	6.296	0.275	5.67	8.73	0.352	0	0	0	0
1202585	7	5.99	6.311	0.306	6.27	9.69	0.376	0	0	0	0
1202586	2	5.99	6.26	0.226	4.51	7.23	0.27	0	0	0	0
1203677	5	5.99	6.318	0.192	5.48	6.07	0.328	0	0	0	0
1203779	7	5.99	6.011	0.278	3.08	9.24	0.184	0	0	0	0
1203780	5	5.99	6.354	0.198	6.08	6.25	0.364	0	0	0	0
1203805	4	5.99	6.12	0.267	2.42	8.74	0.145	0	0	0	0
1203806	5	5.99	6.374	0.147	6.41	4.61	0.384	0	0	0	0

Variability Gauge

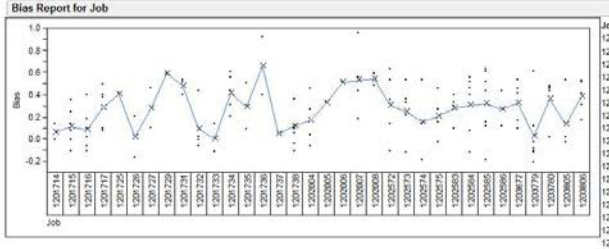
G996-7



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1201714	0.060000
1201715	0.111667
1201716	0.081667
1201717	0.282000
1201725	0.410000
1201726	0.015000
1201727	0.280000
1201729	0.590000
1201731	0.476667
1201732	0.092500
1201733	0.005000
1201734	0.418000
1201735	0.290000
1201736	0.650000
1201737	0.050000
1201738	0.115714
1202004	0.167500
1202005	0.330000
1202006	0.510000
1202007	0.527500
1202008	0.535000
1202572	0.306000
1202573	0.251667
1202574	0.156667
1202575	0.207500
1202583	0.275000
1202584	0.308333
1202585	0.321429
1202586	0.270000
1203677	0.328000
1203779	0.021429
1203780	0.364000
1203805	0.130000
1203806	0.384000

QAQC Analysis for Period From Oct 3, 2012 - Jan 18, 2013 ---- Bureau Veritas Bamako / Mali

CRM Specs

CRM	Value	Sigma
G998-6	0.8	0.06

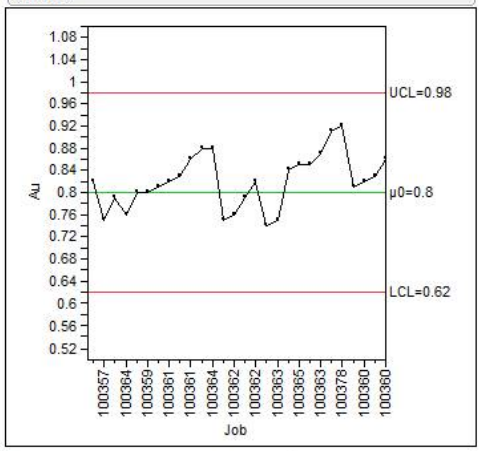
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G998-6	28	0.8	0.82	0.049	5.31	11.92	0.042	1	3.6	0	0

Levey Jennings Control Chart

G998-6

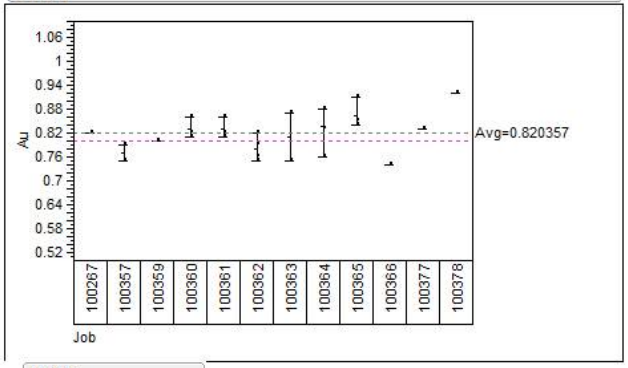


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

G998-6



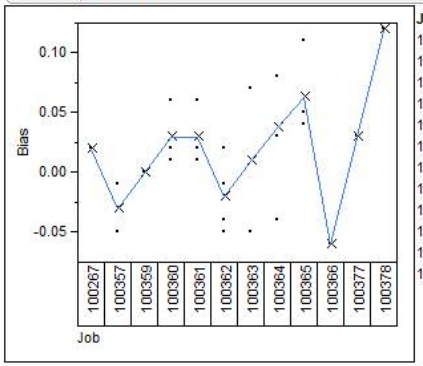
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
100267	1	0.8	0.82	.	2.5	.	0.02	0	0	0	0
100357	2	0.8	0.77	0.028	3.75	7.35	0.03	0	0	0	0
100359	2	0.8	0.8	0	0	0	0	0	0	0	0
100360	3	0.8	0.83	0.026	3.75	6.38	0.03	0	0	0	0
100361	3	0.8	0.83	0.026	3.75	6.38	0.03	0	0	0	0
100362	4	0.8	0.78	0.032	3.75	8.11	0.03	0	0	0	0
100363	2	0.8	0.81	0.085	7.5	20.95	0.06	0	0	0	0
100364	4	0.8	0.838	0.057	7.19	13.56	0.057	0	0	0	0
100365	4	0.8	0.863	0.032	7.81	7.42	0.062	0	0	0	0
100366	1	0.8	0.74	.	7.5	.	0.06	0	0	0	0
100377	1	0.8	0.83	.	3.75	.	0.03	0	0	0	0
100378	1	0.8	0.92	.	15	.	0.12	1	100	0	0

Bias Report for Job



Job	Avg Bias
100267	0.02000
100357	-0.03000
100359	0.00000
100360	0.03000
100361	0.03000
100362	-0.02000
100363	0.01000
100364	0.03750
100365	0.06250
100366	-0.06000
100377	0.03000
100378	0.12000

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 ---- Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G998-6	0.8	0.06

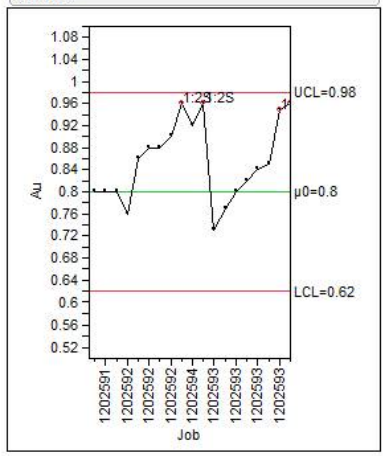
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G998-6	19	0.8	0.855	0.073	8.68	17.01	0.069	5	26.3	0	0

Levey Jennings Control Chart

G998-6



Notes

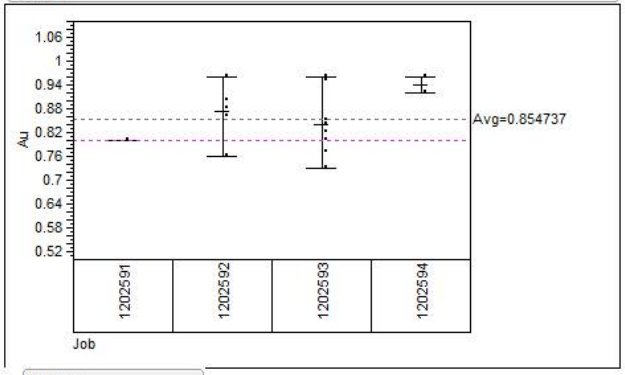
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1202591	3	0.8	0.8	0	0	0	0	0	0	0	0
1202592	6	0.8	0.873	0.065	10.83	14.96	0.087	1	16.7	0	0
1202593	8	0.8	0.84	0.081	8.12	19.22	0.065	2	25	0	0
1202594	2	0.8	0.94	0.028	17.5	6.02	0.14	2	100	0	0

Variability Gauge

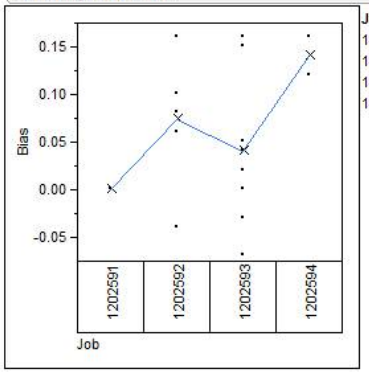
G998-6



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
1202591	0.000000
1202592	0.073333
1202593	0.040000
1202594	0.140000

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 — ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G999-2	0.63	0.06

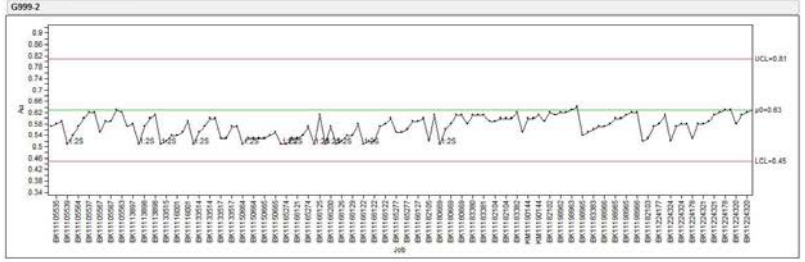
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G999-2	129	0.63	0.572	0.037	9.18	12.98	0.058	12	9.3	0	0

Levey Jennings Control Chart

G999-2



Notes

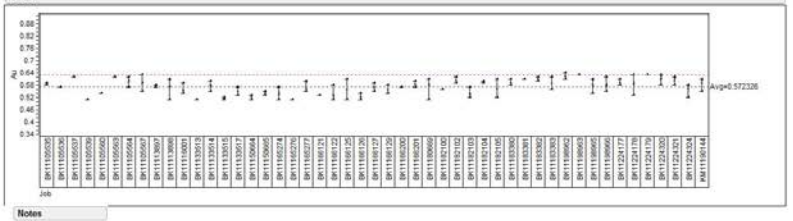
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105535	2	0.63	0.585	0.007	7.14	2.42	0.045	0	0	0	0
BK11105536	1	0.63	0.57	.	9.52	.	0.06	0	0	0	0
BK11105537	1	0.63	0.62	.	1.59	.	0.01	0	0	0	0
BK11105539	1	0.63	0.51	.	19.05	.	0.12	1	100	0	0
BK11105560	1	0.63	0.54	.	14.29	.	0.09	0	0	0	0
BK11105563	1	0.63	0.62	.	1.59	.	0.01	0	0	0	0
BK11105564	3	0.63	0.597	0.025	5.29	8.44	0.033	0	0	0	0
BK11105567	4	0.63	0.59	0.033	6.35	11.07	0.04	0	0	0	0
BK11113897	2	0.63	0.575	0.007	8.73	2.46	0.055	0	0	0	0
BK11113898	4	0.63	0.573	0.045	9.13	15.72	0.058	1	25	0	0
BK11116001	4	0.63	0.555	0.024	11.9	8.58	0.075	0	0	0	0
BK11133513	1	0.63	0.51	.	19.05	.	0.12	1	100	0	0
BK11133514	4	0.63	0.58	0.024	7.94	8.45	0.05	0	0	0	0
BK11133515	2	0.63	0.515	0.007	18.25	2.75	0.115	1	50	0	0
BK11133517	4	0.63	0.55	0.023	12.7	8.4	0.08	0	0	0	0
BK11159664	3	0.63	0.523	0.012	16.93	4.41	0.107	1	33.3	0	0
BK11159665	4	0.63	0.538	0.01	14.68	3.58	0.092	0	0	0	0
BK11165274	5	0.63	0.532	0.025	15.58	9.36	0.098	2	40	0	0
BK11165276	1	0.63	0.51	.	19.05	.	0.12	1	100	0	0
BK11165277	4	0.63	0.575	0.024	8.73	8.28	0.055	0	0	0	0
BK11166121	1	0.63	0.53	.	15.87	.	0.1	0	0	0	0
BK11166122	4	0.63	0.533	0.032	15.48	12.02	0.098	1	25	0	0
BK11166125	2	0.63	0.56	0.071	11.11	25.25	0.07	1	50	0	0
BK11166126	3	0.63	0.523	0.015	16.93	5.84	0.107	1	33.3	0	0
BK11166127	2	0.63	0.57	0.026	9.52	9.92	0.06	0	0	0	0
BK11166129	2	0.63	0.56	0.028	11.11	10.1	0.07	0	0	0	0
BK11166200	1	0.63	0.57	.	9.52	.	0.06	0	0	0	0
BK11166201	2	0.63	0.585	0.021	7.14	7.25	0.045	0	0	0	0
BK11180669	4	0.63	0.578	0.047	8.33	16.34	0.053	1	25	0	0
BK11182100	1	0.63	0.56	.	11.11	.	0.07	0	0	0	0
BK11182102	2	0.63	0.605	0.021	3.97	7.01	0.025	0	0	0	0
BK11182103	2	0.63	0.54	0.026	14.29	9.8	0.09	0	0	0	0
BK11182104	4	0.63	0.595	0.006	5.56	1.94	0.035	0	0	0	0
BK11182105	2	0.63	0.565	0.064	10.32	22.53	0.065	0	0	0	0
BK11183380	2	0.63	0.595	0.021	5.56	7.13	0.035	0	0	0	0
BK11183381	2	0.63	0.61	0	3.17	0	0.02	0	0	0	0
BK11183382	2	0.63	0.61	0.014	3.17	4.64	0.02	0	0	0	0
BK11183383	2	0.63	0.59	0.042	6.35	14.38	0.04	0	0	0	0
BK11198962	4	0.63	0.623	0.013	1.98	4.04	0.013	0	0	0	0
BK11198963	1	0.63	0.63	.	0	.	0	0	0	0	0
BK11198965	5	0.63	0.584	0.029	7.3	9.87	0.046	0	0	0	0
BK11198966	4	0.63	0.58	0.029	7.94	10.15	0.05	0	0	0	0
BK11224177	2	0.63	0.595	0.021	5.56	7.13	0.035	0	0	0	0
BK11224178	4	0.63	0.583	0.041	7.54	14.12	0.048	0	0	0	0
BK11224179	1	0.63	0.63	.	0	.	0	0	0	0	0
BK11224320	4	0.63	0.61	0.022	3.17	7.08	0.02	0	0	0	0
BK11224321	3	0.63	0.603	0.021	4.23	6.9	0.027	0	0	0	0
BK11224324	4	0.63	0.583	0.029	10.71	10.21	0.068	0	0	0	0
KM11190144	4	0.63	0.59	0.027	6.35	9.16	0.04	0	0	0	0

Variability Gauge

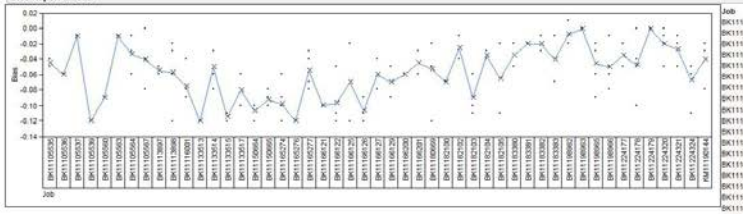
G999-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11105535	-0.04500
BK11105536	-0.06000
BK11105537	-0.01000
BK11105539	-0.12000
BK11105560	-0.09000
BK11105563	-0.01000
BK11105564	-0.03333
BK11105567	-0.04000
BK11113897	-0.05500
BK11113898	-0.05750
BK11116001	-0.07500
BK11133513	-0.12000
BK11133514	-0.05000
BK11133515	-0.11500
BK11133517	-0.08000
BK11159664	-0.10667
BK11165665	-0.09250
BK11165274	-0.09600
BK11165276	-0.12500
BK11165277	-0.05500
BK11166121	-0.10000
BK11166122	-0.09750
BK11166125	-0.07000
BK11166126	-0.10667
BK11166127	-0.06000
BK11166129	-0.07000
BK11166200	-0.06000
BK11166201	-0.04500
BK11180669	-0.05250
BK11182100	-0.07000
BK11182102	-0.02500
BK11182103	-0.09000
BK11182104	-0.03500
BK11182105	-0.06500
BK11183380	-0.03500
BK11183381	-0.02000
BK11183382	-0.02500
BK11183383	-0.04000
BK11198962	-0.00750
BK11198963	0.00000
BK11198965	-0.04000
BK11198966	-0.05000
BK11224177	-0.03500
BK11224178	-0.04750
BK11224179	0.00000
BK11224320	-0.02000
BK11224321	-0.02667
BK11224324	-0.06750
KM11190144	-0.04000

QAQC Analysis for Period From Feb 15, 2012 - Jun 19, 2012 - ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
G999-2	0.63	0.06

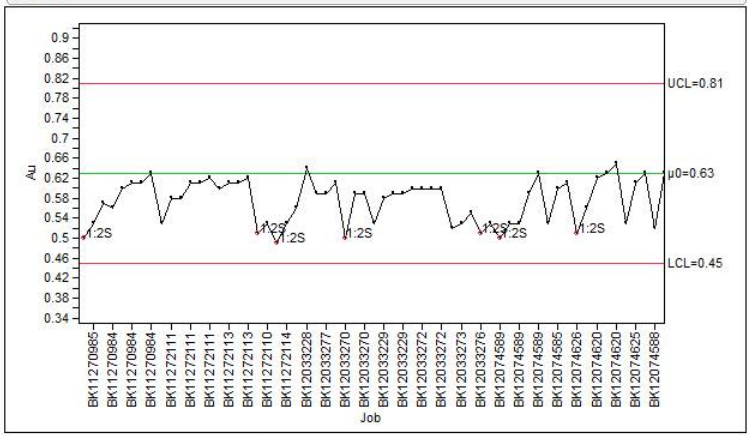
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G999-2	61	0.63	0.575	0.044	8.95	15.21	0.056	7	11.5	0	0

Levey Jennings Control Chart

G999-2



Notes

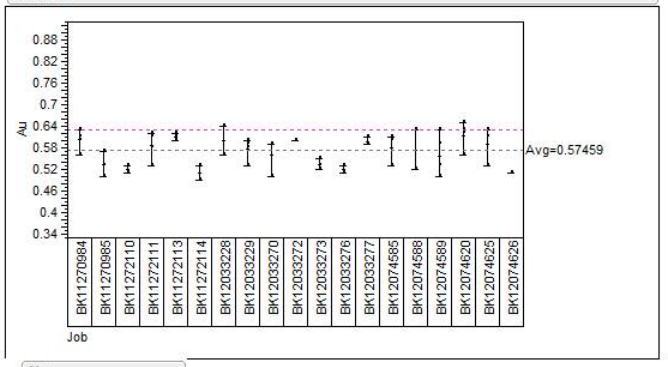
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11270984	5	0.63	0.602	0.026	4.44	8.6	0.028	0	0	0	0
BK11270985	3	0.63	0.533	0.035	15.34	13.17	0.097	1	33.3	0	0
BK11272110	2	0.63	0.52	0.014	17.46	5.44	0.11	1	50	0	0
BK11272111	6	0.63	0.588	0.033	6.61	11.26	0.042	0	0	0	0
BK11272113	4	0.63	0.61	0.008	3.17	2.68	0.02	0	0	0	0
BK11272114	2	0.63	0.51	0.028	19.05	11.09	0.12	1	50	0	0
BK12033228	2	0.63	0.6	0.057	6.35	18.86	0.04	0	0	0	0
BK12033229	5	0.63	0.578	0.028	8.25	9.6	0.052	0	0	0	0
BK12033270	3	0.63	0.56	0.052	11.11	18.56	0.07	1	33.3	0	0
BK12033272	3	0.63	0.6	0	4.76	0	0.03	0	0	0	0
BK12033273	3	0.63	0.533	0.015	15.34	5.73	0.097	0	0	0	0
BK12033276	2	0.63	0.52	0.014	17.46	5.44	0.11	1	50	0	0
BK12033277	3	0.63	0.597	0.012	5.29	3.87	0.033	0	0	0	0
BK12074585	3	0.63	0.58	0.044	7.94	15.03	0.05	0	0	0	0
BK12074588	2	0.63	0.575	0.078	8.73	27.05	0.055	0	0	0	0
BK12074589	5	0.63	0.556	0.053	11.75	18.97	0.074	1	20	0	0
BK12074620	4	0.63	0.615	0.039	3.97	12.6	0.025	0	0	0	0
BK12074625	3	0.63	0.59	0.053	6.35	17.94	0.04	0	0	0	0
BK12074626	1	0.63	0.51	.	19.05	.	0.12	1	100	0	0

Variability Gauge

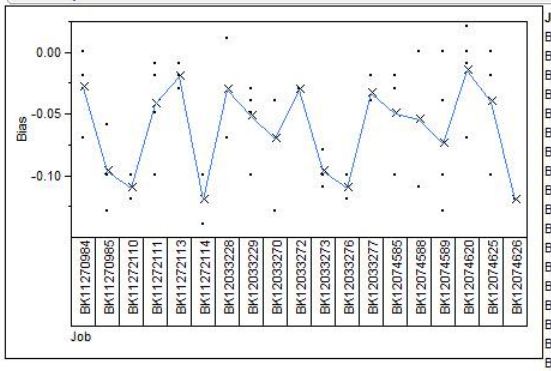
G999-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Job	Avg Bias
BK11270984	-0.02800
BK11270985	-0.09667
BK11272110	-0.11000
BK11272111	-0.04167
BK11272113	-0.02000
BK11272114	-0.12000
BK12033228	-0.03000
BK12033229	-0.05200
BK12033270	-0.07000
BK12033272	-0.03000
BK12033273	-0.09667
BK12033276	-0.11000
BK12033277	-0.03333
BK12074585	-0.05000
BK12074588	-0.05500
BK12074589	-0.07400
BK12074620	-0.01500
BK12074625	-0.04000
BK12074626	-0.12000

QAQC Analysis for Period From Jul 10, 2012 - Jan 1, 2013 — Intertek Tarkwa / Ghana

CRM Specs

CRM	Value	Sigma
G999-2	0.63	0.06

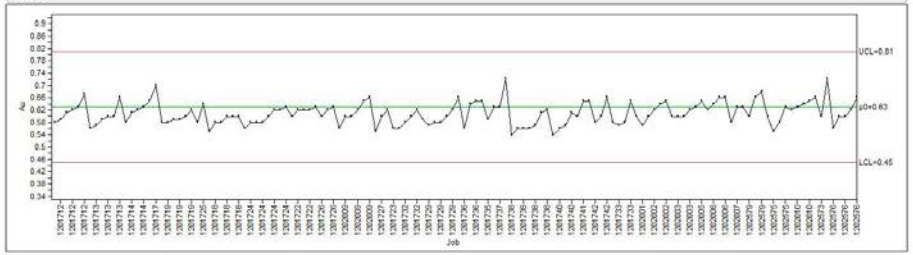
Manufactured By
Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
G999-2	136	0.63	0.609	0.035	5.37	11.63	0.034	0	0	0	0

Levey Jennings Control Chart

G999-2

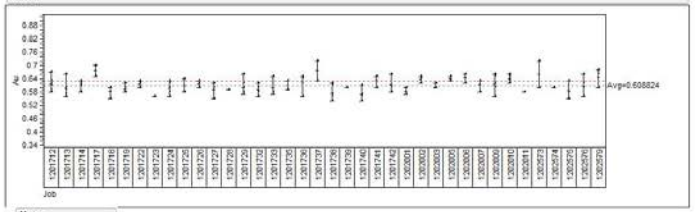


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

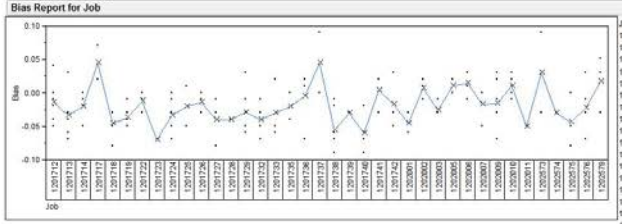
G999-2



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Bias Report for Job



Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
1201712	6	0.63	0.617	0.032	4.23	10.39	0.027	0	0	0	0
1201713	6	0.63	0.597	0.035	6.88	11.74	0.043	0	0	0	0
1201714	4	0.63	0.61	0.022	3.17	7.08	0.02	0	0	0	0
1201717	2	0.63	0.675	0.035	7.14	10.48	0.045	0	0	0	0
1201718	6	0.63	0.585	0.02	7.14	8.75	0.045	0	0	0	0
1201719	6	0.63	0.593	0.015	5.82	5.07	0.037	0	0	0	0
1201722	5	0.63	0.616	0.011	1.9	3.55	0.012	0	0	0	0
1201723	1	0.63	0.56	.	11.11	.	0.07	0	0	0	0
1201724	8	0.63	0.596	0.025	5.36	8.4	0.034	0	0	0	0
1201725	2	0.63	0.61	0.042	4.76	13.91	0.03	0	0	0	0
1201726	3	0.63	0.617	0.015	2.12	4.95	0.013	0	0	0	0
1201727	3	0.63	0.59	0.036	6.35	12.22	0.04	0	0	0	0
1201728	1	0.63	0.59	.	6.35	.	0.04	0	0	0	0
1201729	6	0.63	0.602	0.034	6.08	11.21	0.038	0	0	0	0
1201732	4	0.63	0.59	0.026	6.35	8.75	0.04	0	0	0	0
1201733	3	0.63	0.6	0.044	6.88	14.53	0.043	0	0	0	0
1201735	2	0.63	0.61	0.028	3.17	9.27	0.02	0	0	0	0
1201736	4	0.63	0.625	0.044	4.76	13.96	0.03	0	0	0	0
1201737	2	0.63	0.675	0.064	7.14	10.66	0.045	0	0	0	0
1201738	7	0.63	0.574	0.029	8.84	10.22	0.056	0	0	0	0
1201739	1	0.63	0.6	.	4.76	.	0.03	0	0	0	0
1201740	4	0.63	0.57	0.029	9.52	10.33	0.06	0	0	0	0
1201741	3	0.63	0.633	0.029	3.7	9.12	0.023	0	0	0	0
1201742	3	0.63	0.613	0.042	5.82	13.56	0.037	0	0	0	0
1202001	2	0.63	0.585	0.021	7.14	7.25	0.045	0	0	0	0
1202002	3	0.63	0.637	0.015	2.12	4.8	0.013	0	0	0	0
1202003	4	0.63	0.605	0.01	3.97	3.31	0.025	0	0	0	0
1202005	2	0.63	0.64	0.014	1.59	4.42	0.01	0	0	0	0
1202006	4	0.63	0.645	0.019	3.17	5.94	0.02	0	0	0	0
1202007	3	0.63	0.613	0.029	2.65	9.41	0.017	0	0	0	0
1202009	6	0.63	0.615	0.037	5.03	11.95	0.032	0	0	0	0
1202010	5	0.63	0.64	0.016	2.22	4.94	0.014	0	0	0	0
1202011	1	0.63	0.58	.	7.94	.	0.05	0	0	0	0
1202573	2	0.63	0.66	0.085	9.52	25.71	0.06	0	0	0	0
1202574	1	0.63	0.6	.	4.76	.	0.03	0	0	0	0
1202575	3	0.63	0.567	0.04	6.88	13.78	0.043	0	0	0	0
1202576	5	0.63	0.608	0.036	5.4	11.95	0.034	0	0	0	0
1202579	3	0.63	0.647	0.042	5.82	12.88	0.037	0	0	0	0

Job	Avg Bias
1201712	-0.01333
1201713	-0.03333
1201714	-0.02000
1201717	0.04500
1201718	-0.04500
1201719	-0.03667
1201722	-0.01200
1201723	-0.07000
1201724	-0.03375
1201725	-0.02000
1201726	-0.01333
1201727	-0.04000
1201728	-0.04000
1201729	-0.02833
1201732	-0.04000
1201733	-0.03000
1201735	-0.02000
1201736	-0.00500
1201737	0.04500
1201738	-0.05571
1201739	-0.03000
1201740	-0.06000
1201741	0.00333
1201742	-0.01667
1202001	-0.04500
1202002	0.00667
1202003	-0.02500
1202005	0.01000
1202006	0.01500
1202007	-0.01667
1202009	-0.01500
1202010	0.01000
1202011	-0.05000
1202573	0.03000
1202574	-0.03000
1202575	-0.04333
1202576	-0.02200
1202579	0.01667

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
GLG304-1	0.154	0.017

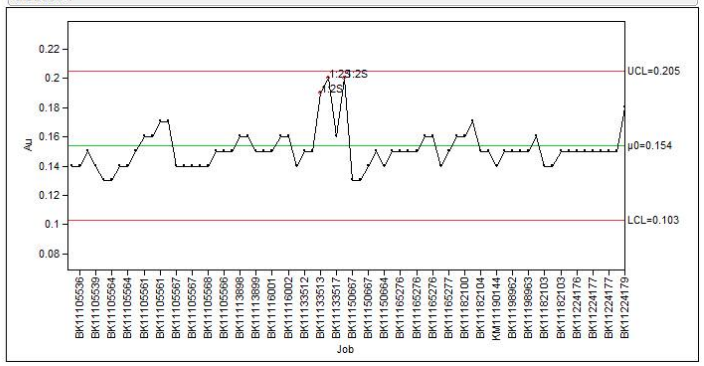
Manufactured By Geostats

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
GLG304-1	70	0.154	0.151	0.014	6.75	18.17	0.01	3	4.3	0	0

Levey Jennings Control Chart

GLG304-1

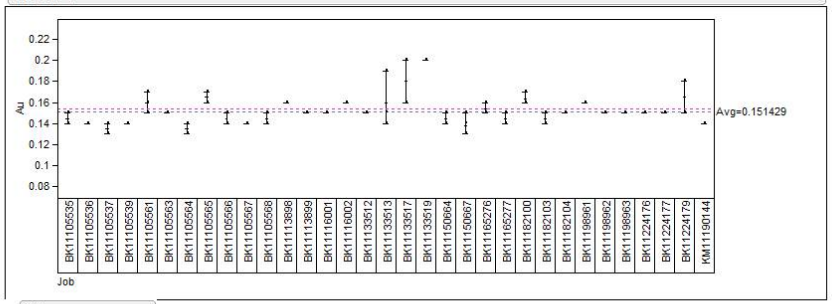


Notes

Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Variability Gauge

GLG304-1



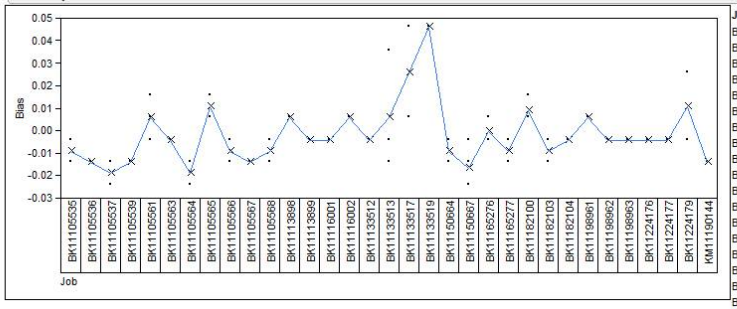
Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105535	2	0.154	0.145	0.007	5.84	9.75	0.009	0	0	0	0
BK11105536	1	0.154	0.14	.	9.09	.	0.014	0	0	0	0
BK11105537	2	0.154	0.135	0.007	12.34	10.48	0.019	0	0	0	0
BK11105539	1	0.154	0.14	.	9.09	.	0.014	0	0	0	0
BK11105561	3	0.154	0.16	0.01	5.63	12.5	0.009	0	0	0	0
BK11105563	1	0.154	0.15	.	2.6	.	0.004	0	0	0	0
BK11105564	2	0.154	0.135	0.007	12.34	10.48	0.019	0	0	0	0
BK11105565	2	0.154	0.165	0.007	7.14	8.57	0.011	0	0	0	0
BK11105566	2	0.154	0.145	0.007	5.84	9.75	0.009	0	0	0	0
BK11105567	3	0.154	0.14	0	9.09	0	0.014	0	0	0	0
BK11105568	2	0.154	0.145	0.007	5.84	9.75	0.009	0	0	0	0
BK11113898	2	0.154	0.16	0	3.9	0	0.006	0	0	0	0
BK11113899	2	0.154	0.15	0	2.6	0	0.004	0	0	0	0
BK11116001	1	0.154	0.15	.	2.6	.	0.004	0	0	0	0
BK11116002	2	0.154	0.16	0	3.9	0	0.006	0	0	0	0
BK11133512	1	0.154	0.15	.	2.6	.	0.004	0	0	0	0
BK11133513	3	0.154	0.16	0.026	11.69	33.07	0.018	1	33.3	0	0
BK11133517	2	0.154	0.18	0.028	16.88	31.43	0.026	1	50	0	0
BK11133519	1	0.154	0.2	.	29.87	.	0.046	1	100	0	0
BK11150664	2	0.154	0.145	0.007	5.84	9.75	0.009	0	0	0	0
BK11150667	4	0.154	0.138	0.01	10.71	13.93	0.016	0	0	0	0
BK11165276	5	0.154	0.154	0.005	3.12	7.11	0.005	0	0	0	0
BK11165277	2	0.154	0.145	0.007	5.84	9.75	0.009	0	0	0	0
BK11182100	3	0.154	0.163	0.006	6.06	7.07	0.009	0	0	0	0
BK11182103	4	0.154	0.145	0.006	5.84	7.96	0.009	0	0	0	0
BK11182104	2	0.154	0.15	0	2.6	0	0.004	0	0	0	0
BK11198961	1	0.154	0.16	.	3.9	.	0.006	0	0	0	0
BK11198962	3	0.154	0.15	0	2.6	0	0.004	0	0	0	0
BK11198963	1	0.154	0.15	.	2.6	.	0.004	0	0	0	0
BK11224176	2	0.154	0.15	0	2.6	0	0.004	0	0	0	0
BK11224177	3	0.154	0.15	0	2.6	0	0.004	0	0	0	0
BK11224179	2	0.154	0.165	0.021	9.74	25.71	0.015	0	0	0	0
KM11190144	1	0.154	0.14	.	9.09	.	0.014	0	0	0	0

Bias Report for Job



Job	Avg Bias
BK11105535	-0.00900
BK11105536	-0.01400
BK11105537	-0.01900
BK11105539	-0.01400
BK11105561	0.00600
BK11105563	-0.00400
BK11105564	-0.01900
BK11105565	0.01100
BK11105566	-0.00900
BK11105567	-0.01400
BK11105568	-0.00900
BK11113898	0.00600
BK11113899	-0.00400
BK11116001	-0.00400
BK11116002	0.00600
BK11133512	-0.00400
BK11133513	0.00600
BK11133517	0.02600
BK11133519	0.04600
BK11150664	-0.00900
BK11150667	-0.01650
BK11165276	0.00000
BK11165277	-0.00900
BK11182100	0.00933
BK11182103	-0.00900
BK11182104	-0.00400
BK11198961	0.00600
BK11198962	-0.00400
BK11198963	-0.00400
BK11224176	-0.00400
BK11224177	-0.00400
BK11224179	0.01100
KM11190144	-0.01400

QAQC Analysis for Period From Jun 29, 2011 - Dec 12, 2011 --- ALS Bamako / Mali

CRM Specs

CRM	Value	Sigma
ST16/5357	0.52	0.03

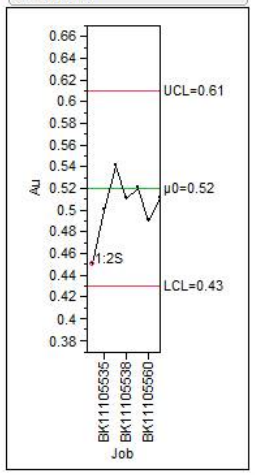
Manufactured By

Summary Lab Performance

CRM	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
ST16/5357	7	0.52	0.503	0.028	4.4	11.18	0.023	1	14.3	0	0

Levey Jennings Control Chart

ST16/5357



Notes

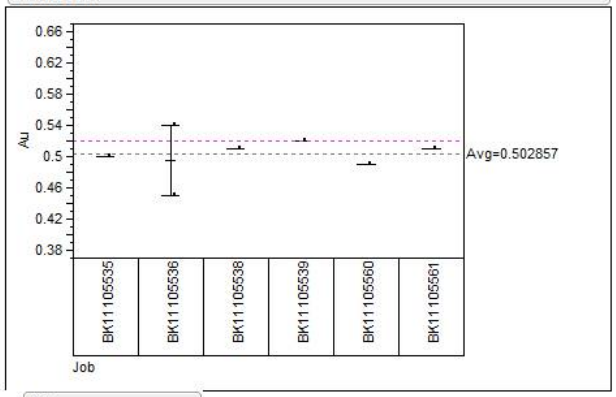
Red Line - 3 Std Dev
Westgard Rule 1 2s - point beyond 2 Std Dev
Graph Window +- 5 Std Dev

Lab Performance by Job

Job	N Samples	Expected Value	Achieved Value	Std Dev	Accuracy %	Precision %	MAD	N Failed 2 Std Dev	% Failed	N Failed 3 Std Dev	% Failed
BK11105535	1	0.52	0.5	.	3.85	.	0.02	0	0	0	0
BK11105536	2	0.52	0.495	0.064	8.65	25.71	0.045	1	50	0	0
BK11105538	1	0.52	0.51	.	1.92	.	0.01	0	0	0	0
BK11105539	1	0.52	0.52	.	0	.	0	0	0	0	0
BK11105560	1	0.52	0.49	.	5.77	.	0.03	0	0	0	0
BK11105561	1	0.52	0.51	.	1.92	.	0.01	0	0	0	0

Variability Gauge

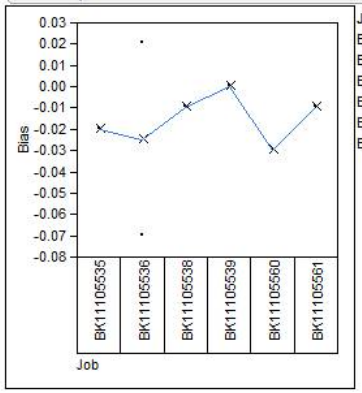
ST16/5357



Notes

Purple Line - Expected Value
Graph Window +- 5 Std Dev

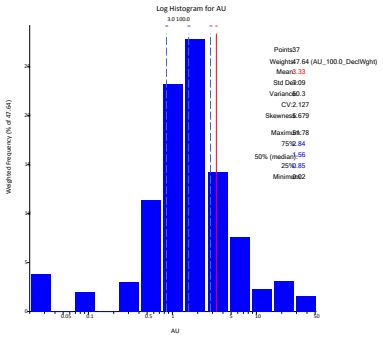
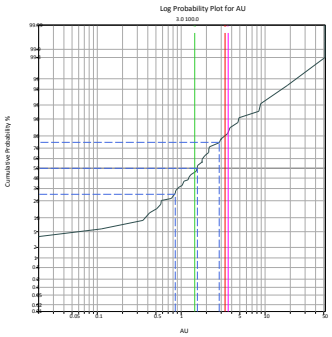
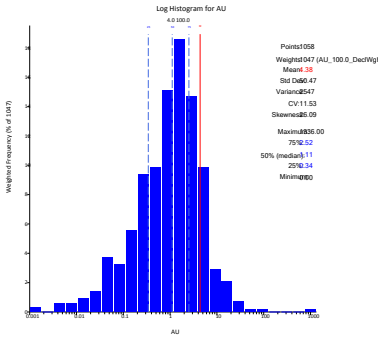
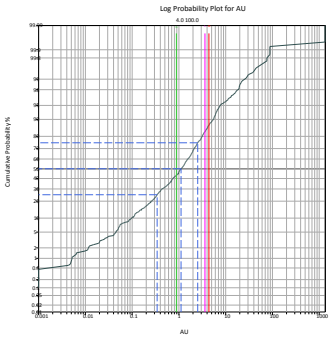
Bias Report for Job



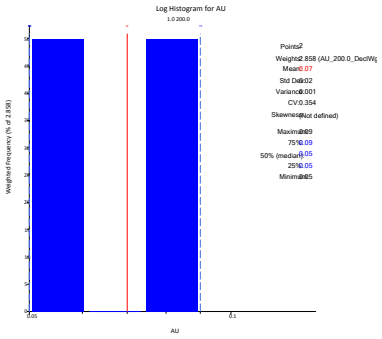
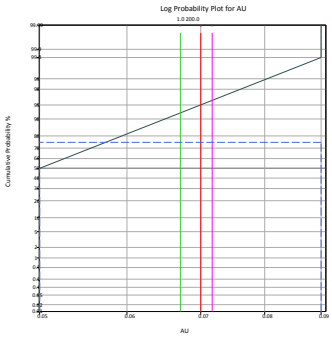
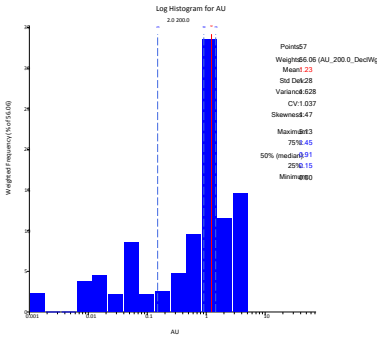
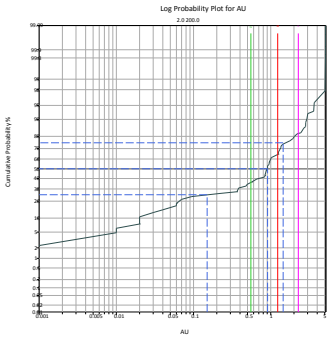
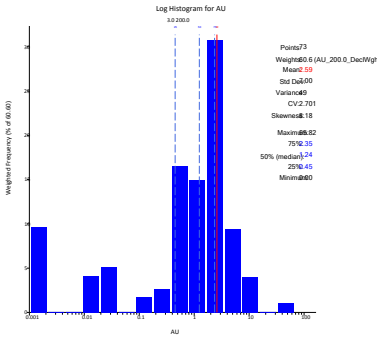
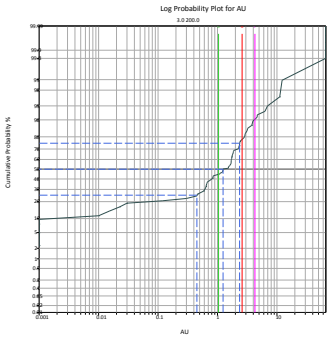
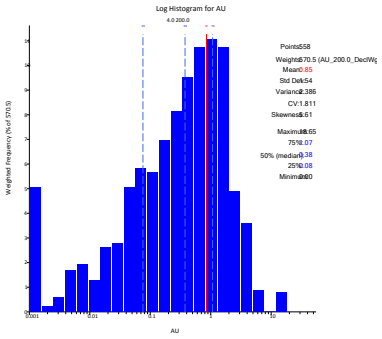
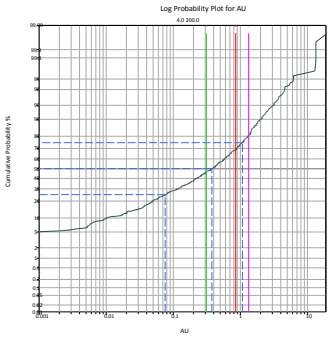
Job	Avg Bias
BK11105535	-0.02000
BK11105536	-0.02500
BK11105538	-0.01000
BK11105539	0.00000
BK11105560	-0.03000
BK11105561	-0.01000

Appendix C Histograms and log-probability plots

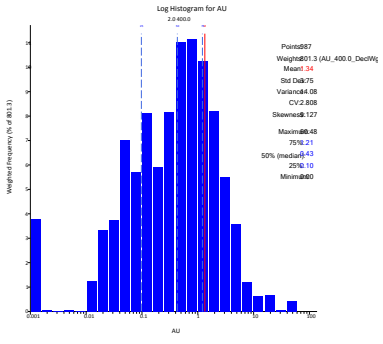
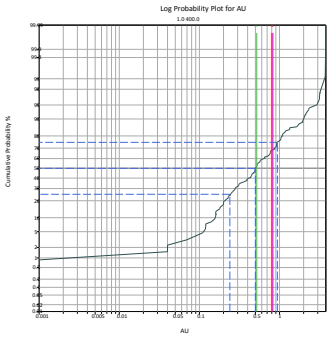
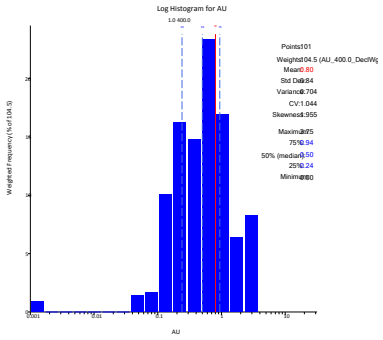
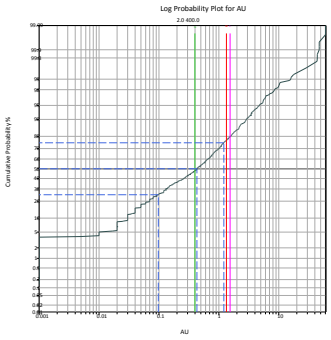
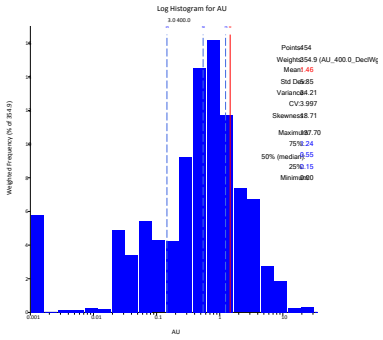
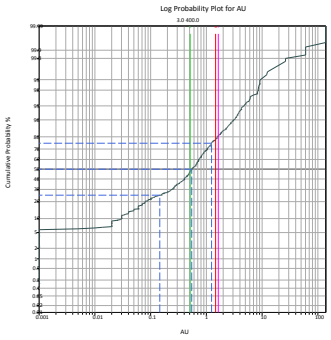
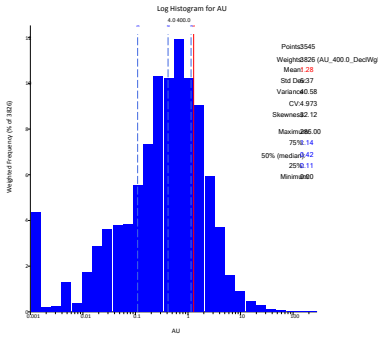
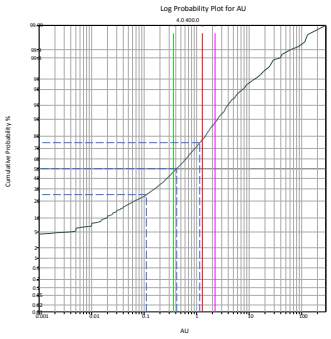
Southern granites

Transitional		
Fresh		

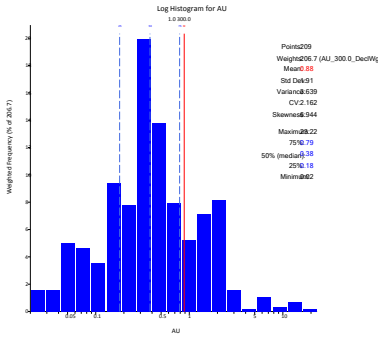
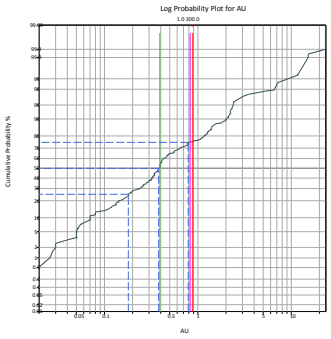
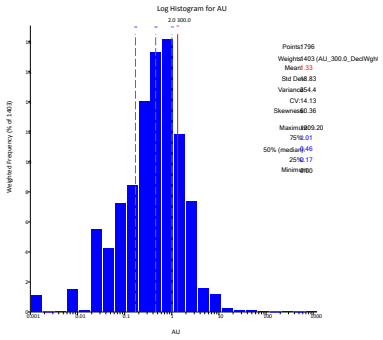
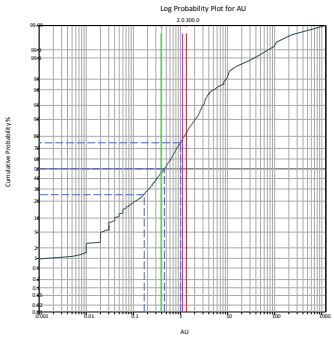
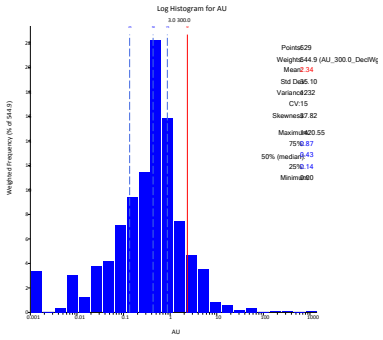
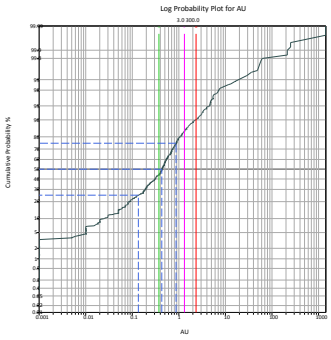
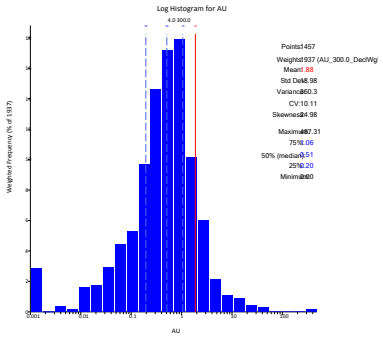
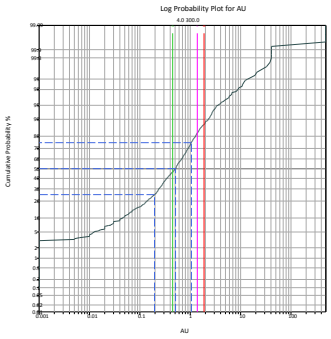
Northern granites

Laterite		
Oxide		
Transitional		
Fresh		

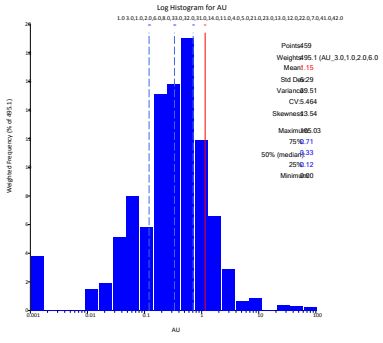
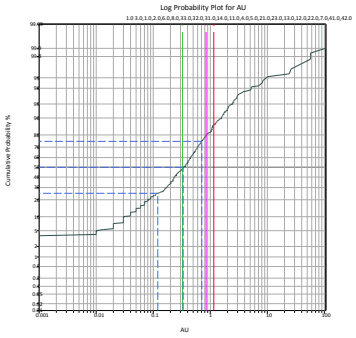
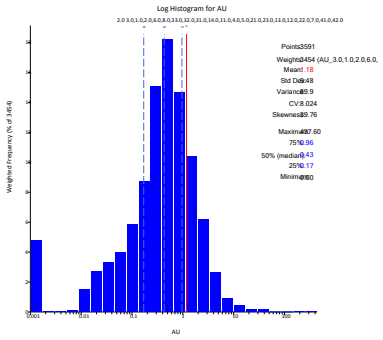
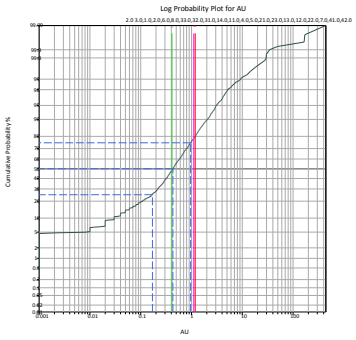
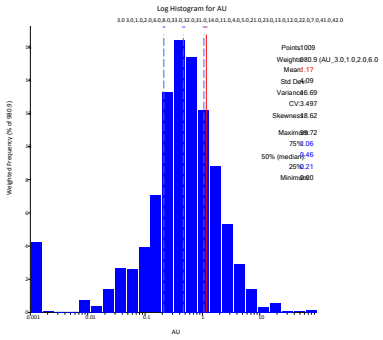
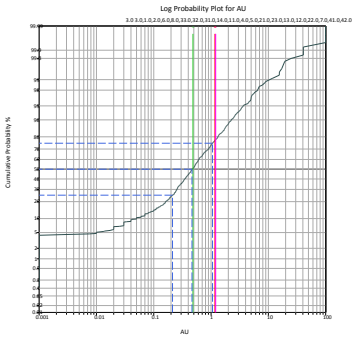
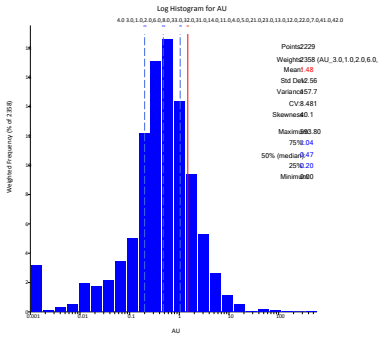
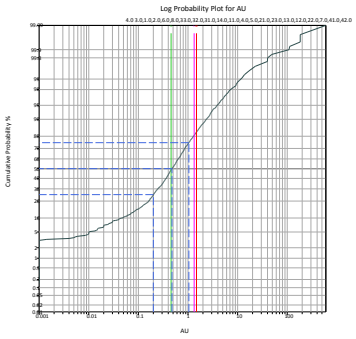
Southern sediments

Laterite		
Oxide		
Transitional		
Fresh		

Northern sediments

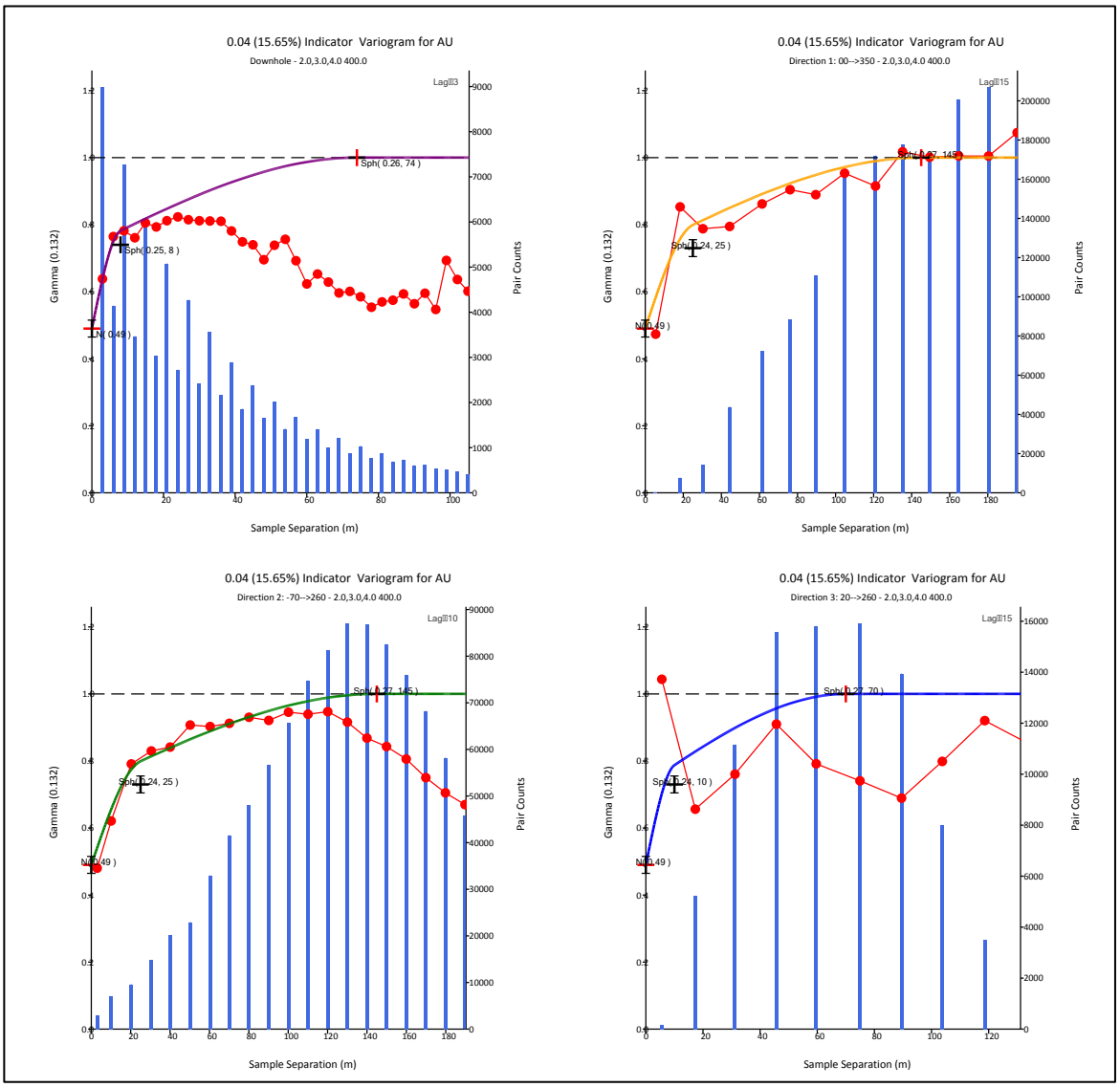
Laterite		
Oxide		
Transitional		
Fresh		

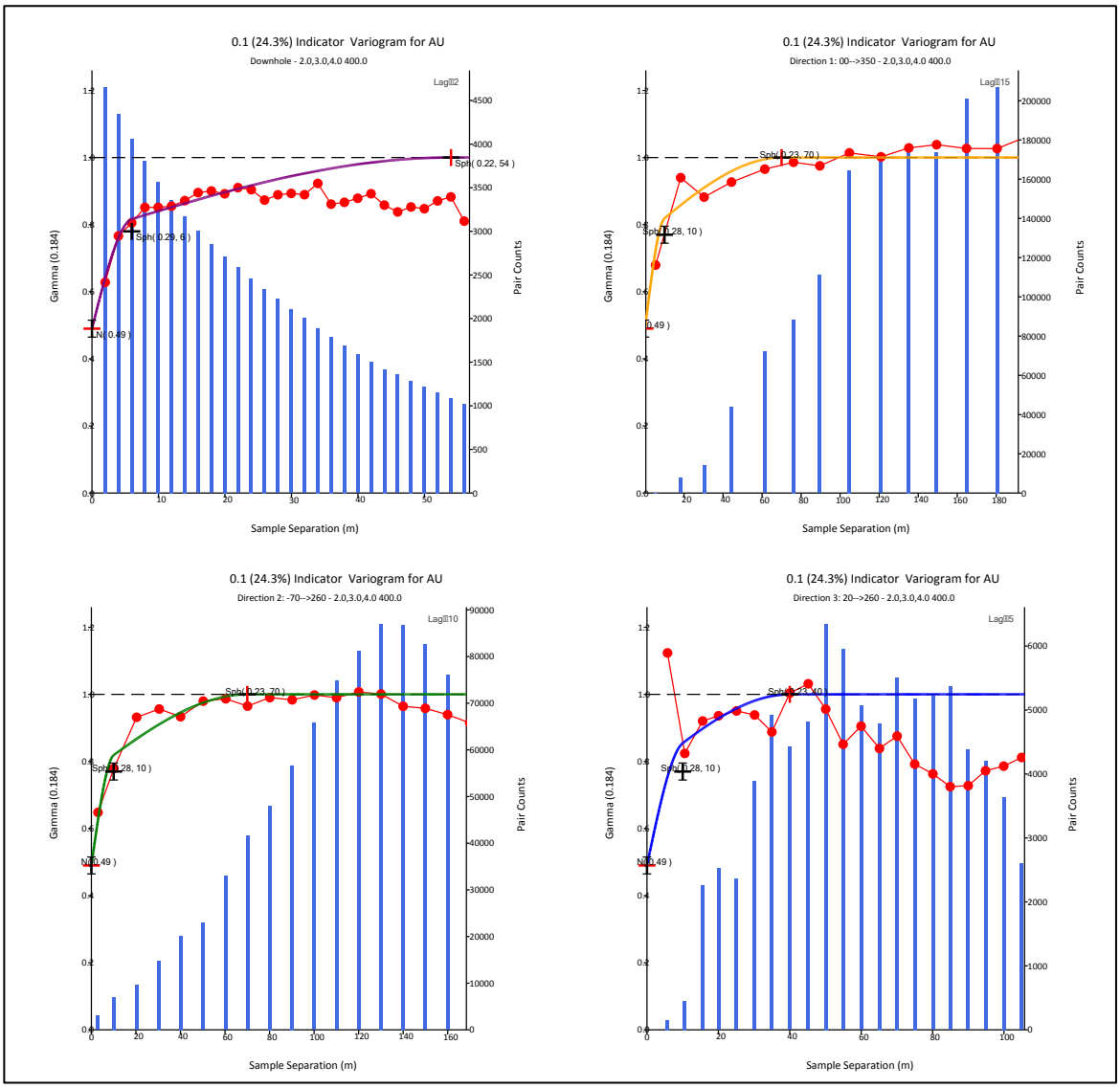
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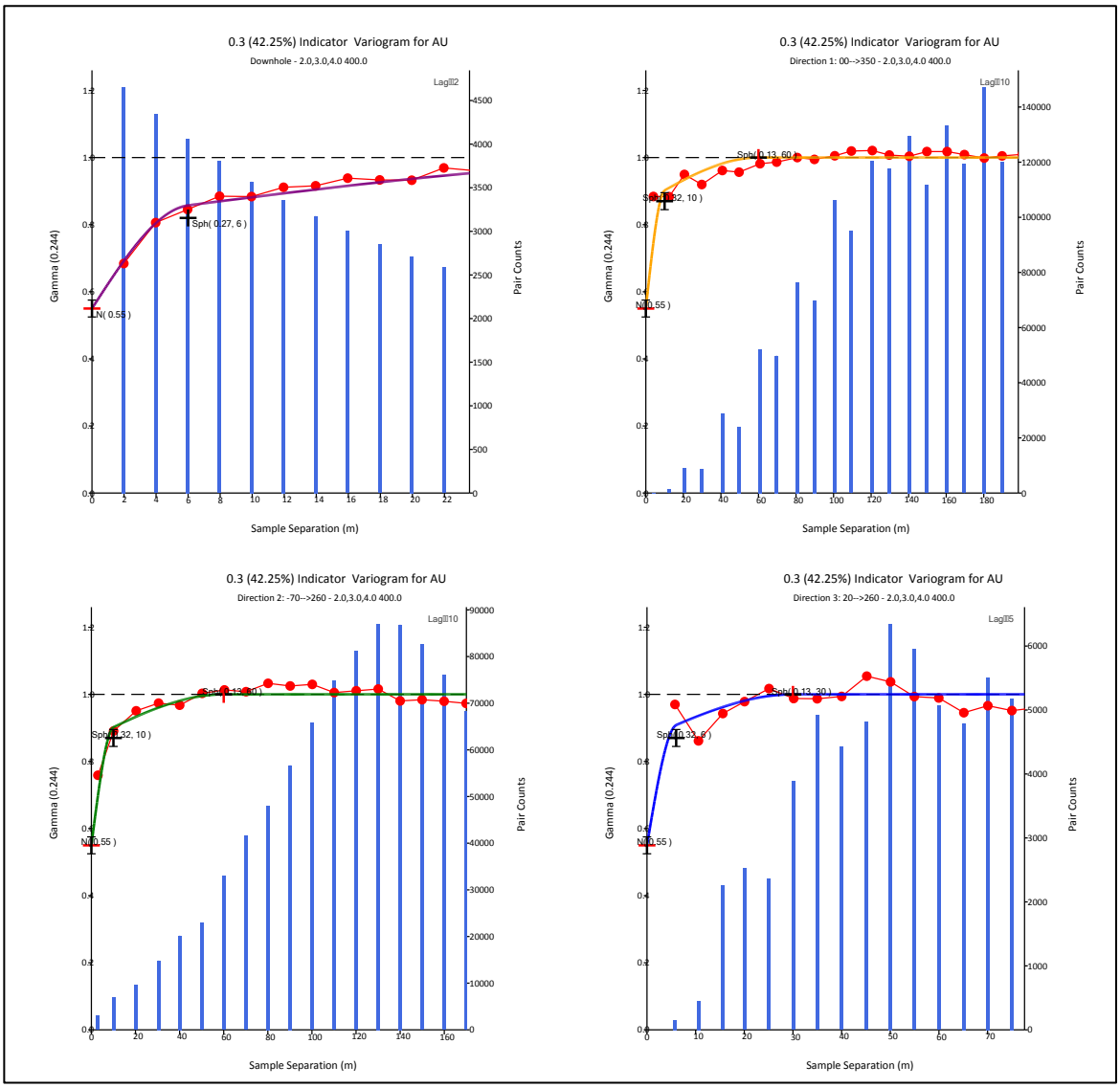
Laterite		
Oxide		
Transitional		
Fresh		

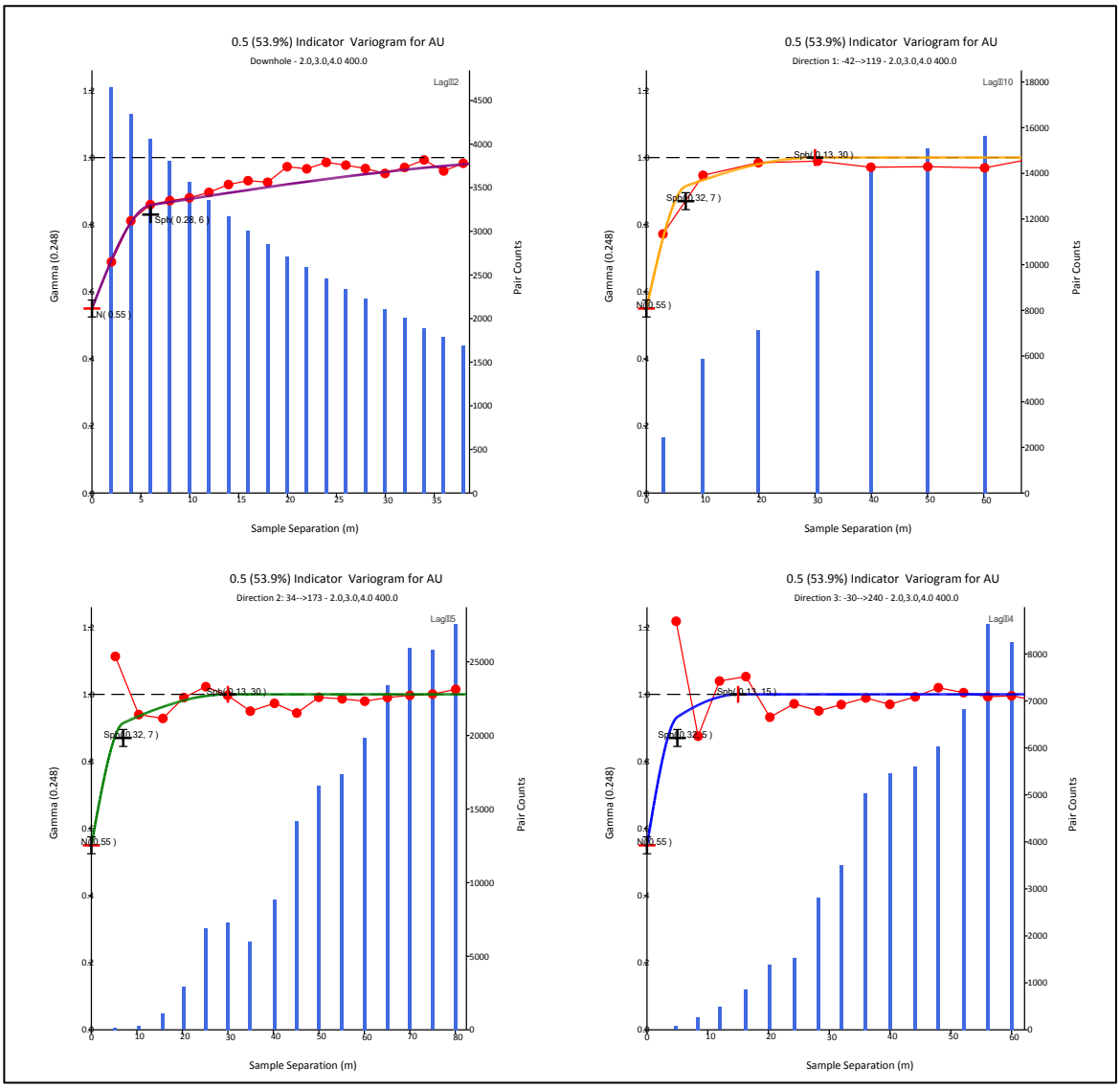
Appendix D Indicator variogram models – granites and sediments

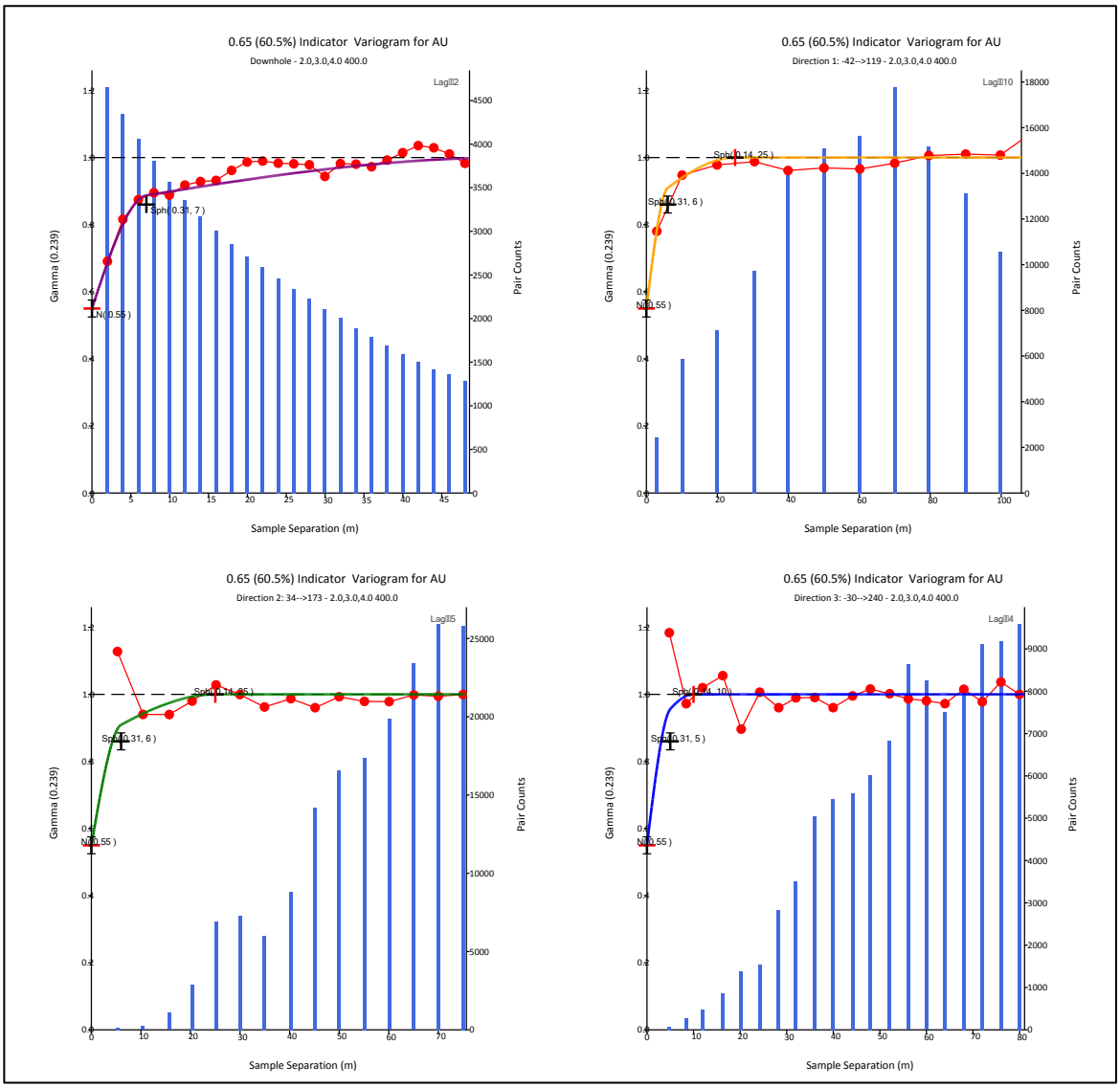
Southern sediments

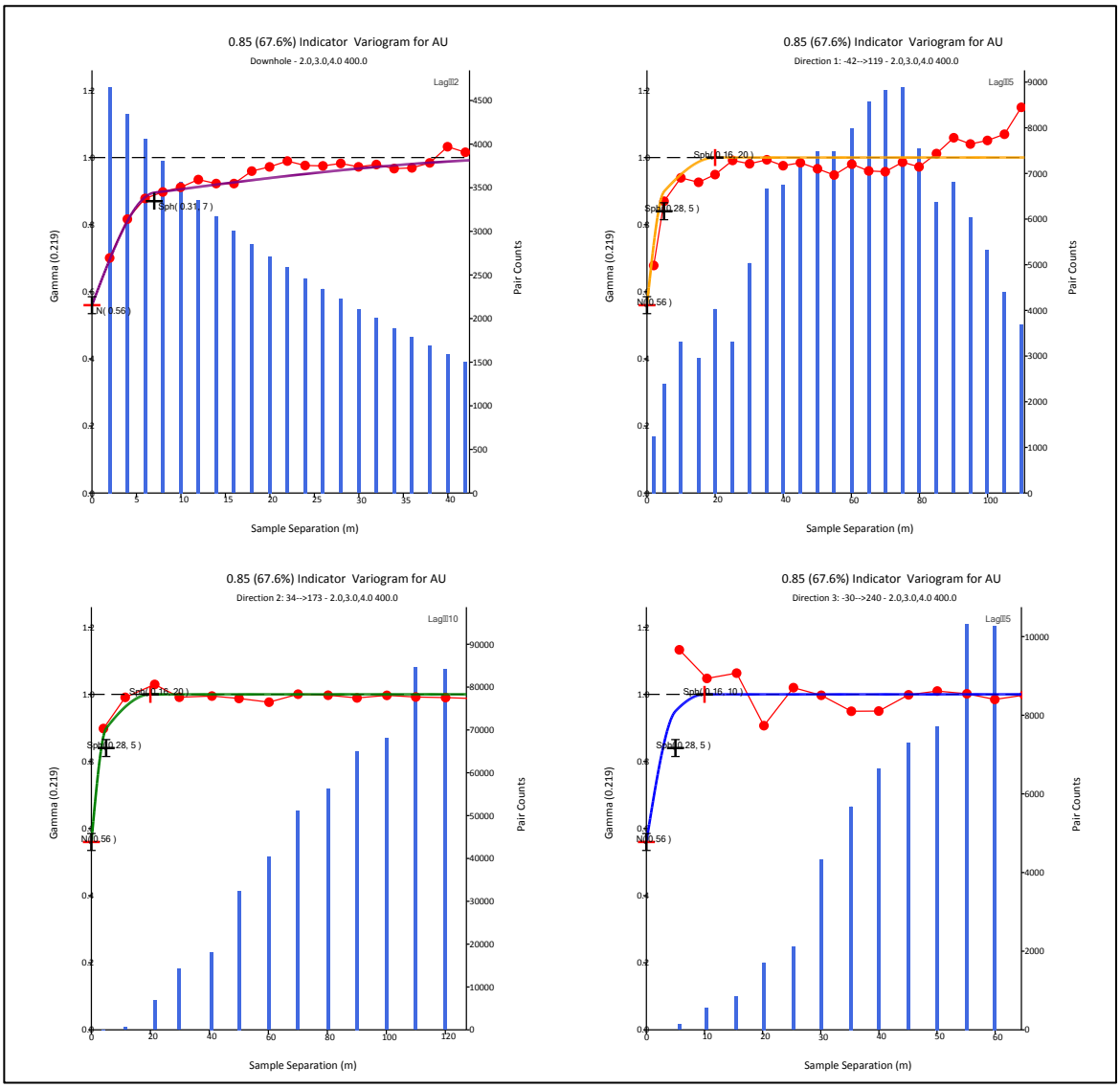


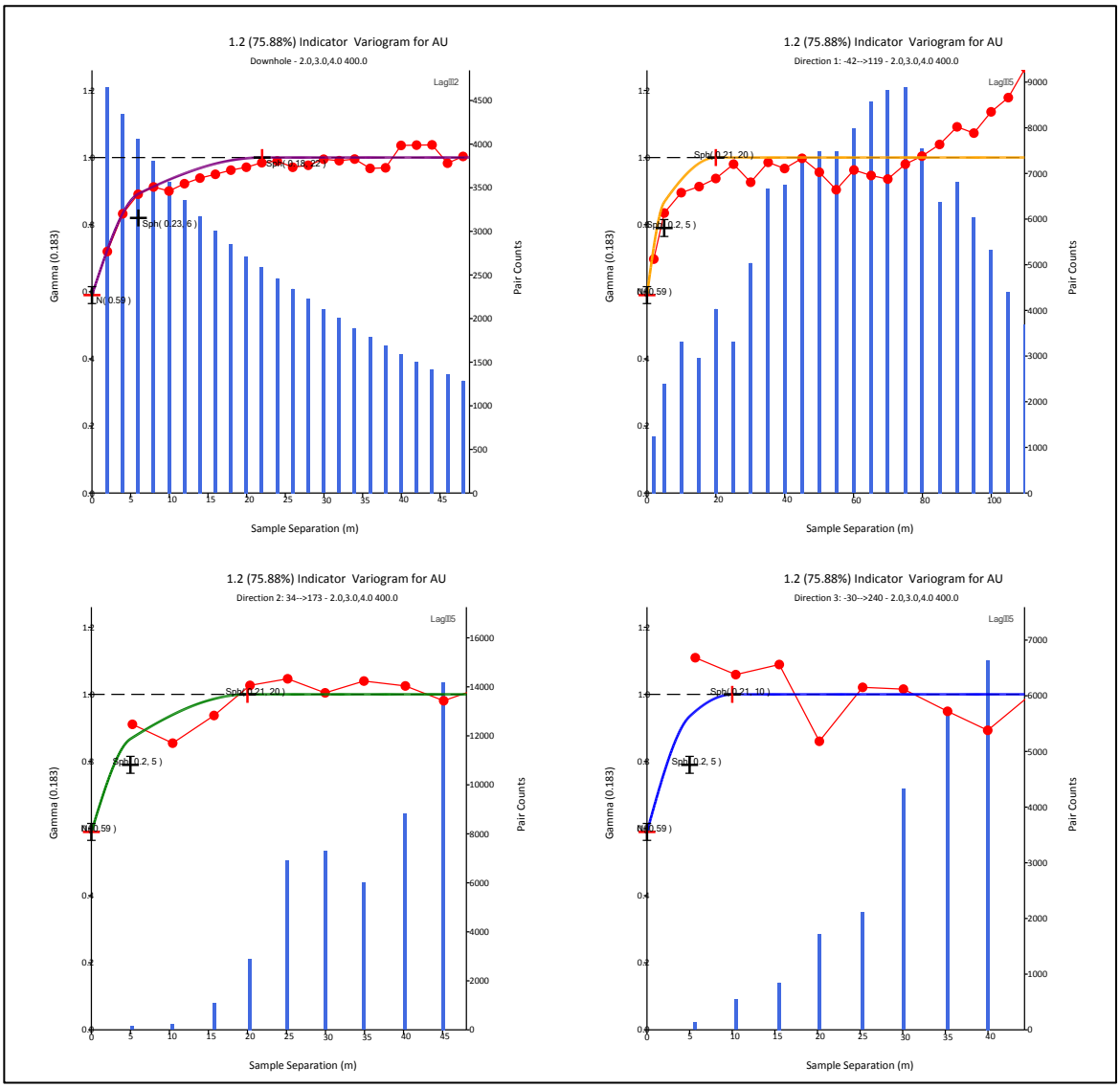

0.1 (24.3%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag⊞2
Sph(0.22, 54)
Sph(0.29, 6)
Nug(0.49)
Gamma (0.184)
Pair Counts
Sample Separation (m)
0.1 (24.3%) Indicator Variogram for AU
Direction 1: 00-->350 - 2.0,3.0,4.0 400.0
Lag⊞15
Sph(0.23, 70)
Sph(0.28, 10)
0.49)
Gamma (0.184)
Pair Counts
Sample Separation (m)
0.1 (24.3%) Indicator Variogram for AU
Direction 2: -70-->260 - 2.0,3.0,4.0 400.0
Lag⊞10
Sph(0.23, 70)
Sph(0.28, 10)
Nug(0.49)
Gamma (0.184)
Pair Counts
Sample Separation (m)
0.1 (24.3%) Indicator Variogram for AU
Direction 3: 20-->260 - 2.0,3.0,4.0 400.0
Lag⊞5
Sph(0.23, 40)
Sph(0.28, 10)
Nug(0.49)
Gamma (0.184)
Pair Counts
Sample Separation (m)

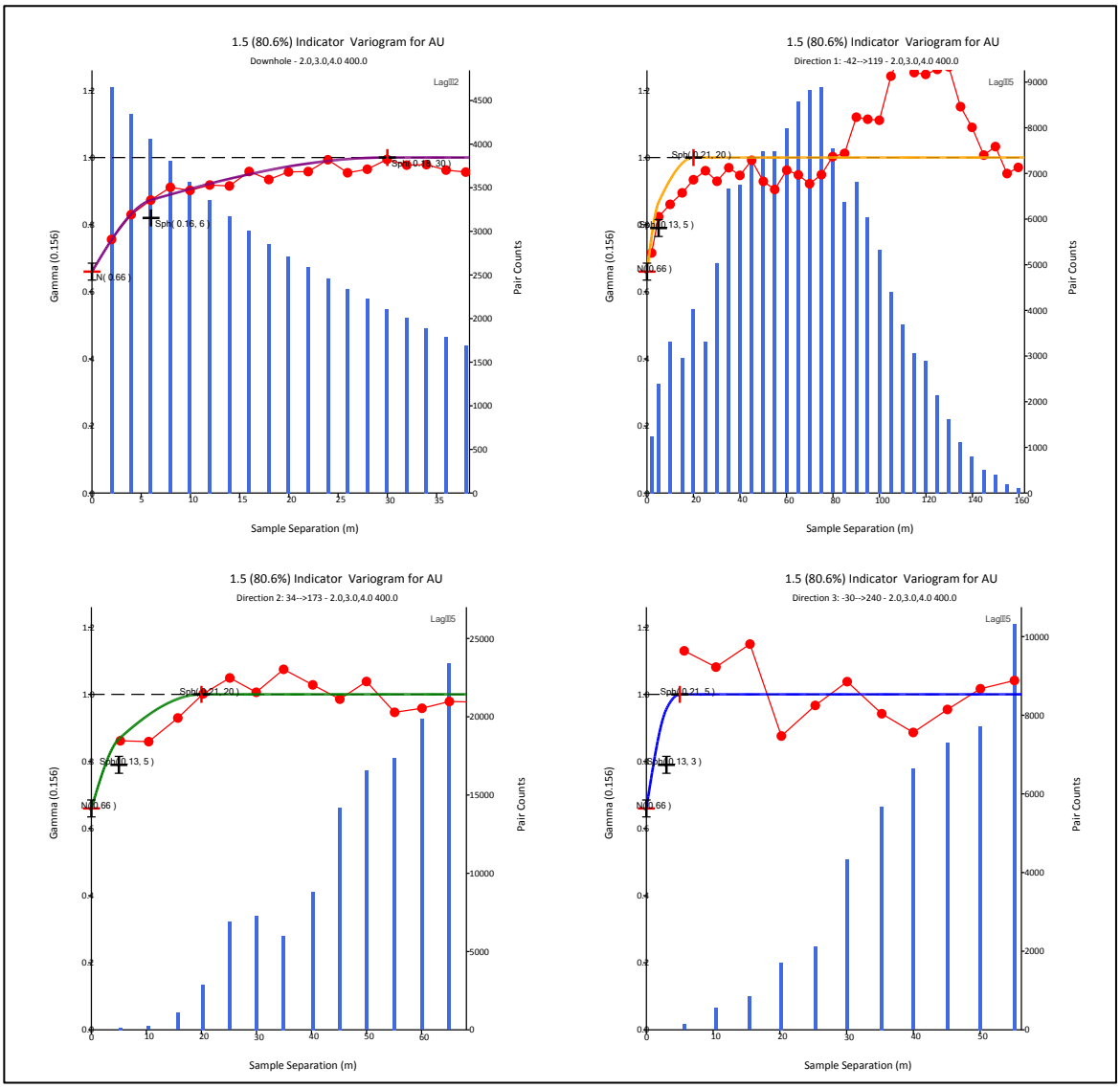

0.3 (42.25%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag 2
Nug(0.55)
Sph(0.27, 6)
Gamma (0.244)
Pair Counts
Sample Separation (m)
0.3 (42.25%) Indicator Variogram for AU
Direction 1: 00-->350 - 2.0,3.0,4.0 400.0
Lag 10
Sph(0.13, 60)
Sph(0.32, 10)
Nug(0.55)
Gamma (0.244)
Pair Counts
Sample Separation (m)
0.3 (42.25%) Indicator Variogram for AU
Direction 2: -70-->260 - 2.0,3.0,4.0 400.0
Lag 10
Sph(0.13, 60)
Sph(0.32, 10)
Nug(0.55)
Gamma (0.244)
Pair Counts
Sample Separation (m)
0.3 (42.25%) Indicator Variogram for AU
Direction 3: 20-->260 - 2.0,3.0,4.0 400.0
Lag 5
Sph(0.13, 20)
Sph(0.32, 8)
Nug(0.55)
Gamma (0.244)
Pair Counts
Sample Separation (m)

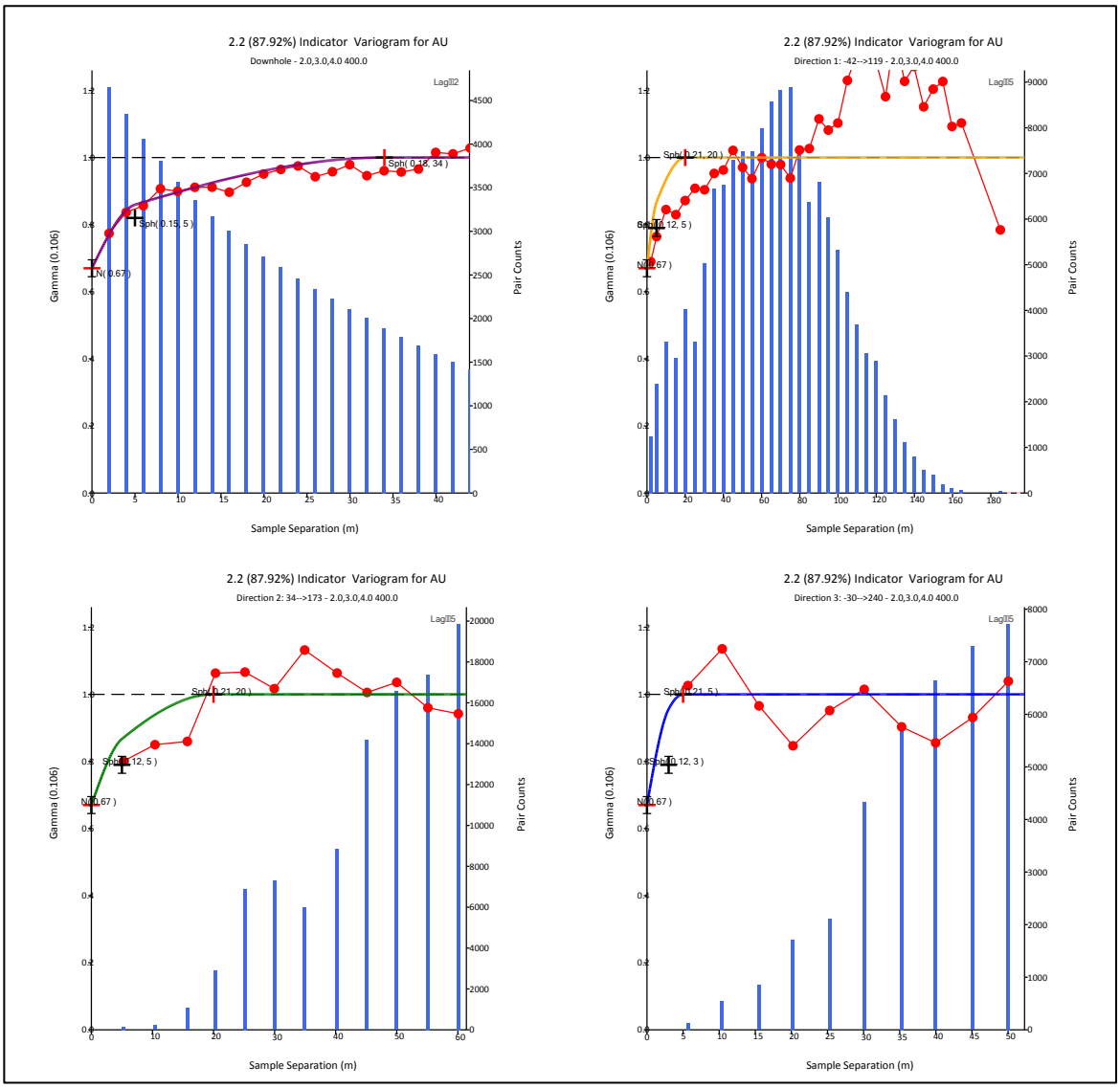

0.5 (53.9%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag 2
Nug(0.55)
Sph(0.28, 6)
Gamma (0.248)
Pair Counts
Sample Separation (m)
0.5 (53.9%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag 10
Sph(0.13, 30)
Sph(0.32, 7)
Nug(0.55)
Gamma (0.248)
Pair Counts
Sample Separation (m)
0.5 (53.9%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag 5
Sph(0.13, 30)
Sph(0.32, 7)
Nug(0.55)
Gamma (0.248)
Pair Counts
Sample Separation (m)
0.5 (53.9%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag 4
Sph(0.13, 15)
Sph(0.32, 5)
Nug(0.55)
Gamma (0.248)
Pair Counts
Sample Separation (m)

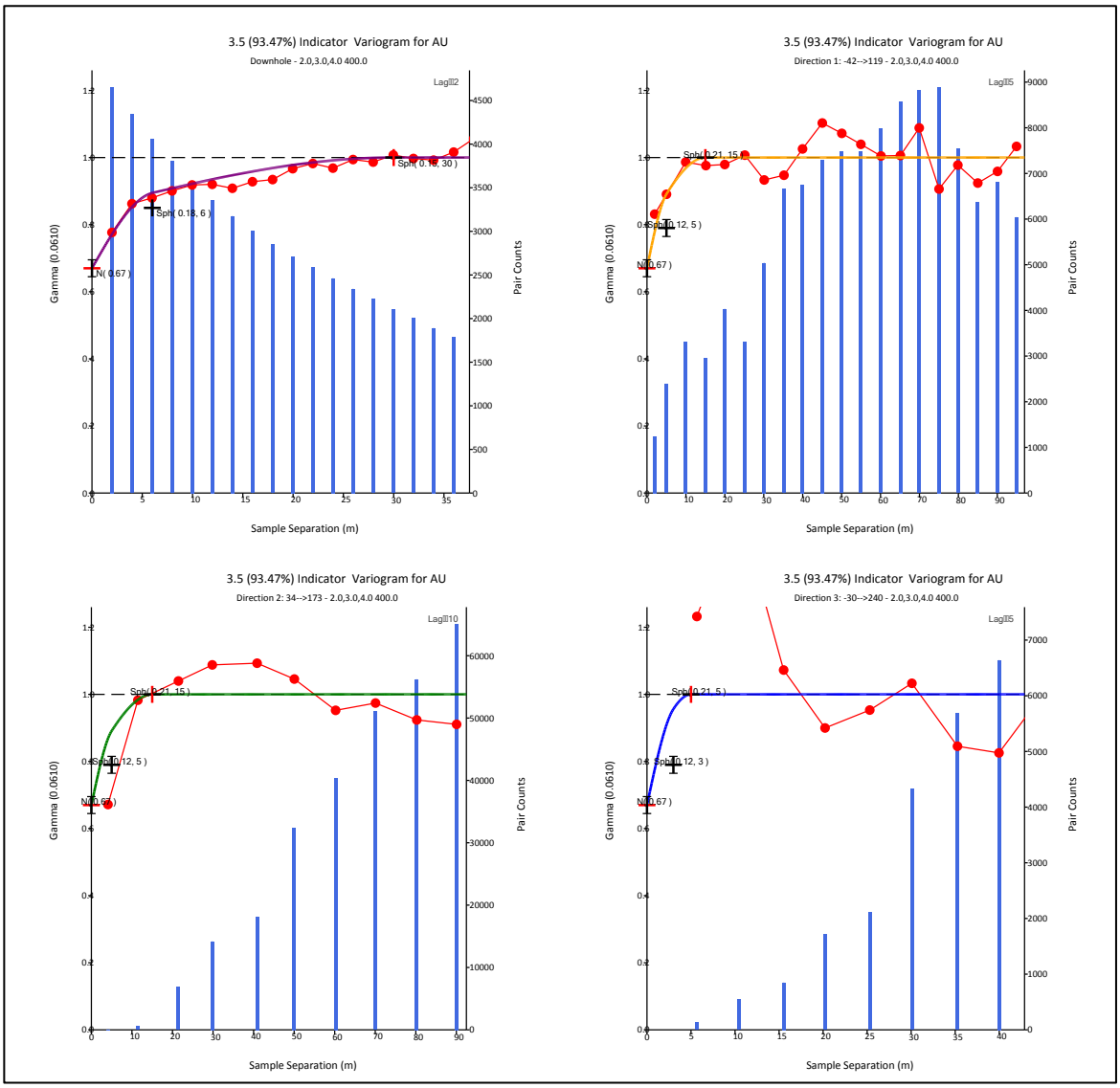

0.65 (60.5%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag 2
Nug(0.55)
Sph(0.33, 7)
Gamma (0.239)
Pair Counts
Sample Separation (m)
0.65 (60.5%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag 10
Nug(0.55)
Sph(0.31, 6)
Sph(0.14, 25)
Gamma (0.239)
Pair Counts
Sample Separation (m)
0.65 (60.5%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag 5
Nug(0.55)
Sph(0.31, 6)
Sph(0.14, 25)
Gamma (0.239)
Pair Counts
Sample Separation (m)
0.65 (60.5%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag 4
Nug(0.55)
Sph(0.31, 5)
Sph(0.24, 10)
Gamma (0.239)
Pair Counts
Sample Separation (m)

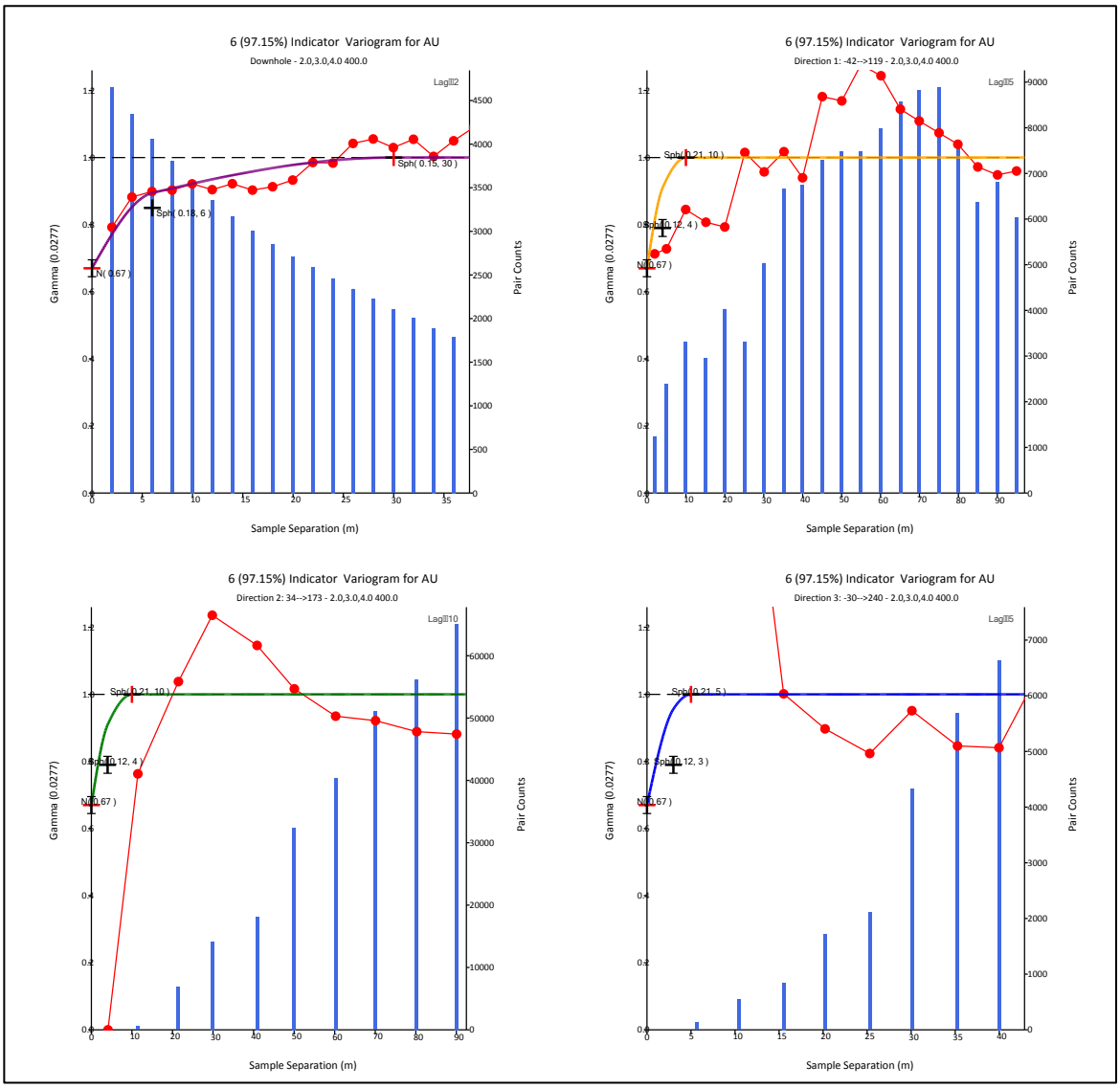

0.85 (67.6%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag= 2
Sph(0.31, 7)
Nug(0.56)
Gamma (0.219)
Pair Counts
Sample Separation (m)
0.85 (67.6%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag= 5
Sph(0.16, 20)
Sph(0.28, 5)
Nug(0.56)
Gamma (0.219)
Pair Counts
Sample Separation (m)
0.85 (67.6%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag= 10
Sph(0.16, 20)
Sph(0.28, 5)
Nug(0.56)
Gamma (0.219)
Pair Counts
Sample Separation (m)
0.85 (67.6%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag= 5
Sph(0.16, 10)
Sph(0.28, 5)
Nug(0.56)
Gamma (0.219)
Pair Counts
Sample Separation (m)

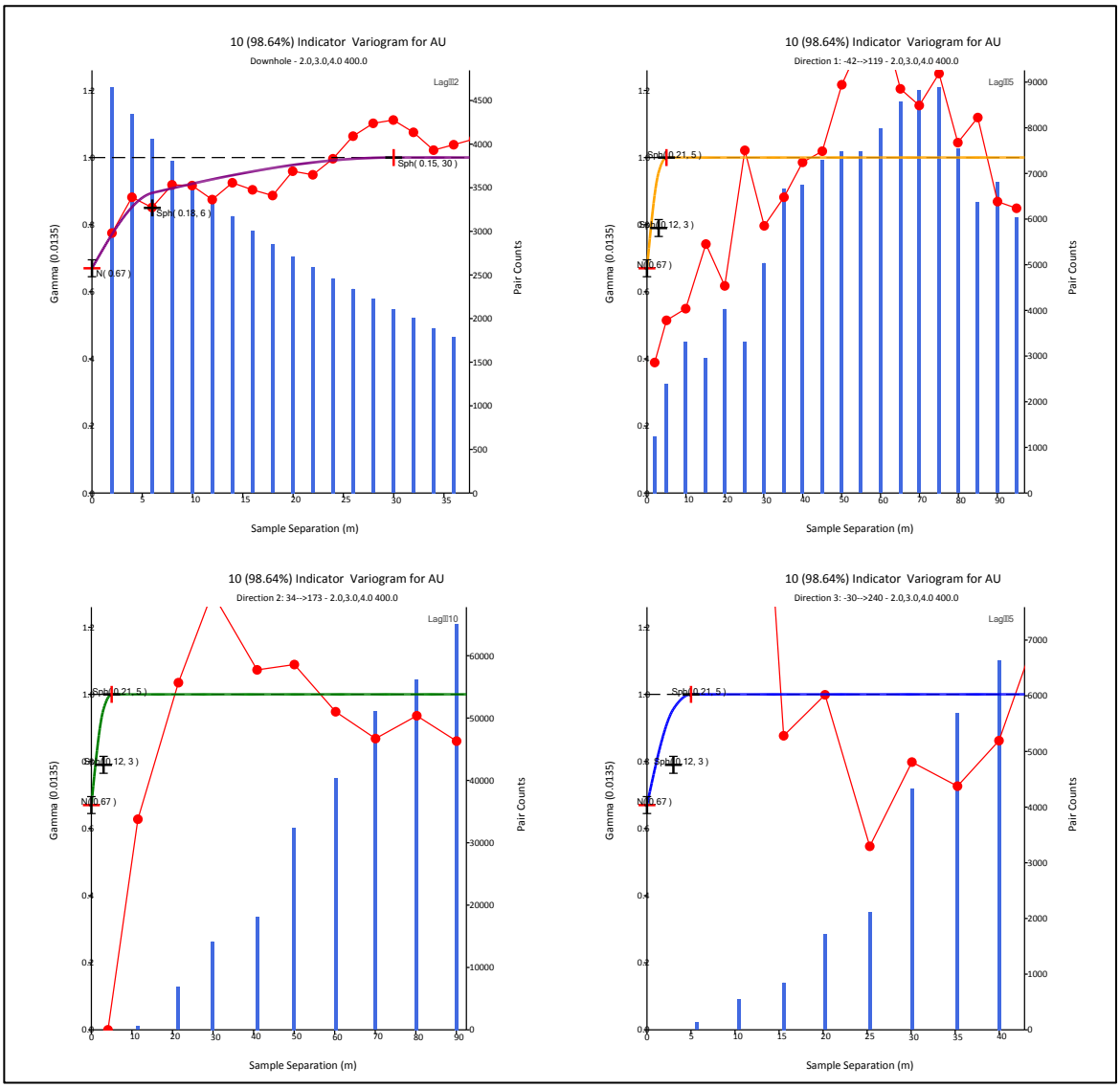

1.2 (75.88%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag||2
Nug(0.59)
Sph(0.23,6)
Sph(0.18,22)
Gamma (0.183)
Pair Counts
Sample Separation (m)
1.2 (75.88%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag||5
Nug(0.59)
Sph(0.2,5)
Sph(0.21,20)
Gamma (0.183)
Pair Counts
Sample Separation (m)
1.2 (75.88%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag||5
Nug(0.59)
Sph(0.2,5)
Sph(0.21,20)
Gamma (0.183)
Pair Counts
Sample Separation (m)
1.2 (75.88%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag||5
Nug(0.59)
Sph(0.2,5)
Sph(0.21,10)
Gamma (0.183)
Pair Counts
Sample Separation (m)


1.5 (80.6%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag= 2
Nug(0.66)
Sph(0.16, 6)
Sph(0.18, 30)
Gamma (0.156)
Pair Counts
Sample Separation (m)
1.5 (80.6%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag= 5
Nug(0.66)
Sph(0.13, 5)
Sph(0.21, 20)
Gamma (0.156)
Pair Counts
Sample Separation (m)
1.5 (80.6%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag= 5
Nug(0.66)
Sph(0.13, 5)
Sph(0.21, 20)
Gamma (0.156)
Pair Counts
Sample Separation (m)
1.5 (80.6%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag= 5
Nug(0.66)
Sph(0.13, 3)
Sph(0.21, 5)
Gamma (0.156)
Pair Counts
Sample Separation (m)

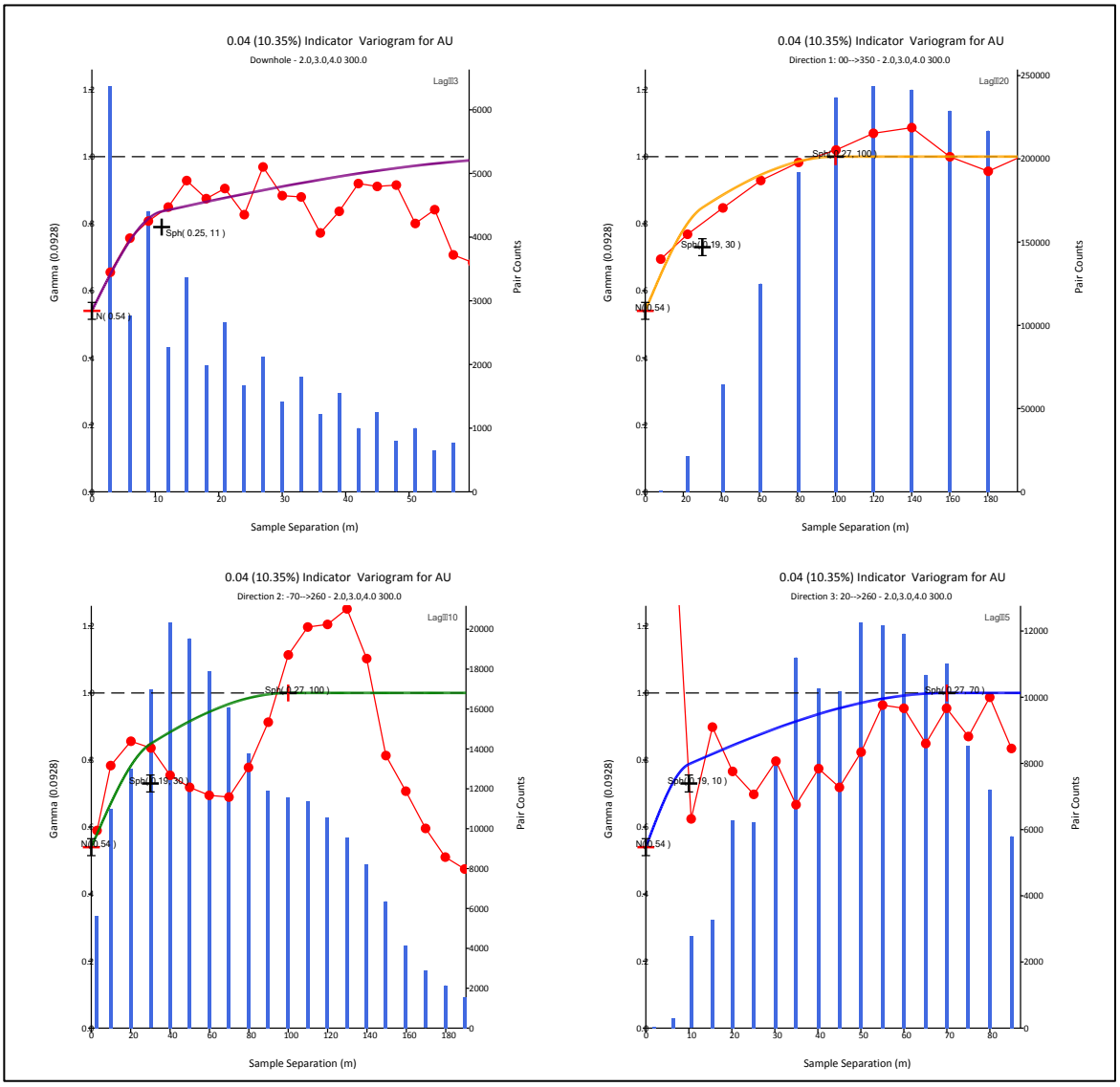

2.2 (87.92%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag# 2
Sph(0.18, 34)
Sph(0.15, 5)
N(0.67)
Gamma (0.106)
Pair Counts
Sample Separation (m)
2.2 (87.92%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag# 5
Sph(0.21, 20)
Sph(0.12, 5)
N(0.67)
Gamma (0.106)
Pair Counts
Sample Separation (m)
2.2 (87.92%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag# 5
Sph(0.21, 20)
Sph(0.12, 5)
N(0.67)
Gamma (0.106)
Pair Counts
Sample Separation (m)
2.2 (87.92%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag# 5
Sph(0.21, 5)
Sph(0.12, 3)
N(0.67)
Gamma (0.106)
Pair Counts
Sample Separation (m)

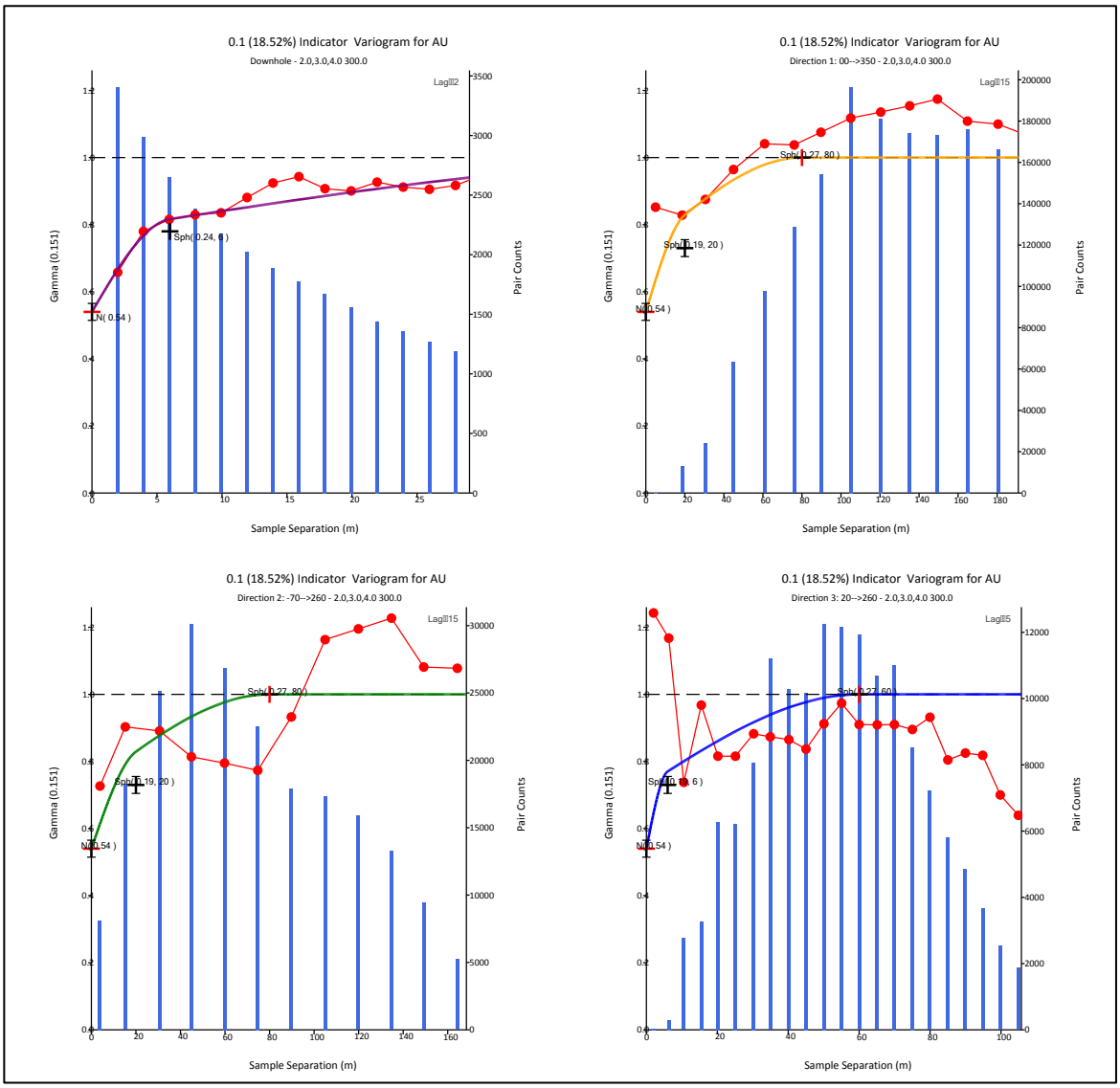

3.5 (93.47%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag 2
Nug(0.67)
Sph(0.18, 6)
Sph(0.18, 30)
Gamma (0.0610)
Pair Counts
Sample Separation (m)
3.5 (93.47%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag 5
Sph(0.21, 15)
Sph(0.12, 5)
Nug(0.67)
Gamma (0.0610)
Pair Counts
Sample Separation (m)
3.5 (93.47%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag 10
Sph(0.21, 15)
Sph(0.12, 5)
Nug(0.67)
Gamma (0.0610)
Pair Counts
Sample Separation (m)
3.5 (93.47%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag 5
Sph(0.21, 5)
Sph(0.12, 3)
Nug(0.67)
Gamma (0.0610)
Pair Counts
Sample Separation (m)

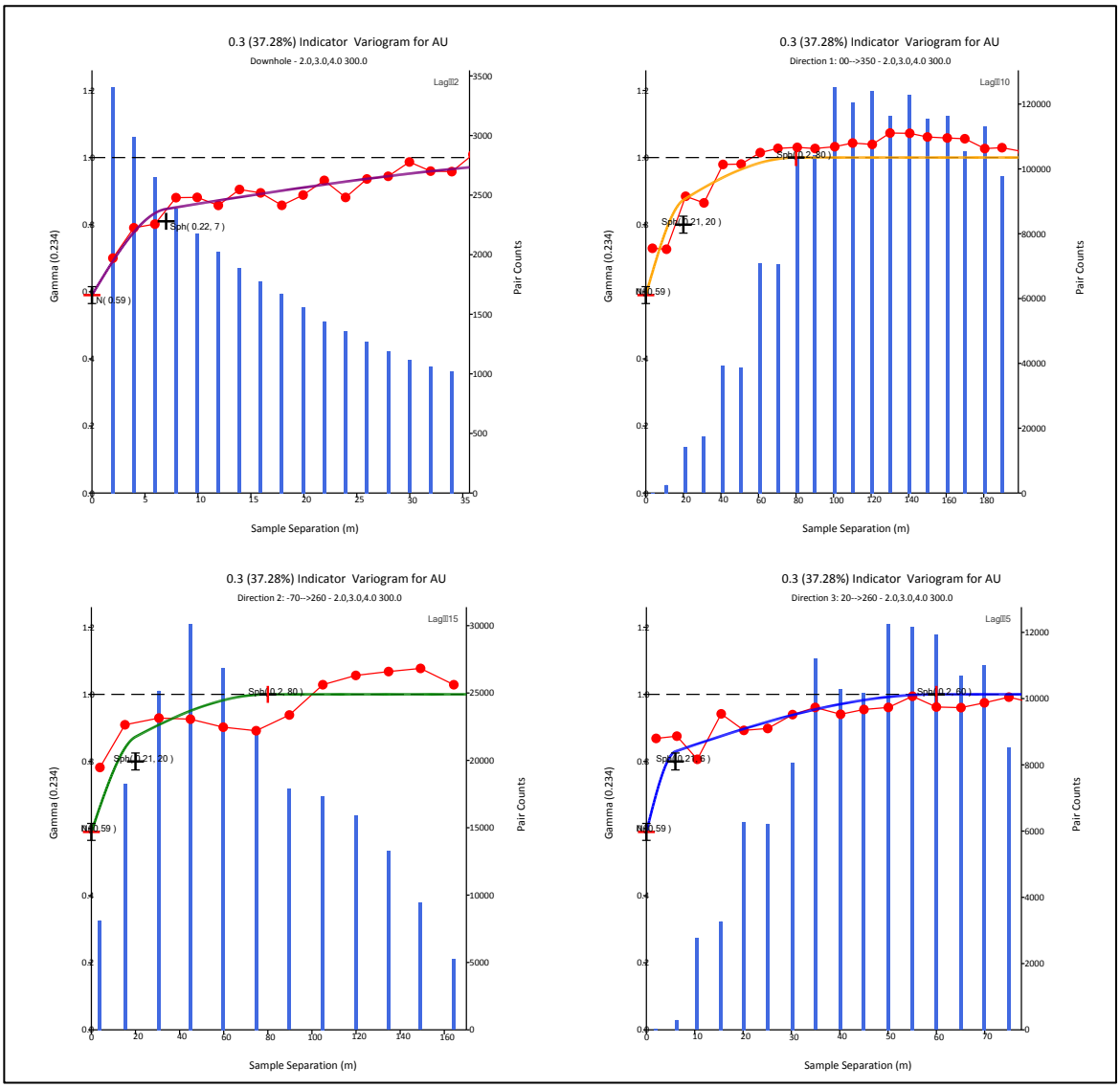

6 (97.15%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag#2
Sph(0.15, 30)
Sph(0.18, 6)
N(0.67)
Gamma (0.0277)
Pair Counts
Sample Separation (m)
6 (97.15%) Indicator Variogram for AU
Direction 1: -42->119 - 2.0,3.0,4.0 400.0
Lag#5
Sph(0.21, 10)
Sph(0.12, 4)
N(0.67)
Gamma (0.0277)
Pair Counts
Sample Separation (m)
6 (97.15%) Indicator Variogram for AU
Direction 2: 34->173 - 2.0,3.0,4.0 400.0
Lag#10
Sph(0.21, 10)
Sph(0.12, 4)
N(0.67)
Gamma (0.0277)
Pair Counts
Sample Separation (m)
6 (97.15%) Indicator Variogram for AU
Direction 3: -30->240 - 2.0,3.0,4.0 400.0
Lag#5
Sph(0.21, 5)
Sph(0.12, 3)
N(0.67)
Gamma (0.0277)
Pair Counts
Sample Separation (m)

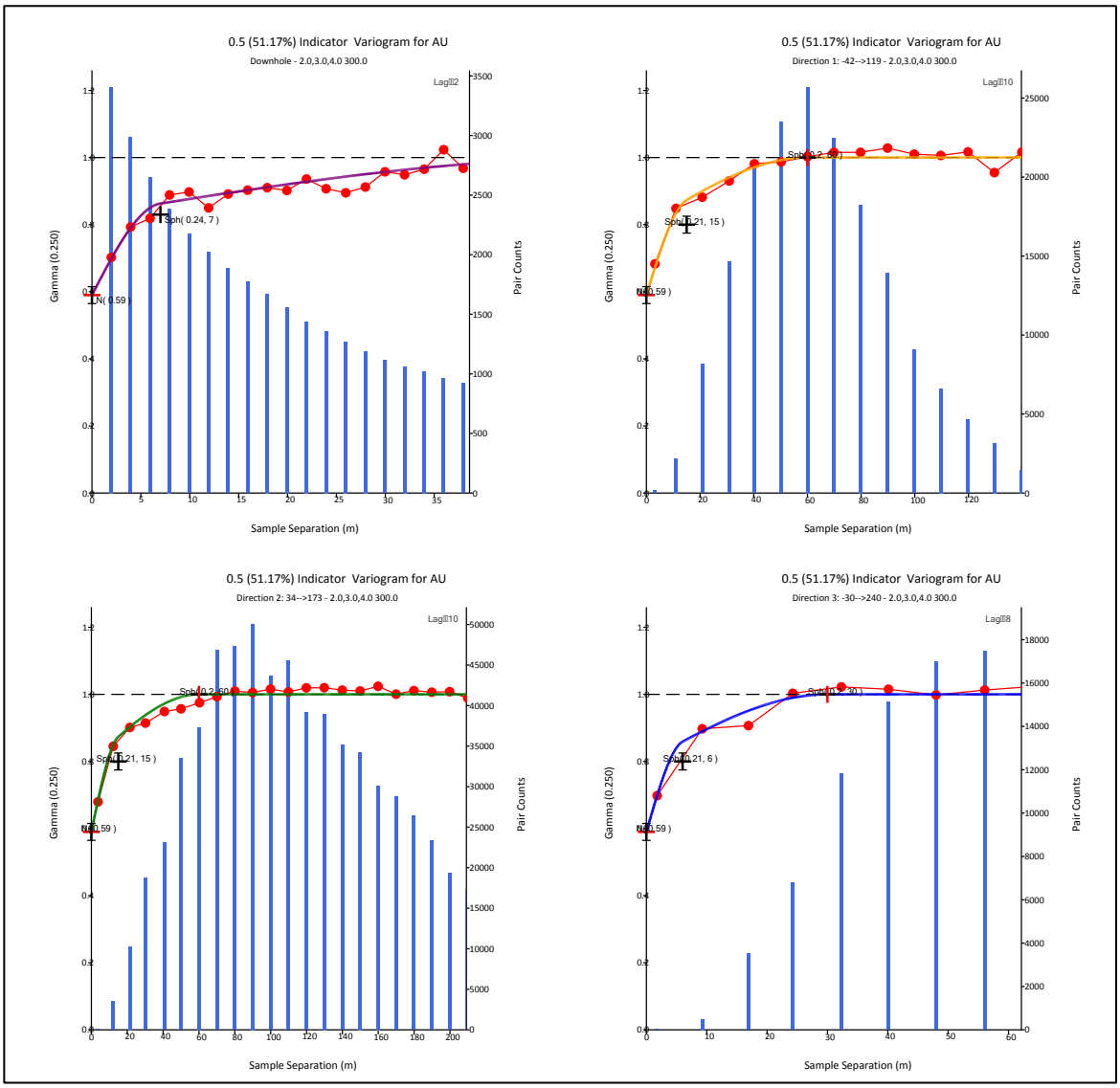

10 (98.64%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 400.0
Lag# 2
Sph(0.15, 30)
Sph(0.18, 6)
N(0.67)
Gamma (0.0135)
Pair Counts
Sample Separation (m)
10 (98.64%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 400.0
Lag# 5
Sph(0.21, 5)
Sph(0.12, 3)
N(0.67)
Gamma (0.0135)
Pair Counts
Sample Separation (m)
10 (98.64%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 400.0
Lag# 10
Sph(0.21, 5)
Sph(0.12, 3)
N(0.67)
Gamma (0.0135)
Pair Counts
Sample Separation (m)
10 (98.64%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 400.0
Lag# 5
Sph(0.21, 5)
Sph(0.12, 3)
N(0.67)
Gamma (0.0135)
Pair Counts
Sample Separation (m)

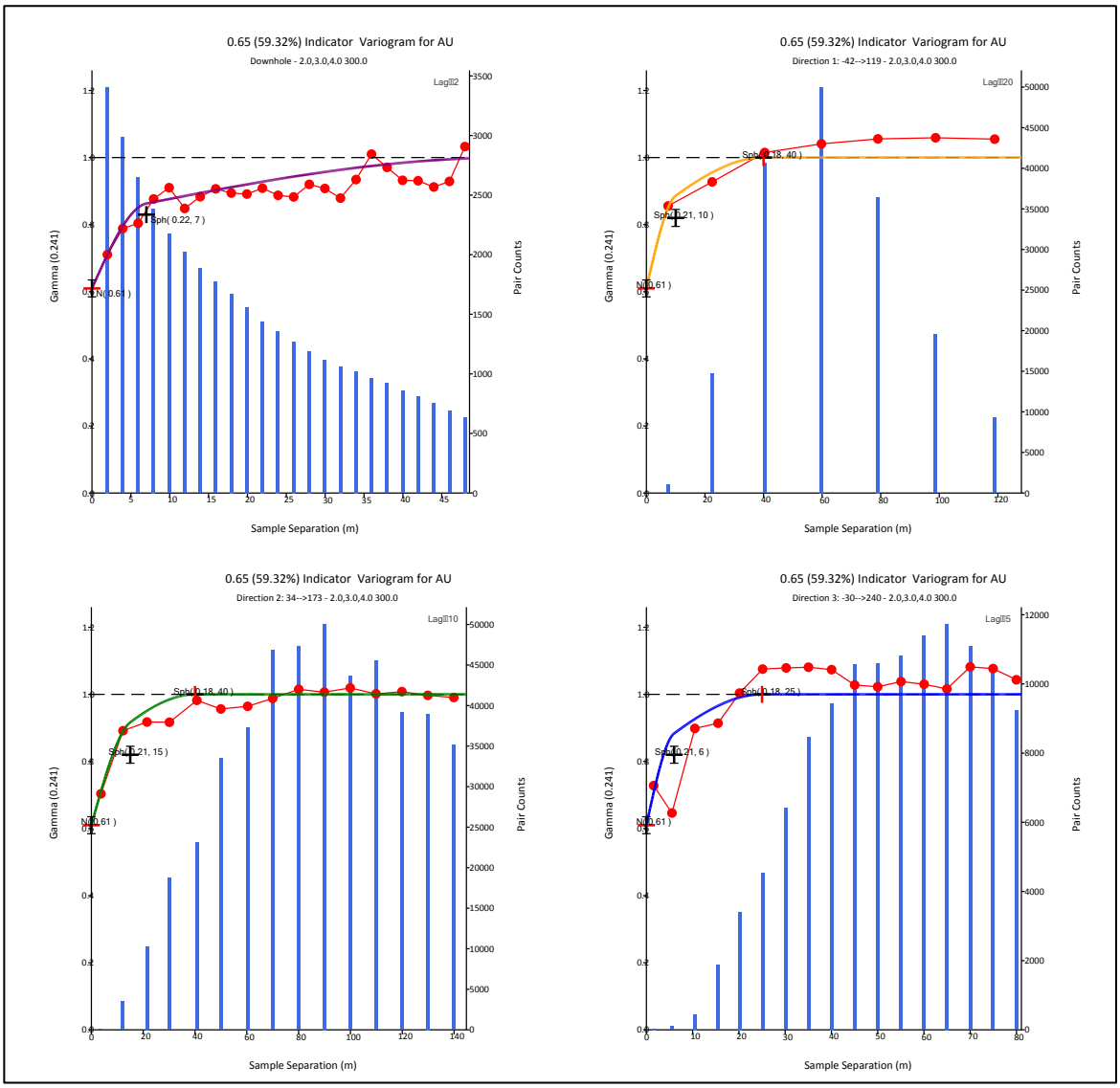
Southern sediments

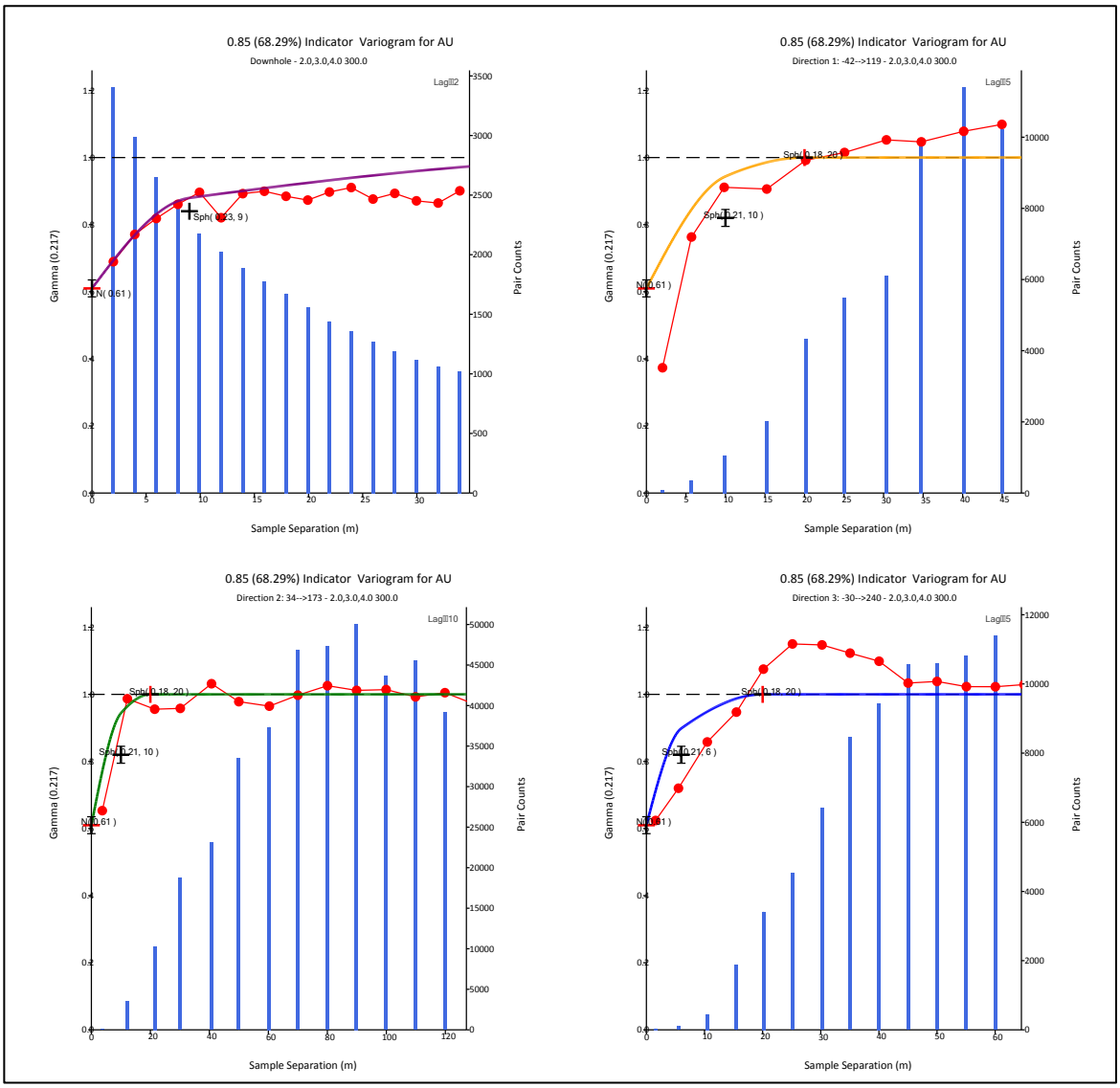


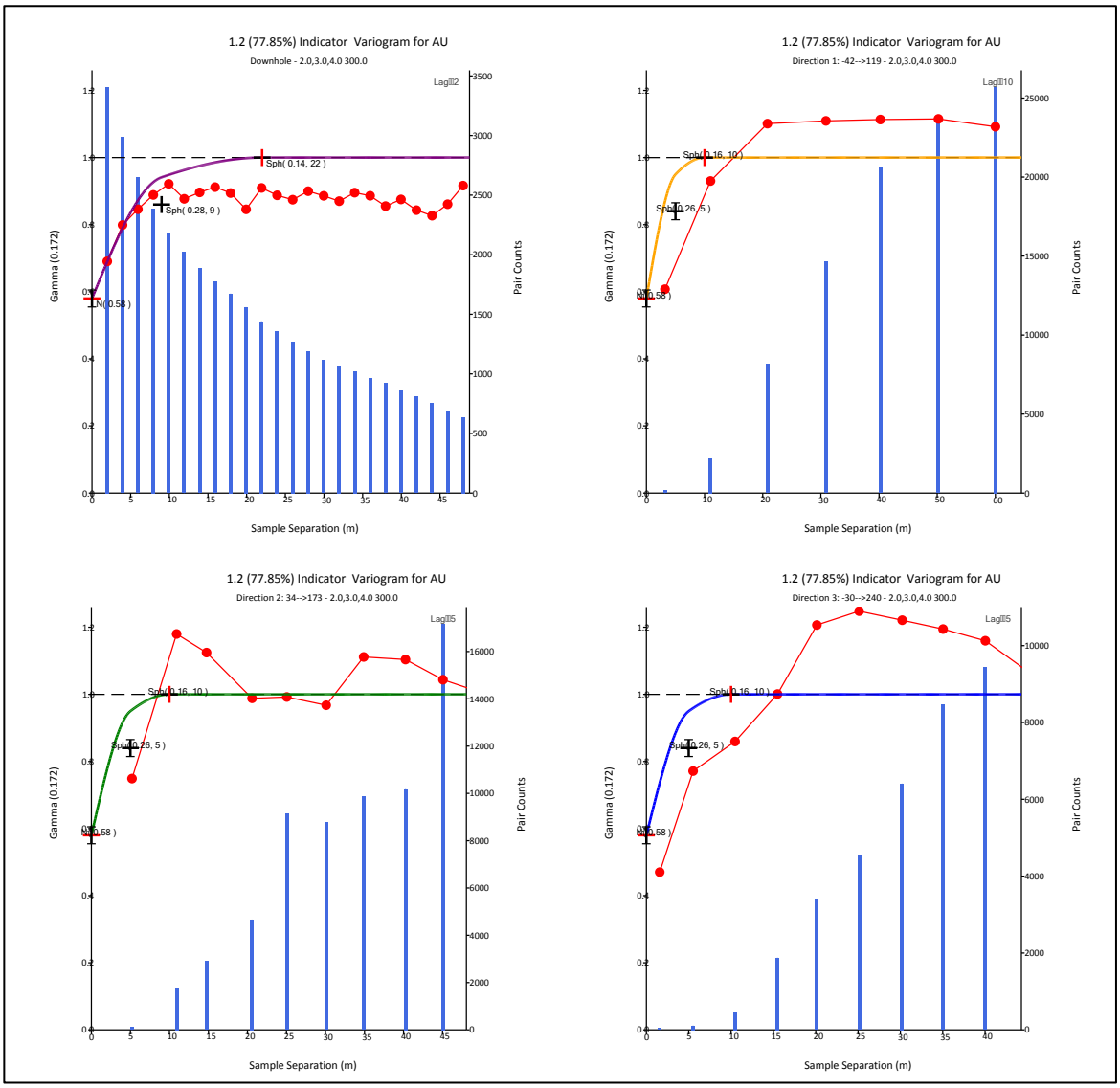

0.1 (18.52%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag= 2
Nug(0.54)
Sph(0.24, 6)
Gamma (0.151)
Pair Counts
Sample Separation (m)
0.1 (18.52%) Indicator Variogram for AU
Direction 1: 00-->350 - 2.0,3.0,4.0 300.0
Lag= 15
Sph(0.27, 80)
Sph(0.19, 20)
Nug(0.54)
Gamma (0.151)
Pair Counts
Sample Separation (m)
0.1 (18.52%) Indicator Variogram for AU
Direction 2: -70-->260 - 2.0,3.0,4.0 300.0
Lag= 15
Sph(0.27, 80)
Sph(0.19, 20)
Nug(0.54)
Gamma (0.151)
Pair Counts
Sample Separation (m)
0.1 (18.52%) Indicator Variogram for AU
Direction 3: 20-->260 - 2.0,3.0,4.0 300.0
Lag= 5
Sph(0.27, 60)
Nug(0.54)
Gamma (0.151)
Pair Counts
Sample Separation (m)

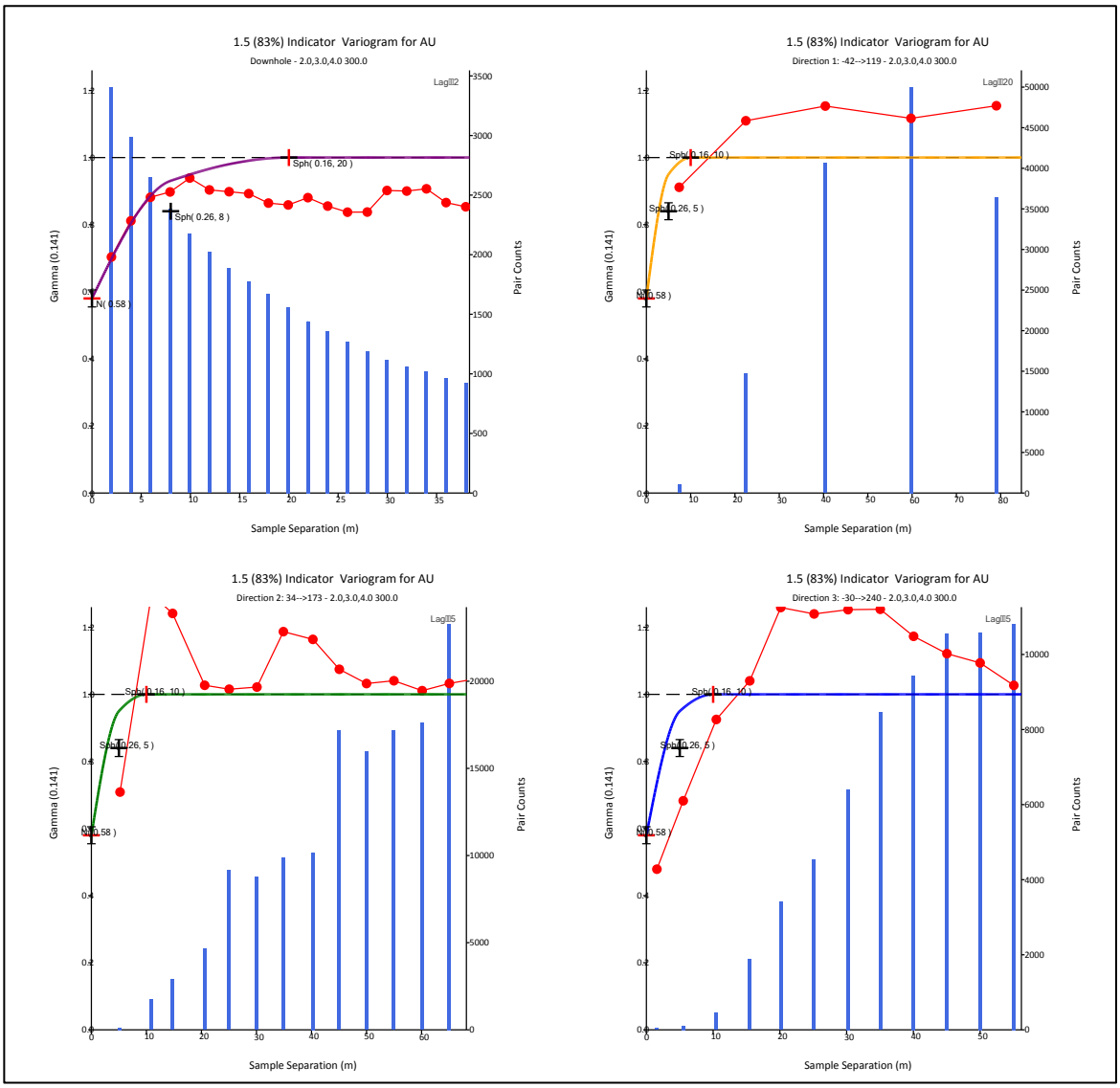

0.3 (37.28%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag≅2
Nug(0.59)
Sph(0.22, 7)
Gamma (0.234)
Pair Counts
Sample Separation (m)
0.3 (37.28%) Indicator Variogram for AU
Direction 1: 00-->350 - 2.0,3.0,4.0 300.0
Lag≅10
Nug(0.59)
Sph(0.21, 20)
Sph(0.2, 80)
Gamma (0.234)
Pair Counts
Sample Separation (m)
0.3 (37.28%) Indicator Variogram for AU
Direction 2: -70-->260 - 2.0,3.0,4.0 300.0
Lag≅15
Nug(0.59)
Sph(0.21, 20)
Sph(0.2, 80)
Gamma (0.234)
Pair Counts
Sample Separation (m)
0.3 (37.28%) Indicator Variogram for AU
Direction 3: 20-->260 - 2.0,3.0,4.0 300.0
Lag≅5
Nug(0.59)
Sph(0.21, 6)
Sph(0.2, 60)
Gamma (0.234)
Pair Counts
Sample Separation (m)

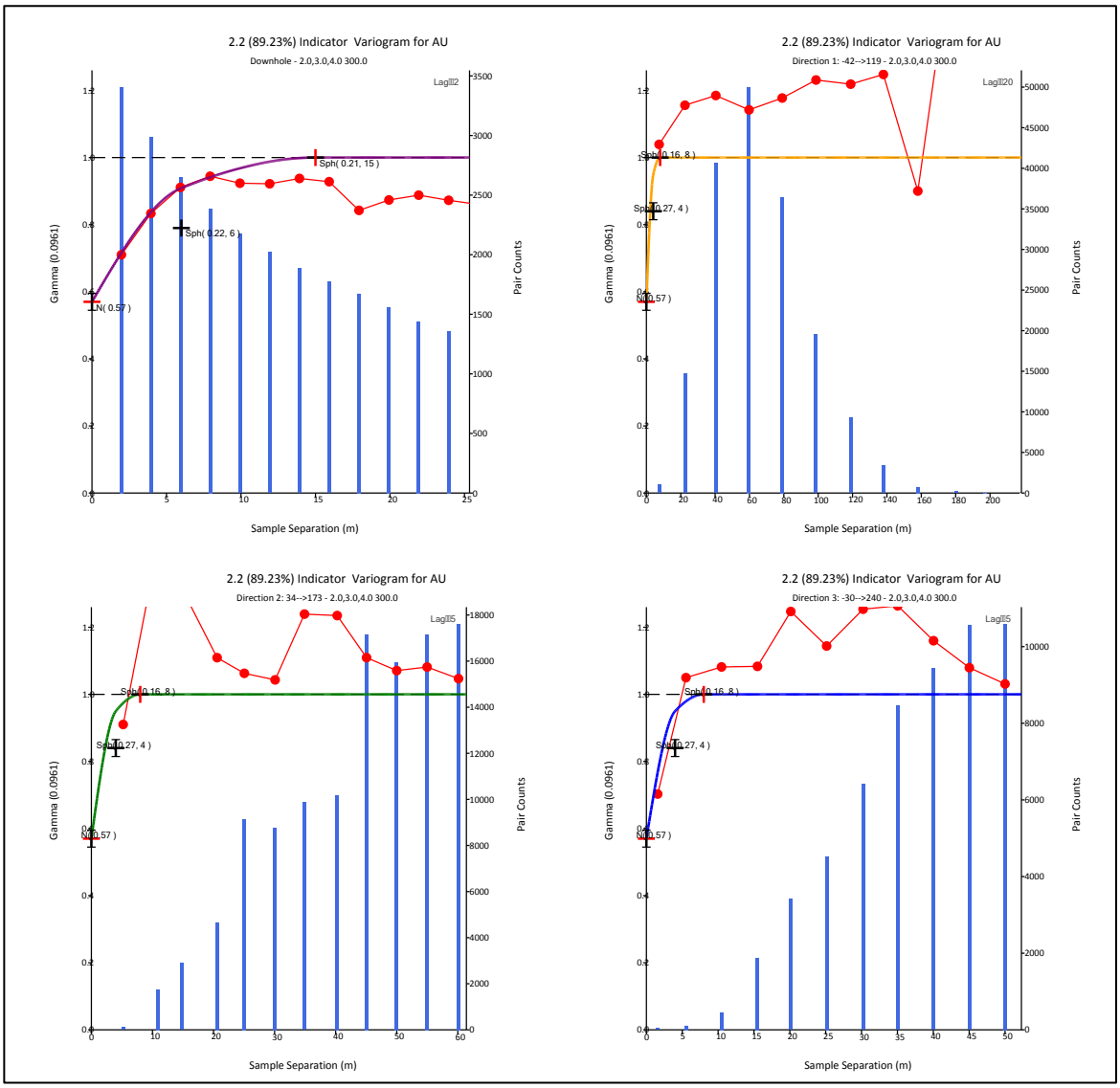

0.5 (51.17%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag≅2
Sph(0.24, 7)
Nug(0.59)
Gamma (0.250)
Pair Counts
Sample Separation (m)
0.5 (51.17%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag≅10
Sph(0.2, 60)
Sph(0.21, 15)
Nug(0.59)
Gamma (0.250)
Pair Counts
Sample Separation (m)
0.5 (51.17%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag≅10
Sph(0.2, 60)
Sph(0.21, 15)
Nug(0.59)
Gamma (0.250)
Pair Counts
Sample Separation (m)
0.5 (51.17%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag≅8
Sph(0.2, 30)
Sph(0.21, 6)
Nug(0.59)
Gamma (0.250)
Pair Counts
Sample Separation (m)

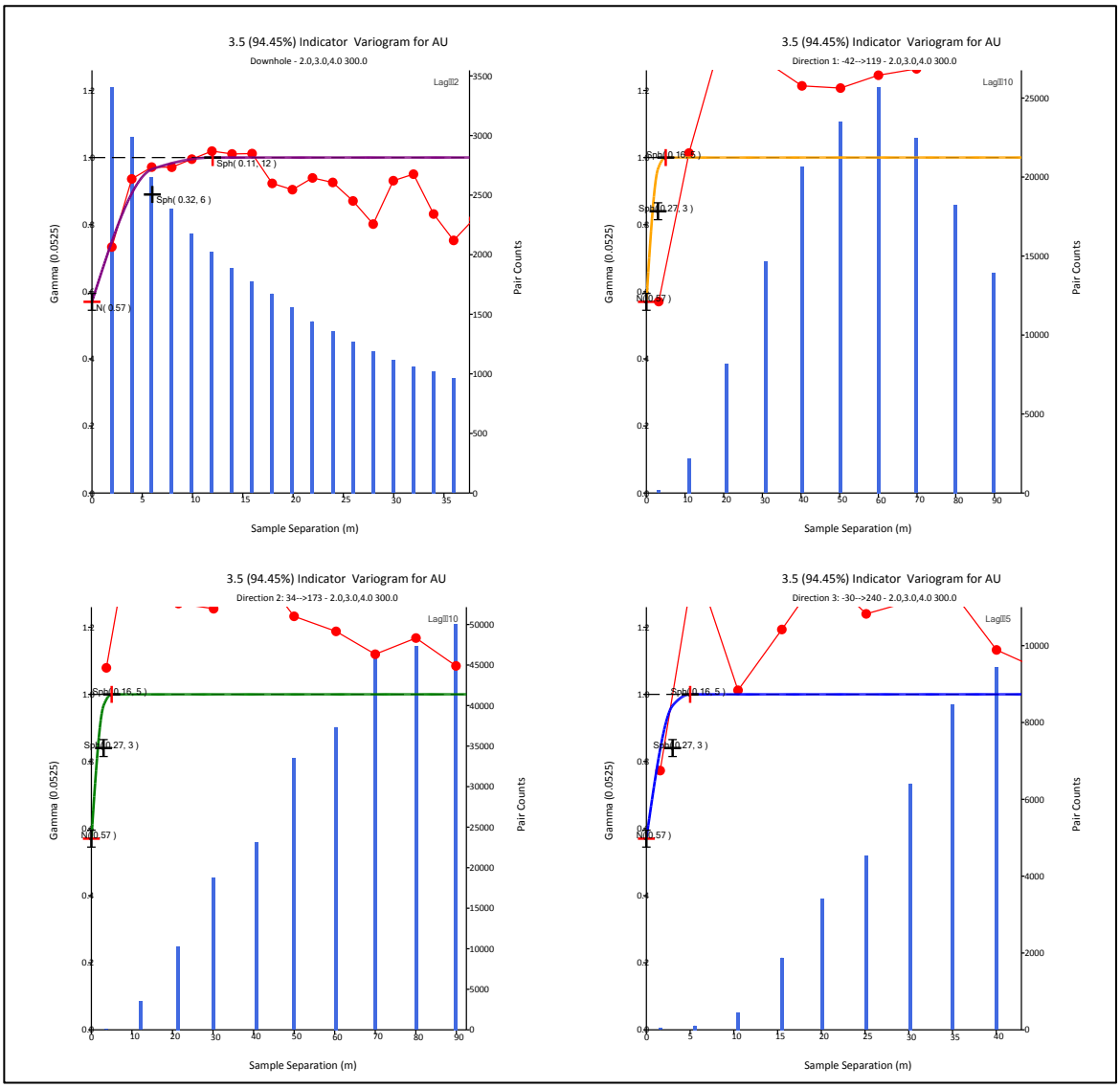

0.65 (59.32%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag 2
Sph(0.22, 7)
Nug(0.61)
Gamma (0.241)
Pair Counts
Sample Separation (m)
0.65 (59.32%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag 20
Sph(0.18, 40)
Sph(0.21, 10)
Nug(0.61)
Gamma (0.241)
Pair Counts
Sample Separation (m)
0.65 (59.32%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag 10
Sph(0.18, 40)
Sph(0.21, 15)
Nug(0.61)
Gamma (0.241)
Pair Counts
Sample Separation (m)
0.65 (59.32%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag 5
Sph(0.18, 25)
Sph(0.21, 6)
Nug(0.61)
Gamma (0.241)
Pair Counts
Sample Separation (m)

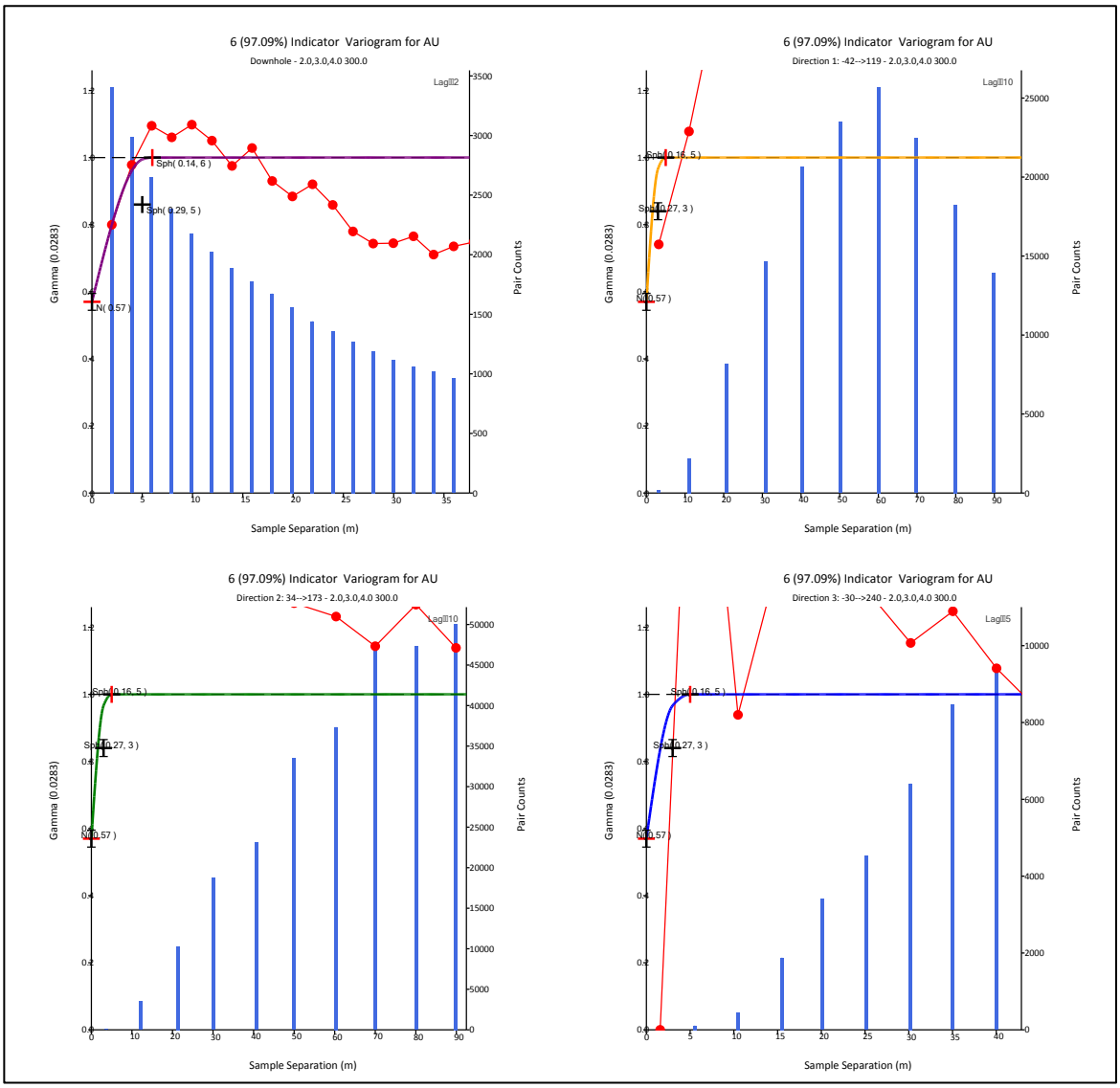

0.85 (68.29%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag# 2
Nug(0.61)
Sph(0.23, 9)
Gamma (0.217)
Pair Counts
Sample Separation (m)
0.85 (68.29%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag# 5
Nug(0.61)
Sph(0.18, 20)
Sph(0.21, 10)
Gamma (0.217)
Pair Counts
Sample Separation (m)
0.85 (68.29%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag# 10
Nug(0.61)
Sph(0.18, 20)
Sph(0.21, 10)
Gamma (0.217)
Pair Counts
Sample Separation (m)
0.85 (68.29%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag# 5
Nug(0.61)
Sph(0.18, 20)
Sph(0.21, 6)
Gamma (0.217)
Pair Counts
Sample Separation (m)

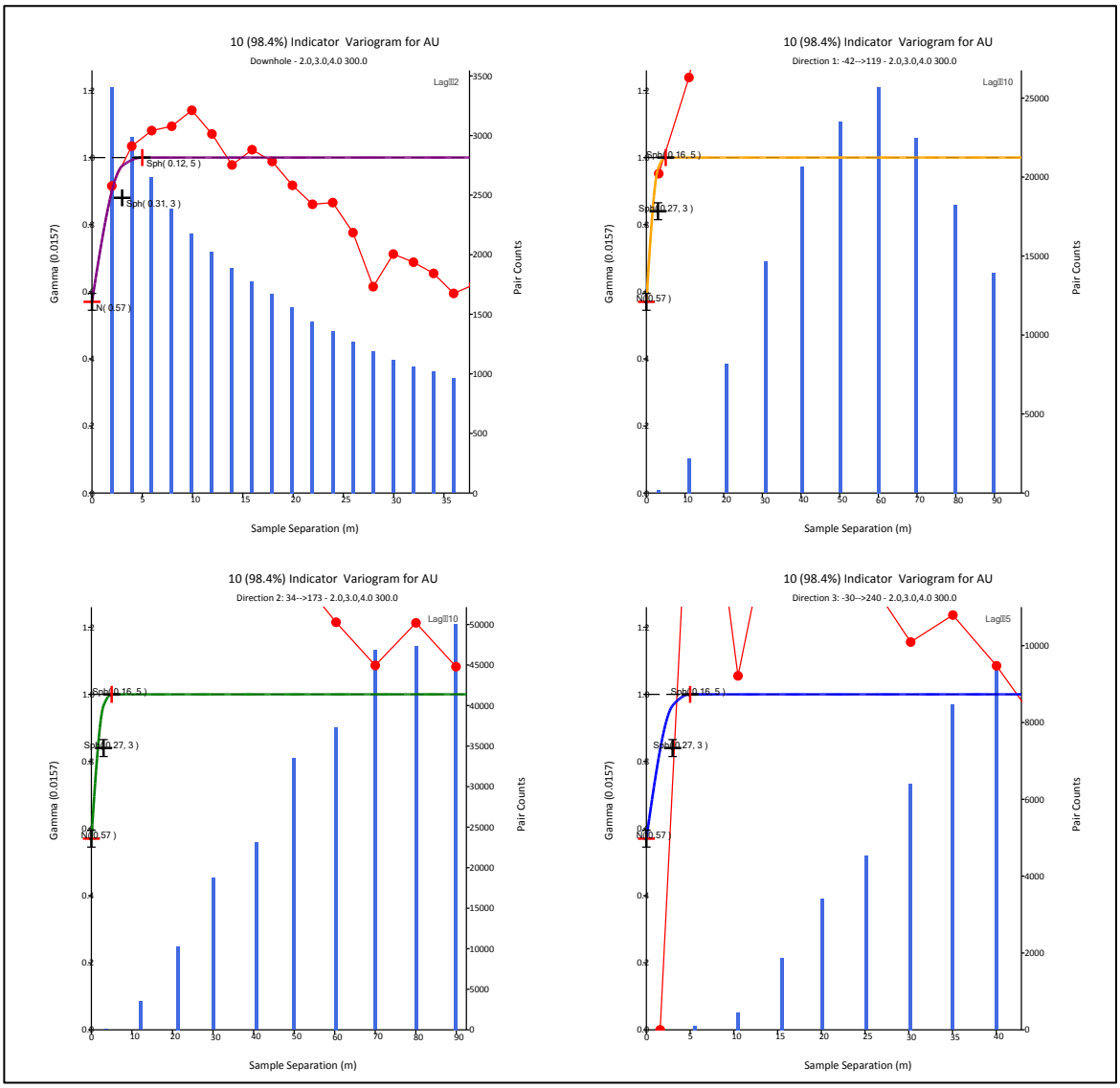

1.2 (77.85%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag 2
Sph(0.14, 22)
Sph(0.28, 9)
Nug(0.58)
Gamma (0.172)
Pair Counts
Sample Separation (m)
1.2 (77.85%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag 10
Sph(0.16, 10)
Sph(0.26, 5)
Nug(0.58)
Gamma (0.172)
Pair Counts
Sample Separation (m)
1.2 (77.85%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag 5
Sph(0.16, 10)
Sph(0.26, 5)
Nug(0.58)
Gamma (0.172)
Pair Counts
Sample Separation (m)
1.2 (77.85%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag 5
Sph(0.16, 10)
Sph(0.26, 5)
Nug(0.58)
Gamma (0.172)
Pair Counts
Sample Separation (m)


1.5 (83%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag⊞2
Sph(0.16, 20)
Sph(0.26, 8)
Nug(0.58)
Gamma (0.141)
Pair Counts
Sample Separation (m)
1.5 (83%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag⊞20
Sph(0.16, 10)
Sph(0.26, 5)
Nug(0.58)
Gamma (0.141)
Pair Counts
Sample Separation (m)
1.5 (83%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag⊞5
Sph(0.16, 10)
Sph(0.26, 5)
Nug(0.58)
Gamma (0.141)
Pair Counts
Sample Separation (m)
1.5 (83%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag⊞5
Sph(0.16, 10)
Sph(0.26, 5)
Nug(0.58)
Gamma (0.141)
Pair Counts
Sample Separation (m)


2.2 (89.23%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag# 2
Sph(0.21, 15)
Sph(0.22, 6)
Nug(0.57)
Gamma (0.0961)
Pair Counts
Sample Separation (m)
2.2 (89.23%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag# 20
Sph(0.16, 8)
Sph(0.27, 4)
Nug(0.57)
Gamma (0.0961)
Pair Counts
Sample Separation (m)
2.2 (89.23%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag# 5
Sph(0.16, 8)
Sph(0.27, 4)
Nug(0.57)
Gamma (0.0961)
Pair Counts
Sample Separation (m)
2.2 (89.23%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag# 5
Sph(0.16, 8)
Sph(0.27, 4)
Nug(0.57)
Gamma (0.0961)
Pair Counts
Sample Separation (m)

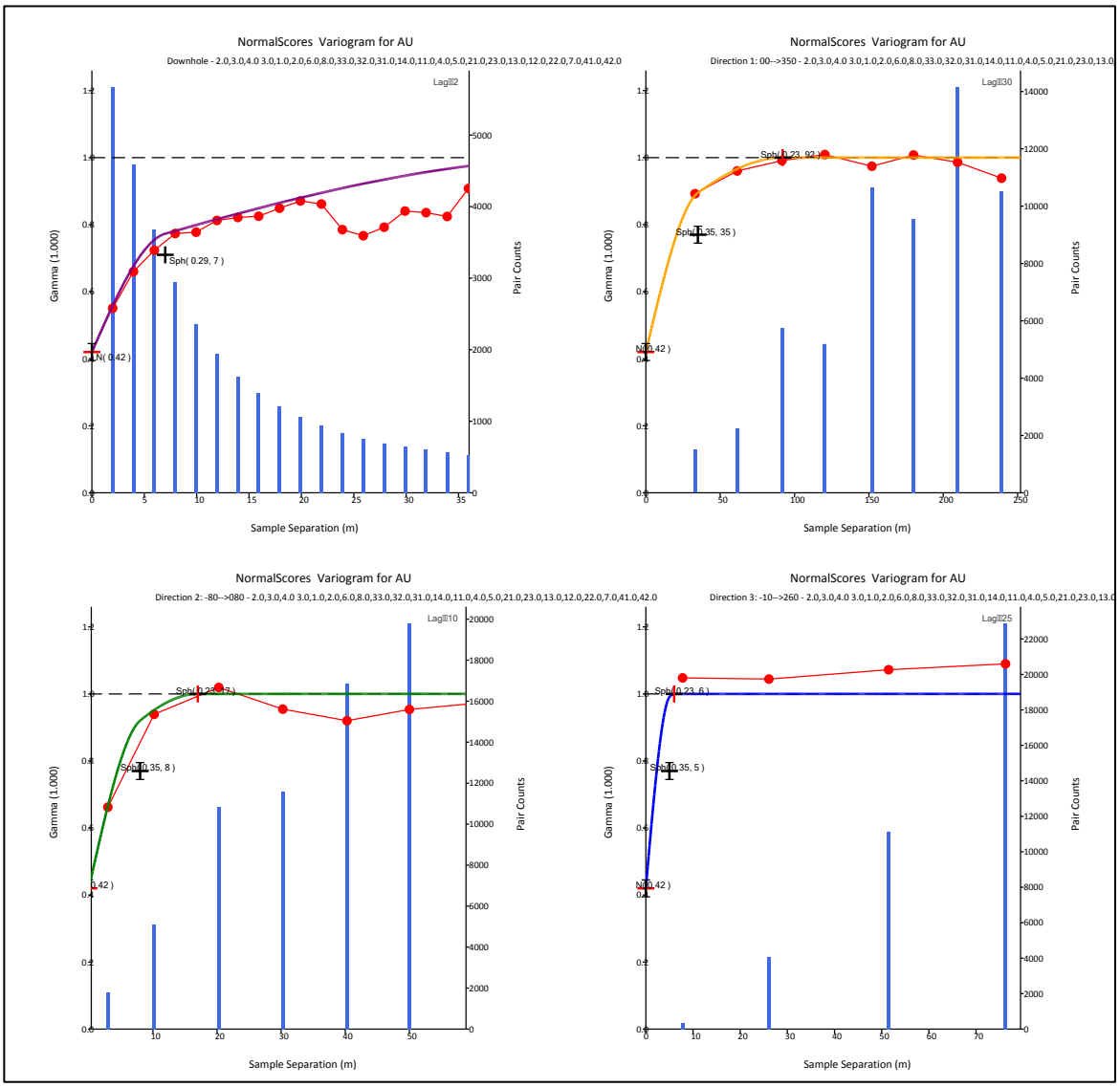

3.5 (94.45%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag= 2
Nug(0.57)
Sph(0.32, 6)
Sph(0.11, 12)
Gamma (0.0525)
Pair Counts
Sample Separation (m)
3.5 (94.45%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag= 10
Nug(0.57)
Sph(0.27, 3)
Sph(0.16, 5)
Gamma (0.0525)
Pair Counts
Sample Separation (m)
3.5 (94.45%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag= 10
Nug(0.57)
Sph(0.27, 3)
Sph(0.16, 5)
Gamma (0.0525)
Pair Counts
Sample Separation (m)
3.5 (94.45%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag= 5
Nug(0.57)
Sph(0.27, 3)
Sph(0.16, 5)
Gamma (0.0525)
Pair Counts
Sample Separation (m)


6 (97.09%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag= 2
Nug(0.57)
Sph(0.14, 6)
Sph(0.29, 5)
Gamma (0.0283)
Pair Counts
Sample Separation (m)
6 (97.09%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag= 10
Sph(0.16, 5)
Sph(0.27, 3)
Nug(0.57)
Gamma (0.0283)
Pair Counts
Sample Separation (m)
6 (97.09%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag= 10
Sph(0.16, 5)
Sph(0.27, 3)
Nug(0.57)
Gamma (0.0283)
Pair Counts
Sample Separation (m)
6 (97.09%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag= 5
Sph(0.16, 5)
Sph(0.27, 3)
Nug(0.57)
Gamma (0.0283)
Pair Counts
Sample Separation (m)


10 (98.4%) Indicator Variogram for AU
Downhole - 2.0,3.0,4.0 300.0
Lag 2
Sph(0.12, 5)
Sph(0.31, 3)
Nug(0.57)
Gamma (0.0157)
Pair Counts
Sample Separation (m)
10 (98.4%) Indicator Variogram for AU
Direction 1: -42-->119 - 2.0,3.0,4.0 300.0
Lag 10
Sph(0.16, 5)
Sph(0.27, 3)
Nug(0.57)
Gamma (0.0157)
Pair Counts
Sample Separation (m)
10 (98.4%) Indicator Variogram for AU
Direction 2: 34-->173 - 2.0,3.0,4.0 300.0
Lag 10
Sph(0.16, 5)
Sph(0.27, 3)
Nug(0.57)
Gamma (0.0157)
Pair Counts
Sample Separation (m)
10 (98.4%) Indicator Variogram for AU
Direction 3: -30-->240 - 2.0,3.0,4.0 300.0
Lag 5
Sph(0.16, 5)
Sph(0.27, 3)
Nug(0.57)
Gamma (0.0157)
Pair Counts
Sample Separation (m)

Appendix E Normal scores models - dykes

Dykes

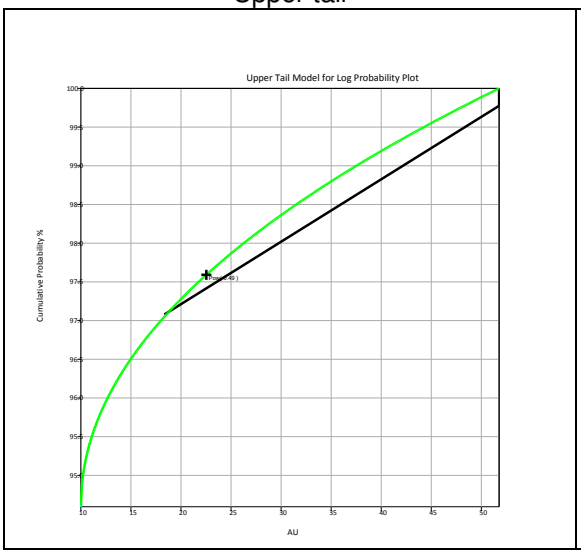

NormalScores Variogram for AU
Downhole - 2.0,3.0,4.0 3.0,1.0,2.0,6.0,8.0,33.0,32.0,31.0,14.0,11.0,4.0,5.0,21.0,23.0,13.0,12.0,22.0,7.0,41.0,42.0
Lag≅2
Sph(0.29, 7)
N(0.42)
Gamma (1.000)
Pair Counts
Sample Separation (m)
NormalScores Variogram for AU
Direction 1: 00-->350 - 2.0,3.0,4.0 3.0,1.0,2.0,6.0,8.0,33.0,32.0,31.0,14.0,11.0,4.0,5.0,21.0,23.0,13.0
Lag≅30
Sph(0.23, 92)
Sph(0.35, 35)
N(0.42)
Gamma (1.000)
Pair Counts
Sample Separation (m)
NormalScores Variogram for AU
Direction 2: -80-->080 - 2.0,3.0,4.0 3.0,1.0,2.0,6.0,8.0,33.0,32.0,31.0,14.0,11.0,4.0,5.0,21.0,23.0,13.0,12.0,22.0,7.0,41.0,42.0
Lag≅10
Sph(0.23, 7)
Sph(0.35, 8)
0.42)
Gamma (1.000)
Pair Counts
Sample Separation (m)
NormalScores Variogram for AU
Direction 3: -10-->260 - 2.0,3.0,4.0 3.0,1.0,2.0,6.0,8.0,33.0,32.0,31.0,14.0,11.0,4.0,5.0,21.0,23.0,13.0
Lag≅25
Sph(0.23, 6)
Sph(0.35, 5)
N(0.42)
Gamma (1.000)
Pair Counts
Sample Separation (m)

Appendix F Upper and lower tail models

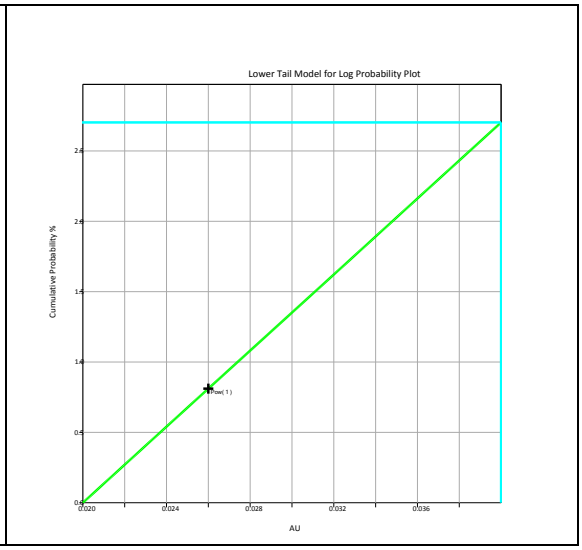
Southern granites

Laterite, oxide and transitional data

Upper tail

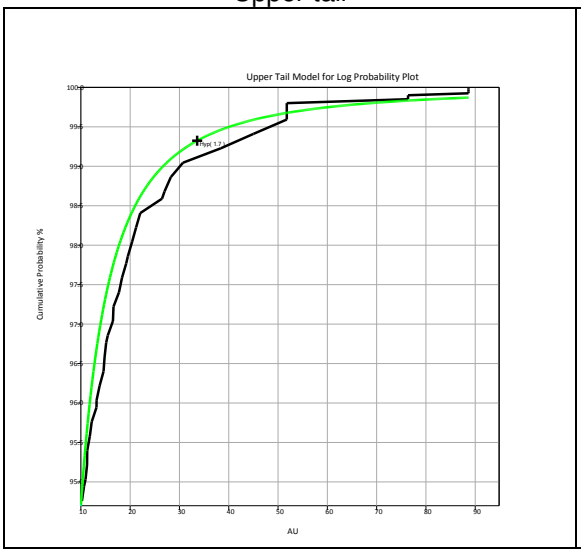


Lower tail

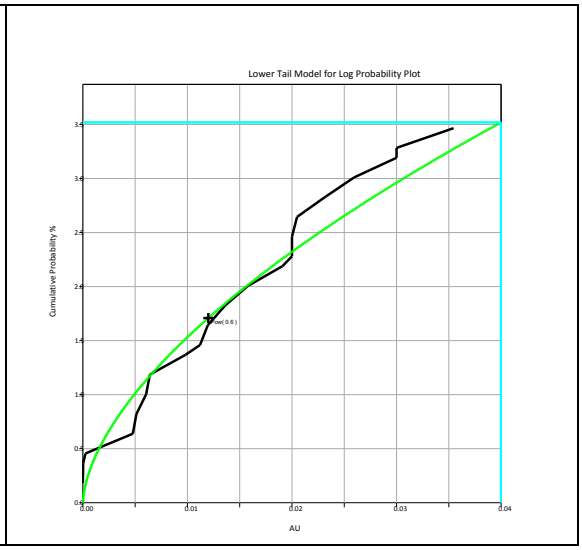


Transitional and fresh data

Upper tail



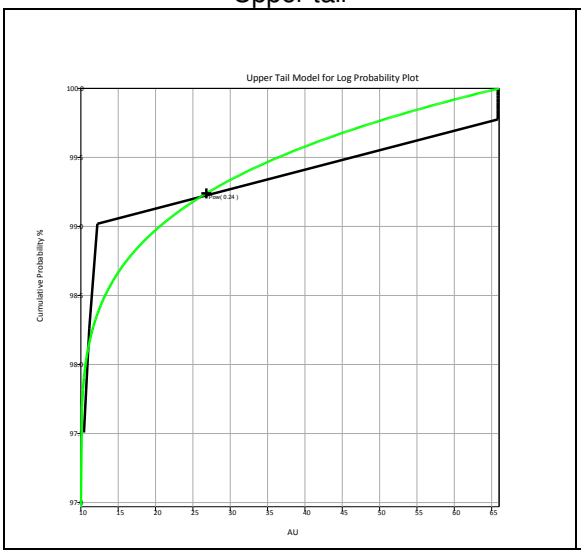
Lower tail



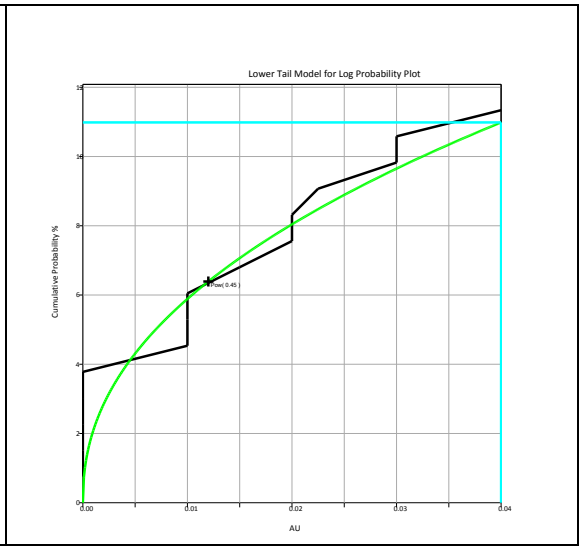
Northern granites

Laterite, oxide and transitional data

Upper tail

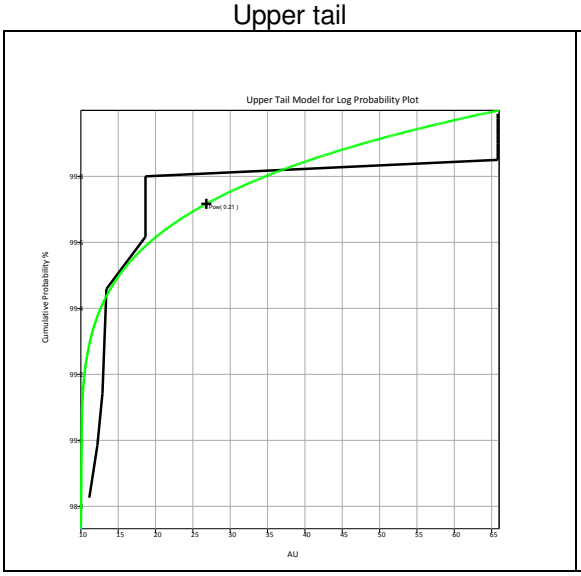


Lower tail

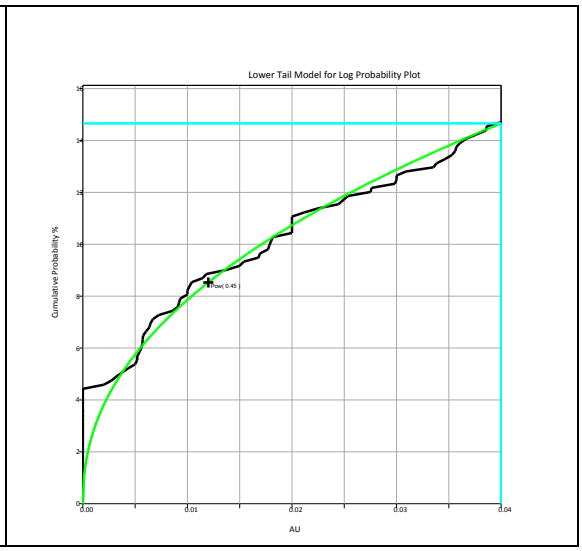


Transitional and fresh data

Upper tail



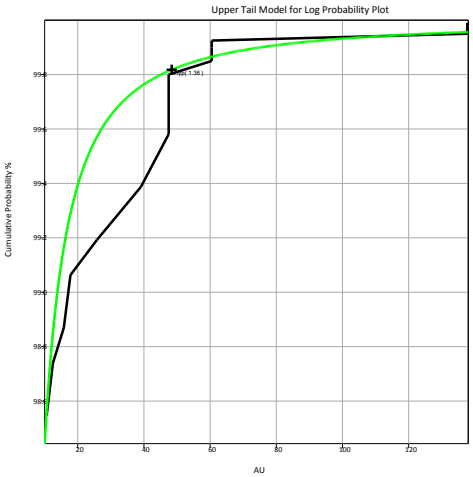
Lower tail



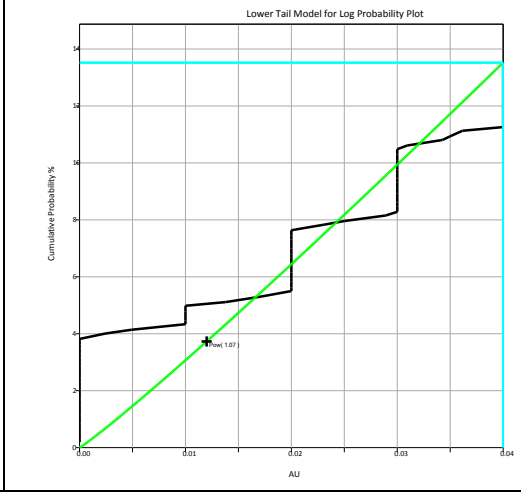
Southern sediments

Laterite, oxide and transitional data

Upper tail

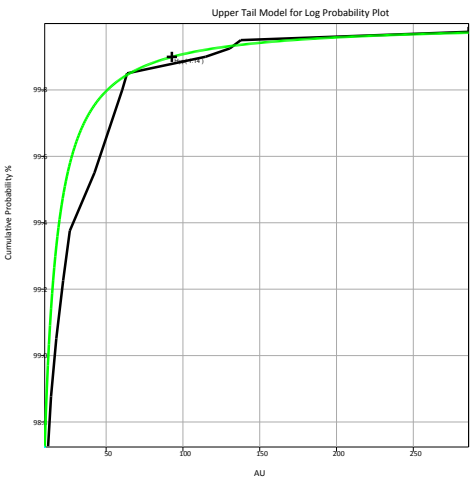


Lower tail

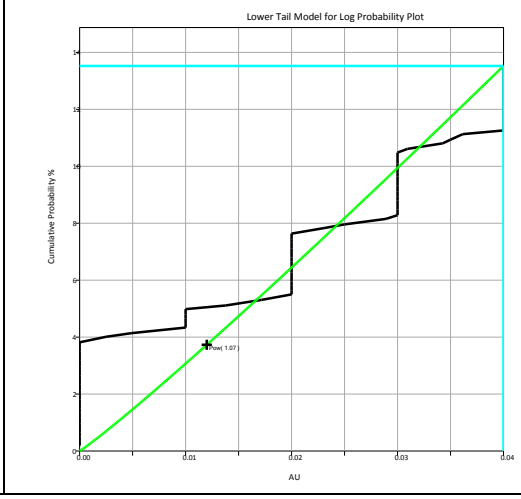


Transitional and fresh data

Upper tail



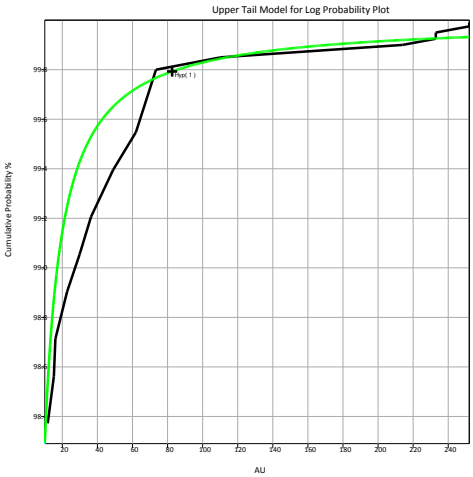
Lower tail



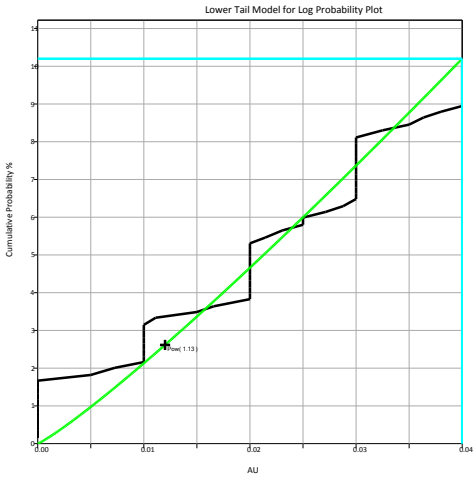
Northern sediments

Laterite, oxide and transitional data

Upper tail

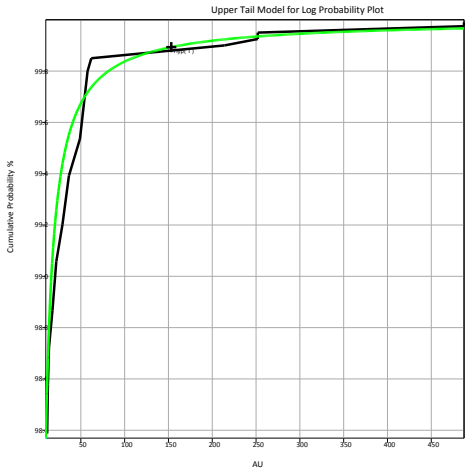


Lower tail

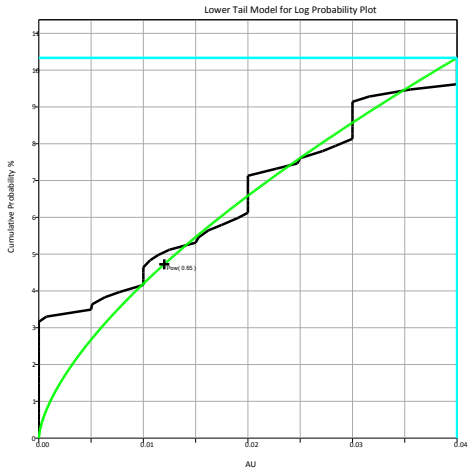


Transitional and fresh data

Upper tail

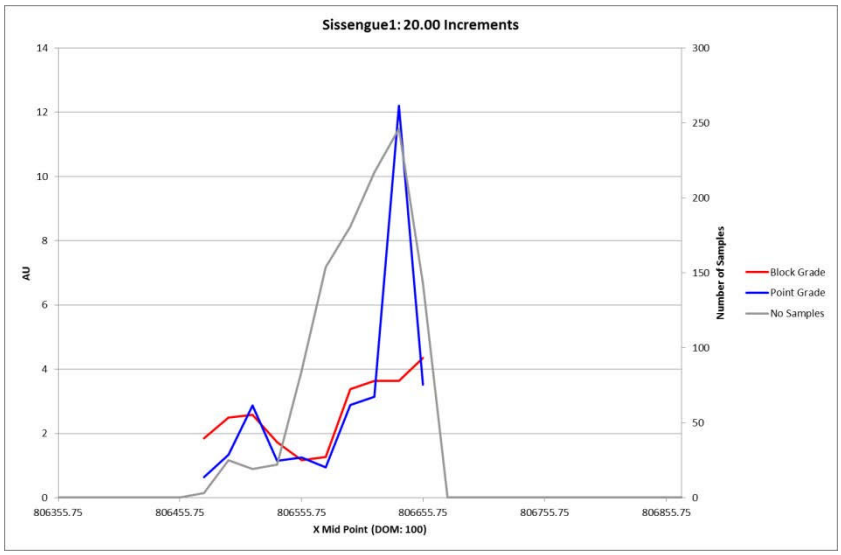


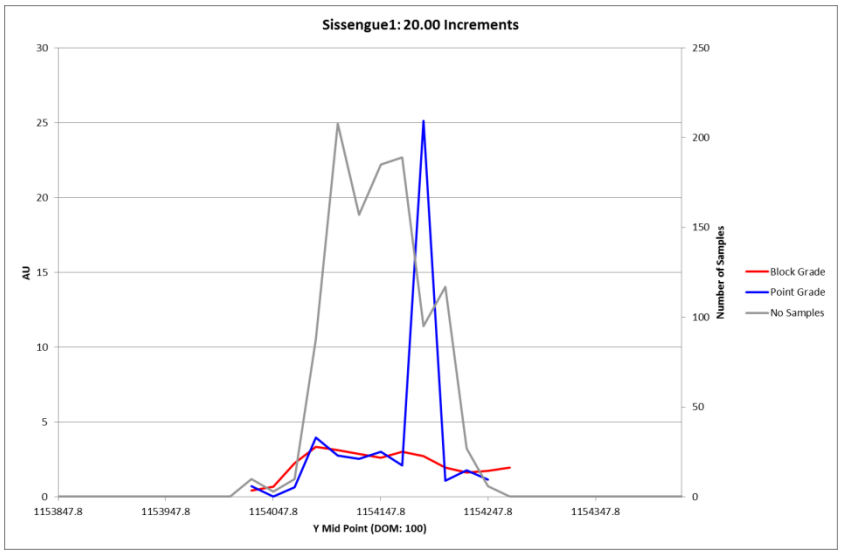
Lower tail

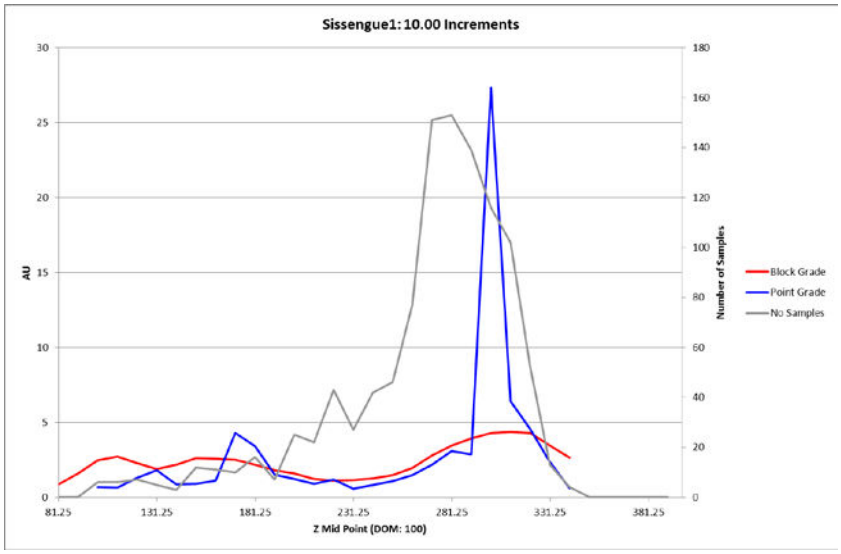


Appendix G Validation trend plots

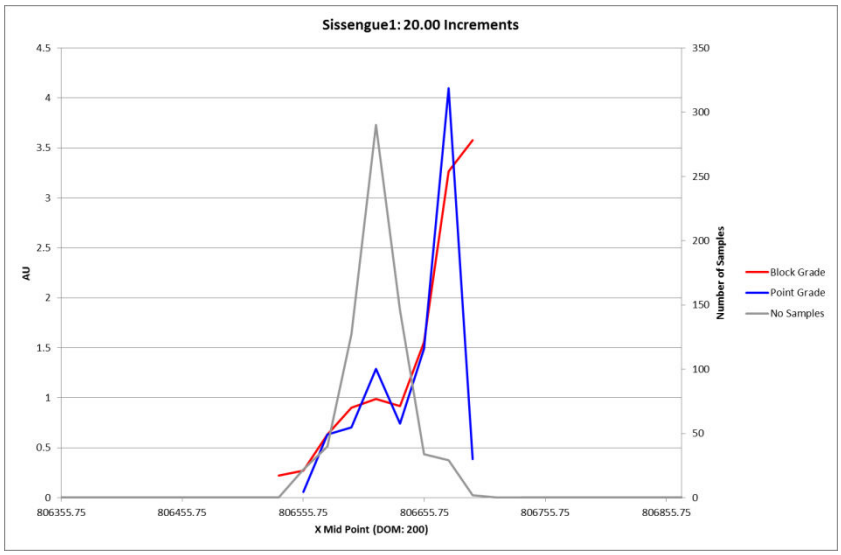
Southern granite

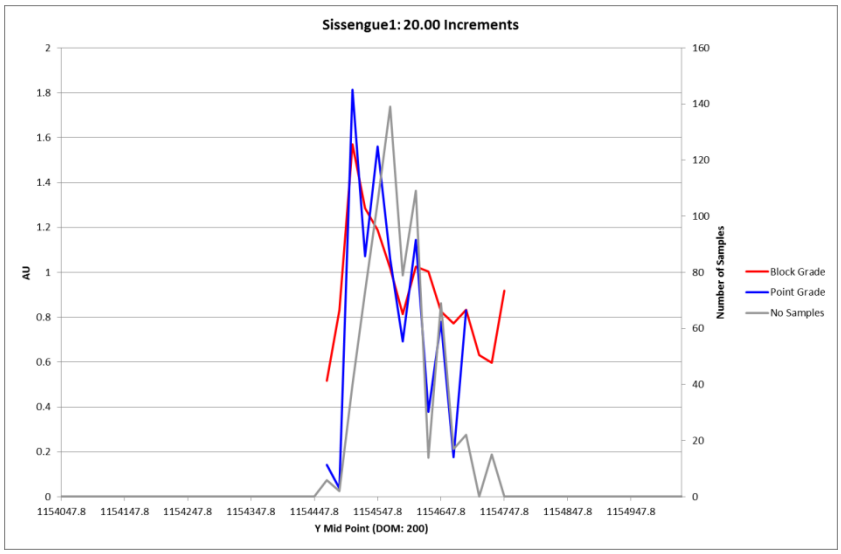


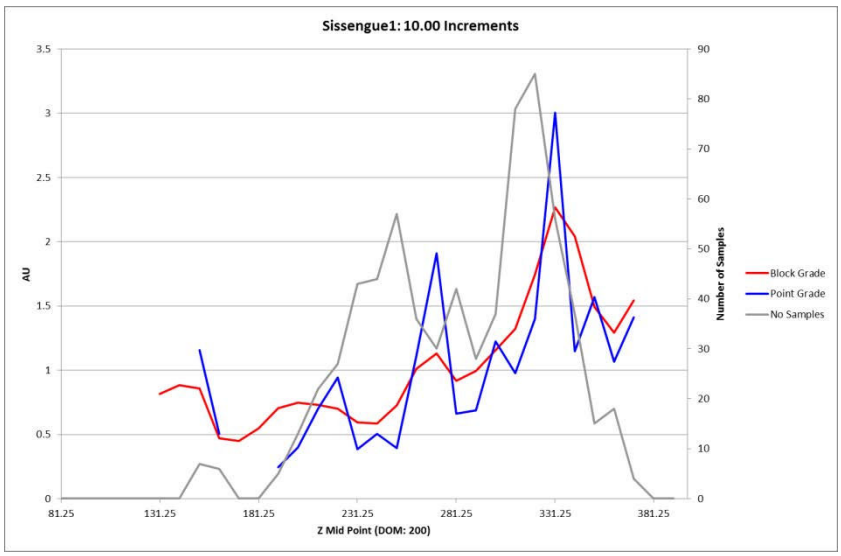




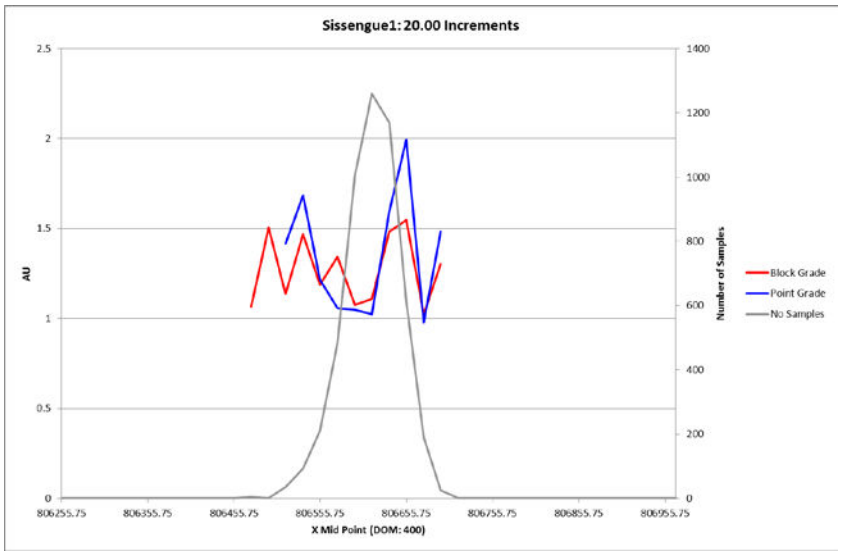
Northern granite

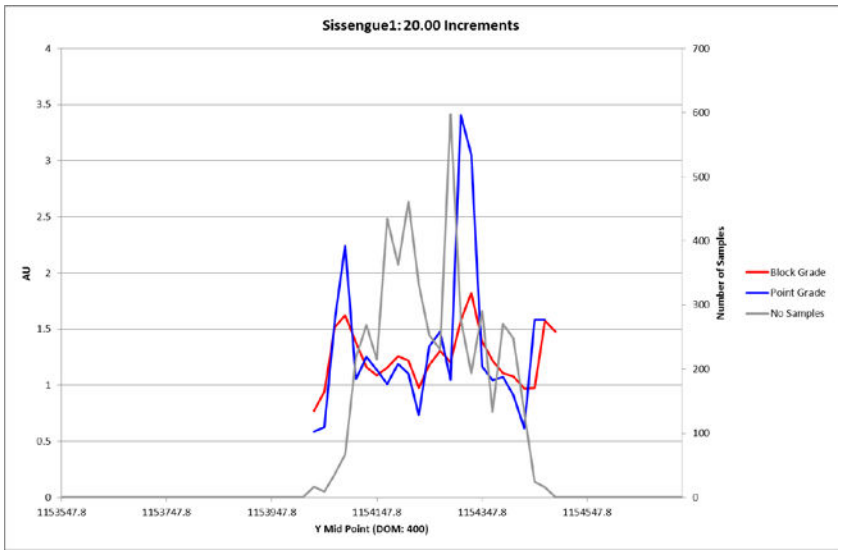


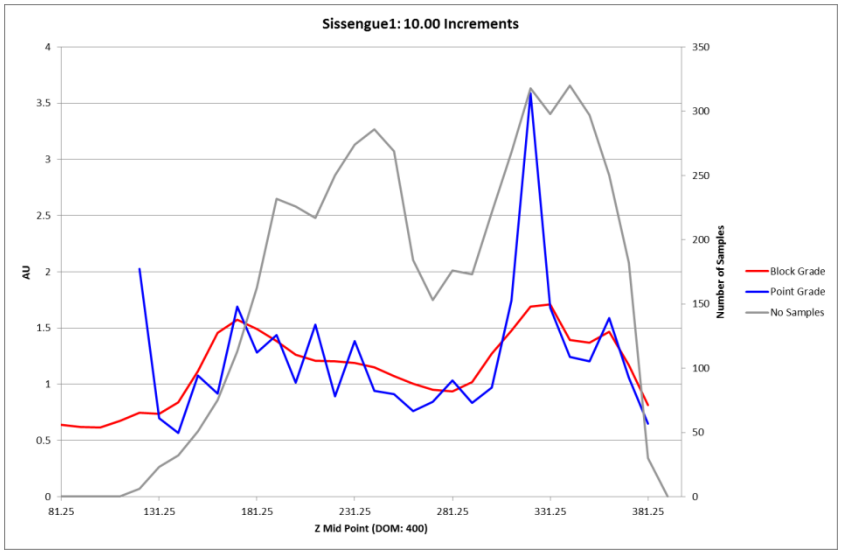




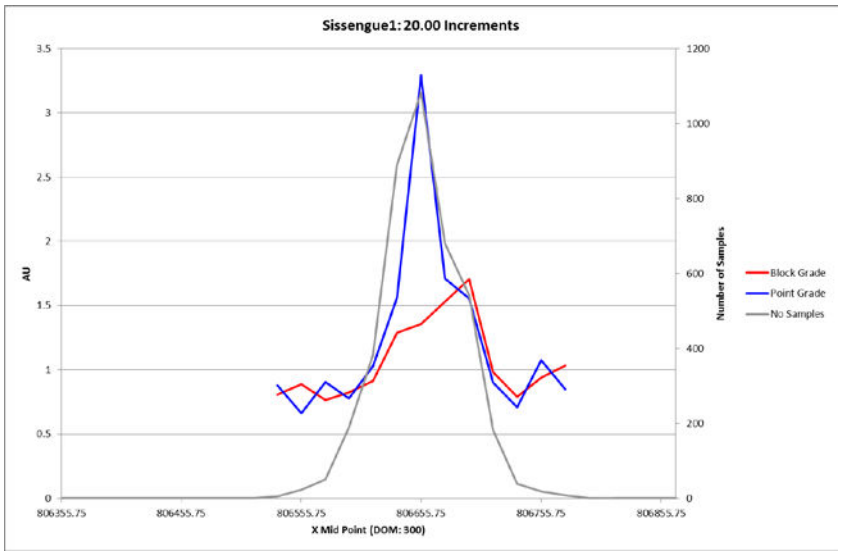
Southern sediments

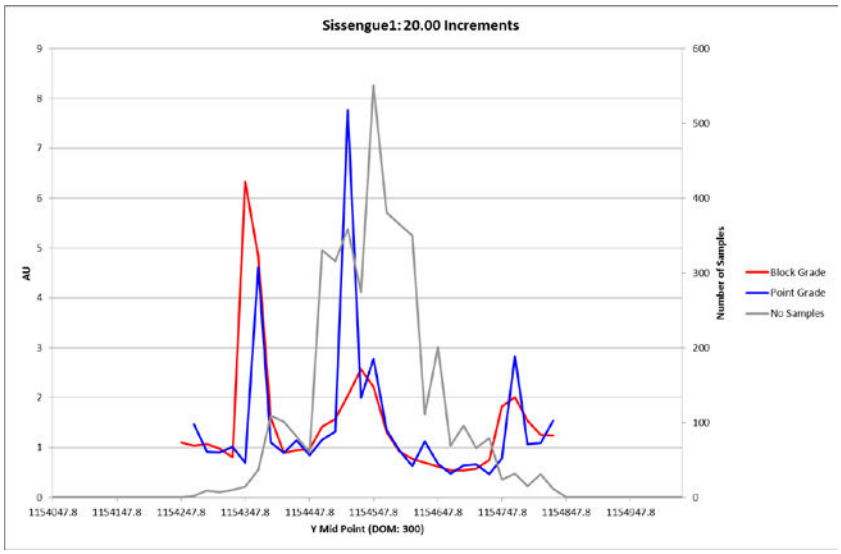


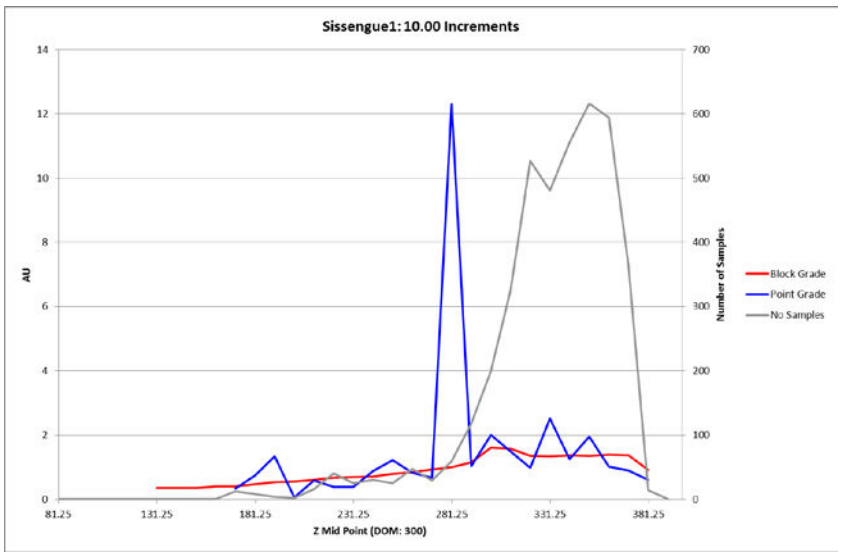




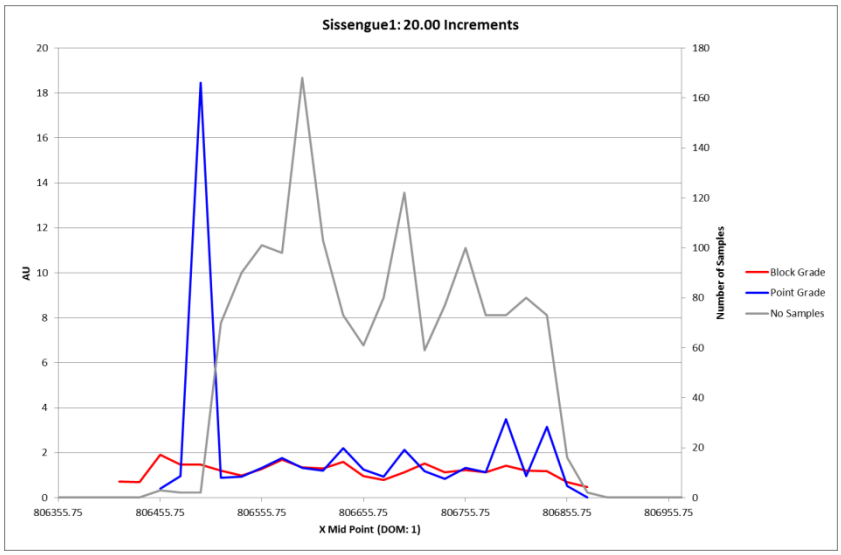
Northern sediments

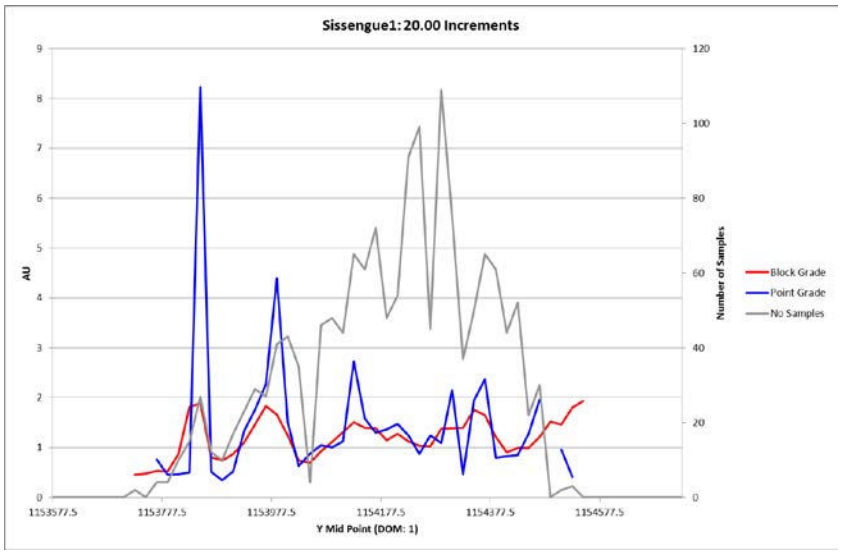


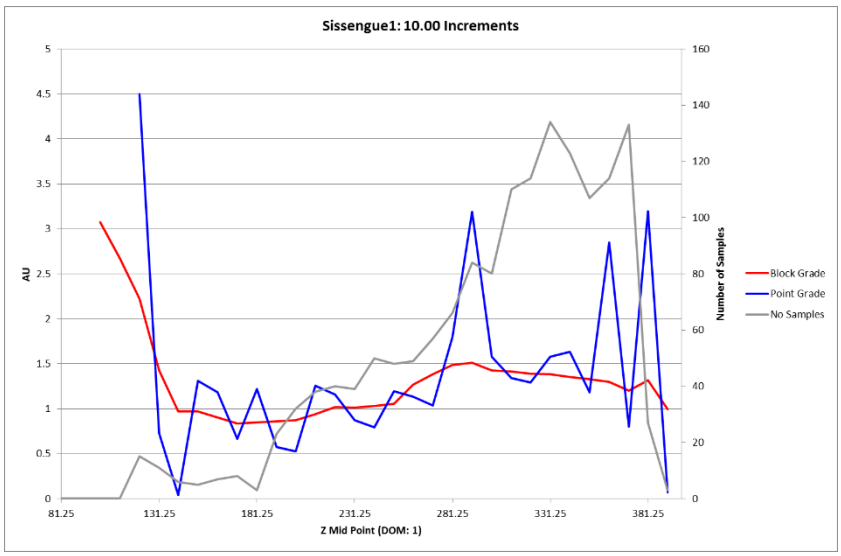




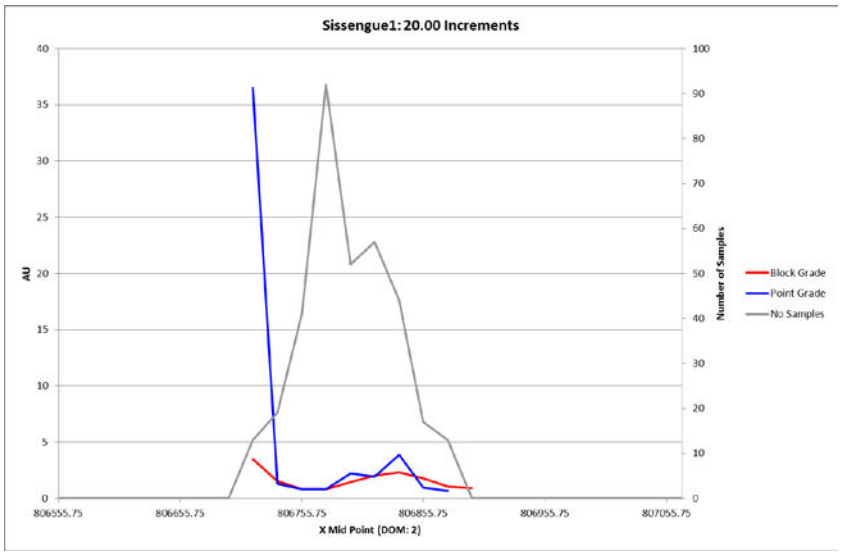
Dyke main 1

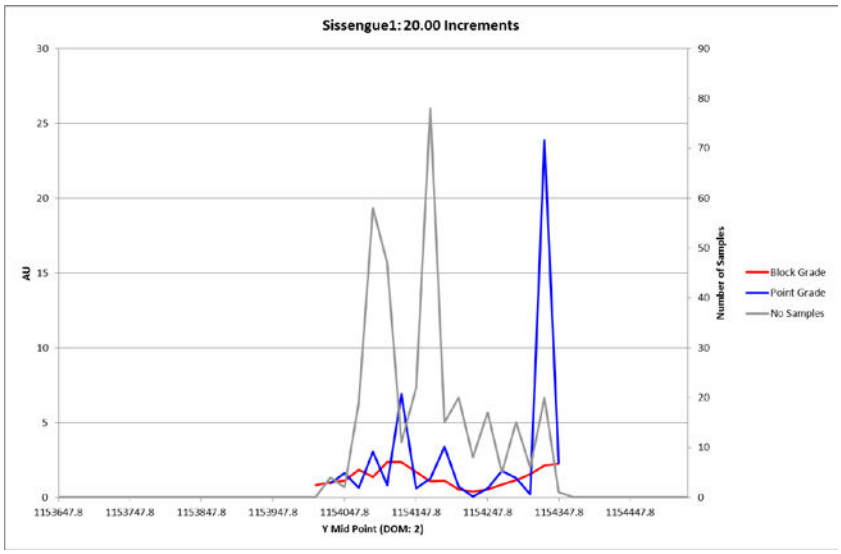

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 1)
Block Grade
Point Grade
No Samples

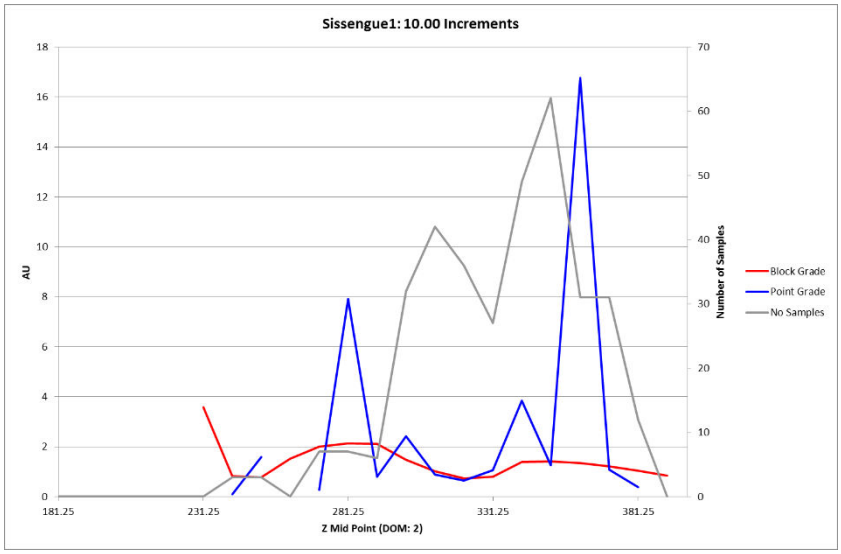

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 1)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 1)
Block Grade
Point Grade
No Samples

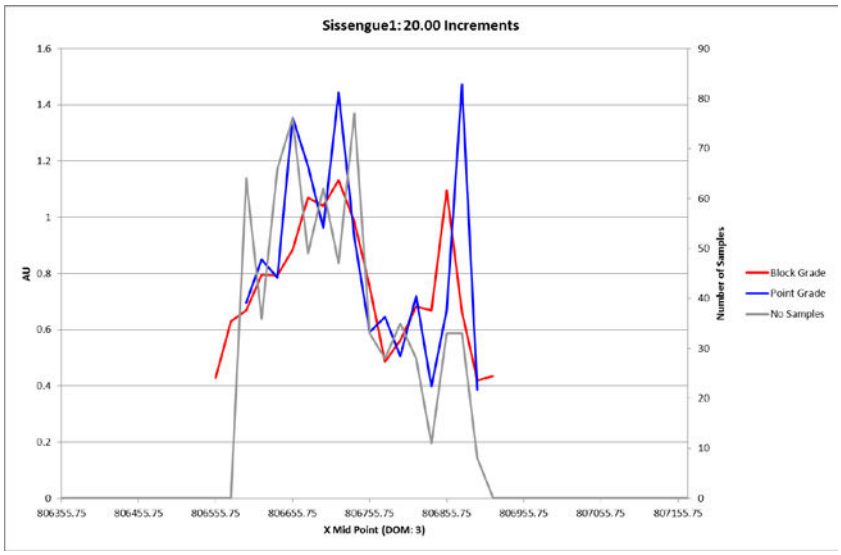
Dyke main 2

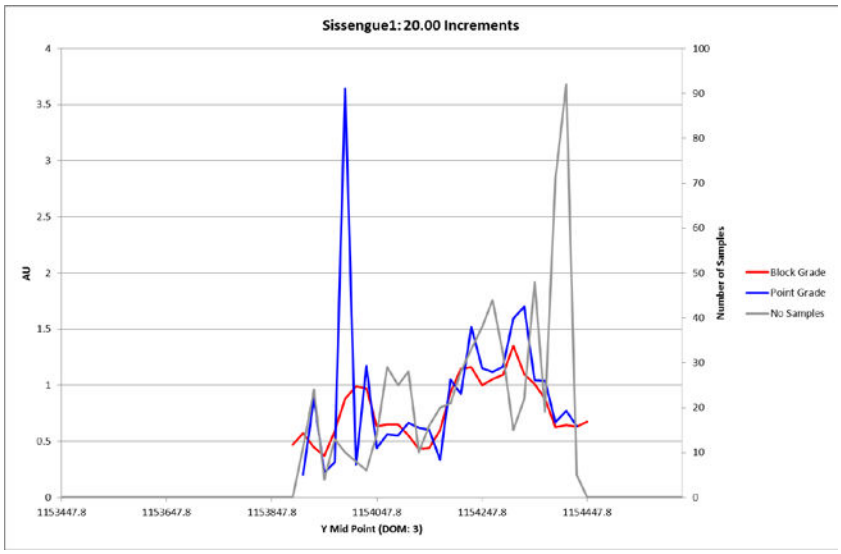

Sissengue1: 20.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
X Mid Point (DOM: 2)

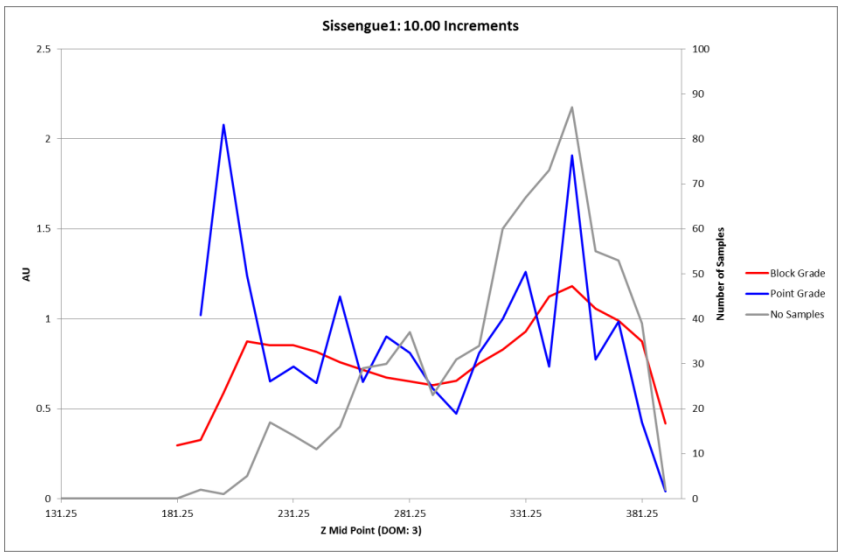

Sissengue1: 20.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
Y Mid Point (DOM: 2)


Sissengue1: 10.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
Z Mid Point (DOM: 2)

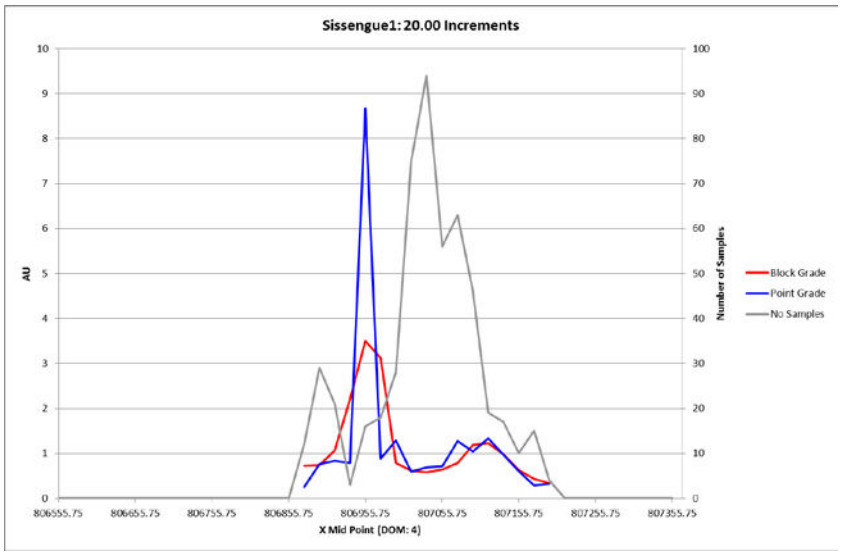
Dyke main 3

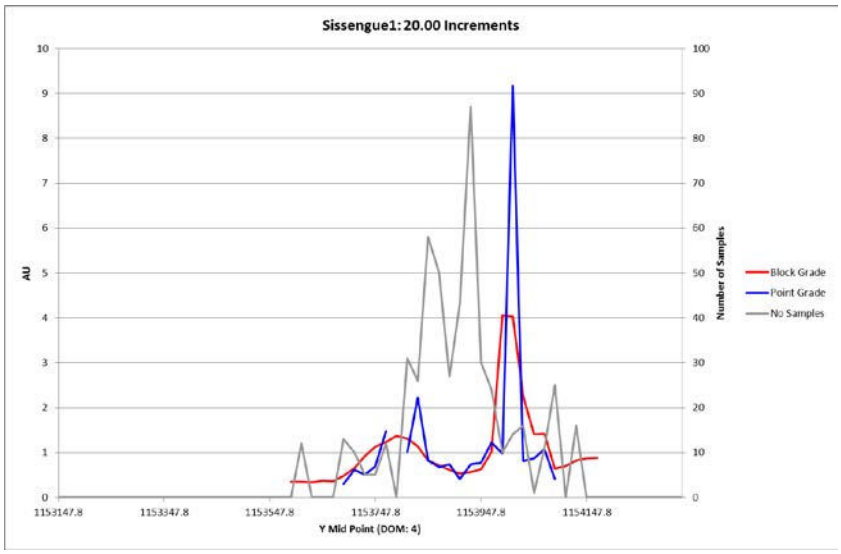

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 3)
Block Grade
Point Grade
No Samples

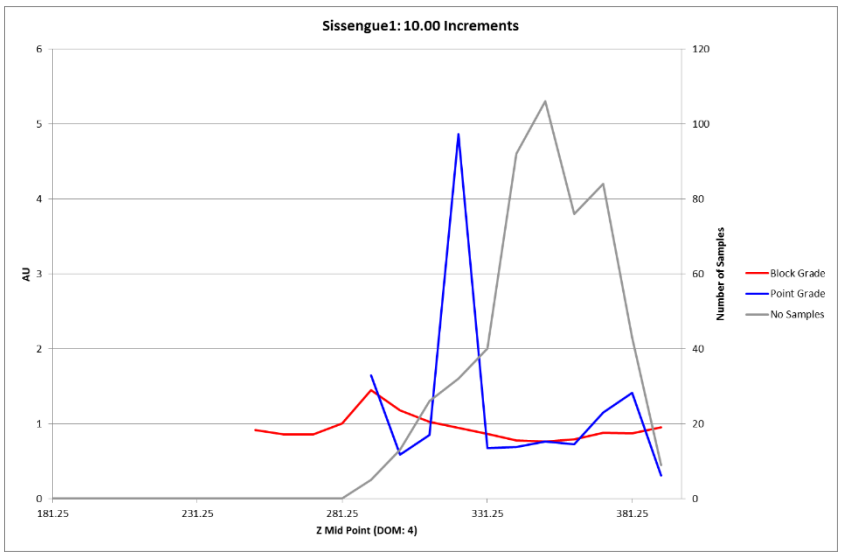

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 3)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 3)
Block Grade
Point Grade
No Samples

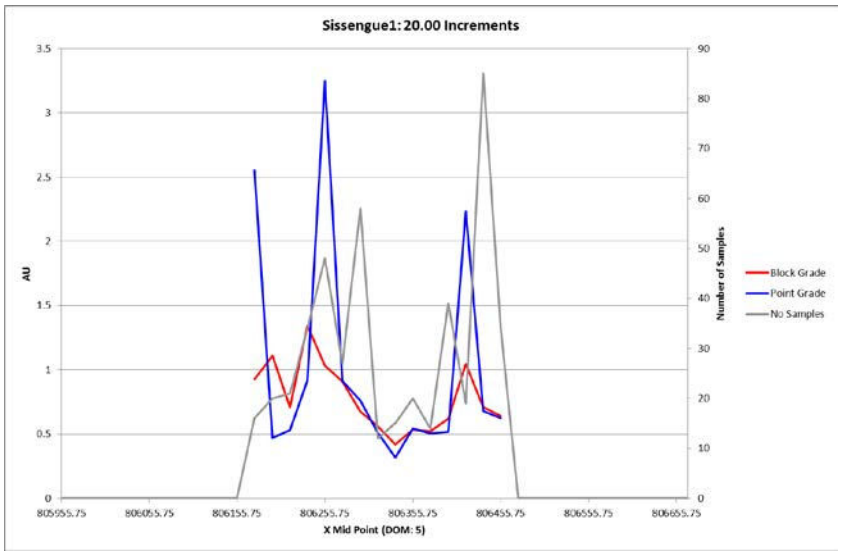
Dyke main 4

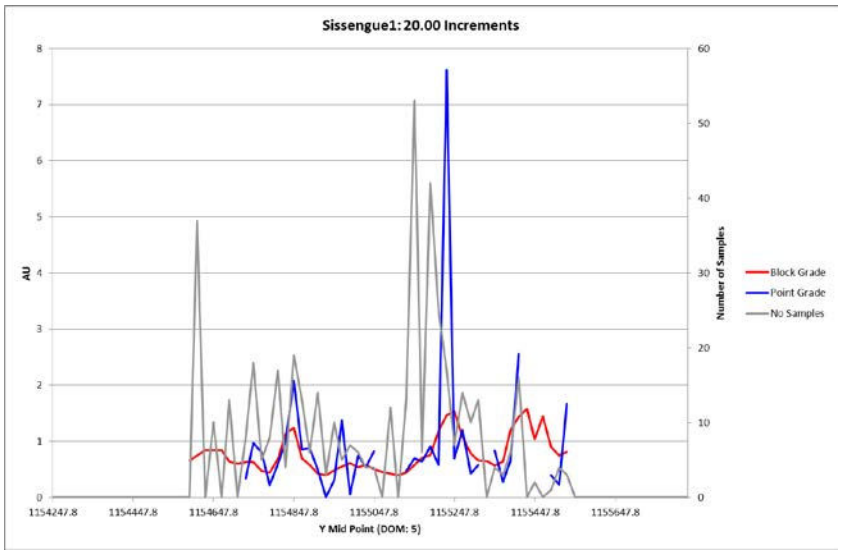

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 4)
Block Grade
Point Grade
No Samples

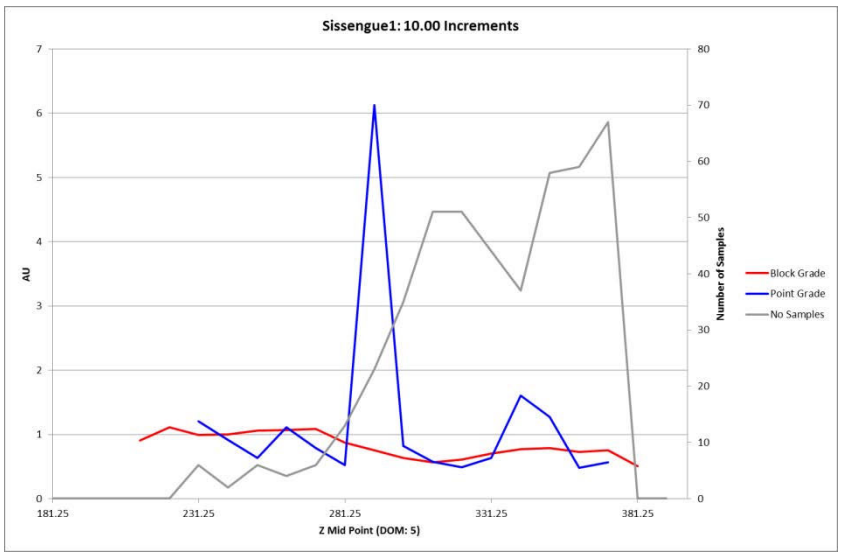

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 4)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 4)
Block Grade
Point Grade
No Samples

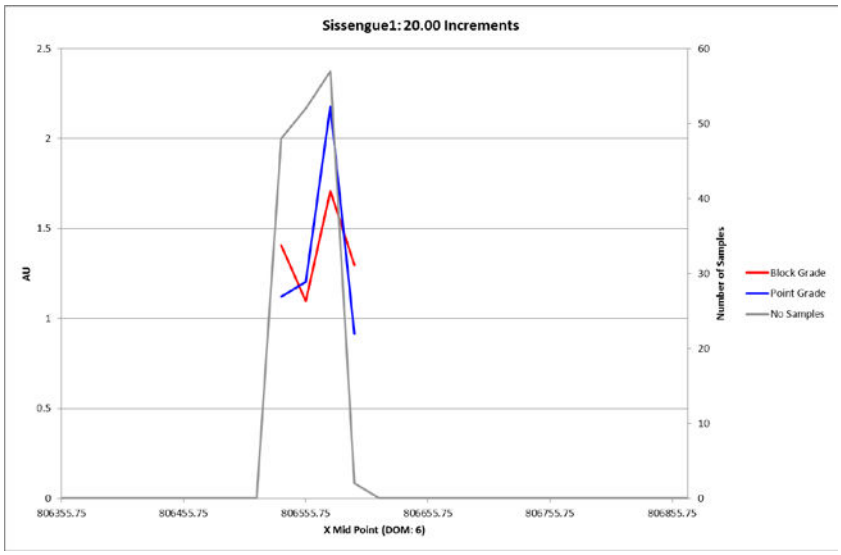
Dyke main 5

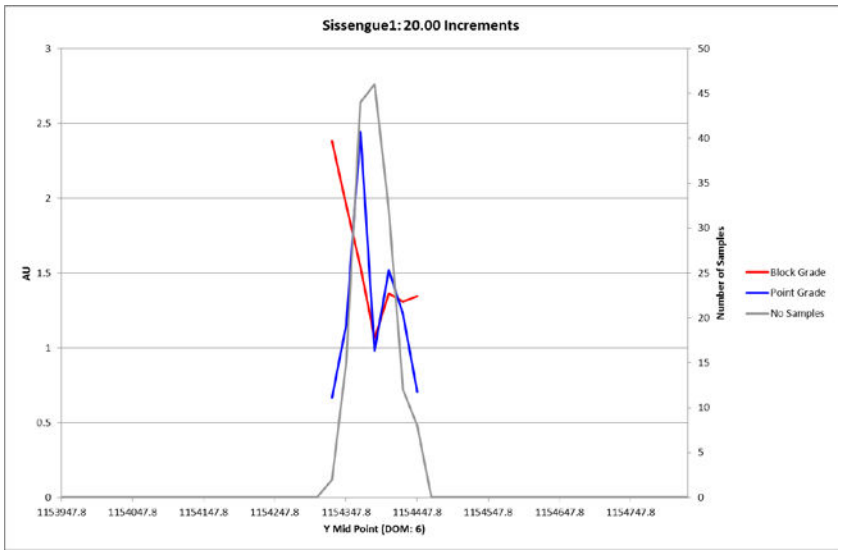

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 5)
Block Grade
Point Grade
No Samples

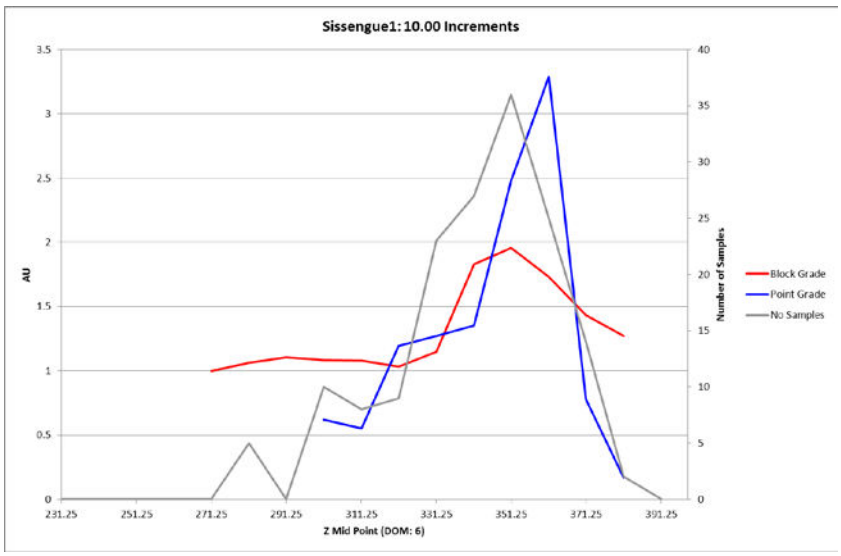

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 5)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 5)
Block Grade
Point Grade
No Samples

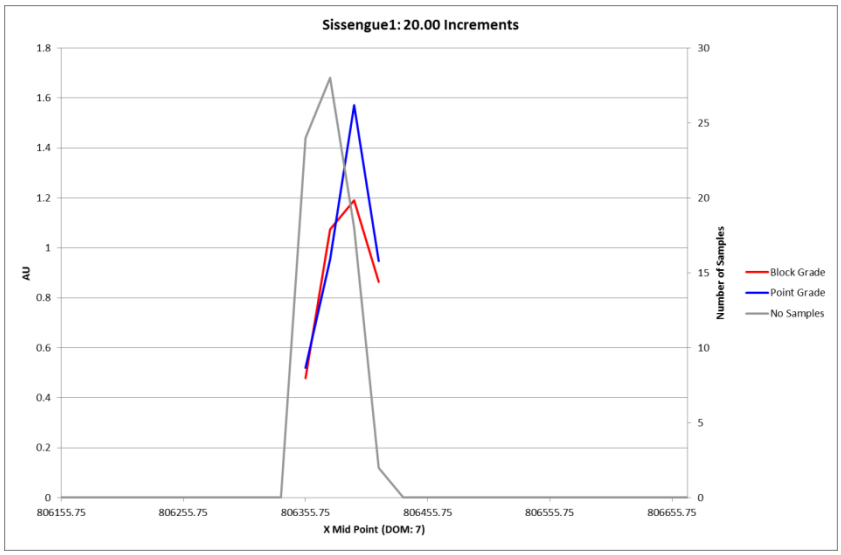
Dyke main 6

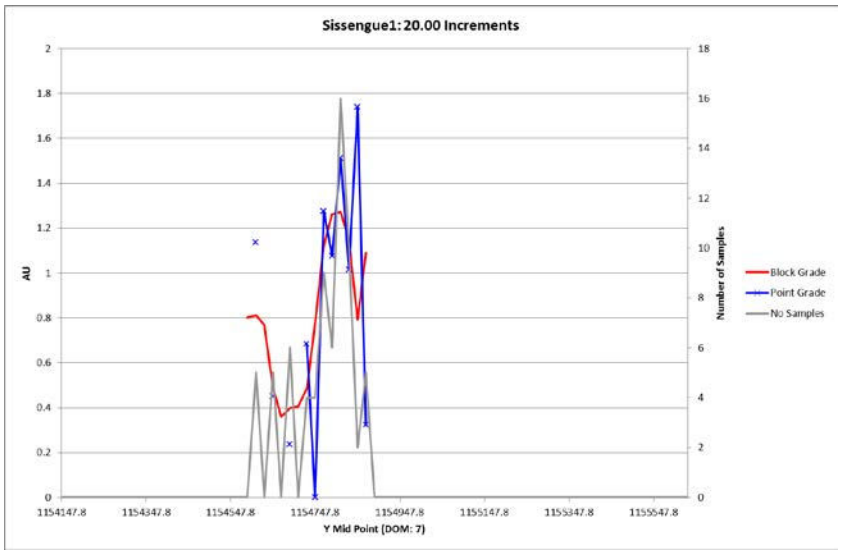

Sissengue1: 20.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
X Mid Point (DOM: 6)

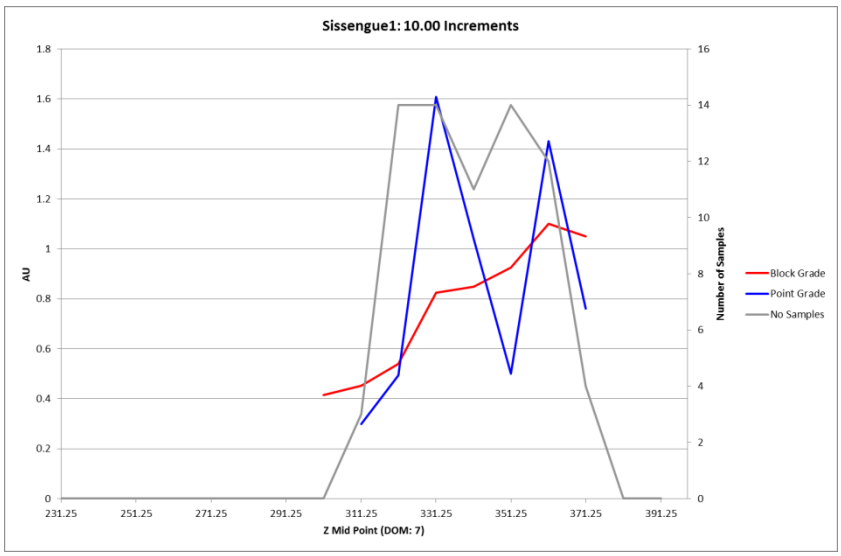

Sissengue1: 20.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
Y Mid Point (DOM: 6)


Sissengue1: 10.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
Z Mid Point (DOM: 6)

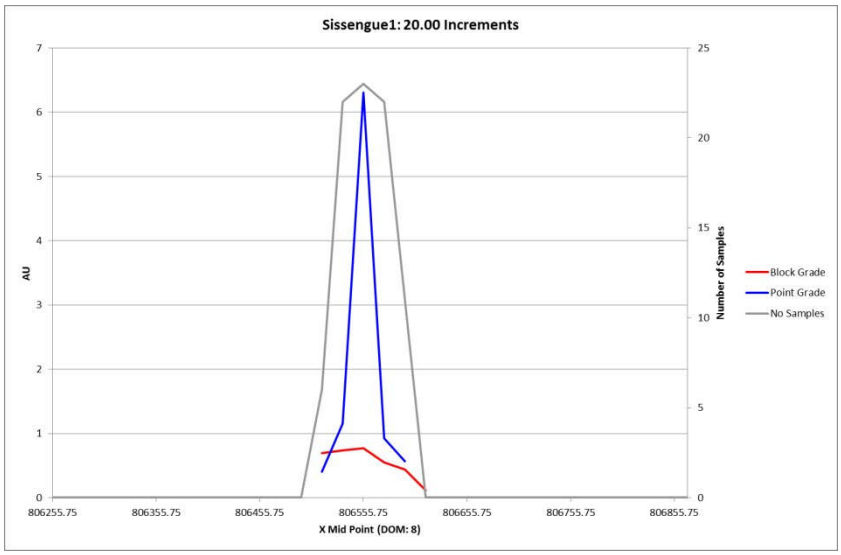
Dyke main 7

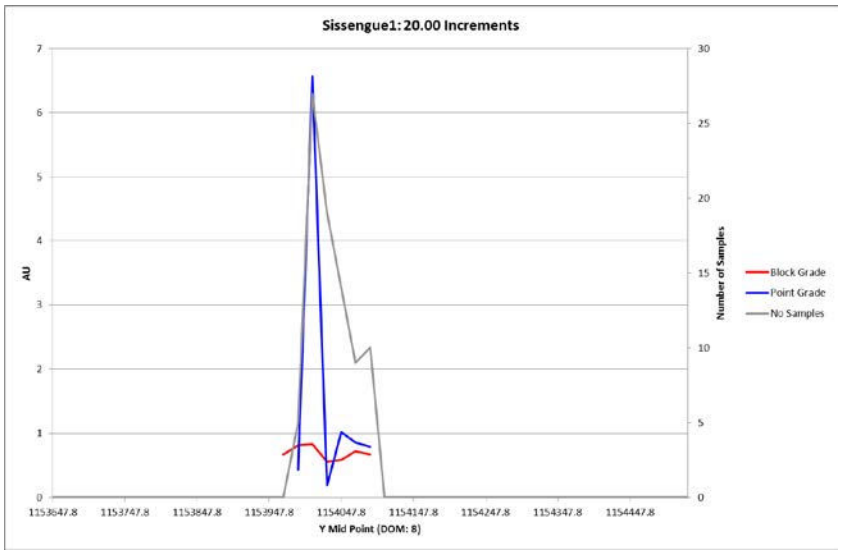

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 7)
Block Grade
Point Grade
No Samples

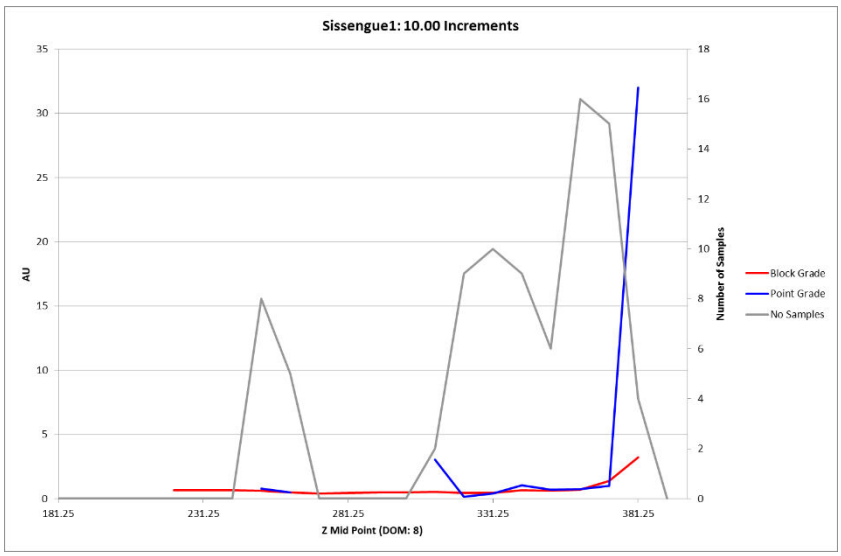

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 7)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 7)
Block Grade
Point Grade
No Samples

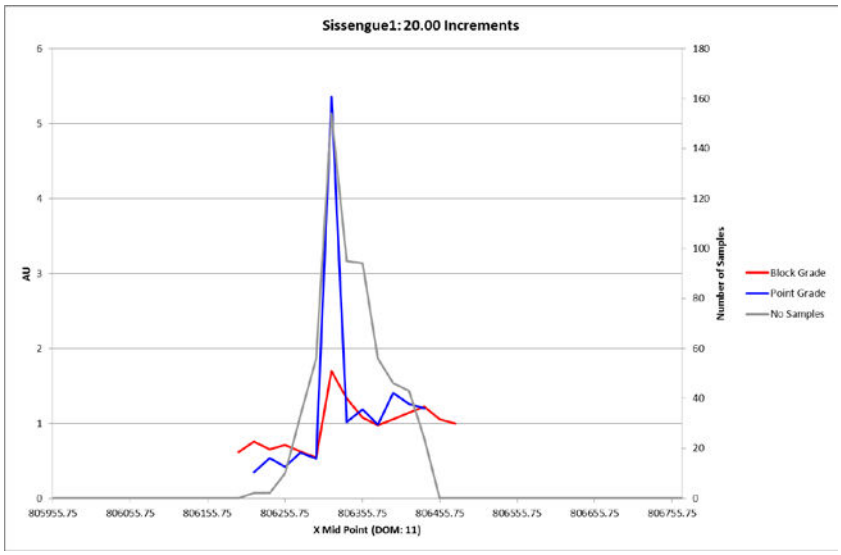
Dyke main 8

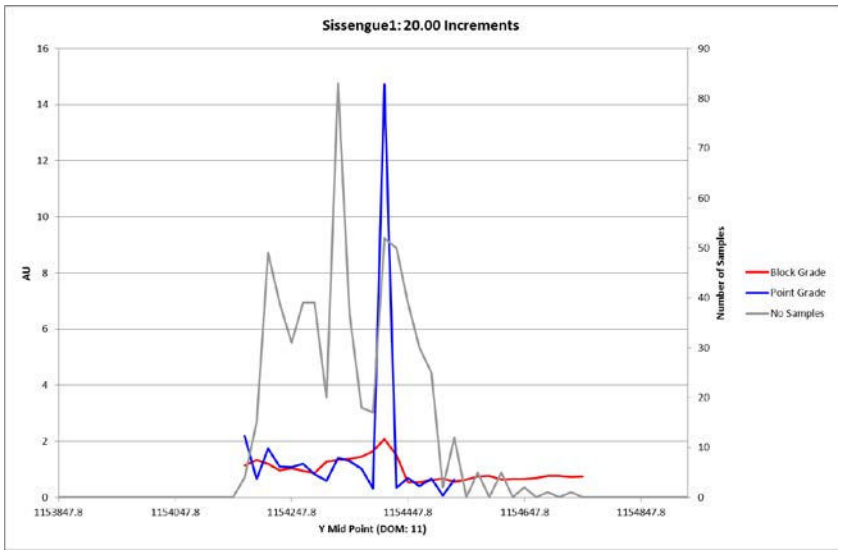

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 8)
806255.75
806355.75
806455.75
806555.75
806655.75
806755.75
806855.75
Block Grade
Point Grade
No Samples

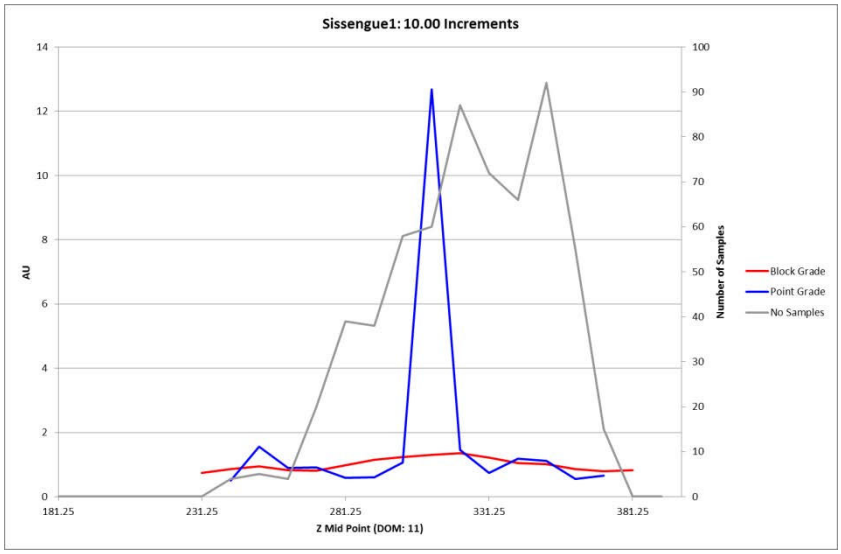

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 8)
1153647.8
1153747.8
1153847.8
1153947.8
1154047.8
1154147.8
1154247.8
1154347.8
1154447.8
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 8)
181.25
231.25
281.25
331.25
381.25
Block Grade
Point Grade
No Samples

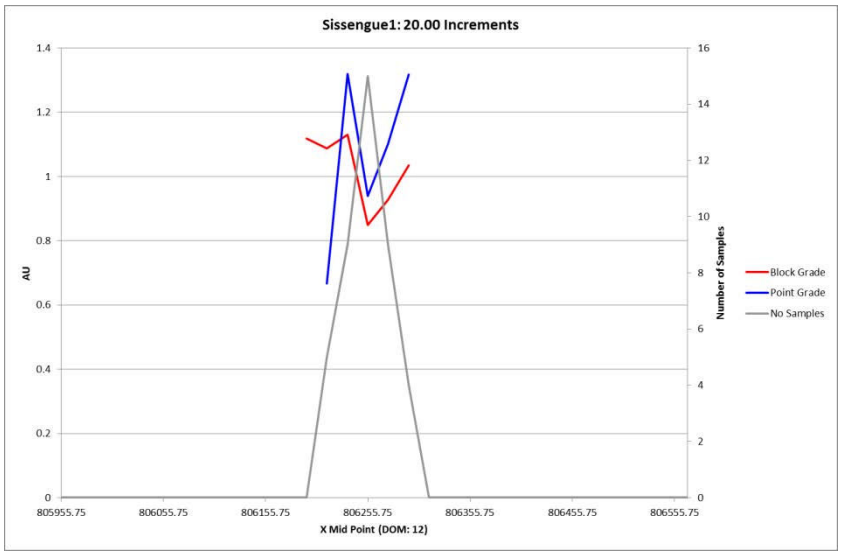
Dyke central 1

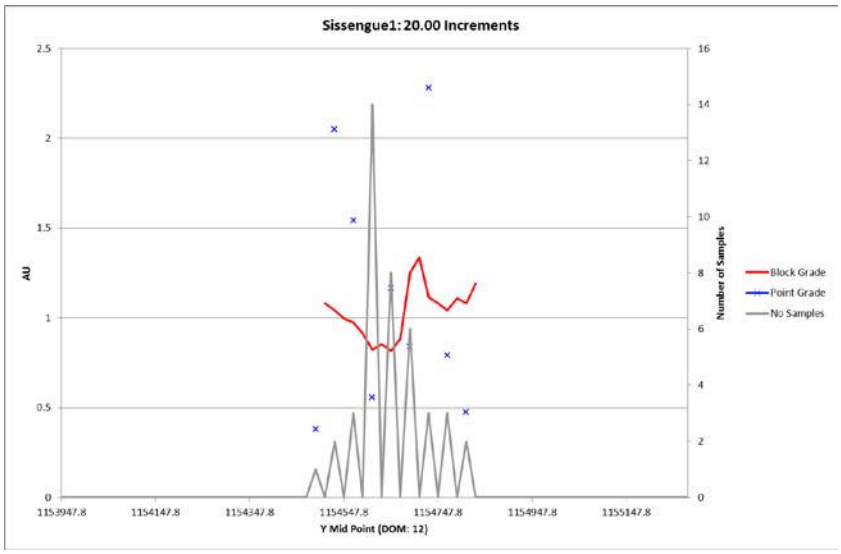


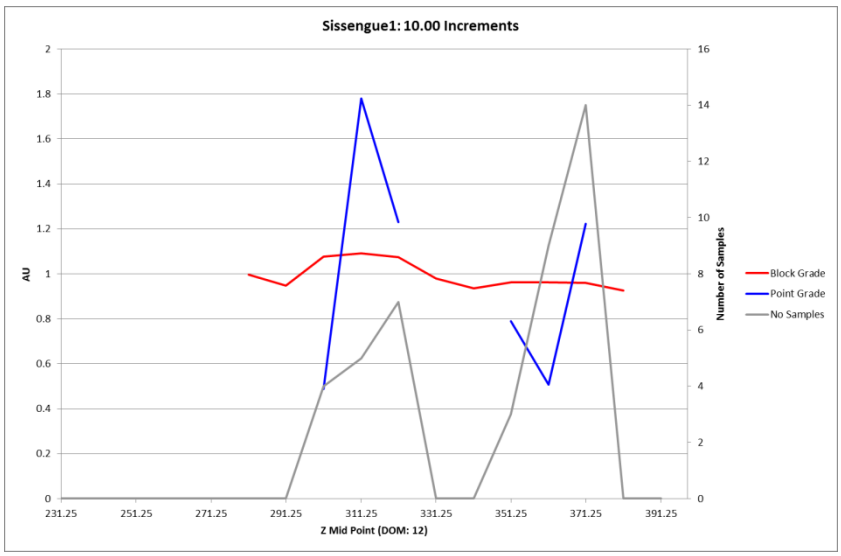




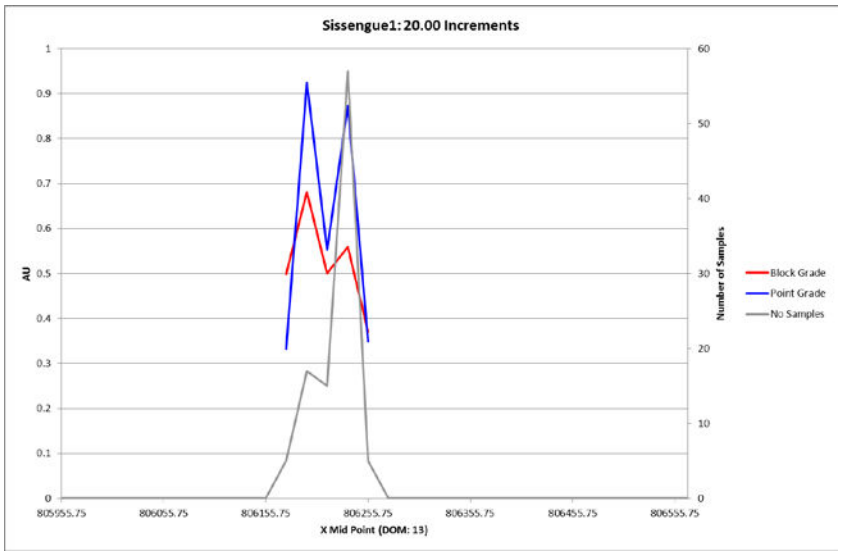
Dyke central 2

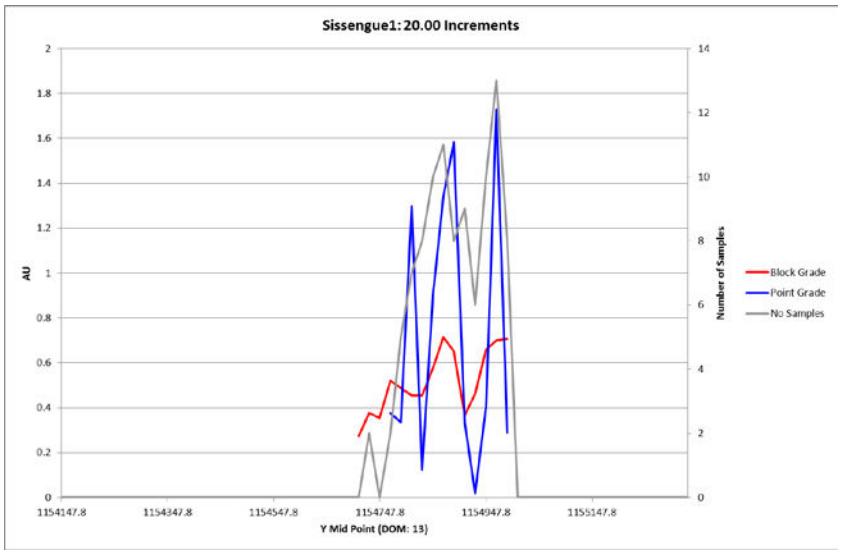


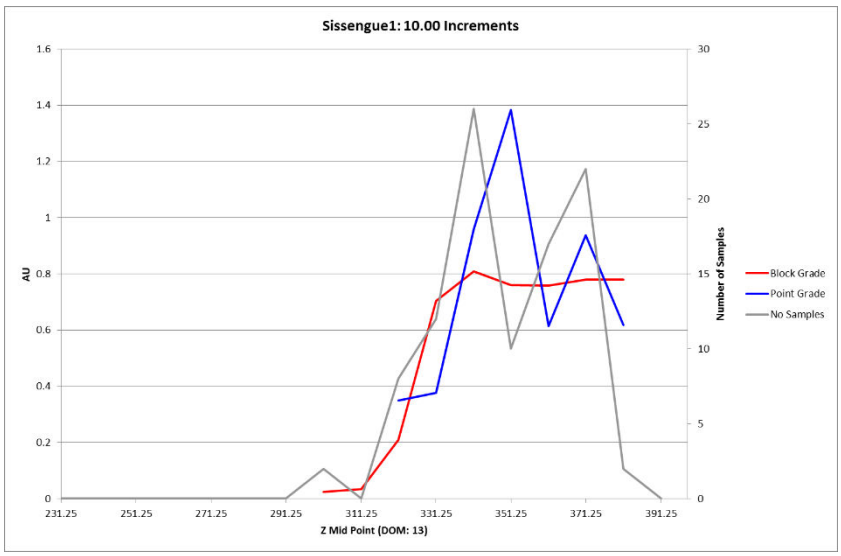




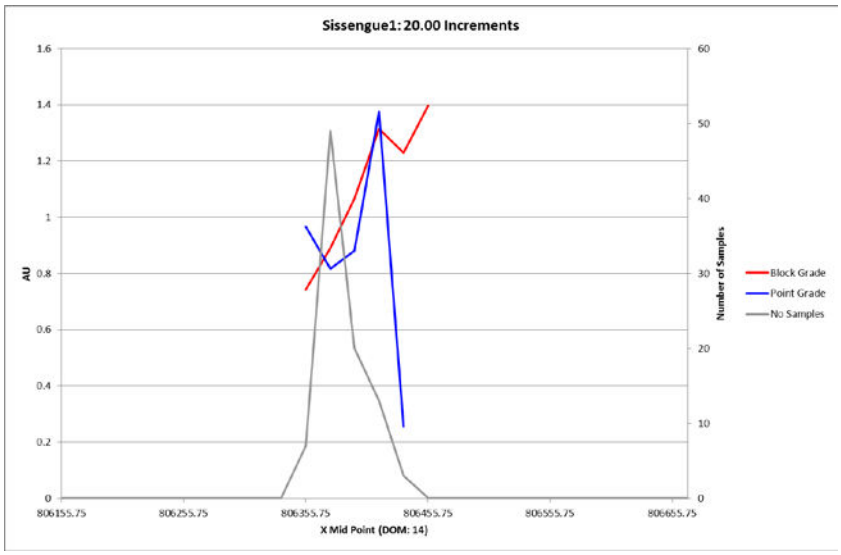
Dyke central 3

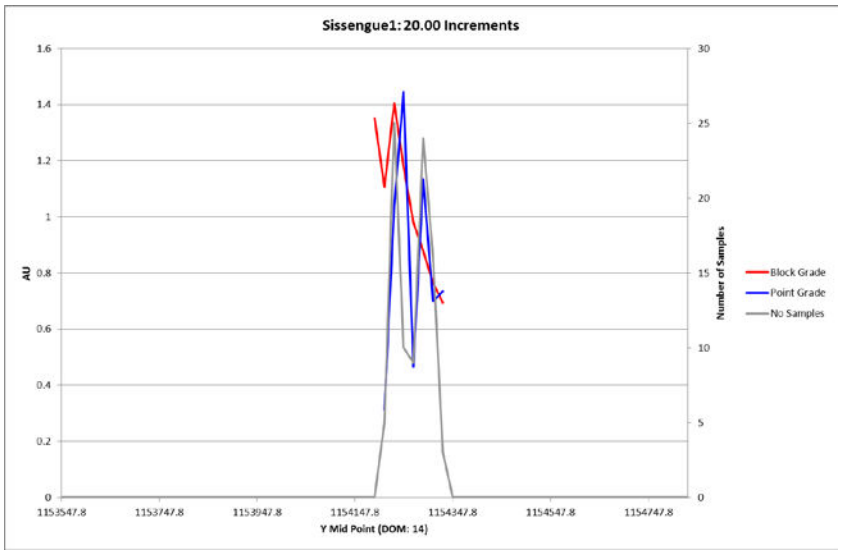


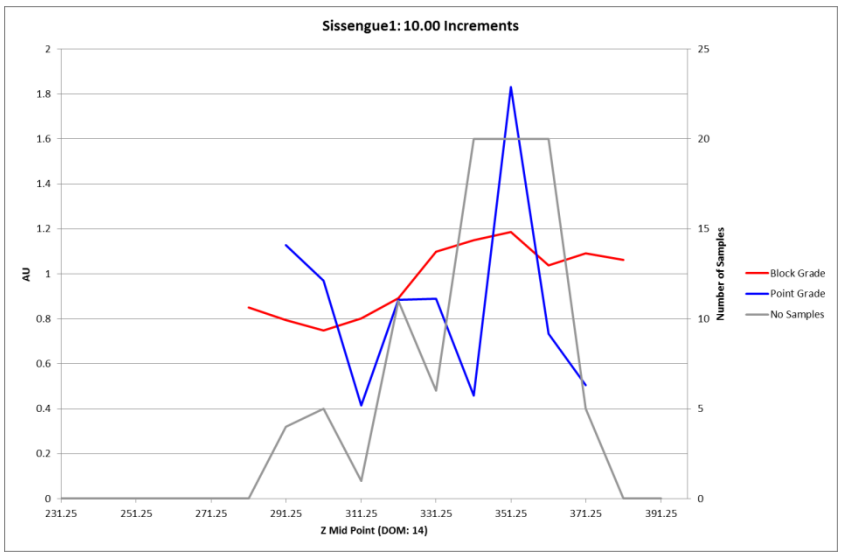




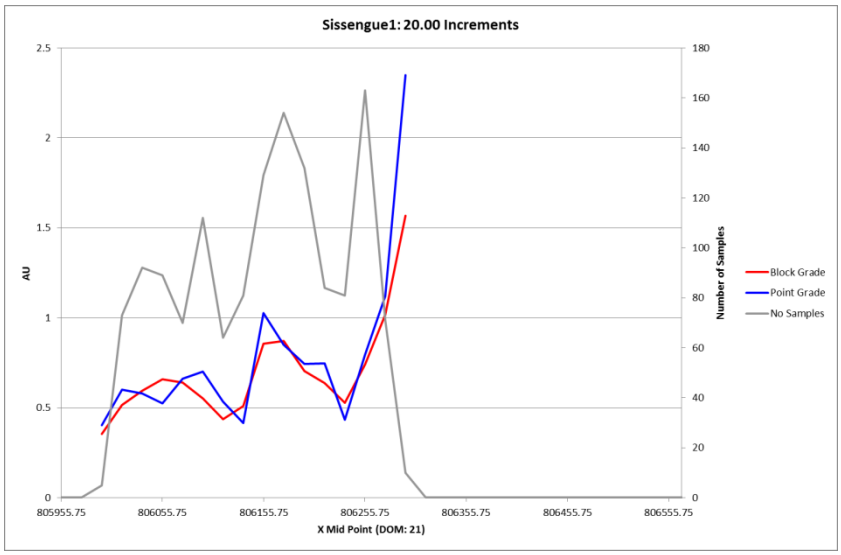
Dyke central 4

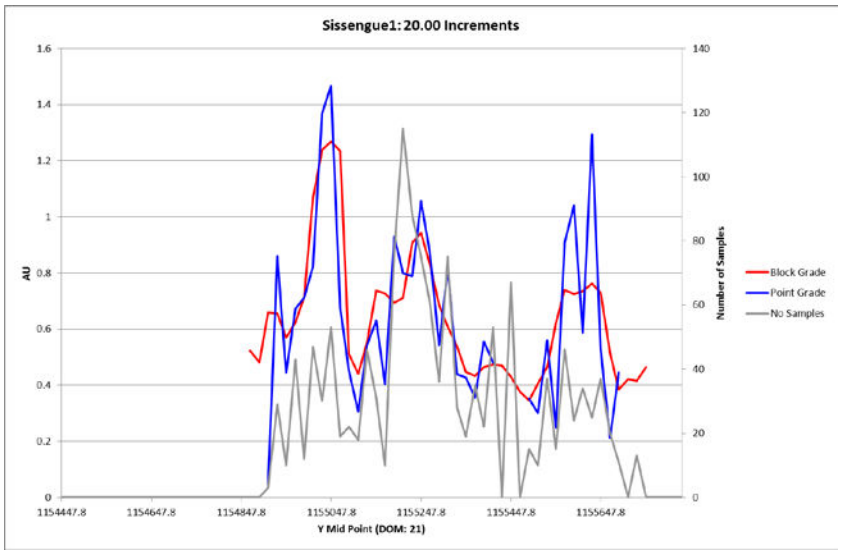


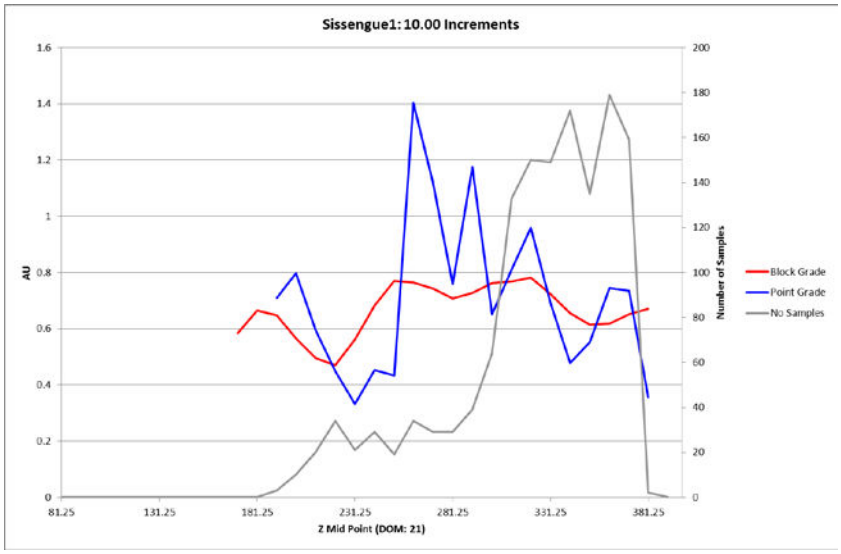




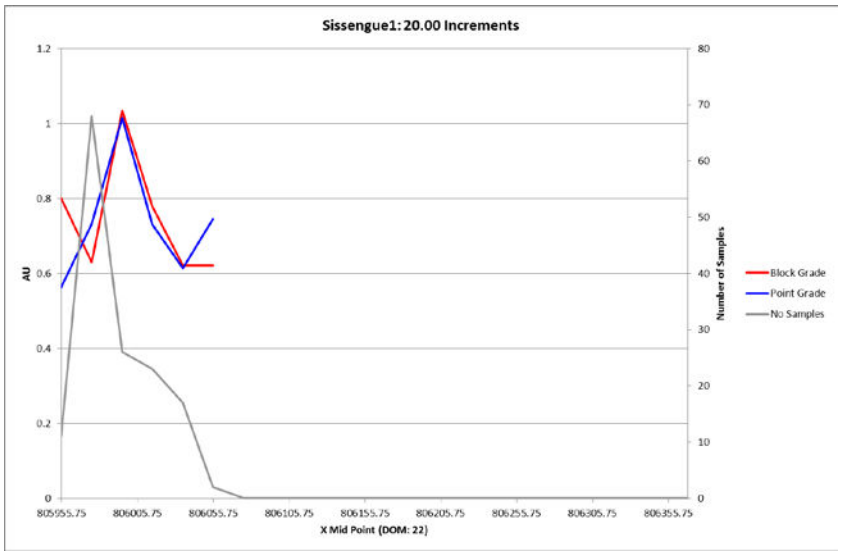
Dyke north 1

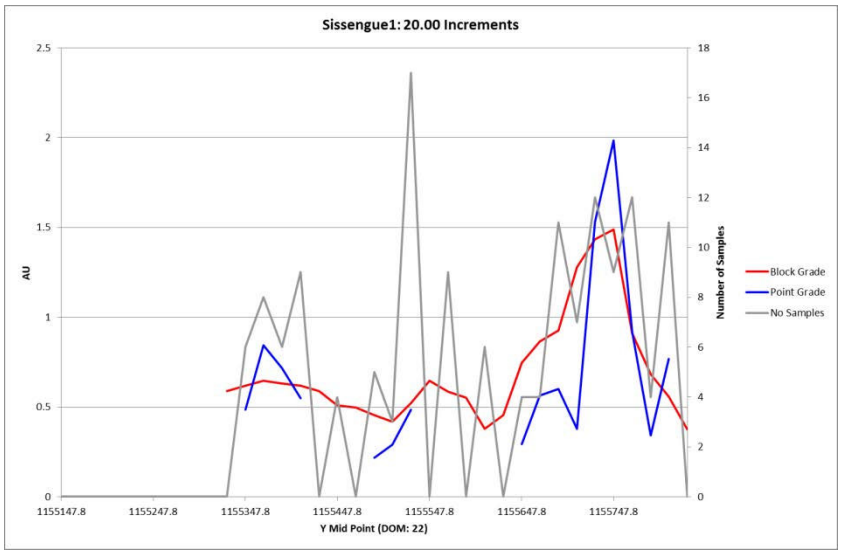

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 21)
Block Grade
Point Grade
No Samples

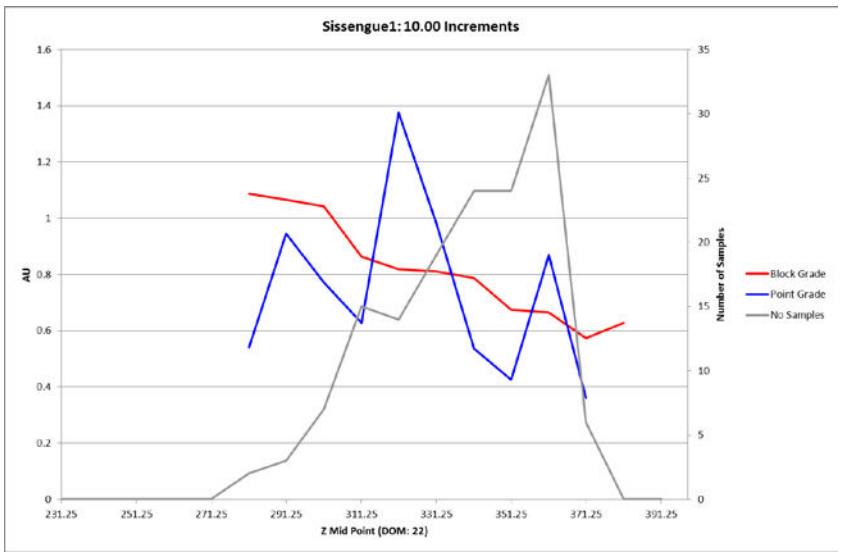

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 21)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 21)
Block Grade
Point Grade
No Samples

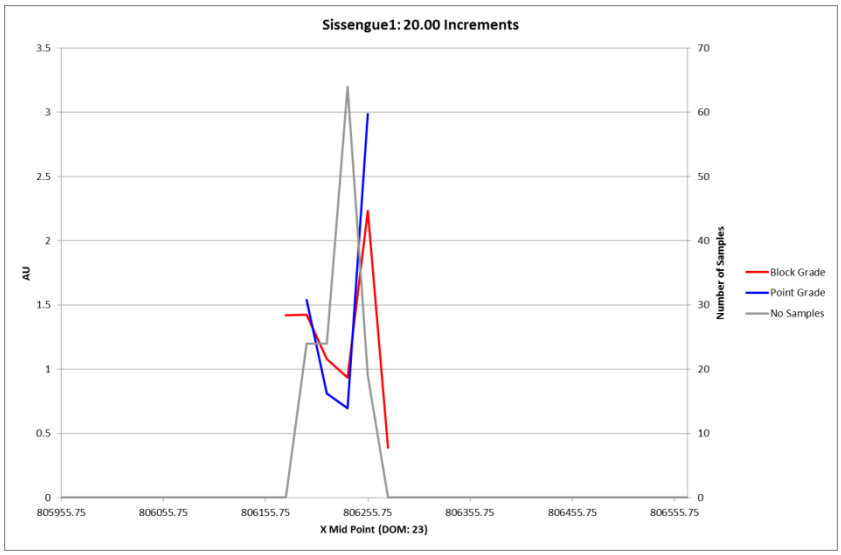
Dyke north 2

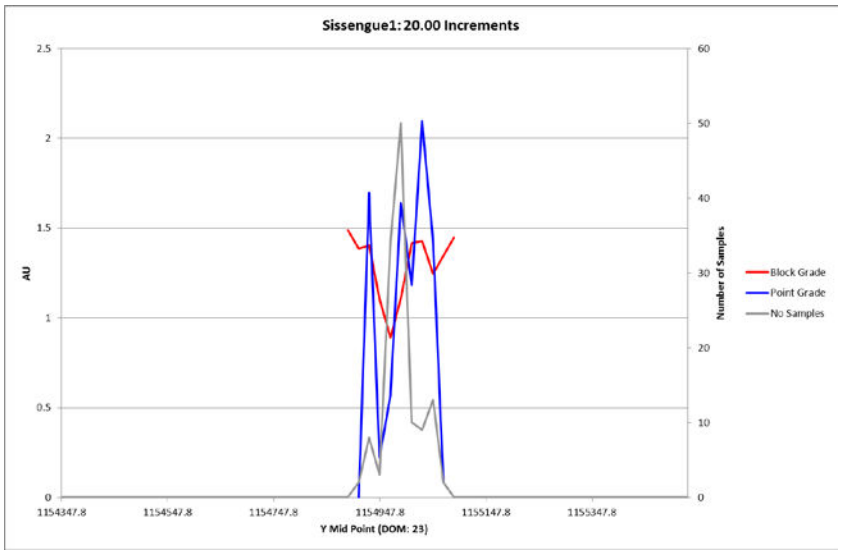

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 22)
Block Grade
Point Grade
No Samples

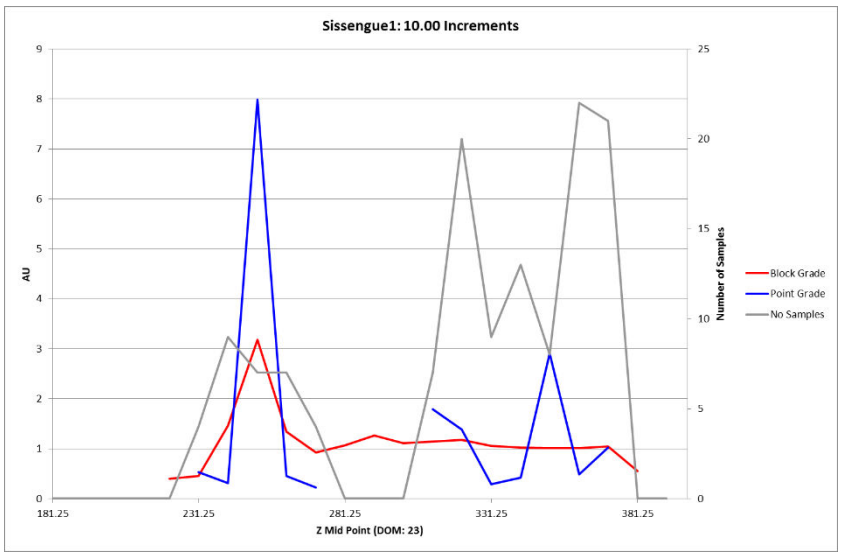

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 22)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 22)
Block Grade
Point Grade
No Samples

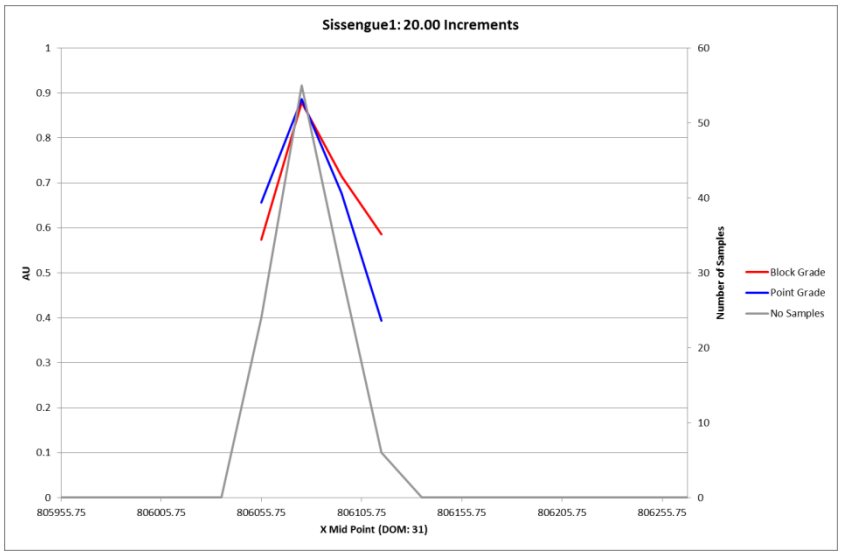
Dyke north 3

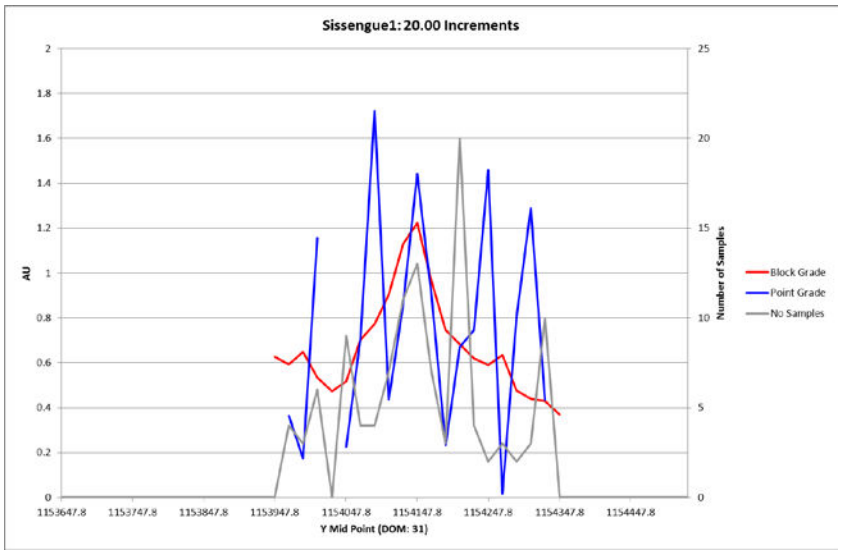

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 23)
Block Grade
Point Grade
No Samples

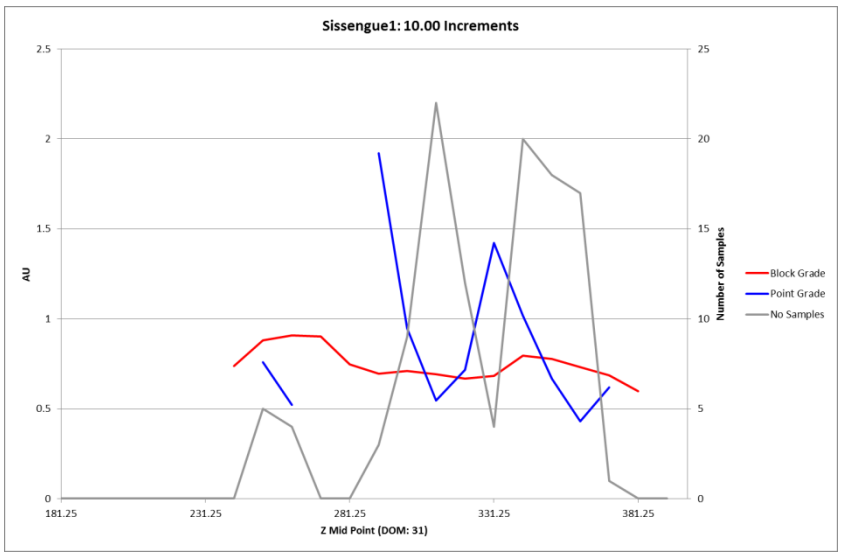

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 23)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 23)
Block Grade
Point Grade
No Samples

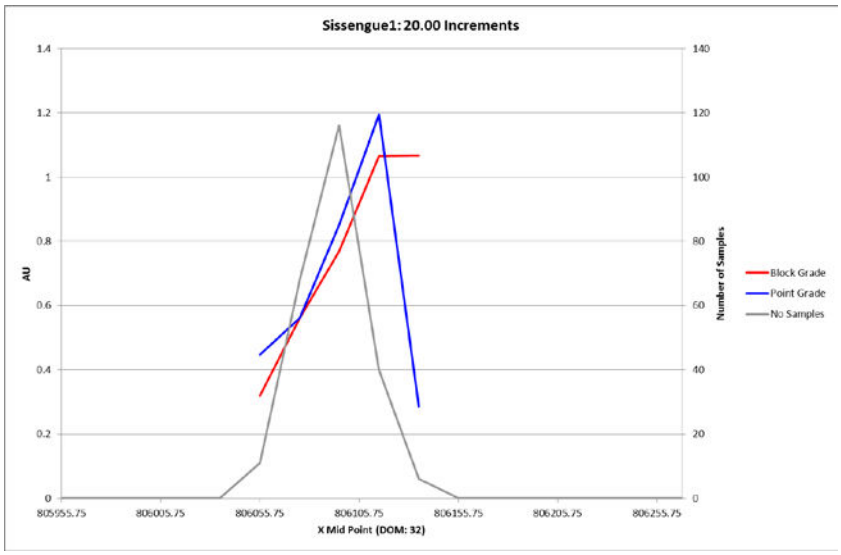
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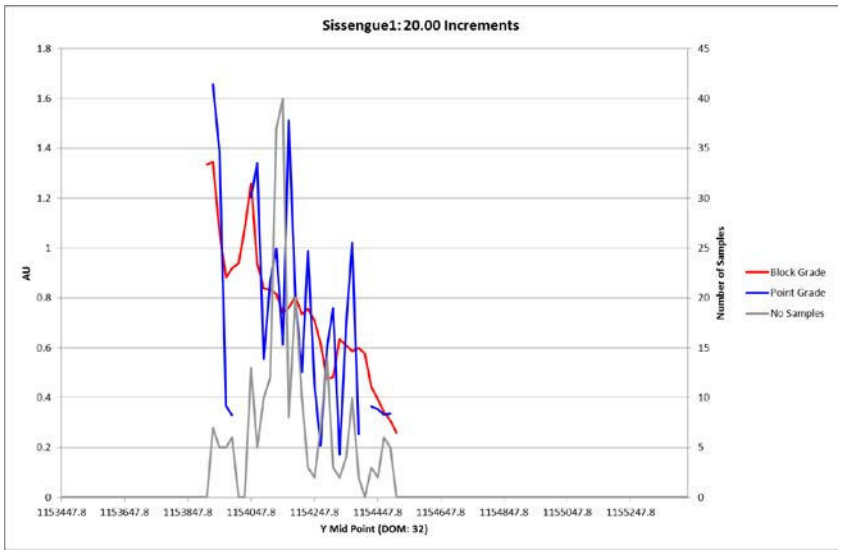

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Block Grade
Point Grade
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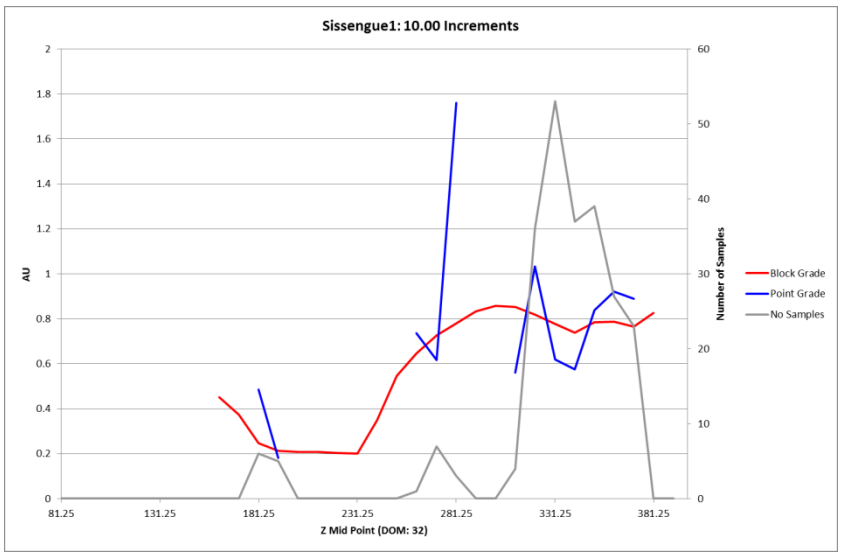

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Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
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Number of Samples
Z Mid Point (DOM: 31)
Block Grade
Point Grade
No Samples

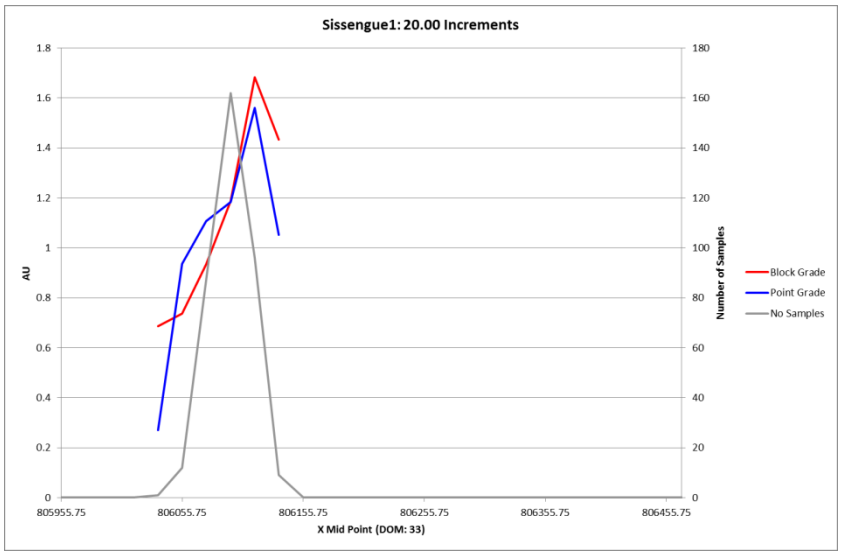
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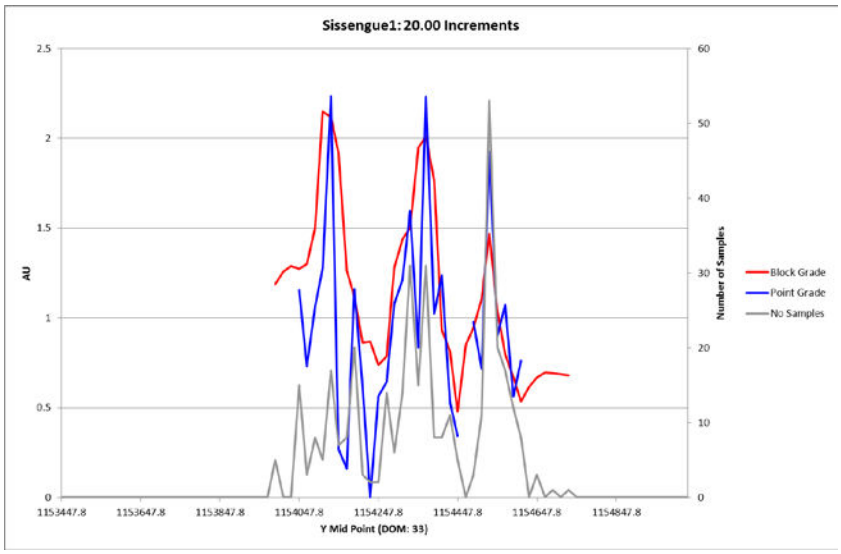

Sissengue1: 20.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
X Mid Point (DOM: 32)

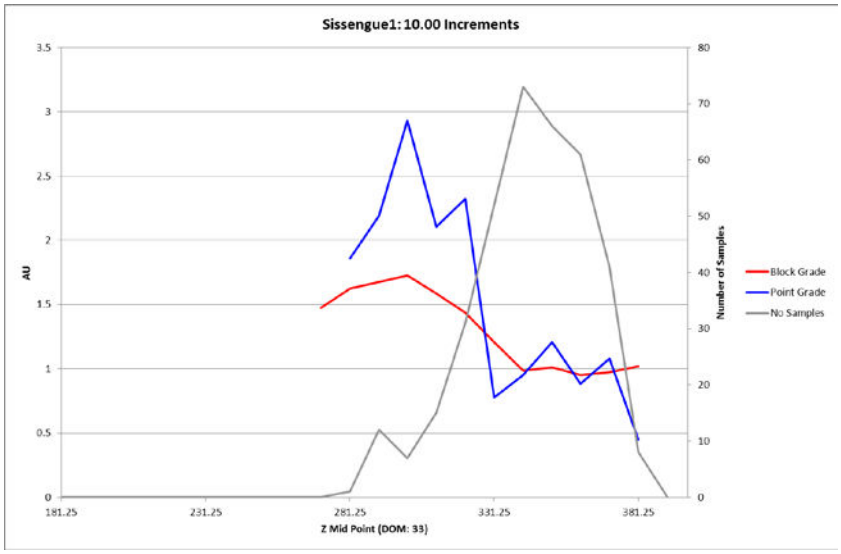

Sissengue1: 20.00 Increments
AU
Number of Samples
Block Grade
Point Grade
No Samples
Y Mid Point (DOM: 32)


Sissengue1: 10.00 Increments
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Number of Samples
Block Grade
Point Grade
No Samples
Z Mid Point (DOM: 32)

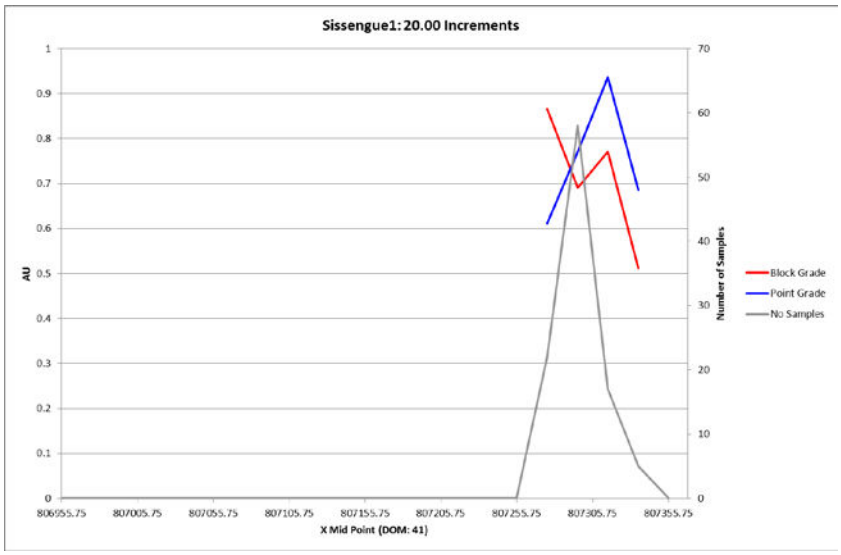
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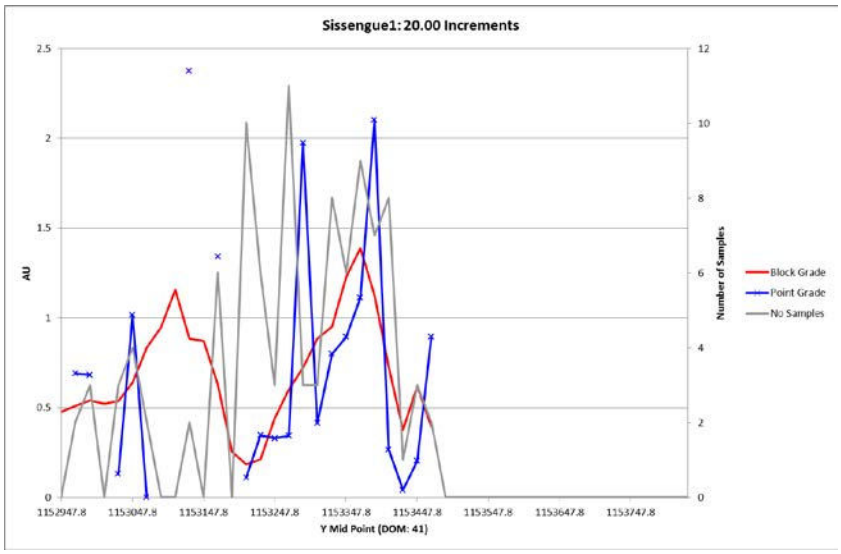

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AU
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X Mid Point (DOM: 33)
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806255.75
806355.75
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Block Grade
Point Grade
No Samples

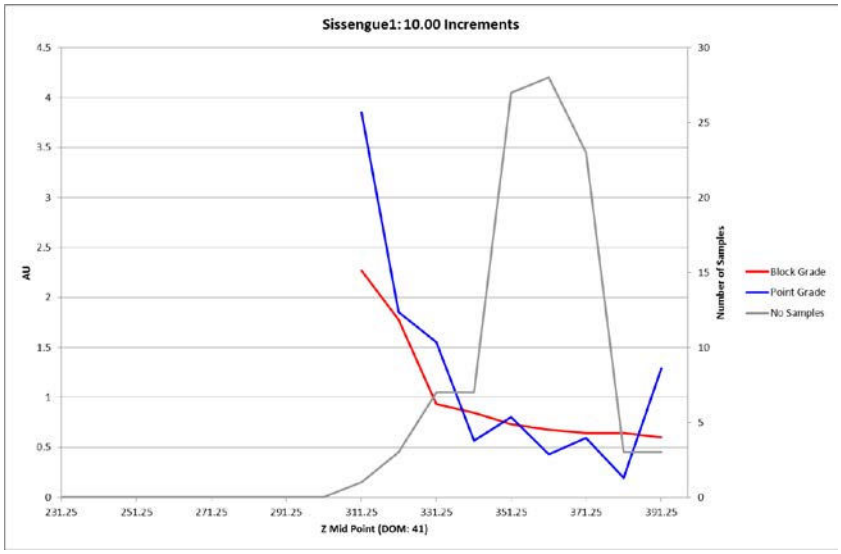

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 33)
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1153647.8
1153847.8
1154047.8
1154247.8
1154447.8
1154647.8
1154847.8
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
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Z Mid Point (DOM: 33)
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231.25
281.25
331.25
381.25
Block Grade
Point Grade
No Samples

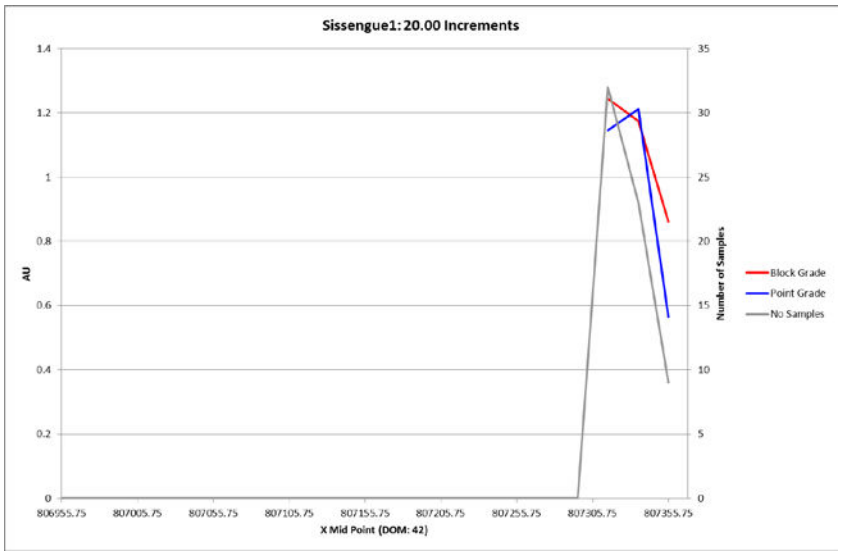
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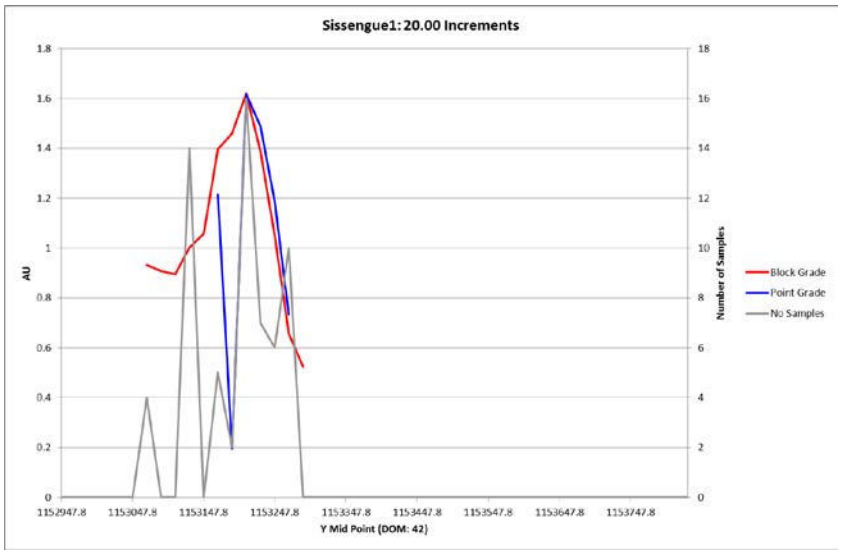

Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 41)
Block Grade
Point Grade
No Samples

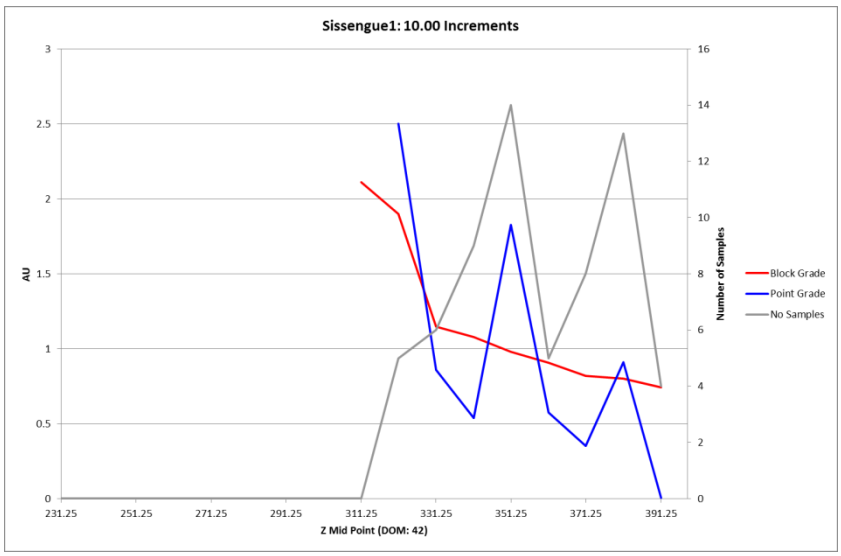

Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 41)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 41)
Block Grade
Point Grade
No Samples

Dyke south 2


Sissengue1: 20.00 Increments
AU
Number of Samples
X Mid Point (DOM: 42)
Block Grade
Point Grade
No Samples


Sissengue1: 20.00 Increments
AU
Number of Samples
Y Mid Point (DOM: 42)
Block Grade
Point Grade
No Samples


Sissengue1: 10.00 Increments
AU
Number of Samples
Z Mid Point (DOM: 42)
Block Grade
Point Grade
No Samples

Appendix H Key assumptions for optimisation

Snowden received the following advice from the client:

- Nominal gold price (for NPV calculations)
  USD1200/oz
- Elevated gold price (for MRE definition)
  USD2400/oz
- Royalties — none
- Recoveries:
  - Oxide — 92%
  - Transition — 91%
  - Granite/Porphyry — 90%
  - Sediments — 78%
- Cut-off grades:
  - Oxide — 0.8 g/t Au
  - Transition — 1.0 g/t Au
  - Granite/Porphyry — 1.1 g/t Au
  - Sediments — 1.3 g/t Au
- Whittle processing cost (treatment cost + general administration cost):
  - Oxide — ($21.79/t+$7.19/t) — $28.98/t
  - Transition — ($25.04/t+$8.14/t) — $33.18/t
  - Granite/Porphyry — ($26.29/t+$8.32/t) — $34.61/t
  - Sediments — ($30.29/t+$9.87/t) — $40.13/t
- Dilution:
  - Mining ore recovery (before diluted) — 97%
  - Mining ore dilution (at 0 g/t Au dilutant grade) — 3%
- Geotechnical parameters:
  - Oxide — (approximately from 390 mRL to 325 mRL) — 31 degrees
  - Transition — (approximately from 325 mRL to 300mRL) — 41 degrees
  - Fresh — (approximately from 300 mRL and below) — 45 degrees
- Mining cost estimate:
  - Contract miner — $3.61/t
  - Fuel ($1.40/L) — $0.59/t
  - Day-works (3%) — $0.13/t
  - Total mining cost — $4.33/t
- Whittle mining cost calculations:
  - Base cost at surface — $4.10/t
  - Incremental trucking cost — $0.005/t/vm

- The Whittle optimisation assumes that processing and infrastructure capital costs have been sunk.
- Snowden has reported the mining inventory within the USD1200/oz Whittle shell.
- No exclusion areas except the geological model limits.
- All costs are in US dollars.

APPENDIX 2

CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF QUALIFIED PERSON

I, Christopher Gorden Waller, MAusIMM(CP), as an author of this report entitled Sissingué Gold Project, Cote d'Ivoire NI 43-101 Technical Report (Revision C), prepared for Perseus Mining Limited and dated 29 May 2015, do hereby certify that:

1) I am Manager of Studies with Lycopodium Minerals Pty Ltd. My office address is Level 5, 1 Adelaide Terrace, East Perth, Western Australia 6004.

2) I am a graduate of the South Australian Institute of Technology (now The University of South Australia) 1978 with a Bachelor of Applied Science degree in Applied Chemistry.

3) I am a Member of the Australasian Institute of Mining and Metallurgy, membership number 101899 and registered as a Chartered Professional with that Institute. I have worked as a metallurgist, operations manager and study manager for a total of thirty seven years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - review and report as a consultant on numerous process facilities and mining projects around the world for due diligence and regulatory requirements
   - Study Manager on a number of feasibility studies in the mining industry in Africa, Australia and Asia.
   - operational roles to the level of General Manager at a number of mining operations in Africa and Australia (including West Africa).

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Sissingué Project site.

6) I am responsible for all of preparation of Item Numbers 1(joint), 2, 3, 13, 17, 18(excluding 18.4, 18.5, 18.6), 21(excluding 21.1.2, 21.2.1), 24, 25.5 and 27(joint) of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

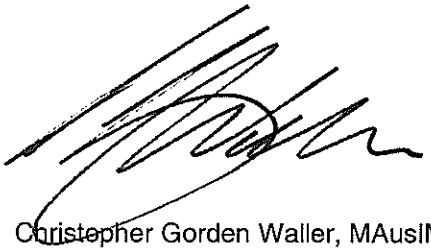
8) I have not been involved in any previous Technical Report on the Sissingué Gold Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.



10) 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.

Dated this 2nd day of June 2015



Christopher Gorden Waller, MAusIMM(CP)

CERTIFICATE OF QUALIFIED PERSON

I, Paul William Thompson, FAusIMM, as an author of this report entitled Sissingué Gold Project, Cote d'Ivoire NI 43-101 Technical Report (Revision C), prepared for Perseus Mining Limited and dated 29 May 2015, do hereby certify that:

1) I am Group General Manager Technical Services with Perseus Mining Limited. My office address is Level 2, 437 Roberts Road, Subiaco, Western Australia 6008.

2) I am a graduate of the University of Liverpool, UK with a Bachelor of Science and the University of Leeds, UK with a Master of Science.

3) I am a Fellow of the Australasian Institute of Mining and Metallurgy, membership number 209231. I have worked as a geotechnical engineer, mine planning engineer, technical services manager, study manager and general manager in the mining industry for a total of 30 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Review and report as a consultant for due diligence requirements;
   - Study Manager on a number of feasibility studies in the mining industry in Africa, Australia, Europe and Asia; and
   - Operational and corporate roles to the level of General Manager for a number of mining companies in Africa and Australia.

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

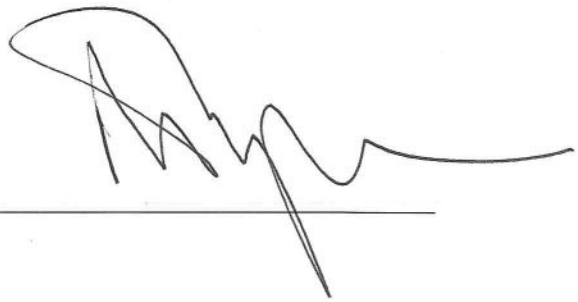
5) I have not visited the Sissingué Project site.

6) I am responsible for all of preparation of Item Numbers 1(joint), 19, 20, 22, 25.1 and 26 of the Technical Report.

7) I am not independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

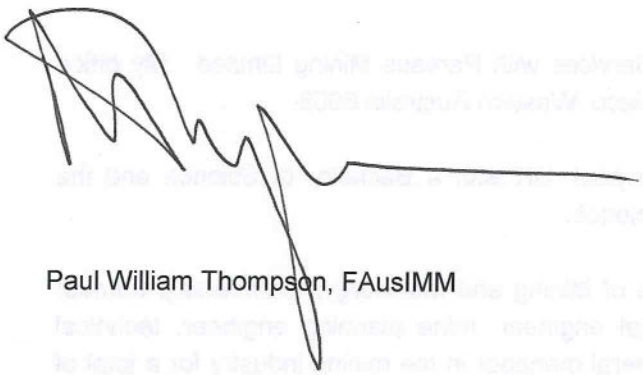
8) I have not been involved in any previous Technical Report on the Sissingué Gold Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.



10) 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.

Dated this 2nd day of June 2015



Paul William Thompson, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

I, Steffen Brammer, MAusIMM(CP), as an author of this report entitled Sissingué Gold Project, Cote d'Ivoire NI 43-101 Technical Report (Revision C), prepared for Perseus Mining Limited and dated 29 May 2015, do hereby certify that:

1) I am Senior Resource Geologist with Perseus Mining Limited. My office address is 4, Chancery Court, 147A Gifford Road, Accra, Ghana.

2) I am a graduate of the Technische Universität München / Germany with a Diploma of Geology (Dipl.-Geol.) 1994

3) I am a Member of the Australasian Institute of Mining and Metallurgy, membership number 209906 and registered as a Chartered Professional with that Institute. I have worked as an exploration geologist and resource geologist for a total of twenty years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Resource modeling and preparation of numerous resource estimates for various companies in West Africa and Central Asia, as employee and consultant; and

   - Involvement with the Project since 1999 throughout exploration and feasibility

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have visited the Sissingué Project site on multiple occasions.

6) I am responsible for all of preparation of Item Numbers 1(joint), 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2 and 27(joint) of the Technical Report.

7) I am not independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

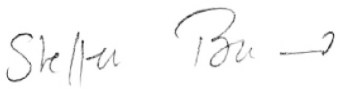
8) I have been involved in the previous Technical Reports on the Sissingué Gold Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Steffen Brammer

10) 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.

Dated this 2nd day of June 2015



Steffen Brammer, MAusIMM(CP)

CERTIFICATE OF QUALIFIED PERSON

I, Joe McDiarmid, as coauthor of this report entitled Sissingué Gold Project, Cote d'Ivoire NI 43-101 Technical Report (Revision C), prepared for Perseus Mining Limited and dated 29 May 2015, do hereby certify that:

1) I am a Principal Mining Engineer with RungePincockMinarco Ltd. My office address is Level 2, 131 St Georges Terrace, Perth WA 6000.

2) I am a graduate of Curtin University of Technology (formally Western Australian School of Mines) and hold a Bachelor of Engineering in Mining.

3) I am a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy membership number 112414. Since 1996, I have been continuously and actively involved in the Mineral Resources sector and engaged in the assessment, development and operation of projects both within Australia and overseas. During this period I have worked in various applicable roles from a Mining Operator to Mine Manager and Principal Mining Engineer.

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Sissingué Project site.

6) I am responsible for Item Numbers 1(joint), 15, 16, 21.1.2, 21.2.1, 25.3, 25.4 and 27(joint) of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8) I have not been involved in any previous Technical Report on the Sissingué Gold Project.

9) I have read the Instrument and Form 43-101F1 (the "Form") and the Technical Report has been prepared in compliance with the Instrument and the Form.

10) 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.

Dated this 2nd day of June 2015



CHARTERED PROFESSIONAL
AusIMM
THE MINERALS INSTITUTE
MINING
JOE McDIARMID

Joe McDiarmid
BEng (Mining), MAusIMM (CP), RPEQ

CERTIFICATE OF QUALIFIED PERSON

I, James Christopher Lane, as an author of this report entitled Sissingué Gold Project, Cote d'Ivoire NI 43-101 Technical Report (Revision C), prepared for Perseus Mining Limited and dated 29th May 2015, do hereby certify that:

1) I am Principal Consultant of WorleyParsons Consulting. My office address is Level 7 QV1 Building 250 St Georges Terrace Perth Western Australia 6000.

2) I am a graduate of Canberra University and University of Sydney in conjunction with the University of New South Wales with a Bachelor of Applied Science (BApp Sc) and an Executive Master of Business Administration (EMBA) respectively.

3) I am a Chartered Engineering Geologist (CGeol - Geological Society London); Registered Professional Geoscientist of the Australian Institute of Geoscientists (AIG - RPGeo 10009) in the fields of Engineering and Environmental Geology; Chartered Geologist of the AusIMM(CP) Member 109219 and a Registered Professional Engineer of Queensland in the field of Geotechnical (mining) RPEQ 13083. I have been working for 40 years in the geotechnical field since graduation as an Engineering Geologist and I have been designing tailings storage facilities for more than 30 years. My relevant experience for the purpose of the Technical Report is:

   - review design and report as a consultant on numerous tailings storage facilities around the world for due diligence and regulatory requirements
   - I have been working on projects in West Africa (Cote D'Ivoire, Ghana, Mali and Senegal) since 1988.

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have visited the Sissingué Project Site on 3 separate trips.

6) I am responsible for all of preparation of Item Numbers1(joint), 18.4, 18.5, 18.6 and 27(joint) of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have been involved in any previous Technical Report on the Sissingué Gold Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Chris Lane

10) 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.

Dated this 2nd day of June 2015

James Christopher Lane