Form 43-101F1

TECHNICAL REPORT

Mineral Resource and Mineral Reserve Estimate for the

Sukari Gold Project, Egypt

Effective Date: 30 June 2015

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1 SUMMARY

1.1 Introduction

This report provides an updated estimate of Mineral Resources and Mineral Reserves for the operating mine site at Sukari, Egypt (defined throughout this document as either the Sukari Gold Project or Sukari). The report has been prepared in accordance with disclosure and reporting requirements set forth in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (NI 43-101) of the Canadian Securities Administrators (including NI 43-101's Companion Policy and Form 43-101F1).

On 10 September 2015, Centamin plc announced on the LSE and TSX that the Sukari estimates of Mineral Resources and Reserves had been updated.

This report is the compilation of work performed by a number of engineering and consultancy organizations commissioned by Centamin plc[1] (defined throughout this document as either Centamin or the Company) on its Sukari Gold Project as well as Centamin employees dating back to 2003 and draws heavily on the previous technical report dated 30 January 2014, which report has been filed on SEDAR at www.sedar.com. Technical information relating to project development, mining and processing has not changed significantly since reported in the technical report dated 30 January 2014 and as such is essentially repeated in this report with minor updates.

The operating company of Sukari is Sukari Gold Mines (SGM), and is jointly owned by Pharaoh Gold Mines NL (PGM) a wholly owned subsidiary of Centamin, and the Egyptian Mineral Resource Authority (EMRA) on a 50/50 basis.

SGM commenced open-pit mining on 1 February 2009. Commissioning of the Stage 1 carbon in-leach (CIL) plant commenced in December 2009 along with initial gold production from this circuit. In April 2010 the Stage 2 Flotation Circuit commissioning also commenced.

The Stage 3 expansion of the process plant incorporated two secondary crushers and associated infrastructure to increase plant throughput from approximately 4.0 million tonnes per annum (Mtpa) to approximately 5.0 Mtpa. The Stage 3 expansion project was completed in July 2011.

Step-out and infill drilling carried out in 2011 continued to identify significant additional mineralization at Sukari in addition to the commencement of initial grade control (close spaced) drilling in the Sukari porphyry. As a result Hellman & Schofield Pty Ltd (H&S) was commissioned by Centamin to update the estimate of Mineral Resources in September 2011. And the Mineral Reserve estimate was subsequently updated in December 2011.

Construction of the Stage 4 expansion commenced during 2012 and comprised the addition of a new primary crusher in the process plant and a second train of milling, flotation, thickening and carbon regeneration, and an upgrade of the regrind circuit to increase plant throughput to a nominal 10.0 Mtpa. Commissioning was completed in the second half of 2014. The expanded plant is expected to achieve a production rate of 11.0 Mtpa from the fourth quarter of 2015 and the Mineral Reserve is estimated on the basis of a production rate of 11.0 Mtpa.

The Sukari underground began production in 2011 with a decline accessing the Amun portion of the orebody. The underground mine has expanded significantly in the past four years. The

[1] Reference to obligations undertaken by Centamin may include undertakings by Centamin's subsidiary entities including PGM and/or SGM.

current underground expansion includes a new decline (Ptah) to its target depth below the existing area of operation.

Centamin commissioned MPR Geological Consultants Pty Ltd to update the estimate of the Mineral Resources, and Cube Consulting Pty Ltd to update the estimate of underground Mineral Resources. The open-pit Mineral Reserve estimate was updated by AMC Consultants Pty Ltd and the underground Mineral Reserve estimate was updated by Crosscut Consulting Pty Ltd, with all estimates as at end June 2015.

1.2 Location

The Sukari gold deposit is located in the Eastern Desert of Egypt at 24º 56’ 50”N 34º 42’ 27”E, about 23 kilometres southwest of the Red Sea coastal town of Marsa Alam. The region has a very long history of mining and exploration carried out through all stages of history, from Pre-dynastic (Ca 3,200 BC), through Ptolemaic, Roman, Arab and British colonial to the present day.

1.3 Ownership

In 1994, PGM negotiated an exploration and mining agreement (Concession Agreement), with the Egyptian Geological Survey and Mining Authority (EGSMA; now the Egyptian Mineral Resources Authority (EMRA)) and the Egyptian Government, to explore for gold and associated minerals in the Eastern Desert of Egypt. The Concession Agreement (CA) was declared into Law 222 of 1994 and came into effect on 29 January 1995 pursuant to which PGM had the right to explore and develop gold and associated metal deposits within the concession area.

On 4 November 2001 PGM was formally notified by EMRA that, in accordance with the terms of the CA, the feasibility study submitted by PGM and dated the 26th of October 2000 relating to the Sukari Gold Project, had been accepted by EMRA and had demonstrated the existence of a “Commercial Discovery” at the Sukari Gold Project. In April 2005 EMRA and PGM agreed that an exploitation area of 160 km^2 containing the proposed Sukari Gold Project and surrounding prospects was appropriate and the Minister of Petroleum approved an exploitation lease covering this area on 24 May 2005.

Under the terms of the CA, PGM has title for a period of 30 years from 24 May 2005, renewable at PGM’s election for a further period of 30 years.

Following demonstration of the commercial discovery, PGM and EMRA were required to establish an operating company. SGM was incorporated under the laws of Egypt on 13 April 2006 to conduct exploration, development, exploitation and marketing operations in accordance with the CA. Responsibility for the day-to-day management of SGM rests with the Sukari General Manager who is appointed by PGM.

The validity of the exploitation lease is currently the subject of a case in the Egyptian courts. The details of this action are set out in detail in Centamin’s regulatory filings including its quarterly, half yearly and annual results and the latest update can be found in Note 7 of Centamin’s results for the six month period ending June 2015.

1.4 Geology and Mineralization

The Sukari felsic porphyry outcrop is located in an easterly dipping sequence of andesite flows, serpentinites and associated volcanoclastic sediments, mainly tuffs and epiclastics. It strikes for 2.3 km and is 100 m to 600 m thick. It forms a jagged-toothed, strong topographic high up to 250 m above wadi level (390 m above sea level, (ASL)). Wadi drainage plains pass to the east and west of the outcrop, and the sharply incised green-brown Red Sea Hills surround that. The area is arid and almost bare of vegetation.

The host to gold mineralization is the Sukari felsic porphyry unit that contains a variable series of sub-units ranging from minor acid and felsic rhyolite and dacite to coarser grained feldspar and quartz porphyries, quartz diorites and granodiorites, dipping from moderate east in the south, through sub-vertical to slightly overturned “hangingwall” contact in the north. The porphyry units are envisaged to have formed during a more acid event in an overall intermediate environment in which some rhyolite–dacite flows were later intruded by high level porphyry as a multiple event (Cavaney, 2005).

1.5 Mineral Resource

The total Mineral Resource estimate is 13 million ounces (Moz) contained gold (Au). The total Mineral Resource is reported as an open-pit Mineral Resource at a 0.3 g/t cut-off grade.

An underground Mineral Resource has also been estimated. The underground resource is located within the boundaries of the open-pit resource, and is estimated from the same mineralization. As such, the underground Mineral Resource is a sub-set of the total or open-pit Mineral Reserve estimate and these estimates are not cumulative.

The underground Mineral Resource estimate is 1.0 Moz contained Au at a 2.0 g/t cut-off grade.

1.5.1 Open-pit Mineral Resource

MPR Geological Consultants Pty Ltd (MPR) was most recently commissioned by Centamin in 2015 to undertake an estimate of mineral resources for the Sukari gold prospect. Estimates were prepared with reference to the Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards (2005) and CIM Best Practice Guidelines (2003) for preparing mineral resources and mineral reserves.

Estimation of mineral resources has been undertaken using Multiple Indicator Kriging (MIK) with block support adjustment. The method estimates the histogram of drillhole sample grades in large panels (20 m east x 25 m north x 10 m elevation), using variograms of gold grade indicators at 14 thresholds. The size of model panels was based on drillhole spacing. A block support adjustment was then applied to estimate the histogram of grades of selective mining units in each panel.

For Sukari, a selective mining unit (SMU) measuring 5 m east x 8 m north x 10 m elevation has been assumed. The shape of the local block gold grade distribution was assumed lognormal and an additional adjustment correction for Information Effect has also been applied, to make allowance for the fact that ore selection will be based on an estimate of the grade of a mining block generated from grade control sampling, not perfect knowledge of block grades.

The mineral resource within each panel has been allocated a confidence category based on the spatial distribution of samples in the Kriging neighbourhood. This classification scheme takes into account the uncertainty in estimates resulting from the proximity and locations of the informing samples.

The Mineral Resource at Sukari, estimated to approximately 1,350 metres depth below surface[2], is shown in Table 1.5-1. The resource dataset comprise 252,449 two-metre downhole composites and surface rock chip samples. The resource estimate at multiple cut-off grades is reported below, with the preferred cut-off grade emboldened, and are based on the open-pit mined surface as at 30 June 2015 and have also been reduced by the volume mined by current and historical underground mining.

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[2] Surface is defined as the initial topography, which includes the Sukari hill and base ground below the hill.

Table 1.5-1 Total Mineral Resource for Sukari

Cut-off	Measured		Indicated		Total Measured & Indicated			Inferred		
	Tonnes (Mt)	Grade (g/t)	Tonne s (Mt)	Grade (g/t)	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
0.3	**198**	**1.05**	**188**	**1.02**	**386**	**1.03**	**12.9**	**33**	**1.0**	**1.1**
0.4	160	1.22	152	1.18	312	1.20	12.0	26	1.2	1.0
0.5	133	1.38	124	1.34	257	1.36	11.2	21	1.3	0.9
0.7	95	1.69	87	1.66	182	1.68	9.8	15	1.7	0.8
1.0	62	2.14	56	2.12	118	2.13	8.1	9	2.1	0.6

Notes to table:

- The Mineral Resource estimate is based on the open-pit mined surface as at 30 June 2015 and depleted for underground mine workings as at 30 June 2015.
- All available assays as at February 2015.
- Resource dataset comprises 252,449 two metre down hole composites and surface rock chip samples.
- Mineral Resource is reported inclusive of those resources converted to Proven and Probable Mineral Reserves.
- The resource is an estimate of recoverable tonnes and grades using Multiple Indicator Kriging with block support correction.
- Measured Resources lie in areas where drilling is available at a nominal 25 x 25 metre spacing, Indicated Resources occur in areas drilled at approximately 25 x 50 metre spacing and Inferred Resources exist in areas of broader spaced drilling.
- The resource model extends from 9,700 mN to 12,200 mN and to a maximum depth of 0 mRL (a maximum depth of approximately 1,000 metres below wadi level).

1.5.2 Underground Mineral Resource

Cube Consulting Pty Ltd (Cube) was commissioned by Centamin in 2015 to undertake an update of the underground Mineral Resource estimate for the Sukari gold prospect. The underground Mineral Resource estimate, shown in Table 1.5-2, is included within the total estimated Mineral Resource shown in Table 1.5 1.

The estimates were prepared with reference to the Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards (2005) and CIM Best Practice Guidelines (2003) for preparing mineral resources and mineral reserves.

Table 1.5-2 Underground Mineral Resource for Sukari

Resource	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	1.85	6.5	0.39
Indicated	2.82	7.0	0.63
Total Measured and Indicated	**4.67**	**6.8**	**1.02**
Inferred	6.97	5.6	1.24

Notes to table:

- Totals may not equal the sum of the components due to rounding adjustments.
- The underground Mineral Resource is reported above 2 g/t cut-off within interpreted mineralized domains.
- The underground Mineral Resource estimate is depleted for underground mine workings as at 30 June 2015.
- All available information has been used including mapping from underground mining and assays as at June 2015.
- Available resource data resulted in 21,369 one metre down hole composites used for grade estimation.
- The underground Mineral Resource is estimated utilising a single Indicator weighted Kriging method (IK) to estimate gold for each of the mineralization domains.
- Underground Measured Mineral Resources are defined by a drill spacing of at least 20 m x 20 m and confined to the interpreted mineralization defined by underground mine development. Underground Indicated Mineral Resources are defined as areas outside the Measured Mineral Resources and defined by approximately 20 m x 20 m drill spacing. Underground Inferred Mineral Resources include all remaining estimated mineralization defined by a drill spacing of approximately 50 m x 50 m.
- Underground Mineral Resource is reported inclusive of those resources converted to Proven and Probable Mineral Reserves.
- The underground resource is located within the boundaries of the open-pit resource, and is included within that total.

The underground Mineral Resource is reported above a cut-off grade of 2 g/t Au within the interpreted mineralized domains. The mineral resource grade does not include allowances for underground mining dilution or ore loss.

The total underground Measured and Indicated gold ounces have increased moderately since last reported as a result of the closer spaced drilling and exposure from underground mining confirming confidence in the geology and grade continuity. The Inferred tonnes have increased significantly, mainly due to the inclusion of the Ptah mineralization with the Sukari underground mineral resource.

1.6 Mineral Reserve

The total combined open-pit and underground Mineral Reserve estimate for contained gold is 8.8 Moz, which is an increase of 7% from the previous 8.2 Moz estimated at 30 September 2013. The increase is due to lower operating mining and processing costs associated with lower international fuel prices, and continued drilling from underground to move ounces up through the resource categories and increase the underground Mineral Reserve.

1.6.1 Total Mineral Reserve

The total Mineral Reserve estimate, open-pit and underground, is tabulated in Table1.6-1.

Table 1.6-1 Total Combined (Open-pit and Underground) Mineral Reserve for Sukari

	Proven		Probable		Total		
	Tonnes (Mt)	Grade (g/t)	Tonnes (Mt)	Grade (g/t)	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
New Mineral Reserve (1-3)	**152**	**1.05**	**101**	**1.15**	**253**	**1.09**	**8.8**
Previous Mineral Reserve (4)	119	1.06	111	1.17	230	1.11	8.2

Notes to table:

Totals may not equal the sum of the components due to rounding adjustments.

(1) Based on a metal price of US$1,300/oz Au and includes:
Open-pit reserve totalling 229 Mt @ 1.09g/t for 8.0 Moz
Underground reserve totalling 2.7 Mt @ 6.0g/t for 0.5 Moz
Surface stockpiles totalling 21 Mt @ 0.42g/t for 0.3 Moz

(2) Based on open-pit mined surfaces as at 30 June 2015 and underground workings as at 30 June 2015

(3) Final open-pit design has a waste to ore ratio of 5.9:1 (including the in-pit dump leach ore, but not stockpiles)

(4) As at 30 September 2013 using US$1,300/oz Au

The Mineral Reserve is based on a gold price of US$1,300/oz. The open-pit cut-off grades are 0.08 g/t Au for oxide (including dump leach material) and 0.42 g/t for both transition and sulphide material. The underground cut-off grade is 3.0 g/t. The reference point for the Mineral Reserve estimate is the mill feed, reported as mined ore delivered to the plant or dump leach processing facilities.

The work satisfies the reporting requirements of the CIM (2004) guidelines for reporting mineral reserves.

1.6.2 Open-pit Mineral Reserve

The open-pit Mineral Reserve for Sukari was estimated by AMC Consultants Pty Ltd (AMC). The open-pit Mineral Reserve estimate was based upon the 2015 Mineral Resource estimate model prepared by MPR. The open-pit Mineral Reserve is reported as at the end of June 2015.

The Mineral Reserve for the open-pit, including stockpiles, was estimated to be 250 Mt of ore at an average grade of 1.03 g/t Au, containing 8.3 Moz of gold. The open-pit Mineral Reserve is summarized by category in Table 1.6-2. The open-pit Mineral Reserve is contained within designed and scheduled open-pits which were based upon the results of Lerchs Grossman pit optimizations of Measured and Indicated Resources. The Inferred Resources that occur within the pit design are treated as mineralized waste in the production schedule and contribute no value to the economic evaluation of the Mineral Reserve.

Table 1.6-2 Open-pit Mineral Reserve for Sukari

Mineral Reserve Category	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Proven – in pit	130	1.11	4.6
Probable – in pit	99	1.07	3.4
Proven – Stockpile	21	0.42	0.3
Total Mineral Reserve	**250**	**1.03**	**8.3**

Notes to table:

- Totals may not equal the sum of the components due to rounding adjustments.
- Mineral Resources are reported inclusive of those resources converted to Proven and Probable Mineral Reserves.

The open-pit resource model contains an allowance for dilution within the limits of the chosen SMU. The Qualified Person (QP), Patrick Smith, notes that the open-pit resource model contains inbuilt allowances for dilution and mining losses, and that it is reasonable, at this stage, to not include any further allowance for these effects in the Mineral Reserve estimate.

The Sukari pit produces ores of varying stages of weathering. The processing costs and metallurgical recoveries are modelled to vary with the extent of the weathering, and hence the cut-off grade applied also varies with the weathering state. The processing plant cut-off grades for the oxide, transition and sulphide rock types were 0.40 g/t, 0.42 g/t and 0.42 g/t, respectively. The dump leach cut-off grade for the oxide rock type was 0.08 g/t.

The Sukari pit will be developed in a number of open-pit mining stages.

1.6.3 Underground Mineral Reserve

The Underground Mineral Reserve for Sukari was estimated by Crosscut Consulting (Crosscut).

The Mineral Reserve for the Sukari underground mine is based on the underground resource model produced by Cube Consulting in July 2015. All resources are in a fresh state of weathering.

Dilution and loss have been applied based on the type of stoping being undertaken (detailed in Section 16.2.3). The cut-off grade used for the underground Mineral Reserve is 3.0 g/t, using the same gold price assumptions and gold recoveries as used in the open-pit Mineral Reserve. The underground Mineral Reserve is based on designed stopes and development within the resource model (detailed in Sections 16.2.2 and 16.2.4), and costed using current costs and assumption of continuation of the current production rate of 1 Mtpa from combination of both stoping and development.

The Mineral Reserve for the Sukari underground mine is estimated to be 2.7 Mt of ore at an average grade of 6.0 g/t Au, containing 520 thousand ounces (koz) of gold. The Mineral Reserve is summarized by category in Table 1.6-3Table 15.3-1.

Table 1.6-3 Underground Mineral Reserve for Sukari

Mineral Reserve Category	Tonnes (Mt)	Grade (g/t)	Contained Gold (koz)
Proven	1.02	6.1	200
Probable	1.70	5.9	320
Total Mineral Reserve	**2.72**	**6.0**	**520**

Notes to table:

- Totals may not equal the sum of the components due to rounding adjustments.
- Mineral Resources are reported inclusive of those resources converted to Proven and Probable Mineral Reserves.

The underground Mineral Reserve estimate includes broken stocks (at 30 June 2015) of 111 thousand tonnes (kt) of ore at an average grade of 9.5 g/t Au and containing 34 koz of gold. The balance of the underground Mineral Reserve was estimated from development and stope designs accessed via the Amun and Ptah Declines.

Stoping dilution varied based on the stoping method employed, and varied from 10% at 0.8 g/t Au for flat dipping room and pillar stopes, to 15% at 0.4 g/t Au for steeply dipping long hole stopes. Development headings were diluted by 5% at 0.8 g/t Au.

Mining losses varied based on type of stoping used, and varied from 10% for steeply dipping long hole stopes to 50% for flat dipping long hole stopes. Development was assumed to have 0% mining loss.

Development and stope designs were based on the 2015 underground resource model prepared by Cube, and depleted for underground workings as at end of June 2015. A cut-off grade of 3.0 g/t Au was used for design purposes, and the calculated breakeven grade was 2.0 g/t Au.

1.7 Open-pit Mining

All ore and waste is being mined using open-pit gold mining methods. The mine currently uses four (4) CAT 6040 face-shovels, four (4) O&K RH120 backhoe excavators and forty-eight (48) CAT 785C haul trucks to carry out the bulk of the ore and waste mining. All ore and waste requires blasting, with no “free dig” material. Ore delineation is determined through data collected in reverse circulation (RC) drilling campaigns and estimated using conditional simulation.

Ore is hauled to the run-of-mine (ROM) pad next to the processing plant and either direct tipped to the crusher or stockpiled for future reclaim to maintain the 11 Mtpa throughput rate.

Waste is used for construction or is hauled to the waste dumps on the east, north and south sides of the pit. Waste is used to construct ramps to provide access on the hillside and to provide fill for the tailings storage facility (TSF).

1.8 Underground

The Sukari underground mine is a trackless diesel mine, with all equipment rubber tyred and self-powered. The workings are accessed via a ramp system declining at a gradient of 1 in 7. Levels are cut every 15 vertical metres, commencing at the 920 level.

Underground mining operates a fleet of conventional rubber-tyred 50-tonne capacity trucks and Elphinstone R2900 loaders for haulage of broken rock. Ore is sourced from development ore drives, infrastructure drives designed in the mineralized porphyry and production stoping areas. Ore is hauled to the ROM pad and dumped for later reclaim to feed the process plant, as the underground trucks do not have sufficient clearance between the tub ducktail and the dump pocket to enable direct tip.

Waste is generated mainly from the declines. It is hauled to an underground specific surface waste dump adjacent to the portal.

Blast holes are bored by rubber-tyred, electro-hydraulic drilling rigs, using 1,000 volt reticulated power. The carriers have diesel engines to move between drilling sites. A fleet of two boom drill jumbos is used for horizontal development and rock bolting, using 43 mm diameter holes. Stoping holes are drilled using a boom mounted long hole drill with carousel, capable of drilling 89 mm diameter holes to a depth of 50 m.

The Amun decline accesses the Hapi, Amun Deeps, Osiris and Horus zones. The Ptah decline accesses the northern portion of the resource. The Horus decline links these declines between 709 metres Relative Level (mRL) on Ptah and 639 mRL on Amun, a distance of 500 metres. The exhaust ventilation system, which comprises a set of declines and rises, also provides a second means of egress.

The mine has a number of four-man and sixteen-man mobile refuge chambers. Fixed permanent fresh air bases are also in place or planned. An emergency set of services runs through the exhaust system, providing an independent source of compressed air and fire-fighting water. Communications are generally by a leaky feeder radio system, with an emergency copper-wired telephone system also in place.

Ground support for development drives uses wet sprayed fibrecrete. The fibres are polymer and are used to reinforce the sprayed concrete shell. The decline walls and back are sprayed from the sill, while the porphyry drives are sprayed shoulder to shoulder. All openings have split set rock bolts installed. These are grouted in the weaker decline rock and the kaolinite rich shear zones. Intersections are also supported with six-metre long steel cable bolts that are plated after grouting.

1.9 Metallurgy

The metallurgical behaviour of Sukari ores has been established through various testwork programmes dating back to 2000 performed variously by Lakefield Oretest Pty Ltd, Independent Metallurgical Testing Laboratories Pty Ltd (IML) and AMMTEC Ltd (AMMTEC), and subsequently the testwork results were combined with production data results, comminution, flotation and cyanidation of flotation concentrate and tailings were the dominant processes tested.

The ore is relatively hard and competent being hosted in porphyry and is suitable for semi-autogenous grind (SAG) milling. The gold is fine and associated with pyrite that is readily floated and ultra-fine grinding renders the gold amenable to cyanidation.

Near surface ore has undergone varying degrees of oxidation and core logging has classified the ore into five stages of oxidation ranging from M1 (un-oxidized sulphide ore) through M5 (highly weathered oxide ore). Definitive testwork by AMMTEC on the five ore classifications provided a basis for prediction of gold recovery based on flotation response and resulted in selection of three processing routes or circuits:

- Circuit 1 - Whole ore direct cyanidation of “oxide ore” (M5).
- Circuit 2 – Flotation of “mixed ore” with separate cyanidation of reground concentrate and flotation tail (M2, M3 and M4).
- Circuit 3 – Flotation of “sulphide ore” with cyanidation of reground concentrate and discarding flotation tail (M1).

1.10 Process Plant

The process plant has been constructed and operated over a number of staged iterations.

Stage 1 comprised the coarse ore stockpile, milling and CIL circuits relocated from Kori Kollo combined with a refurbished primary crusher and new elution/gold room facility for the treatment of oxide ore, using whole ore direct cyanidation at the commencement of the operation. Commissioning of this first stage commenced in December 2009 along with initial gold production from this circuit.

Stage 2 comprised the addition of a new flotation, thickening, regrind, concentrate CIL and concentrate elution circuits for the treatment of sulphide bearing ore at a design rate of 4 Mtpa of mill feed. Operation of this stage commenced in April 2010.

Stage 3 comprised of the addition of a secondary crushing circuit designed to reduce the ore feed size to the SAG mill to allow an increase in the plant throughput rate from the initial design of 500 tonnes per hour (tph) (4 Mtpa) to a new nominal 625 tph (5 Mtpa). The Stage 3 expansion project was completed in July 2011 and operated at 6 Mtpa, 20% above nameplate. The secondary crushing circuit was designed and built for a nominal rate of 10 Mtpa to allow for future, Stage 4, plant expansion.

Stage 4 comprised the addition of a new primary crusher and second train of milling, flotation and thickening, as well as upgrading the existing regrind circuit. The plant expansion was designed to increase the nominal capacity of the process plant from 5 Mtpa to 10 Mtpa. In addition, a new regeneration kiln designed specifically for highly fouled carbon and sized for the gold loadings on carbon at 10 Mtpa was installed. Construction commenced during 2012 and commissioning was completed in the second half of 2014. It is expected that the plant will operate at 11 Mtpa, 10% higher than the nameplate capacity of 10 Mtpa.

Gold room facilities are customized for each Zadra elution circuit each comprising electrowinning cells and smelting furnace. The customization is due to the significant differential in required carbon handling capacity through each circuit and the security risk in transporting loaded carbon or cathodes.

Due to the lack of a local source of fresh water in the Sukari area, seawater is used for process water and is pumped a distance of 25 km from the Red Sea for process plant and mining requirements.

A dump leach facility is operated in parallel with the process plant for the treatment of low grade oxide ore.

Pregnant solution from the dump leach is pumped from the pregnant liquor pond either to the processing plant Stage 1 CIL circuit for recovery of gold onto carbon along with the plant feed, or, as is currently the case, pumped predominantly to a series of carbon columns at the dump leach facility with the tails solution recycled over the heap. The loaded carbon is transported to the gold room at the plant for stripping and electrowinning.

Power is generated by two power station facilities and is distributed to area substations.

1.11 Services and Infrastructure

On-site infrastructure comprises the following facilities:

- Power is generated from three power station installations totalling 75 MW.
- A fuel storage facility comprising 2 x 2,030 m^3, 2 x 80 m^3 and 1 x 987 m^3 diesel tanks complete with off-loading facilities and distribution pumps.
- On site buildings for administration offices, first-aid clinic and security.

- On site buildings for the process plant including sub-stations, control rooms, plant offices, plant maintenance workshop, main warehouse, laboratory and reagent storage buildings.
- Mine facilities including mine offices, mine maintenance workshop and change room.
- Fire protection systems including fire pump with reserve storage and hydrants.
- A low security area fence is provided around the plant site offices/buildings, fuel depot, power station and water ponds. A high security area fence is provided around the process plant and gold room area.
- A communications network with satellite and terrestrial connections to existing Egyptian communication networks.
- An accommodation village to house up to 800 permanent operations personnel, complete with kitchen/mess, ablution blocks, laundry and recreation area.

Off-site infrastructure comprises the following facilities:

- The main access road, 26 km in length, running through the Umm Tunduba wadi from Marsa Alam to the project site.
- Two seawater supply systems, incorporating seawater intake pumps and coastal wells, booster pumping stations, and pipelines running parallel from the Red Sea coast to Sukari. The pipelines have been buried to a depth of 1 m.

1.12 Tailings Storage Facility

Knight Piésold Ltd (KP) was engaged by PGM to design the tailings storage facility for the Sukari mine site.

Site surveys and geotechnical investigations were carried out at alternative locations and a site was chosen immediately adjacent to the proposed plant site.

To avoid tailings and strongly saline water penetrating the wadi groundwater, the TSF basin is lined with 1.5 mm thick HDPE material.

Tailings are deposited by sub-aerial techniques and decant water is pumped from a floating barge back to the plant process water tank.

The embankment is designed to be raised in annual increments by upstream construction methods. The most recent design allows for a total tailings capacity of 68 Mt.

1.13 Environment

In 2007, an Environmental and Social Impact Assessment (ESIA) was completed by Environics, an Egyptian company that specializes in Environmental Management Systems, and approved by the Egyptian Environmental Affairs Agency. Various addenda have subsequently been prepared and submitted to the authorities, and approved.

The risk management framework adopted for the determining, assessing and managing the environmental risks was the Equator Principles and all the system elements, including public consultation, were considered during the preparation of the ESIA.

Public Consultation meetings were held in Marsa Alam during the course of preparing the ESIA. All concerns raised were satisfactorily answered or mitigated and no significant objections are outstanding.

The ESIA and its addenda set the framework for the environmental management scheme that is currently adopted at Sukari and delineate measures that should be taken to prevent or minimize potential impacts. We maintain high standards of environmental performance and meet, and when practical exceed relevant legal requirements.

1.14 Project Implementation

Stage 1 and 2

SGM completed the Stage 1 and 2 projects for a capital cost of US$267.5 million. Production ramp-up continued throughout 2010 and 2011.

Stage 3 Secondary Crushing Expansion

The Stage 3 expansion of the process plant incorporated two secondary crushers and associated infrastructure to increase plant throughput to approximately 5.0 Mtpa.

The Stage 3 expansion project was completed in July 2011 for a capital cost of US$15.4 million. Production ramp-up continued throughout the second half of 2011.

Stage 4 Expansion

The Stage 4 expansion of the process plant included a new primary crusher, a third secondary crusher, coarse ore stockpile, milling, flotation and thickening circuits, along with all associated infrastructure. The expansion has been designed to increase plant throughput to approximately 11 Mtpa.

SGM completed the Stage 4 expansion of the process plant for a capital costs US$331.2 million. Commissioning was completed in the second half of 2014.

1.15 Capital Costs

Centamin has, in 18 May 2015, published a five-year cost forecast from 2015 to 2019. The cost forecast includes sustaining capital of circa US$80 million per annum. No material expansion capital expenditure is planned.

1.16 Operating Cost Estimate

Operating costs were derived from actual production and cost performance data and projected costs and input into the open-pit optimization and underground evaluations to estimate the Mineral Reserves. Costs detailed in completed and active contracts, updated labour and consumable costs (mainly diesel fuel) and recent quotations for reagent supply and maintenance consumables.

The open-pit mine activity costs include all operating costs for excavating, loading and dumping of ore and waste as well as drill and blast costs, grade control costs and low grade rehandling costs.

The open-pit unit mining cost is US$1.47/t of material mined at 1,100 metres relative level (mRL) and increasing with depth and height. This cost will increase to a maximum of US$3.79/t of material mined at the base of the final pit at 560 mRL. The unit mining cost is based on a delivered diesel fuel price of US$0.70 per litre.

The underground mining cost is US$58.04/t mined. The CIL processing cost is US$14.53/t processed.

The Operating Cost estimate excludes contingencies, escalation, accuracy provisions, and corporate overhead charges, financing costs, royalties, expenditures classified as capital, gold shipping and insurance. Sustaining capital, rehabilitation and closure costs, and other Owner's Costs including overhead costs, process costs and underground costs are included in the Financial Analysis.

1.17 Financial Evaluation

As Centamin is a producing issuer, it has excluded information required by Item 22 of Form 43-101F1 as any forecast increase in plant throughput in this technical report is not considered to be a material expansion of current production.

1.18 Interpretations and Conclusions

Sukari is viewed as a long-term strategic asset for Centamin. The open-pit mine is forecast to operate for approximately 19 years using the current underground Mineral Reserve as complementary mill feed. If the underground mine is extended beyond the life of the current underground Mineral Reserve, that contribution will allow the operation to extend beyond 19 years.

The recent Stage 4 expansion of the processing plant has been successfully implemented and commissioned, with nameplate capacity of 10 Mtpa already exceeded and the planned throughput of 11 Mtpa expected to be achieved in the coming year.

The underground operations are progressing and extracting high-grade gold from the structurally-controlled zones of mineralization at depth. Interaction of the open-pit and underground is expected to occur when mining nears the base of Stage 3A of the open-pit, with the upper benches of Stages 3A and 3B currently being mined.

Development of the open-pit continues as planned, with further productivity improvements of the mining equipment expected to achieve total material movement of at least 66 Mtpa from 2015 onwards. Current supply of explosives is sufficient to meet the current production rates, and additional supplies of explosives can be secured to meet increased production rates, as required.

The strip ratio in the first five years of operation (2009 to 2014) was less than the LOM average. Strip ratio is forecast to increase to the LOM average (5.9:1), and in some future periods is likely to peak higher than the LOM average when waste pre-stripping is undertaken for the final three pit stages.

The Mineral Reserve was estimated at a gold price of US$1,300/oz and used a 10% discount rate. The financial analysis demonstrates that the project is economic.

1.19 Recommendations

1.19.1 Reconciliation

The QP, Patrick Smith, recommends a site-wide reconciliation process be developed and implemented to analyse all aspects of the production chain. With the major ramp-up in mining and processing rates at Sukari in recent years this process is needed to allow robust analysis of resource, reserve, and grade control models.

The reconciliation study would be undertaken as part of operational activities and budget.

1.19.2 Underground Mining Recovery

The QP, Declan Franzmann, recommends that SGM investigates stope backfilling opportunities in high-grade mineralization areas. Backfilling reduces the use of pillars for local stope stability, and extraction of the mineralization in these pillars can improve stope recovery.

The backfill study would be undertaken as part of operational activities and budget.

Cautionary Notes

Forward-Looking Information

This document contains “forward-looking information” which may include, but is not limited to, statements with respect to the future financial or operating performance of Centamin plc (‘Centamin’ or ‘the Company’), its subsidiaries (together ‘the Group’), affiliated companies, its projects, the future price of gold, the estimation of mineral reserves and mineral resources, the realization of mineral reserve and resource estimates, the timing and amount of estimated future production, revenues, margins, costs of production, estimates of initial capital, sustaining capital, operating and exploration expenditures, costs and timing of the development of new deposits, costs and timing of future exploration, requirements for additional capital, foreign exchange risks, governmental regulation of mining operations and exploration operations, timing and receipt of approvals, consents and permits under applicable mineral legislation, environmental risks, title disputes or claims, limitations of insurance coverage and regulatory matters. Often, but not always, forward-looking statements can be identified by the use of words such as “plans”, “expects”, “is expected”, “budget”, “scheduled”, “estimates”, “forecasts”, “intends”, “targets”, “aims”, “anticipates” or “believes” or variations (including negative variations) of such words and phrases, or may be identified by statements to the effect that certain actions, events or results “may”, “could”, “would”, “should”, “might” or “will” be taken, occur or be achieved.

Forward-looking statements involve known and unknown risks, uncertainties and a variety of material factors, many of which are beyond the Company’s control which may cause the actual results, performance or achievements of Centamin, its subsidiaries and affiliated companies to be materially different from any future results, performance or achievements expressed or implied by the forward-looking statements. Readers are cautioned that forward-looking statements may not be appropriate for other purposes than outlined in this document. Such factors include, among others, future price of gold; general business, economic, competitive, political and social uncertainties; the actual results of current exploration and development activities; conclusions of economic evaluations and studies; fluctuations in the value of the US dollar relative to the local currencies in the jurisdictions of the Company’s key projects; changes in project parameters as plans continue to be refined; possible variations of ore grade or projected recovery rates; accidents, labour disputes or slow-downs and other risks of the mining industry; climatic conditions; political instability, insurrection or war, civil unrest or armed assault; labour force availability and turnover; delays in obtaining financing or governmental approvals or in the completion of exploration and development activities; as well as those factors referred to in the section entitled “Risks and Uncertainties” section of the Management Discussion & Analysis filed on SEDAR at www.sedar.com and on the National Storage Mechanism. The reader is also cautioned that the foregoing list of factors is not exhausted of the factors that may affect the Company’s forward-looking statements.

Although the Company has attempted to identify important factors that could cause actual actions, events or results to differ materially from those described in forward-looking statements, there may be other factors that cause actions, events or results to differ from those anticipated, estimated or intended. Forward-looking statements contained herein are made as of the date of this document and, except as required by applicable law, the Company disclaims any obligation to update any forward-looking statements, whether as a result of new information, future events or results or otherwise. There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, readers should not place undue reliance on forward-looking statements.

2 INTRODUCTION

2.1 Request for Technical Report, Purpose, and Terms of Reference

The purpose of this Technical Report is to provide an update to the 30 January 2014 Technical Report in light of further development of the underground mine, updated estimates of mineral resources and mineral reserves, and completed commissioning of the Stage 4 process plant expansion for the Sukari gold mine located in Egypt.

This Technical Report with updated estimates of Mineral Resources and Mineral Reserves at Sukari gold mine was prepared in accordance with Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), NI 43-101 Form F1, and Canadian Institute of Mining, Metallurgy and Petroleum (CIM) “Best Practices and Reporting Guidelines.”

2.2 Units and Currency

In this report, all currency is in US Dollars (US$) unless otherwise stated.

Quantities are generally stated in Systeme International d’Unites (SI) metric units, the standard for Canadian and international practice, including metric tonnes (t) for weight, and kilometres (km) and metres (m) for distance.

2.3 Effective Date

The resource dataset has been addressed in Sections 10 and 11 and has an effective date of 30 June 2015, with immaterial timing differences as noted below. Accordingly, the effective date of this report is 30 June 2015.

The open-pit resource model has been reported to the topographic surface of 30 June 2015, and uses drilling data with an effective date of 28 February 2015. Drilling completed between 28 February and 30 June 2015 is immaterial to the open-pit Mineral Resource estimate.

The underground resource model has been reported to the underground workings status of 30 June 2015 and uses drilling data with an effective date of 3 July 2015. Drilling completed between 30 June and 3 July 2015 is immaterial to the underground Mineral Resource estimate.

The open-pit Mineral Reserve estimate has been reported to the topographic surface of 30 June 2015. The underground Mineral Reserve estimate has been reported to the underground workings status of 30 June 2015.

2.4 Qualified Persons

The Qualified Persons (QP) responsible for the interpretation or supervision or approval of the information contained in this report are summarized in Table 2.4-1.

Table 2.4-1 Qualified Persons and Other Contributors

Qualified Persons Responsible for the Preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of Centamin?**	**Date of Last Site Visit**	**Professional Designation**	**Sections of Report**
Mr P Smith	Principal Mining Engineer	AMC Consultants Pty Ltd	Yes	4 to 6 December 2013	MAusIMM CP(Min)	1.1, 1.6.1, 1.6.2, 1.7, 1.18, 1.19.1, 2, 3, 15.1, 15.2, 16.1, 24, 25, 26.1, 27, 28
Mr D Franzmann	Consulting Mining Engineer	Crosscut Consultants Pty Ltd	Yes	7 to 10 April 2015	FAusIMM CP(Min)	1.6.3, 1.8, 1.19.2, 15.3, 16.2, 26.2
Mr N Johnson	Consulting Geologist	MPR Geological Consultants Pty Ltd	Yes	2 to 8 July 2013	MAIG	1.5.1, 14.1
Mr M Zammit	Principal Consultant	Cube Consulting Pty Ltd	Yes	9 to 16 March 2015	MAIG	1.5.2, 11.5, 12, 14.2
Other Experts who assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of Centamin?**	**Date of last Site Visit**	**Sections of Report**	
Mr R Osman	Business Development Manager	Centamin plc	No	20 to 23 October 2015	1.2, 1.3, 1.4, 1.11, 1.13, 1.14, 1.15, 1.16, 1.17, 4, 5, 6, 7, 8, 9, 10, 11.1, 11.2, 11.3, 11.4, 18, 19, 20, 21, 22, 23	
Mr R Longley	Metallurgical Manager	Sukari Gold Mines	No	Works on site	1.9, 1.10, 1.12, 13, 17	
Mr C Boreham	Underground Manager	Sukari Gold Mines	No	Works on site	1.6.3, 1.8, 15.1, 15.3, 16.2	
Mr J Poniewierski	Principal Mining Engineer	AMC Consultants Pty Ltd	Yes	20 July to 8 August 2015	1.6.2, 1.7, 15.2, 16.1	

Patrick Smith is the QP responsible for the preparation of Sections 1.1, 1.6.1–1.6.2, 1.7, 1.18–1.19.1, 2–3, 15.1–15.2, 16.1, 24–28 and, as lead author of this Technical Report, responsible for the review of Sections noted in Table 2.4-1 for R Osman and R Longley.

The respective certificates for the Qualified Persons can be found in Section 28.

2.5 Sources of Information and Data

This report is based on information provided by Centamin, SGM and/or PGM, and externally-appointed consultants, which reflect various technical and economic conditions prevailing at the time of compilation of the report. These conditions can change significantly over relatively short periods of time and as such the information and opinions contained in this report may be subject to change.

2.6 Previous Technical Reports

Centamin has previously filed technical reports on Sukari gold mine as follows:

- Mineral Resource and Reserve Estimate for the Sukari Gold Project, Egypt, 30 January 2014
- Mineral Resource and Reserve Estimate for the Sukari Gold Project, Egypt, 14 March 2012
- Mineral Resource and Reserve Estimate for the Sukari Gold Project, Egypt, 6 December 2010
- Mineral Resource and Reserve Estimate for the Sukari Gold Project, Egypt, 21 May 2009
- Technical Report - Sukari Gold Project, Egypt, 23 March 2007

2.7 Property Inspections

A site visit was conducted by Mr Declan Franzmann of Crosscut Consultants Pty Ltd from 7 to 10 April 2015 during which Mr Franzmann inspected the underground workings and collected data that was required to estimate the underground mineral reserve.

A site visit was last conducted by Mr Nicolas Johnson of MPR Geological Consultants Pty Ltd from 2 to 8 July 2013.

A site visit was last conducted by Mr Patrick Smith of AMC Consultants Pty Ltd from 4 to 6 December 2013 during which Mr Smith had the opportunity to inspect the open-pit and the underground workings, and to enable data collection and discussions relevant to the estimation of the open-pit mineral reserve.

A site visit was conducted by Mr Mark Zammit of Cube Consulting Pty Ltd from 9 to 16 March 2015 to inspect the Sukari Gold Project in order to review the controls on mineralization, the geological interpretation and to review the data collection.

3 RELIANCE ON OTHER EXPERTS

The sources of information found in this report, including data and supporting reports, were supplied by Centamin and/or SGM and PGM personnel, as well as documents referenced in Section 27.

The lead author of this Technical Report, Patrick Smith, BE (Mining) MAusIMM CP(Min) RPEQ of AMC Consultants Pty Ltd is not qualified to provide comments on issues relating to mining and exploration titles and land tenure, royalties, and permitting and legal matters. The assessment of data pertaining to these sections (4.3 and 4.4) relies on information provided directly by representative experts employed by Centamin and/or SGM and PGM, which otherwise has not been independently verified by the author.

Similarly, the lead author of this Technical Report is not qualified to provide extensive comment on environmental issues associated with the Sukari Gold Project included in Section 20 of this report. The assessment of data pertaining to these disciplines relies on information provided by Environics, an environmental management consulting firm based in Giza, Egypt, which has not been independently verified by the authors.

Centamin, SGM and/or PGM provided MPR Geological Consultants Pty Ltd and Cube Consulting Pty Ltd with electronic files that included drill logs, survey and collar data, lithology, and assay results for the complete dataset that was used in the estimate of mineral resources.

Centamin, SGM and/or PGM provided AMC Consultants Pty Ltd and Crosscut Consultants Pty with the electronic files that included mine production and cost data, topographical and underground mine void data, and the resource models that were used in the estimate of mineral reserves.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Background Information on Egypt

4.1.1 Geography

The Arab Republic of Egypt is a Northern African country of approximately 1,000,000 km^2. It borders the Mediterranean Sea (to the north), Libya (to the west) and the Gaza Strip and Israel (to the east), and the Red Sea north of Sudan, and includes the Asian Sinai Peninsula. It lies approximately at geographic coordinates 27ºN and 30ºE and operates on time zone GMT+2h00. Egypt has a population of approximately 85 million people generally concentrated on the fertile banks of the Nile River. Around half of Egypt's residents live in urban areas, with the majority spread across the densely populated centres of greater Cairo (which is the largest city in Africa and the Middle East), Alexandria and other major towns in the Nile Delta.

Arabic is the official language with English and French widely understood by educated classes. The main ethnic group is Egyptian 98%, followed by Berber, Nubian, Bedouin, and Beja 1%, Greek, Armenian and other European (primarily Italian and French) 1%. The rapidly growing population is young, with approximately one-third of the total under 15 years of age. Despite improvements in health care, infant mortality is high and about half of all deaths occur among children less than five years of age. Life expectancy, however, increased from about 33 years in 1927 to almost 72 years by 2008.

Egypt is predominantly Muslim, at approximately 90% of the population, with the majority being adherents of the Sunni branch of Islam. A significant number of Muslim Egyptians also follow native Sufi orders, and a minority of Shi'a. Christians represent about 10% of the population.

Egypt is bisected by the highly fertile Nile valley where most economic activity takes place. The Nile divides the desert plateau through which it flows into two unequal sections, the Western Desert between the river and the Libyan frontier; and the Eastern Desert extending to the Suez Canal, the Gulf of Suez, and the Red Sea. The Western Desert is arid and without wadis (dry beds of seasonal rivers), while the Eastern Desert is extensively dissected by wadis and fringed by rugged mountains in the east. The desert of central Sinai is open country, broken by isolated hills and scored by wadis. The coastal regions of Egypt, with the exception of the Delta, are everywhere hemmed in either by desert or by mountain; they are arid or of very limited fertility. The coastal plain, in both the north and east, tends to be narrow; it seldom exceeds a width of 50 km. With the exception of the cities of Alexandria, Port Said, and Suez and a few small ports and resorts, the coastal regions are sparsely populated and underdeveloped.

4.1.2 Political System

Egypt has been subject to political disturbance in the last few years. However, a new constitution was adopted in January 2014; President Abdul Fattah al-Sisi was elected in May 2014; and parliamentary elections are expected to take place in October 2015.

4.1.3 Legal System

The Legal System is based on English common law, Islamic law, and Napoleonic codes. Judicial review is by the Supreme Court and the Administrative Court which oversees validity of administrative decisions.

4.1.4 Economy

Egypt has one of Africa’s most prosperous economies despite the fact that it is extremely vulnerable to external factors such as the continuing conflict in the surrounding region and the subsequent drop in tourism, which is one of the country’s main economic sectors in addition to the oil and gas sectors.

The country is classified as middle-income and continues to record economic growth although there is increased pressure by international donors to implement reforms. These include the lifting of price controls, the reduction of subsidies as well as the relaxation of restrictions on trade and investment.

4.1.5 Transport and Communications

Egypt is served by a range of international airlines, which fly to Cairo and Alexandria, as well as the more tourism focused cities of Luxor on the Nile and Hurghada on the Red Sea. Internal air services service other points of the country. The nearest airport to the Sukari Project is located approximately 60 km north of Marsa Alam, with Marsa Alam being approximately 30 km from the mine site. There are bus and rail services along the Nile River valley. The road system in the country is good with only small amounts of the system remaining unpaved.

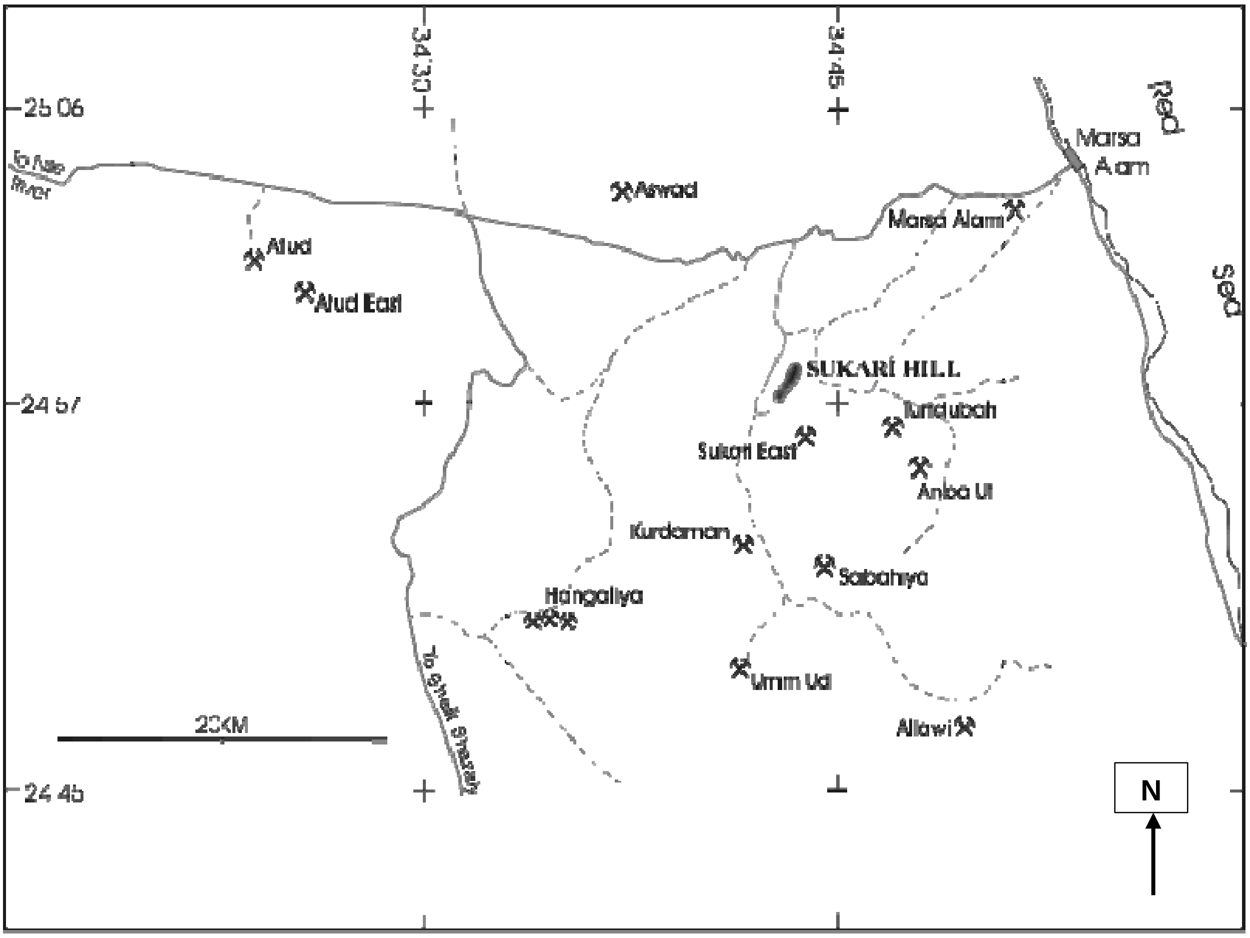
In spite of its long coastline, Egypt has only three ports of any significance – Alexandria, Port Said, and Suez. Alexandria handles most of the country's imports and exports, as well as the bulk of passenger traffic. Port Said, at the northern entrance to the Suez Canal, lacks the berthing and loading facilities of Alexandria. Suez's main function is that of an entry port for petroleum and minerals from the Egyptian Red Sea coast and for goods from the Far East.

There is a large, well-established telephone network covering the whole country, which underwent extensive upgrading during the 1990s. Cellular phone services are available in the major centres. Internet access is available in most centres.

4.2 Project Location

The Sukari Gold Project is located in bare hills in the arid Eastern Desert of Egypt some 30 km westerly from the growing township of Marsa Alam on the Red Sea (Figure 4.2-1).

Figure 4.2-1 Sukari Location and Road Access



4.3 The Project Tenement and Area

According to information supplied by Centamin, PGM entered into a Concession Agreement in 1994 with the Egyptian Geological Survey and Mining Authority (EGSMA, now known as the Egyptian Minerals and Resource Authority or EMRA) and the Egyptian Government to explore, develop, mine and sell gold and associated minerals in three areas in the Eastern Desert. The Concession Agreement was declared into Egyptian Law 222 of 1994, which came into effect on 29 January 1995.

In accordance with the Concession Agreement, a “Commercial Discovery” was declared in November 2001 following the submission of a feasibility study by PGM; EMRA and PGM then agreed on the borders of the area to be converted from exploration to exploitation lease comprising an area of 160km^2. The Minister of Petroleum approved a 30-year lease in respect of 160 km^2 on 24 May 2005, extendable for a further 30 years upon PGM providing appropriate commercial justification.

The Sukari concession is shown in Figure 4.3-1. The concession covers an area of 160 km^2, containing the proposed Sukari mine site and surrounding prospects. The points defining its boundary are listed in a clockwise fashion from the northwest corner in Table 4.3-1. Key obligations on the Sukari concession are profit share and royalties—failure to comply with which might lead to the licence being terminated.

Figure 4.3-1 Concession Area

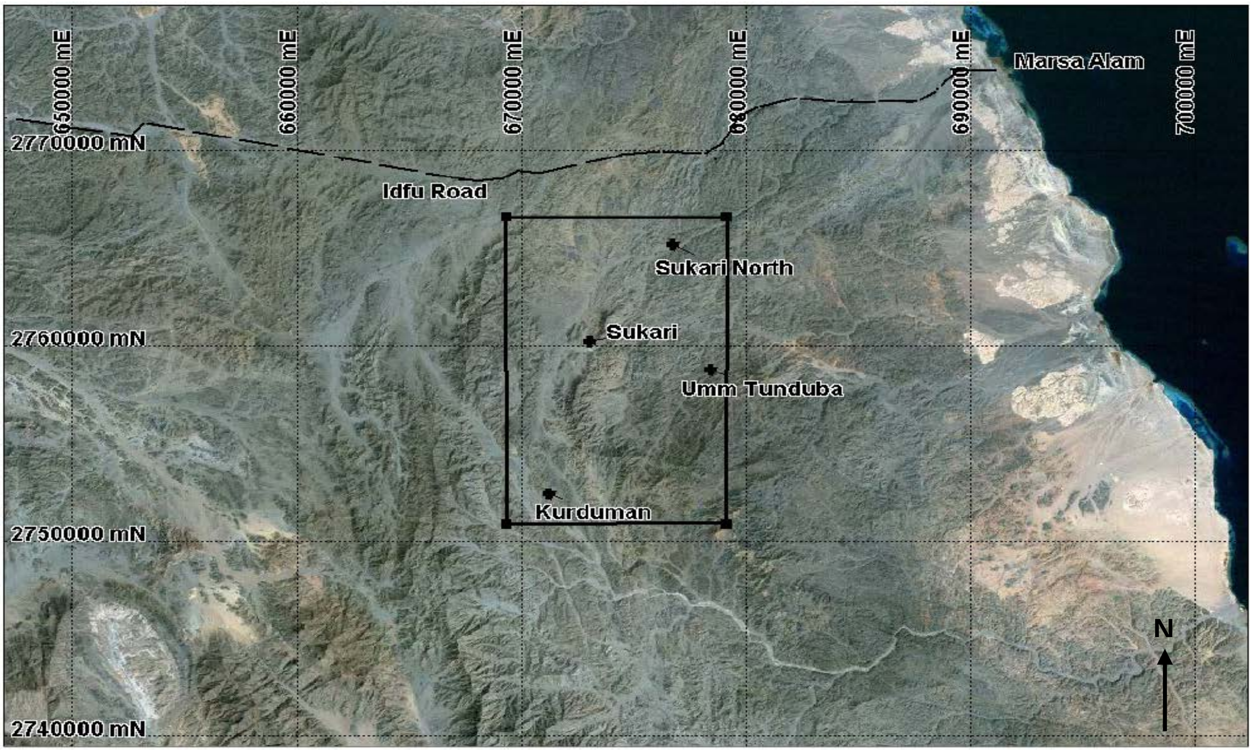


Table 4.3-1 Sukari Project, List of Tenement Corners

Pillar	Longitude (East)	Latitude (North)
A	34º 40' 40"	25º 00' 20"
B	34º 46' 30"	25º 00' 20"
C	34º 46' 30"	24º 51' 45"
D	34º 40' 40"	24º 51' 45"

Information pertaining to the project tenement and area has been provided by Centamin and has not been independently verified by the authors.

4.4 Royalties, Payments, Agreements, and Encumbrances

According to information supplied by Centamin, PGM and EMRA have established an operating company, SGM, 50%-owned by each party (see Section 1.1 above). The articles of association of this company have not been translated from Arabic and have therefore not been independently verified by the authors.

The Government of Egypt receives a royalty of 3% of net sale revenue, payable in cash each calendar half year.

PGM shall be entitled to recover the following costs and expenses payable from sales revenue (excluding the government royalty):

- All current operating expenses incurred and paid after the initial commercial production;
- Exploration costs, including those accumulated prior to the commencement of commercial production at the rate of 33.3% per annum;
- Exploitation capital costs, including those accumulated prior to the commencement of commercial production at the rate of 33.3% per annum.

If costs recoverable by PGM exceed the sales revenue (excluding the royalty payable to the government) in any financial year, the excess shall be carried forward for recovery in the next financial year or years until fully recovered but in no case after the termination of the agreement.

After deduction of royalty payments and recoverable expenses, the remainder of the sale revenue will be shared equally between PGM and EMRA. Except that for the first and second years in which there are net proceeds for the entire year, an additional 10% of those proceeds shall be paid to PGM as incentive (PGM 60%, EMRA 40%). For each of the next two years in which there are net proceeds for the entire year, 5% of the net proceeds shall be paid to PGM (PGM 55%, EMRA 45%), thereafter the net proceeds shall be shared equally (PGM 50%, EMRA 50%).

Starting from the date of commercial production PGM will enjoy a 15-year exemption from taxes imposed by the Egyptian government. PGM and EMRA also agree that the operating company will in due course file an application to extend the tax free period for a further 15 years.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

A coastal highway runs along the west coast of the Red Sea from the border with Sudan in the south to Suez in the north, passing through Marsa Alam, Qseir, Safaga, Hurghada, Ras Gharib and Ein Sokna. Another highway connects Cairo directly to Ein Sokna. The distance from Cairo to Marsa Alam by highway is about 750 km, about ten hours by supply truck. There is also a bitumen highway from Idfu on the Nile to Marsa Alam. There is a new international airport north of Marsa Alam.

From Marsa Alam the Idfu bitumen highway first runs west up gravel outwash plains to the mouth of Wadi Khariga, then it follows the winding narrow wadi to Bir Umm Khariga Well, some 20 km from Marsa Alam. From Umm Khariga Well, a corrugated gravel road runs southerly for 10 km where it crosses a low divide and goes down into Wadi Sukari, and then on to the old Sukari gold mine. The Sukari road then turns east from the Wadi Sukari road and continues on for about 1.5 km north-west to the Sukari Gold Mine operations complex.

5.2 Climate

Egypt has a dry climate. It is hot in the summer, with temperatures averaging between 80 and 90°F (27-32°C). Winters are warm, with temperatures averaging between 55 and 70°F (13-21°C). Mining and milling activities are thus able to be conducted year round. A steady wind from the northwest helps to lower the temperature near the coast. The Khamaseen is a wind that blows from the south in Egypt, usually in spring or summer, bringing sand and dust, and sometimes raises the temperature in the desert to more than 100°F (38°C).

Rain seldom falls in Egypt. Along the Mediterranean Coast, the average yearly rainfall is 8 inches (20 cm). Farther south, only about one inch of rain falls every year. During winter snow falls on Sinai's mountains and some coastal cities, such as Baltim, Damiatta, Sidi Barrany and Alexandria.

The climate in which SGM operations are located is considered to be harsh. The average temperature during the winter months (October to March) ranges from 17°C to 27°C and during the summer months (April to September) from 26°C to 36°C with maximum temperatures exceeding 40°C. Humidity is normally very low but has been known to exceed 80% at the seawater intake near the coast, especially during the winter months. Precipitation is almost non-existent with rainfall rarely exceeding 10 mm per year and in some years, there is no rainfall recorded at all.

5.3 Physiography

The Sukari Gold Mine site is in the Eastern Desert of Egypt and it is estimated that nearly all of the gold mineralization occurs within a porphyry outcrop which is expressed as a 2,500 m long jagged-toothed, strong topographic high rising to 350 m above the local wadi (intermittent water course) level. Wadi drainage plains pass to the east and west of the outcrop and the sharply incised green-brown Red Sea Hills surround that. The area is arid and almost bare of vegetation.

5.4 Local Resources and Infrastructure

The Sukari Gold Mine operation is located in stark desert with little or no vegetation. There is no permanent population in the immediate area and it has been visited only by people tending nomadic livestock herds in recent times.

The mine site has access via bitumen road 5 km from the mine site operations boundary. The nearest local town is Marsa Alam (25 km distant) on the Red Sea, which is a fast growing tourism, focused suburban town area with population estimated at approximately 5,000 persons. This town offers a local hospital and also police presence in the area. There are two banks with ATM cash outlets and several local restaurants. There are numerous resort complexes located along the coast line and within close proximity of the town that also offer available resources including ATMs, restaurants and shops. The town has many small stores and is able to support a moderate-sized construction workforce, as demonstrated by the recent completion of the Sukari Stage 4 process plant upgrade capital works between 2012 and 2013.

Sukari mine site provides all the power and water services to support its operations.

6 HISTORY

6.1 Ownership History

Gold was mined at Sukari in Pharaonic and Roman times, with perhaps 1,000 kg of gold produced. Small-scale mining was re-established in 1912 to 1914. More substantial operations were undertaken in the period 1937 to 1951, with recorded production of 4,768 kg of gold from underground workings.

PGM negotiated a Concession Agreement (CA) in 1994 to explore for gold and associated minerals. In November 2000, PGM submitted a feasibility study, relating to the Sukari Gold Project, in accordance with the terms of the CA. This was accepted, and on 24 May 2005 an Exploitation Lease covering an area of 160 km^2, containing the proposed Sukari mine site and surrounding prospects, was officially granted.

6.2 Exploration History

The first systematic modern exploration in the Sukari area was carried out in the 1970s by the Egyptian Government with assistance from the former Union of Soviet Socialist Republics (USSR). Work completed consisted of geological mapping, trenching, geochemical sampling and finally, five diamond drillholes were completed at Sukari during the period 1975-1977. Assaying of the drill core confirmed the presence of gold mineralization at depth.

6.3 Resource History

There is no historical estimate of the Sukari deposit, prepared before PGM negotiated the exploration and mining agreement in 1994, to be reported.

6.4 Production History

Gold has been mined at Sukari since Pharaonic and Roman times. Numerous small pits are located over about two kilometres strike on Sukari Ridge. There are also small pits in wadi colluvium along the flanks of the ridge, most notably in Wadi Pharaoh to the east of the northern part of the ridge. It is believed that about 1,000 kg of gold may have been produced.

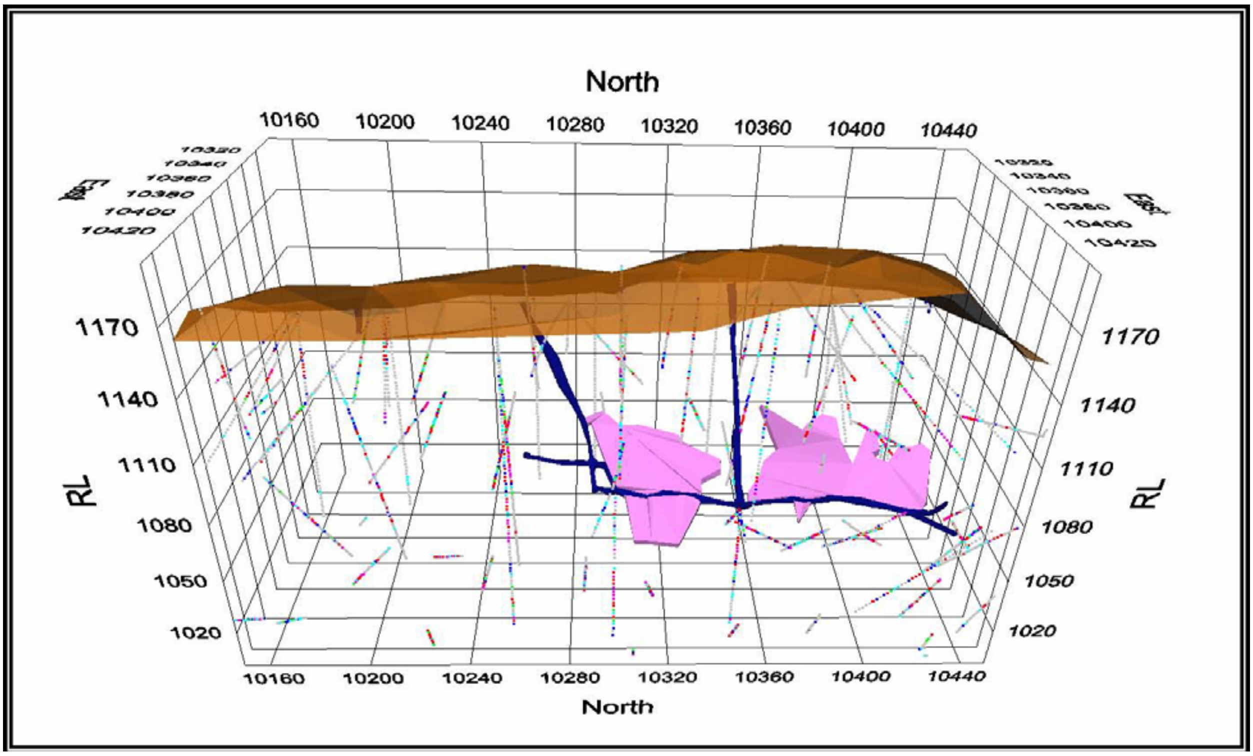
The old Sukari Mine was established on an outcropping quartz vein (the “Sukari Main Lode”) on the south-western flank of Sukari Ridge (Figure 6.4-1). In Pharaonic times, mining of this vein extended to about 50 m from surface, intermittently, along about 200 m strike, with stopes about one metre wide. Small-scale mining was re-established in 1912 by British concerns but appears to have ceased at the outbreak of World War I.

Figure 6.4-1 View of Historic Mine Workings

In 1936, a renewed effort by government authorities to re-establish Egypt's gold mining industry saw Sukari selected as the first mine to be brought back into production. In 1934, "developed ore reserves" were estimated to be about 12,000 t at 20 g/t Au (Hume, 1937), with a further 65,000 t estimated "ore in sight". After preparatory work, production commenced in August 1937 and continued intermittently until February 1951. Recorded gold production for this period is 4,768 kg (Azzaz, et.al., 1978). Ore was sourced from the Main Lode, with the ancient underlay shaft being refurbished and extended to about 185 m depth (on the underlay). An extraction level was established at 110 m depth and stoping above this level extended over about 100 m strike length. Several subsidiary adits and underlay shafts accessed stopes along the length of the mined strike. Ore below the 110 m level has also been stoped over about 50 m strike length. Stopes are generally 2 m to 3 m wide. Figure 6.4-2 shows a 3D view, looking west, of the surveyed underground workings. The total tonnage reported within the surveyed volumes is approximately 26,000 tonnes.

A battery for ore treatment was located at Sukari from 1936 to 1944, at which time it was moved to Marsa Alam and used to treat ore from other prospects in the district.

Figure 6.4-2 Historic Underground Workings



Modern production since resumption of mining by SGM is given in Table 6.4-1.

Table 6.4-1 Sukari Production History

		6 months ended Jun 2015	12 months ended Dec 2014	12 months ended Dec 2013	12 months ended Dec 2012	12 months ended Dec 2011	6 months ended Dec 2010	12 months ended Jun 2010
Ore Mined - Open Pit	('000t)	4,313	10,936	11,664	6,377	6,306	3,805	4,183
Total Mined	('000t)	29,667	44,820	41,718	25,108	21,248	10,891	17,003
Strip Ratio	waste / ore	5.88	3.10	2.58	2.94	2.34	1.85	3.10
Ore Mined - U/ground Development	('000t)	256	464	304	203	172	40	0
Ore Mined - U/ground Stopes	('000t)	290	504	284	190	40	NR	NR
Ore Processed	('000t)	5,145	8,427	5,684	4,526	3,612	1,378	1,906
Head Grade	(g/t)	1.40	1.53	2.12	2.04	1.90	2.06	1.37
Gold Recovery	(%)	89.3%	87.8%	88.5%	86.1%	85.3%	85.4%	87.0%
Gold Produced - Total	(oz)	216,014	377,261	356,943	262,828	202,699	83,432	67,101
Gold Produced - Dump Leach	(oz)	9,529	15,564	12,382	6,686	10,664	5,436	NR
Cash Operating Cost of Production	US$/oz	712	729	663	669	556	549	478
Gold Sold	(oz)	215,417	375,300	363,576	254,959	214,763	66,378	63,753
Average Sales Price	US$/oz	1,202	1,257	1,384	1,667	1,555	1,308	1,152

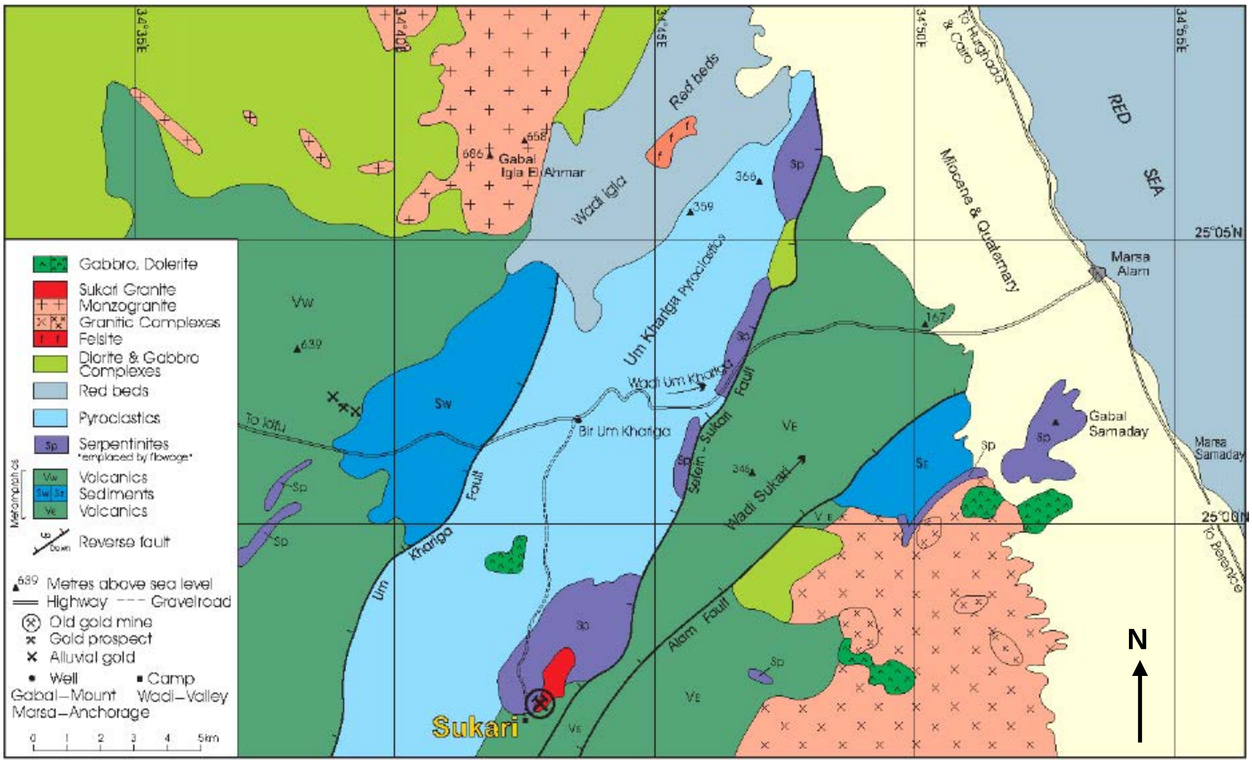
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional/Local Geological Setting

The Sukari gold deposit is located in the Eastern Desert of Egypt at 24º 56' 50"N 34º 42' 27"E, about 23 km due west of the Red Sea coastal town of Marsa Alam. The regional geological setting is shown in Figure 7.1-1.

The rock sequence at Sukari comprises part of the Neoproterozoic (900-650 Ma) Arabian-Nubian Shield, one of a number of areas of African continental crust that accreted and stabilised during the Pan-African Orogeny. At a district scale, the host sequence at Sukari comprises a NNE striking mélange of predominantly calc-alkaline igneous rocks and metasediments representing an accreted island arc or arcs. Several bodies of serpentinite, representing accreted slivers of highly deformed oceanic crustal rocks, occur in the hangingwall of the NNE striking, ESE verging, Sefein-Sukari thrust (Akaad, et al, 1993). This district-scale (~25 km) structure is mapped as passing immediately to the east of Sukari, where it separates rocks of the Um Khariga Metapyroclastics (west of Sukari granitoid and enveloping serpentinite) from the Sukari Metavolcanics (east of Sukari). Vail (1983) assigns an age of 770-660 Ma to rocks of the region. The entire sequence has undergone regional metamorphism to mid-upper greenschist facies.

Figure 7.1-1 Sukari District Geology



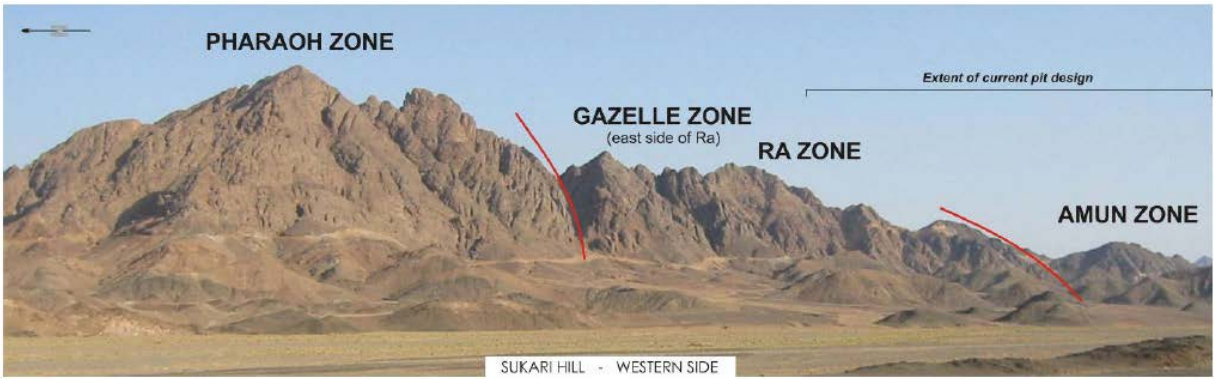
Note: Geology map from Cavaney 2005

7.2 Geology of Property

The Sukari felsic porphyry outcrop is located in an easterly dipping sequence of andesite flows, serpentinites and associated volcanoclastic sediments, mainly tuffs and epiclastics. It strikes for 2.3 km and is 100 m to 600 m thick. Drilling to date indicates that the Sukari Porphyry dips toward the east at between 50º and 75º. The western and eastern contacts of the porphyry are thus regarded as footwall and hangingwall contacts respectively. Porphyry/wall rock contacts are, in places, vertical or overturned.

The Sukari area has been designated four geographical zones namely Amun, Ra, Gazelle and Pharaoh Zones from south to north respectively (Figure 7.2-1).

Figure 7.2-1 Sukari Hill with designated Geographical Zones (looking to the SE)

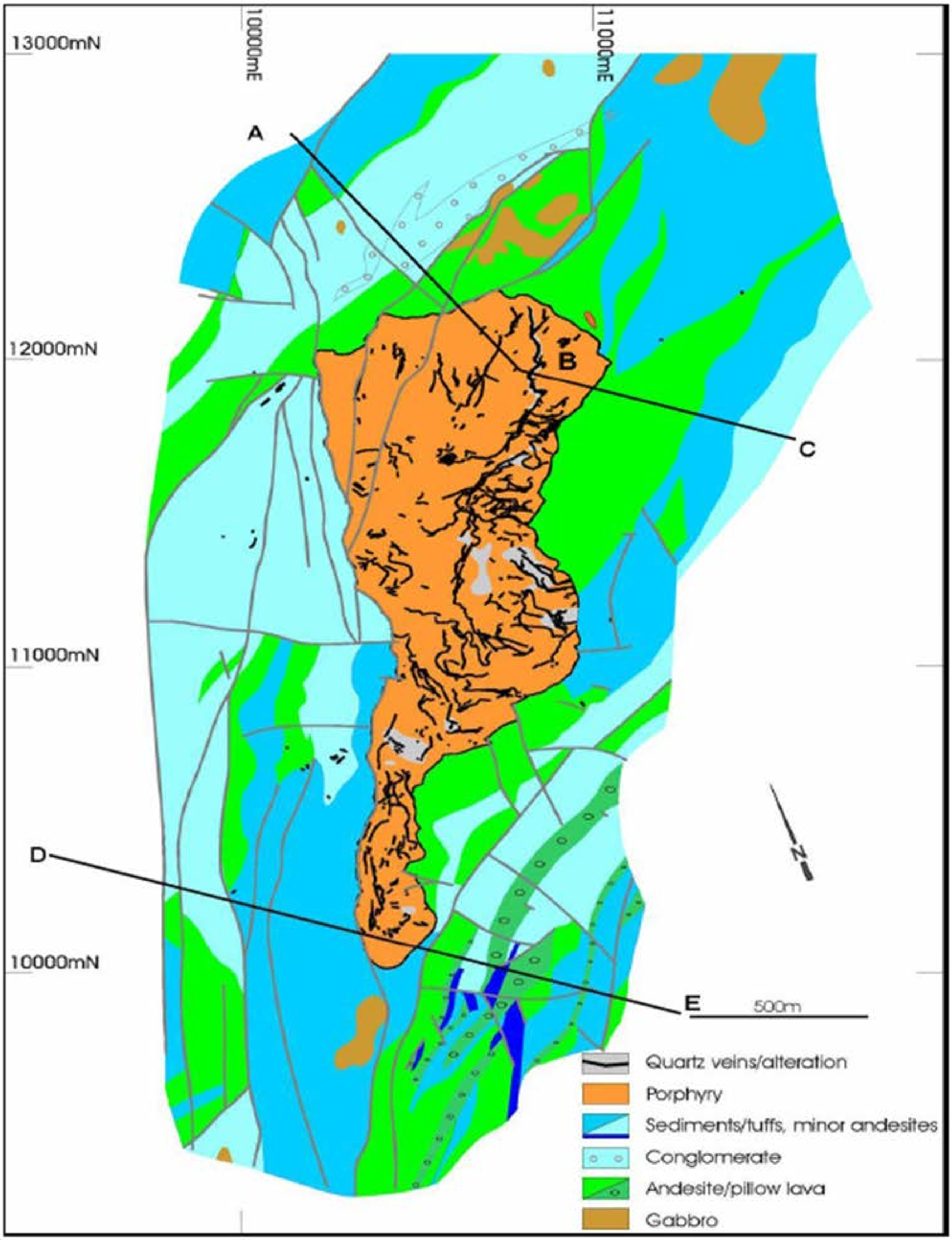


The hangingwall sequence comprises a mixture of serpentinite, meta-conglomerate, lesser fine-grained metasediments, minor basalt and porphyry dykes or sills (Figure 7.2-2). Drillhole logging has failed to define any consistent sequence in these rocks. Although this may be due in part to inconsistencies in the logging, surface exposures indicate that these rocks are complexly deformed. It is reasonable to assume that the porphyry dykes in the hangingwall sequence are genetically and temporally related to the main Sukari Porphyry. The footwall sequence is devoid of porphyry dykes. This may indicate that the entire sequence is overturned as it would be reasonable to expect subsidiary or feeder dykes in the footwall of the main intrusion rather than the hangingwall.

On the eastern slope of Sukari Ridge individual stratigraphic units, normally comprising more competent units in serpentinite, can be traced over tens of metres. Surface mapping is, however, complicated by a lack of persistent stratigraphic markers and the fact that, in many places, schistosity is at a high angle to bedding.

Few drillholes penetrate more than about 50 m into the footwall sequence, where they have intersected fine-grained pelitic metasediments with lesser basalts. Outcrop is poor at the foot of the western slope of Sukari Ridge. Low ridges and mounds in the wadi to the west of Amun Zone are formed by outcropping serpentinite and it is likely that this lithology dominates the footwall sequence. Similar to hangingwall rocks, footwall metasediments and serpentinites show development of very strong schistosity and a lack of stratigraphic markers, combined with poor outcrop, prevents mapping of a stratigraphic sequence.

Figure 7.2-2 Interpreted Geology, Sukari Porphyry and Surrounding Country-rocks



7.3 Mineralization

Gold mineralization at Sukari is hosted exclusively by porphyry. The lack of significant gold grades in chemically reactive serpentinitic wall rocks can be explained in one of two ways:

- The porphyry represented a favourable host either because of its composition, relative to mineralising fluids, or its mechanical properties or both.
- The Sukari Porphyry was relocated, relative to wall rock sequences, by faulting after gold mineralization.

While significant post-mineralization faulting has possibly occurred, the second scenario is unlikely. Porphyry dykes in the hangingwall of the main porphyry body show gold mineralization of essentially the same character as that in the main porphyry and wall rocks immediately adjacent to those dykes are also barren. Those dykes range in thickness from a few centimetres to several metres. It is not reasonable to postulate that they, along with the main porphyry body, were all relocated by faulting after mineralization.

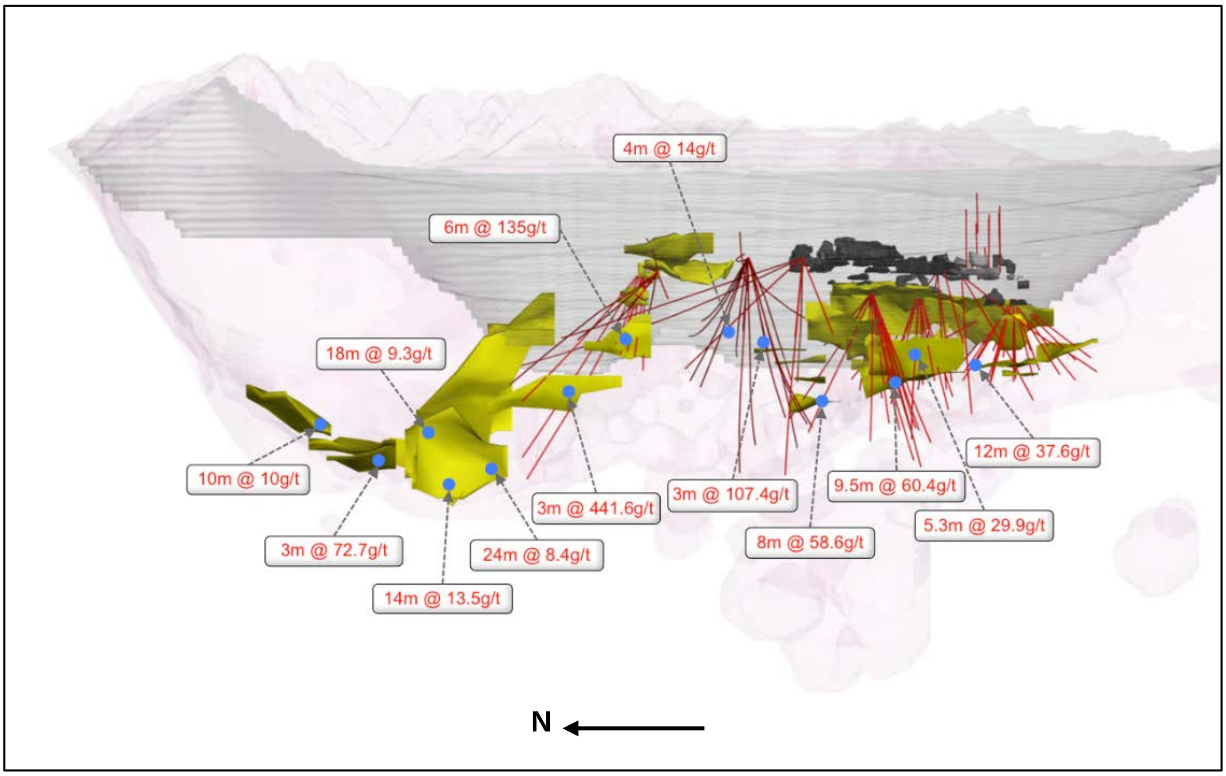
Certainly it is evident that the Sukari Porphyry has acted as a rigid body surrounded by weaker rocks. Footwall and hangingwall rocks have taken up strain by development of strong schistosity, almost certainly accompanied by large decreases in volume. The porphyry has taken up strain by development of predominantly brittle fault structures.

7.4 Mineralization Geometry

The porphyry host for the mineralization has a strike length of approximately 2,300 m, and ranges in thickness from 100 m to approximately 600 m (Cavaney 2005). Gold mineralization within this is not continuous and its deposition has been influenced by major long-lived structures, the most important of which are tabular sheets of crackle breccia. Figure 7.4-1 illustrates the overall shape and size of the porphyry host and the geometry of the high grade Main Reef and Hapi Reef.

Drilling to date indicates that the Sukari Porphyry dips toward the east at between 50º and 75º. The western and eastern contacts of the porphyry are thus regarded as footwall and hangingwall contacts, respectively. Porphyry/wall rock contacts are, in places, vertical or overturned.

Figure 7.4-1 Geometry of Main Zone and Hapi Zone within the Host Porphyry

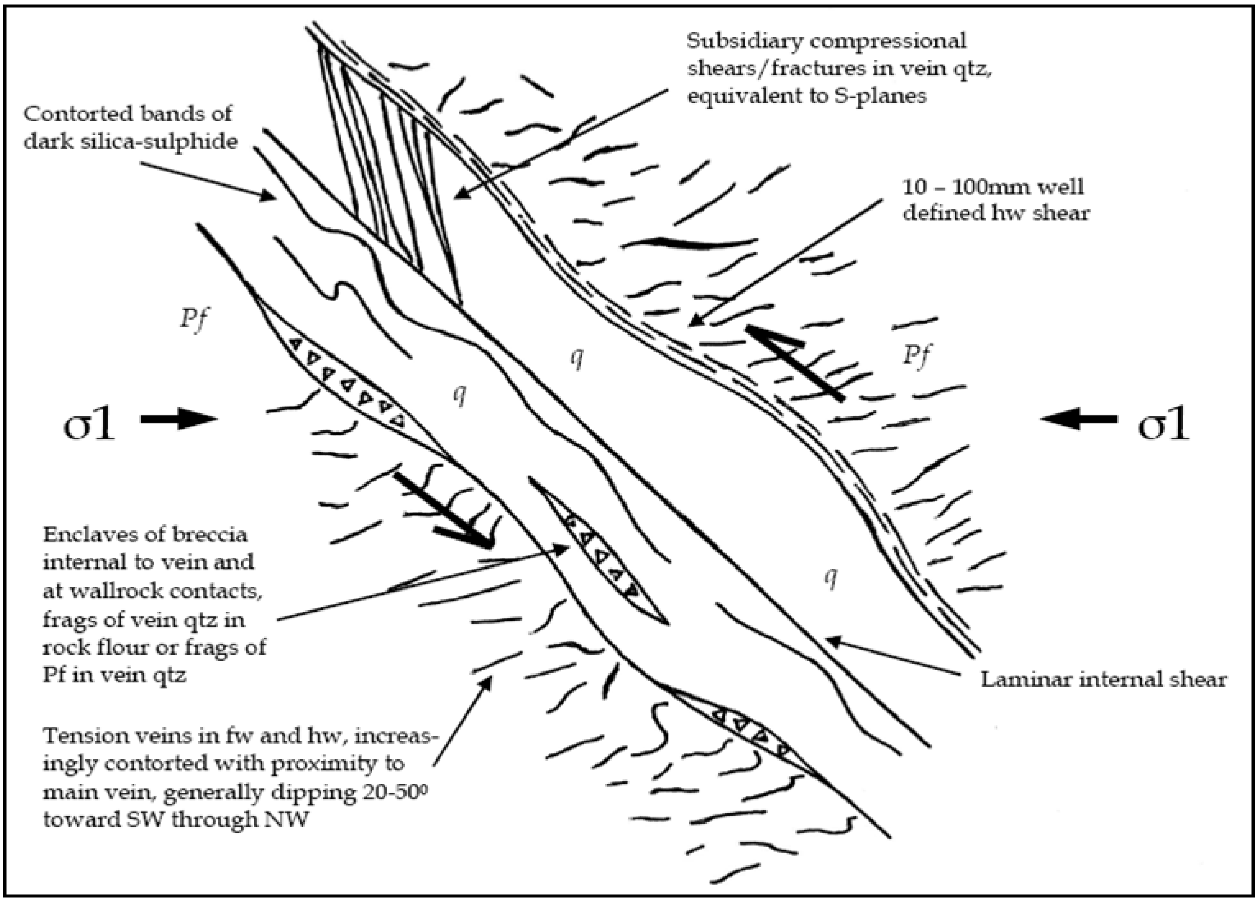


7.5 Deformation Kinematics during Mineralization

There are several features that indicate that mineralization at Sukari took place under a regime of near-horizontal, east-west directed stress.

Figure 7.5-1 shows a sketch diagram of a portion of the Sukari Main Reef. The reef is a complex, multiphase, antitaxial vein. Early vein quartz bands have, in places, been contorted into recumbent folds indicating reverse movement. The vein, and in places the wall rock immediately adjacent to it, shows development of sub-vertical compressional S foliation planes. The orientation of those planes is also consistent with reverse movement.

Figure 7.5-1 Sukari Main Reef – Sketch Diagram



Large-scale slickensides are present in sheared porphyry immediately adjacent to the hangingwall of the Main Reef. These slickensides, and associated pull-apart terminations, are common in underground exposures and their orientation is consistently steep, indicating predominantly reverse movement. A variety of plunges were observed, some indicating a component of dextral shear and some a component of sinistral shear. All, however, indicate a ratio of dip-slip to strike-slip components of 8:1 or greater.

The majority of small-scale extensional veins are sub-horizontal, generally dipping at shallow angles to the SW or NW. Where contorted, such as in the vicinity of the Main Reef, their tips are often sub-horizontal. Being purely extensional veins, with essentially no displacement across them, they must have formed parallel to the direction of maximum principle stress (σ1) and perpendicular to the direction of minimum principle stress (σ3). The variation in dips, from SW to NW, is consistent with the horizontal component of shear displacement varying between dextral and sinistral at various times during mineralization.

7.6 Geometry of Vein Arrays and Shears

A key to understanding brittle vein arrays, and consequently being able to predict mineralization geometry, is to differentiate between through-going structures and discontinuous, extensional veins.

Four types of mineralized quartz veins were observed at Sukari:

- Contorted and banded veins.
- Brecciated veins.
- Shear veins.
- Spaced extensional veins.

The contorted veins are the result of multiple deformation and veining episodes and are similar in character to veins found in low-sulphidation epithermal Au-Ag deposits. Sukari Main Reef represents this type of vein. They represent through-going, long-lived structures on which significant displacement (metres to tens of metres) has occurred. Arsenopyrite mineralization tends to be concentrated in these veins and they represent the highest-grade mineralization at Sukari.

Brecciated veins consist of brecciated vein quartz and porphyry rock fragments or porphyry fragments in a matrix of vein quartz +/- sulphides +/- haematite. They are a variant of the contorted veins and, in some exposures, breccia zones can be seen developed at the contacts of contorted veins and host porphyry. The breccia veins also represent through-going structures.

Shear veins appear to be rare. In the few observed occurrences they are centimetre-scale laminar veins with vague contact-parallel layering possibly formed by aligned inclusion trails. They sometimes have bands of sulphides and or haematite developed at vein-wallrock contacts.

Extensional veins are distinguished by their short strike lengths and, in places, by internal fabrics that indicate purely extensional opening with no displacement across the vein. Extensional veins normally form stacked arrays between thin linking shears. The linking shears are commonly sericite altered and may contain fine sulphides +/- haematite. They have acted as fluid conduits but their orientation relative to the applied stress field has prevented any infill by vein quartz—they were under compression during the deformation that accompanied mineralization. The orientation of the shears, not the extensional veins, indicates the large-scale direction of continuity of a stacked vein array.

7.7 Sulphide Development

Gold mineralization at Sukari is intimately related dominantly to sulphides; pyrite is the most abundant sulphide, followed by arsenopyrite. High gold grades are associated with increased arsenopyrite concentration. The sulphides occur as fine grained, subhedral disseminations in altered porphyry and as blebby sub- to euhedral crystals and finer disseminations in quartz veins, fractures and breccias. Pyrite is found in all the mineralized zones. Deposition of pyrite was continuous throughout the various mineralising stages. Rust coloured goethite is commonly seen to rim pyrite crystals in a semi-weathered environment or fracture surfaces that have been exposed to surficial weathering conditions.

Arsenopyrite is most common in the zones of higher-grade gold mineralization, so it is common in the Main Reef and in the Hapi Zone, and breccias. Arsenopyrite is not abundant in the stacked extensional zones and minor quartz veins. It occurs primarily as fine, sub- to euhedral crystals (sometimes characteristically as diamond shaped needles in cross section) in fine, anastomosing stylolitic F veins, and rimming breccia clasts.

Petrographic study has shown that inclusions of pyrite in arsenopyrite and vice versa are common indicating that they belong to the same paragenesis (Khalil, 2006). Both minerals contain numerous inclusions of rutile and wall rock materials suggesting their formation as a result of sulphidization of pre-existing rocks during the hydrothermal stage. Pyrite and arsenopyrite exhibit deformation and even brecciation textures. Abundant younger native gold is filling stringers and tiny holes in these deformed pyrite and arsenopyrite.

Other sulphides such as galena, chalcopyrite, sphalerite, pyrrhotite have been noted. Some pyrite crystals contain relicts of pyrrhotite and chalcopyrite suggesting the formation of pyrite by pyritization of pre-existing pyrrhotite and probably chalcopyrite. Galena occurs as coarse subhedral crystals in milky extensional quartz veins, generally in areas of strong gold mineralization, but is rare. Sphalerite is sometimes a significant sulphide mineral. Abundant exsolved chalcopyrite bodies are randomly distributed in the sphalerite host. The sphalerite-chalcopyrite association seems to be filling and replacing the older pre-existing pyrite. The observed galena in pyrite seems to be belonging to the same age of the sphalerite-chalcopyrite association

7.8 Gold

Visible gold in core occurs as anhedral grains in milky white extensional and breccia quartz veins and as intergrowths with pyrite and arsenopyrite, commonly in narrow shear veins at quartz vein margins and margins to clasts in hydraulic quartz vein breccias.

Silver is routinely assayed in drill samples; there is no correlation between it and gold and only very low levels returned (generally less than 1 ppm Ag). Out of 70,500 Ag assays in the database, 32 are >10 ppm, one with a peak value of 165 ppm.

SEM and mineragraphic work (Mintek, 2000), determined that high-purity gold occurs free in quartz, on the margins of pyrite and arsenopyrite crystals, and as microfracture fillings. Gold as electrum is paragenetically first as it is often occluded in pyrite and followed secondly by high purity gold (>900 fine) depleted in silver. Gold is fine grained and ranges from 1 μm to 40 μm.

Based on the microscopic investigation, the following paragenesis is postulated (Khalil, 2006):

1. Magnetite and titanium-bearing mineral (probably titanomagnetite or ilmenite) were formed during the magmatic stage. Pyrrhotite and chalcopyrite relicts seem to belong to this magmatic stage.
2. Under contact metamorphism and high strain conditions, probably at the site of shear zones, platy hematite and rutile were formed.
3. Pyrite and arsenopyrite are of hydrothermal origin and seem to be formed as a result of sulphidization of pre-existing rocks.
4. Sphalerite, chalcopyrite, galena and native gold are fracture filling minerals and are syn – to post the host deformed pyrite and arsenopyrite assemblage.
5. Goethite and anatase were formed under weathering conditions.

7.9 Deposit-Scale Architecture

Drillhole logging completed by SGM geologists has provided the geological interpretation illustrated in Figure 7.9-1. Veins and shears that comprise through-going structures have been differentiated from zones of stacked extensional veins. Shear veins can generally be correlated between drillholes although it must be recognised that their character may change over tens of metres, e.g. a structure filled with brecciated porphyry cemented by vein quartz in one hole may appear as a contorted vein in an adjacent hole. Lithologic logging was simplified by recognition of the variable carbonate-silica alteration in serpentinites.

Although several published papers describe the Sukari Main Reef as occurring at the footwall contact of the Sukari Porphyry, that interpretation has been disproved by drilling. The reef and other similar through-going shears (e.g. Hapi Reef) dip at 35º to 50º toward grid east whereas the overall dip of the porphyry body is about 65º. The nature of the mineralized shears almost certainly changes when they pass from the porphyry into surrounding metasediments and serpentinite – their extensions cannot be traced into footwall and hangingwall rocks. It appears likely that they become re-oriented to steeper dipping, compressional structures. Where they cut the porphyry they assume a flatter orientation, possibly comprising Reidel shears in the deposit-scale deformation scheme. It is likely that they cause relatively small (metres to tens of metres) offsets in both footwall and hangingwall porphyry contacts.

A large volume of mineralization is associated with stacked brittle veins. These clearly occur in zones proximal to the through-going mineralized shears but they also occur remote from them and in such areas the overall orientation of vein packages is open to conjecture. Under the deformation regime, extensional veins should occur in stacks dipping at between 35º and 60º to both the east and west. The west-dipping stacks form in reverse faults that are antithetic to the through-going mineralized shears such as the Main Reef and Hapi Reef. Strike directions of both east- and west-dipping stacks are likely to be diverse because of local strain partitioning and their dip and strike extents are also likely to vary from a few metres to tens of metres.

There is good evidence for antithetic shears, at least some of which host mineralized vein arrays. Figure 7.9-2 shows a fault plane dipping at 35º/260º, part of a foliated shear zone that is about 12 m wide. The shear zone is replete with millimetre to centimetre thick contorted, extensional quartz veins dipping at shallow angles to the west and these veins appear to be coeval with reverse movement on the shear. Topographic relief on the western side of Sukari Ridge (Figure 7.9-3) indicates that a number of large-scale faults occur in this orientation but whether all are coeval with mineralization is not known.

Figure 7.9-1   Oblique Schematic Section 10,300 mN to 10,550 mN

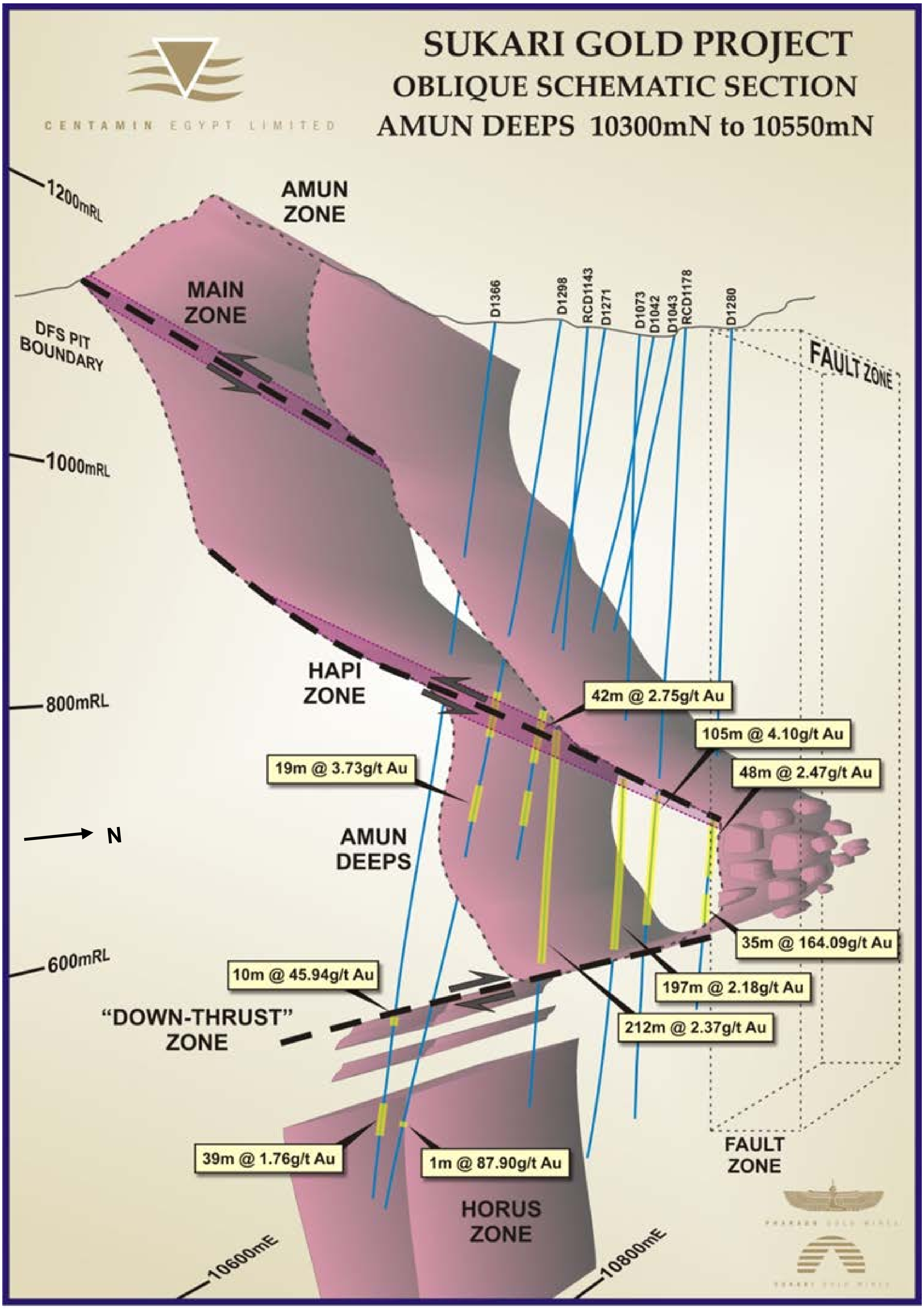


Figure 7.9-2 Antithetic Reverse Shearing

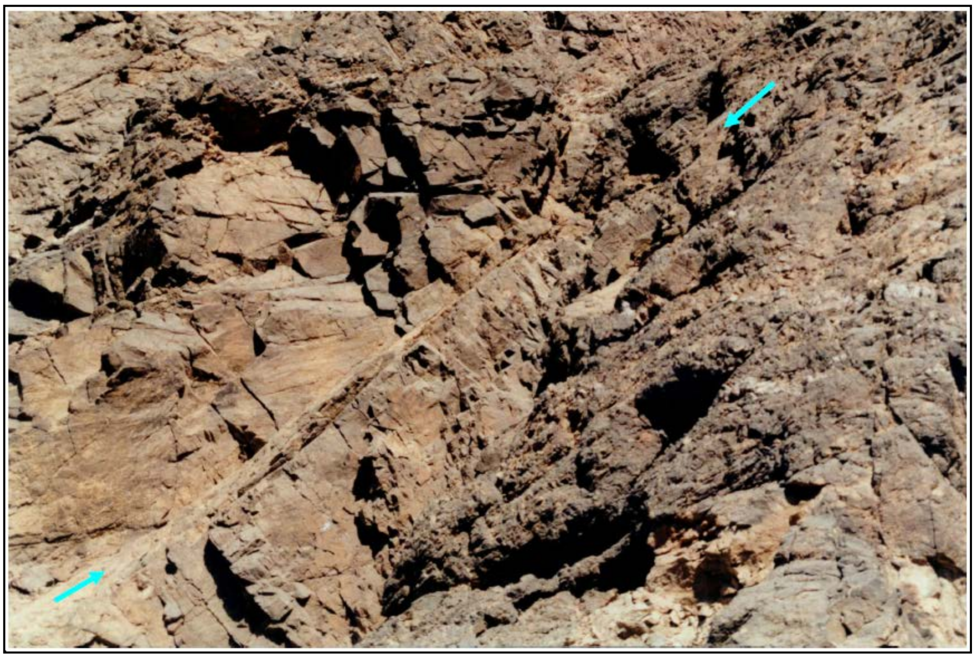
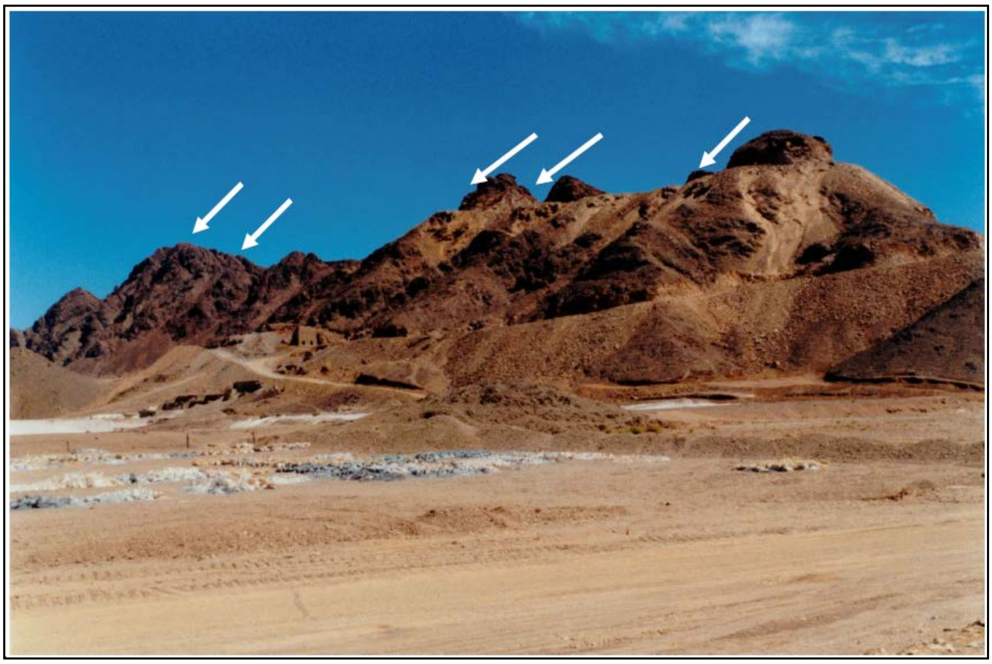


Figure 7.9-3 View of Sukari Ridge showing probable Antithetic Shears



7.10 Post-Mineralization Structures

The only significant post-mineralization structures observed at Sukari to date belong to a set of ENE-WSW trending sub-vertical fractures associated with andesite dykes. The thickness of these dykes ranges from centimetres to one or two metres and the fractures they fill appear to have negligible offset across them. The dykes are not always planar and, being sub-parallel to drill sections, confident interpretation of them between drillholes is not always possible.

The dykes and associated fracture set are clearly visible on Landsat images, forming an ENE trending swarm over about 10 km width, extending to the Red Sea coast. They are possibly related to an incipient transform structure associated with the opening of the Red Sea rift.

7.11 Alteration

Serpentinites in the footwall and hangingwall sequences have been variably carbonatised and silicified to listwaenite, typically a mixture of ferroan or magnesian carbonate and cryptocrystalline silica. Buisson & LeBlanc (1985) describe this type of alteration, sometimes associated with gold mineralization, as being commonplace in serpentinitic rocks of the Arabian-Nubian Shield in Egypt, the Sinai Peninsula and Arabian Peninsula.

Figure 7.11-1 illustrates the progression of this alteration. The barely-visible black-green core on the left of the photograph is least altered serpentinite. This rock often shows fine blood-red haematite veining. From left to right the cores show increasing carbonate and silica replacement until the rock is altered entirely to listwaenite. At its most extreme, and if magnesian carbonate dominates, the altered rock may not be reactive to cold acid. It appears that most, if not all, of the rocks logged in Sukari drill core as rhyolite are in fact altered serpentinite. This misidentification has introduced unwarranted complications in the geology of the wallrock sequences.

Figure 7.11-1 Progressive Alteration of Serpentinite

The timing of carbonate alteration relative to gold mineralization is unknown. Carbonatized serpentinite does, in places, contain fine-grained pyrite and quartz veins but no detectable gold. Although it would make sense to postulate that carbonate alteration of serpentinite wallrocks accompanied the gold mineralization event it is difficult to conceive of a process that would so efficiently partition gold deposition into only the porphyry host.

8 DEPOSIT TYPES

Gold mineralization is hosted by a granitoid body of approximately granodiorite-tonalite composition referred to as the Sukari Porphyry. Granitoids of the Egyptian portion of the Arabian-Nubian Shield are described as “older granites”, these being “syntectonic” intrusions related to arc magmatism, and “younger granites” that clearly post-date arc accretion and have monzonitic, syenitic or potassic compositions (Moghazi et al, 1999). Sharara (1999) ascribes the Sukari Porphyry to the latter, based on an age of 615-570 Ma by Stern and Hedge (1985). Ghoneim et al. (1999) ascribe an age of 559 Ma ± 6 Ma to the Sukari Porphyry based on Rb-Sr and Sm-Nd isotopic dating of “fresh granite” and mineralization age of 522 Ma ± 11 Ma based on the same methods applied to albitised granite and separated albite. In contrast, Sharara and Vennemann (1999) state that gold mineralization at several localities in the Eastern Desert region, including Sukari, pre-dates intrusion of “younger granites”.

Considering the composition of Sukari Porphyry, its deformed nature and its contact relationships with surrounding rocks, it is considered to be a sub-volcanic, “syn-tectonic” granitoid, representing the roots of a rhyolitic arc volcano.

Based on fluid inclusions and stable isotope analyses, Sharara (1999) and Sharara and Vennemann (1999) estimate a mineralization temperature of 270ºC to 370ºC for Sukari and other gold deposits in the district, with gold deposited from a low-salinity carbonic fluid similar to that typically invoked for many Archaean gold deposits.

9 EXPLORATION

9.1 Introduction

Work by PGM commenced in 1995 with the establishment of a camp and work completed consisted of a detailed literature search prior to gridding, traversing, mapping, geochemical sampling, trenching, channel sampling, heavy mineral sampling, augering and surveying.

Exploration at Sukari is at a very advanced stage. It has culminated in the discovery of a gold deposit, and in turn to an operating gold operation, with an open-pit and underground mine and centralised processing facility. However, significant potential remains to increase the known Mineral Resources and Mineral Reserves, and exploration will continue during the life of the mining operation.

9.2 Exploration Methods Used

Gridding and traversing was carried out at 1:10,000 scale.

Mapping was carried out on a 1:500 scale (Amun Zone) and 1:1,000 scale.

Trenching and channel sampling within the cut trenches was undertaken, mainly within zones of intense silicification and sulphidization. Total length of trenching was 1,143 m.

Channel sampling from historical underground workings was completed. A total of 982 samples were submitted for analysis.

Augering was undertaken on two heaps of tailings on a 10 m by 10 m grid to a maximum depth of 1 m. A total of 327 samples were taken for gold analysis.

Rock chips were taken initially on 160 m spaced lines with some supplementary infill lines. In addition, dykes, quartz veins and zones of hydrothermal alteration were grab-sampled. Later rock chip sampling was undertaken on 100 m spaced lines and samples were approximately 1 m to 2 m in length.

Regional sampling and prospecting consisted of rock chip and channel sampling at various small mines in the vicinity.

Heavy mineral sampling was undertaken at various suitable sites in wadis.

Both reverse circulation (RC) and diamond drilling have been employed for exploration and resource definition purposes.

Further information on sampling is referred to in Section 11.

9.3 Use of Contractors

All exploration work has been carried out by, or supervised by PGM and/or SGM geological personnel. The only significant use of contractors has been for drilling and assaying.

The contract drilling is currently being undertaken by Barminco Egypt Underground Mining Services SAE (Barminco Egypt) and Capital Drilling Egypt.

Preparation and assaying of samples produced during the various exploration programs has been undertaken at either Minesite Reference Laboratories (earlier phases of drilling prior to 2005) or Ultra Trace Analytical Laboratories (Ultra Trace), which now operates as Bureau Vertias Minerals, (used during the 2005 and subsequent drilling), both situated in Perth, Western Australia.

9.4 Interpretation of Exploration Information

9.4.1 Introduction

At this stage of the project, interpretation of the exploration information has been largely superseded by the estimation of mineral resources and mineral reserves. However, some general comments may be made on interpretation of the exploration results.

9.4.2 Geological Mapping

Geological mapping in the Sukari area has been successful, despite difficult topography (steep cliffs). The mineralization and alteration can usually be seen at the surface and mapping has been useful in defining the extent and morphology of the porphyry that hosts the mineralization

9.4.3 Rock Chip Sampling

Rock chip samples have been useful in confirming the position of the lode horizon. Rock chip sampling conducted by SGM has been surveyed and rock chip lines are available as point data, which has been included in the resource estimation sampling database. Most surface rock chip samples were collected over 2 m intervals and are deemed compatible with the 2m drillhole composites without any further manipulation.

9.4.4 Trenches

Excavation of trenches as an exploration technique has been very successful. Significant gold intersections in trenches typically overlie sub-surface zones of similar grade and width, as defined by subsequent drilling. Trench sample assays are available as point data located by nominal northings, and with eastings and elevations estimated by interpolation between trench end locations measured by tape and compass.

Trench sampling positions have not been surveyed, and this leads to some doubt about the true locations of trench samples they have been excluded from the sample database used for resource estimation.

9.5 Data Reliability

Data acquired during exploration at Sukari is considered to be very reliable. All work has been carried out by technically-qualified personnel, and has been planned and supervised by highly-trained and experienced geoscientists. The location of all exploration data is known with adequate accuracy. The quality of geochemical analyses has been monitored by the use of blanks, standards, field duplicates, and check analyses via primary and umpire laboratories.

10 DRILLING

10.1 Introduction

The first known drillholes at Sukari are five holes drilled by the EGSMA-Soviet research team in 1976-78 (Azzaz, et.al., 1978). The locations of these holes are known and logging and assay records are available, however, a lack of confidence in the quality of the sampling and assaying from these holes preclude their use in the resource dataset. The geological logs are included in the drilling database and have been used to aid in the interpretations of the Sukari geology.

Drilling by PGM commenced in April 1997 and is ongoing at the time of this report**.** PGM's drilling at Sukari has been by diamond core drilling, using two Atlas Copco Craelius 252 rigs drilling with standard (i.e. not wire-line) TT46 gear to produce 35.3 mm diameter core. These rigs are skid-mounted electric-hydraulic drills normally used for drilling in underground mines. Several holes were drilled from the No. 1 (110 m) level of the old Sukari mine workings. In August 2000, the drilling effort was augmented by a track-mounted Atlas Copco 262 diesel-hydraulic rig. This rig uses standard (i.e. not wire-line) TT46 gear to produce 35.3 mm diameter core.

During the first half of 2002, the drilling capacity at Sukari was significantly enhanced with the introduction of a drilling contractor utilising larger, more flexible, drilling rigs; two multi-purpose diamond and RC capable drilling rigs (one KL600 and one LMP850). In the last quarter of 2002, two CS1000 rigs, in second quarter of 2007 two CS14 (wire-line) configured for HQ and NQ core drilling only were added to the drilling fleet at Sukari, and in 2009 and additional HQ and NQ Atlas Copco CS3001 core drilling rig was added to the fleet. By the end of 2011 Underground Resource drilling commenced with introduction of one diamond rig (Boart Longyear LM90 wire-line with rod handler) configured for HQ and NQ to use in the underground resource definition. In the middle of 2012, an additional underground LM rig commenced. From the second half of 2013, resource drilling was focused from underground with four underground LM rigs. Starting from the middle February 2015, the resource drilling from underground is being conducted with two underground rigs.

Drilling has been conducted on 25 m spaced sections oriented grid east-west for the open-pit resource and 15 m spaced sections for the underground resource. Locations of drillholes used for mineral resource estimation are shown in Figure 10.1-1.

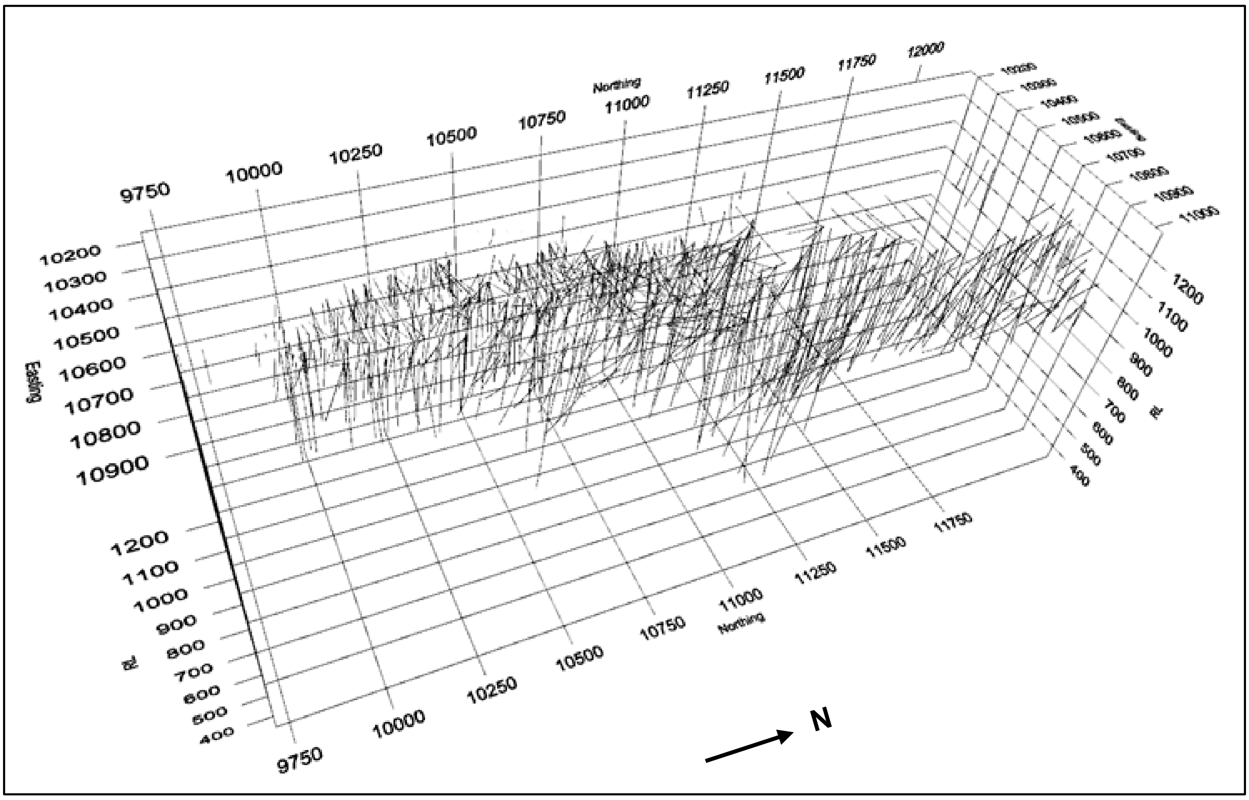
At the time of writing the Sukari resource drillhole database comprises approximately 579,319m of drilling in 1,892 holes, including some holes abandoned because of drilling difficulties and some intervals that are awaiting assays. Five historical diamond holes (D001 to D005) are included in the drill database statistics, however, neither their geological nor sampling information has been used as the recorded information for these holes cannot be verified. Underground resource drilling is undertaken using both HQ and NQ drill strings. Some summary details of the available drillhole dataset are provided in Table 10.1-1.

Table 10.1-1 Summary Details of Sukari Resource Drillhole Dataset

Total	Number
Total number of drillholes	1,892
Total metres (all holes)	579,319
RC total metres	74,961
DD total metres	504,358

Commencing October 2005, the drill-related data has been stored in Century Systems Inc geological database, the complete assay database was imported during August 2006. Century Systems is used to collect geological data in the field, validating data on entry, store the data in a secure open database, and validate quality control from the lab. Data is migrated to mine software by ODBC link and CSV exports.

Figure 10.1-1 Drillhole Locations



Note: View looking down towards grid west

10.2 Survey Controls

A 50 mN by 20 mE control grid was established by a contract surveyor using a point located about 200 m south-west of the EGSMA garages as grid origin (10,000 mN, 10,000 mE, 1,100 mRL). A baseline striking 020º magnetic and cross-lines were established using a theodolite. Stadia measurements were used for distances, with slope corrections as required.

The highest point of Sukari Ridge is marked by a spot height on the Jabal Nuqrus 1:50,000 scale topographic map sheet, with elevation 630 m. The map sheet also shows a survey station at about the middle of Sukari Ridge with elevation 591 m. This point is ground marked by a cairn of stones. Using these two points as references, the elevation of the grid origin is approximately 380 m above sea level.

GPS observations place the grid origin at 24º 56’ 48”N 34º 42’ 36”E. This location reconciles well with the location of a survey control point shown on the 1972 1:1,000 scale geological map of Sukari mine area compiled by EGSMA. The location of that point, marked by a small cairn of stones near the southern end of Sukari Ridge, above the old mine workings, is listed as being 24º 56’ 48.78”N 34º 42’ 48.71”E. Location of that point was presumably by sun observations or by carrying survey from distant controls.

In February 1998, licensed surveyor Mr G. Robinson placed and surveyed control stations around Sukari Ridge and determined the bearing of the Sukari exploration grid baseline. The true bearing of the grid baseline determined from sun observations at the grid origin was calculated as 22º 29’ ± 10’’. This is very close to 020º magnetic as at Year 2000.

In 1999, qualified surveyor Mr F. Howard established several additional local control points, using Robinson’s controls.

Current Survey control is provided by a modern RTK GNSS system. The system hardware comprises a Base-station receiver, Base-station de-coder, VHF radios and associated signal repeaters and also in-pit rover receivers.

GNSS, or Global Navigation Satellite System, uses a combination of both American (GPS) satellites and Russian (GLONASS) satellites.

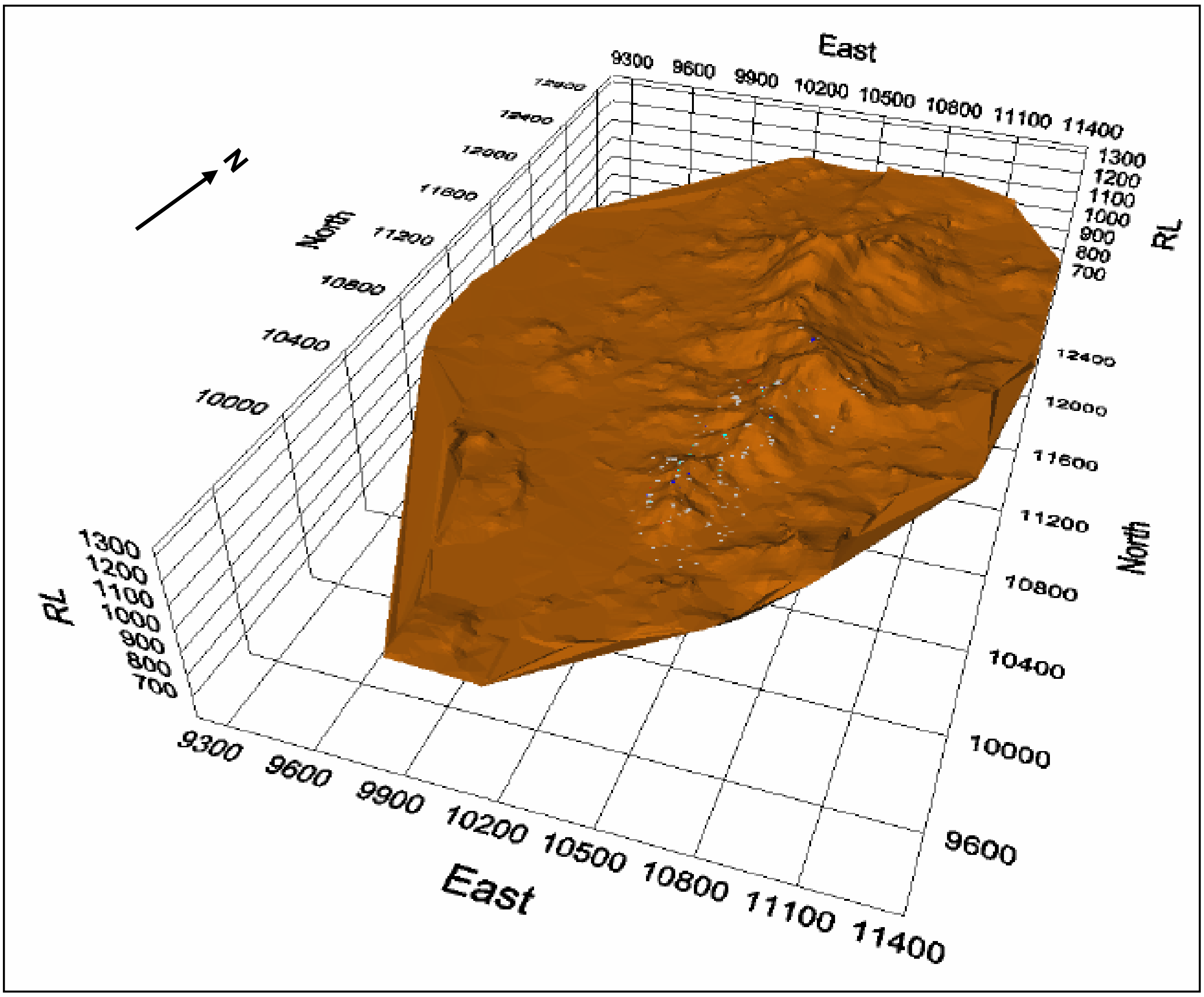
The Base Station pillar coordinates were determined using the previously mentioned control points with checks made to existing features with good correlation. The Base Station is ideally located in an elevated and stable area, well outside the mining area.

The industry-standard Trimble system provides sub-centimeter coordinate accuracy and is the primary source for all survey data within the Sukari Operation.

10.3 Topographic Survey Data

Spot heights and features have been surveyed by SGM surveyors using RTK GNSS survey equipment. Inaccessible areas are surveyed by reflectorless total station with temporary station coordinates provided by RTK GNSS. The survey data is reduced and processed using Surpac software to provide digital terrain model (DTM) and contour files in various formats as required. Surveyed spot heights form the basis for the digital terrain model used in resource estimation and mine design (Figure 10.3-1).

Figure 10.3-1 Sukari Topography DTM



10.4 Downhole Surveys

Most PGM drillholes drilled prior to 2003 have been downhole surveyed at depth intervals of 50 m to 100 m using a single shot camera device. The first survey is normally at about 4 m depth and the dip and azimuth at that point is allocated to the collar point (i.e. zero depth). Magnetic azimuths recorded by the survey instrument are reduced to grid bearing by subtraction of 22º, this correction being derived by taking shots on surface along the grid baseline.

Holes less than 50 m long are generally not surveyed. In addition, several holes have not been surveyed due to unstable ground conditions and twelve holes (Holes 92 to 103) were not surveyed because of survey equipment breakdown. For unsurveyed holes, dip and azimuth of the rig frame is measured on surface and those readings allotted to the top and bottom hole depths in the drillhole database. There has been no attempt to estimate deviations in unsurveyed holes.

During a site visit in 2000, an H&S representative cross-checked 50 survey discs with records in the computer database. Two azimuth readings in the field book and computer database were five degrees different to disc readings. All other readings were in agreement.

Even given the small drillhole diameter, drill path deviations are small. Average change in dip is about 1º per 50 m; average change in azimuth is about 1.5º per 50 m. This is thought to be due to the requirement to drill with low pull-down force because of ground conditions and the use of lightweight rigs. In H&S's opinion, the lack of downhole surveys for a number of holes drilled in the earlier period of PGM's exploration does not introduce any significant error into the resource estimate.

For the recent drilling downhole surveying is conducted on a more rigorous basis. For all PGM and contractor diamond drillholes, the first survey is taken at 10 m and then at 50 m intervals thereafter, where ground conditions allow. Dip only surveys are taken for RC holes. If the RC hole is used as a pre-collar for diamond the RC component is re-surveyed by the diamond rig crew to obtain surveyed azimuths for this proportion of the hole.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Methods

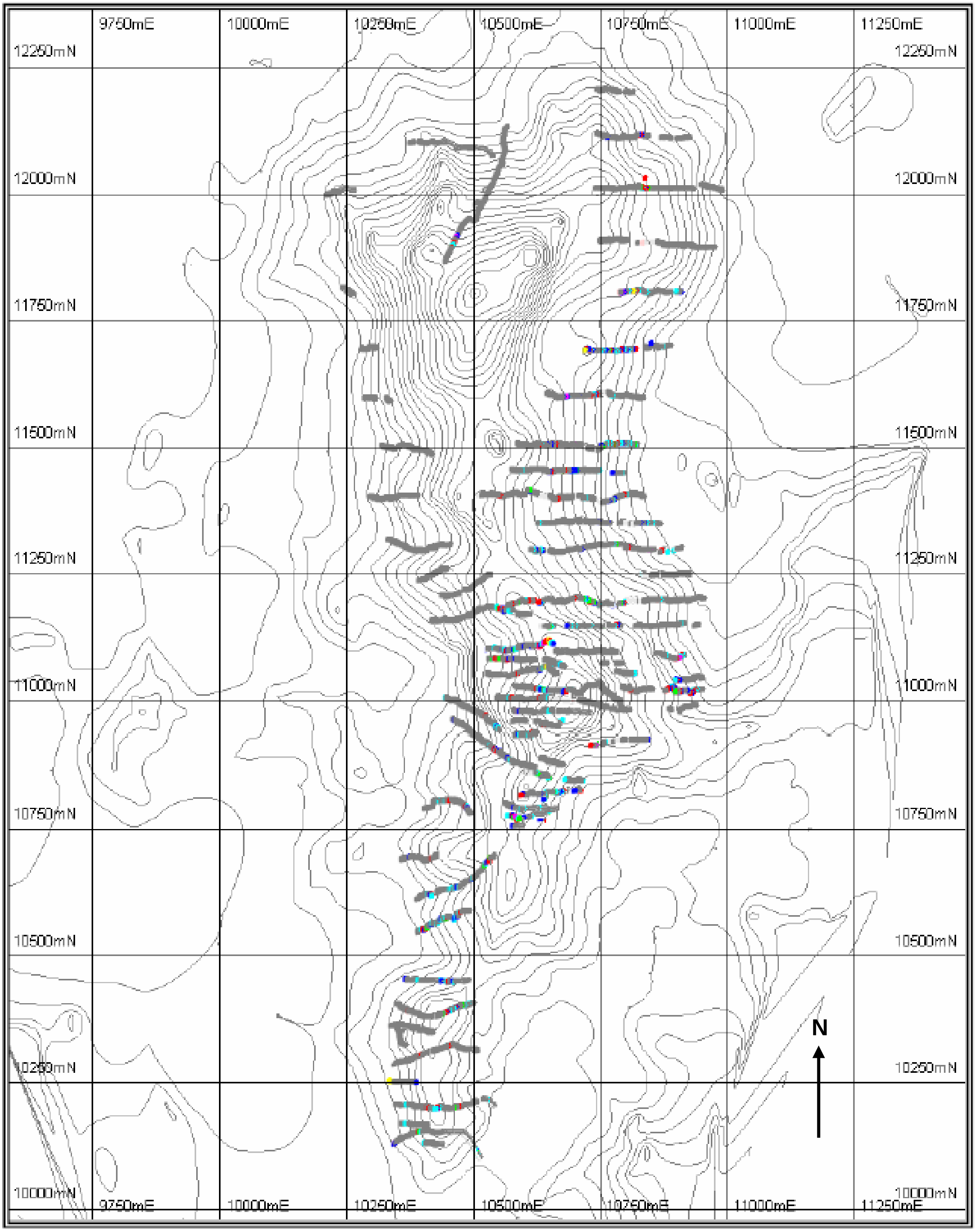
11.1.1 Surface Mapping and Sampling

Two EGSMA geological maps are available for Sukari. A 1:1,000 scale map covers the Amun Zone, in the vicinity of the old Sukari Mine. A 1:2,000 scale map covers the entire 2.5 km strike length of the Sukari Ridge. These maps were used as bases for early PGM evaluation work.

In 1998 surface geology of the Amun Zone was grid-mapped at 1:500 scale by consulting geologist Mr. R.J. Cavaney. PGM geologists have also mapped several surface sample traverses in detail.

Sample trenches excavated by the EGSMA-Soviet study team in the 1970s have been resampled by PGM and additional lines of surface rock chip samples have been taken also. Surface rock chip sample traverses are shown in Figure 11.1-1 and have been included into the resource dataset.

Figure 11.1-1 Surface Rock Chip Sampling



Trench sample locations have not been surveyed and these data have been excluded from the sample database used for resource estimation. Surveyed locations are available for surface rock chip samples and they have been used in resource estimation. The surface rock chip samples total 4,908 (10,540 m of two metre samples) originating from 180 sample lines.

11.1.2 Underground Mapping and Sampling

Rock chip and channel samples have been taken from adits, underlay shafts, drives and stope pillars in the old Sukari gold mine and other workings on Sukari Ridge. Sample locations were surveyed by tape and compass. Assays of underground sampling have not been used for resource estimation.

11.1.3 Logging and Sampling, RC Holes

Sukari geological and technical staff are present during the drilling of all RC holes and are responsible for the taking of the samples for assay. In addition to sample collection, the drill rig geologist is also responsible for recording:

1. Geology – logged at the drill rig onto RC log sheet.
2. Sample Recovery – recorded onto the RC log sheet as a mass (kg) and as a percentage of theoretical sample return.

Samples are routinely taken at one metre intervals. Prior to drilling and sampling the next drilling metre the driller is instructed to clean the sample system on the rig. This is undertaken by lifting the face off the hammer off the bottom of the hole and blowing back through the rods and out through the cyclone to clear all sample from the previous metre. This is undertaken to minimise any potential downhole contamination.

The drilled material for a single one metre sampling interval is collected into a large green plastic bag taken from the bottom of the drill rig cyclone. The total recovered sample is weighed prior to splitting. The sample weight is recorded in kilograms (kg) and also as a percentage recovered on the geological log sheet.

Table 11.1-1 shows the expected recovery with different RC hammer diameters using a bulk density (BD) of 2.68 t/m^3.

Table 11.1-1 Expected Recovery from 1 m RC Sample by Weight

inch	6	5.38	5.06	5.00	4
mm	**152.4**	**136.7**	**128.6**	**127.00**	**100.16**
130%	63.6	51.1	45.2	44.1	28.2
120%	58.7	47.2	41.8	40.7	26.00
110%	53.8	42.2	38.3	37.3	23.9
100%	**48.9**	**39.3**	**34.8**	**33.9**	**21.7**
90%	44.00	35.4	31.3	30.5	19.5
80%	39.10	31.4	27.8	27.10	17.40
70%	34.2	27.5	24.4	23.7	15.20
60%	29.3	23.6	20.9	20.3	13.00
50%	24.5	19.7	17.4	17.00	10.9
40%	19.6	15.7	13.9	13.6	8.70
30%	14.7	11.8	10.4	10.20	6.50

At the time of writing, RC sample recoveries were available for 56,986 RC sample intervals. Statistics of RC recoveries are shown in Figure 11.1-2; average RC sample recovery is about 86% and is considered good. Gold assays and RC recoveries are plotted as a scatter in Figure 11.1-3 and shows that there is no significant correlation between grade and RC sample recovery.

Figure 11.1-2 Histogram of RC Sample Recoveries

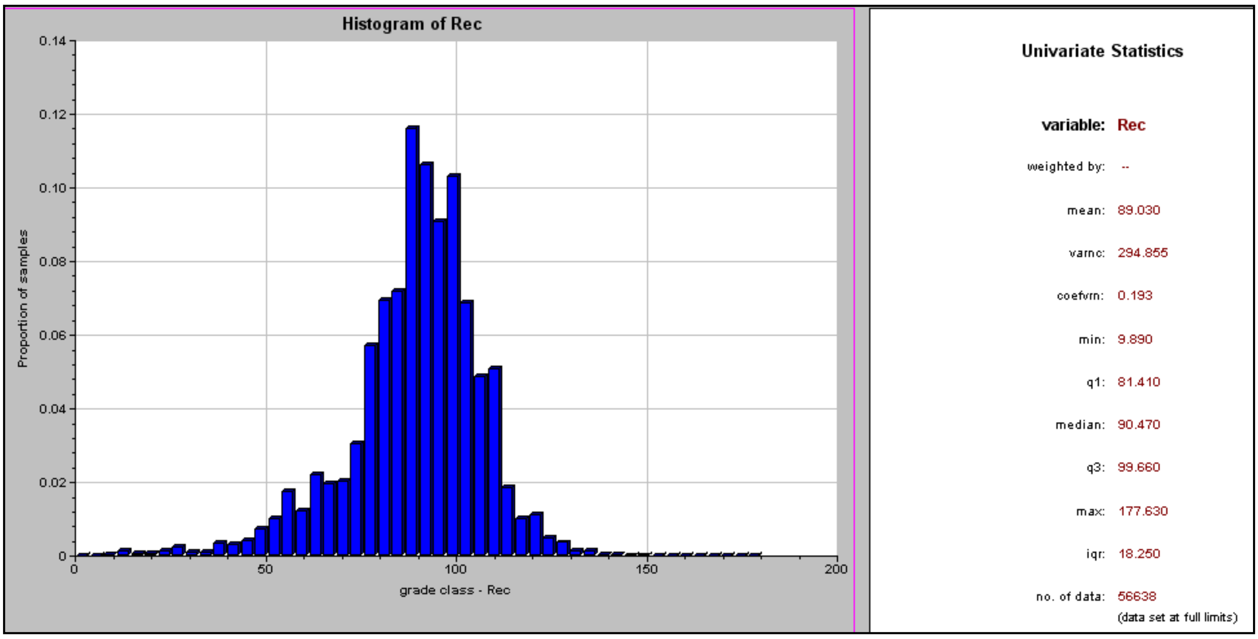
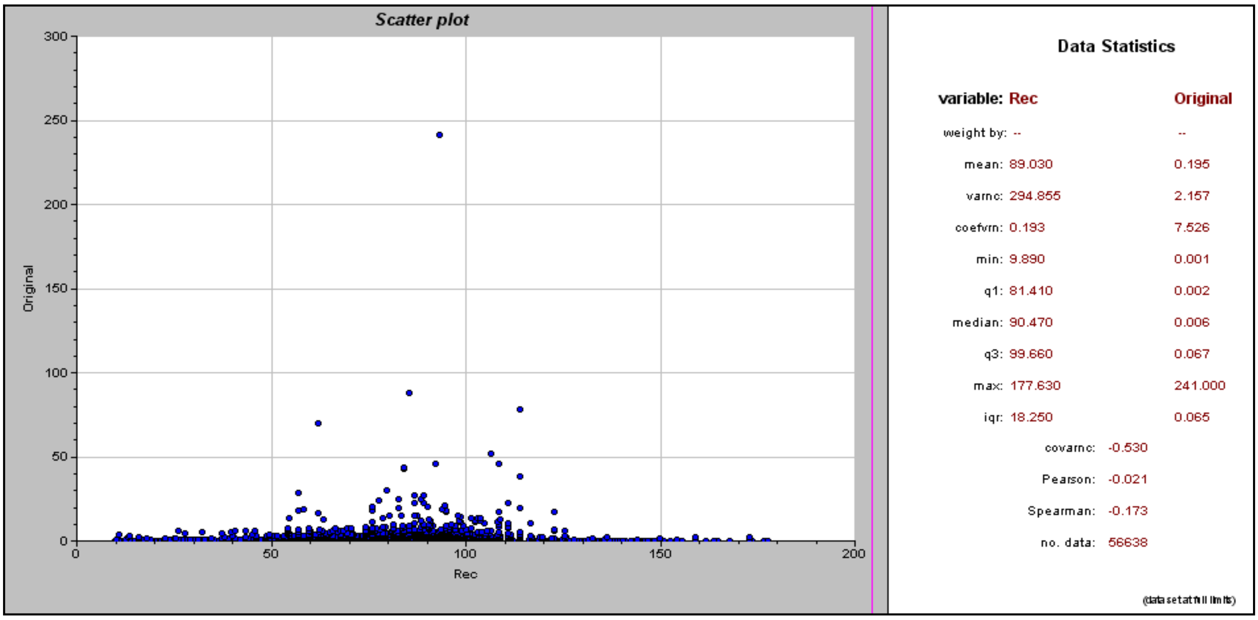


Figure 11.1-3 Scatter Plot, RC Sample Recovery vs Gold Grade



The bulk RC sample (collected in large green plastic bag) is split at the drill rig site to obtain the sample for assay. The bulk sample is reduced in mass by passing the sample through a 50-50 splitter (with 25 mm spaced riffles), until a 4 to 5 kg sample is collected. The final split sample is placed in a numbered calico bag for dispatch to the onsite sample preparation facility. The method for splitting is described as:

"*prior to passing the sample through the splitter place a sheet of plastic over the top of the splitter and pour the sample on to this plastic sheet. Then quickly pull away the plastic sheet to allow the sample to drop through all together. This is important as the sample will be layered in the green plastic and ensures a good mix of the sample, before decreasing the sample mass*."

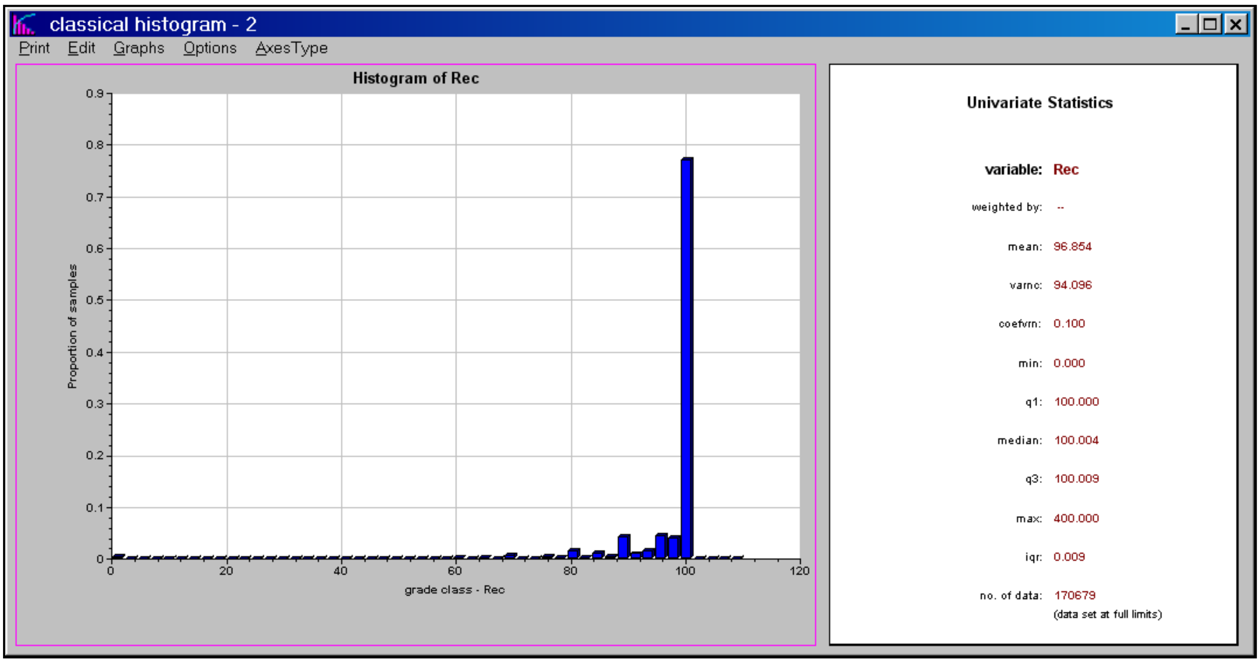
In the case where a wet sample is encountered the sample is collected in a poly weave bag identified by the full BHID (Bore Hole Identification) and metre interval marked in aluminium tag. The sample is allowed to dry for two days and then split in the same way as the dry sample described above. H&S reported that wet drilling conditions were rarely encountered in RC holes at Sukari.

11.1.4 Logging and Sampling, Core Holes

Core is laid out in trays, each of which carries 6 to 7 m of core. Drillers mark each core tray with drillhole number and tray number, and a wooden core block showing downhole depth is placed at the end of each core run. Sampling is based on 1 m intervals as a maximum, though the sampling interval length will be modified to honour lithology boundaries to a minimum sample interval of 0.3 m.

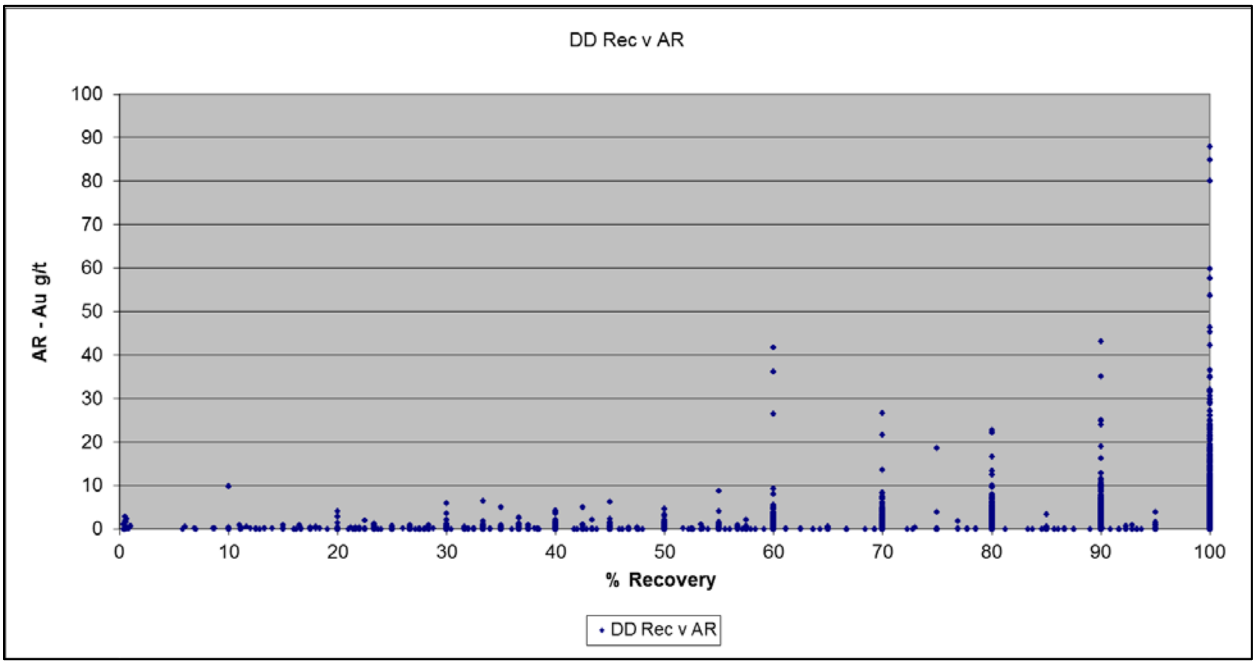
The geologist checks downhole depths marked on core blocks, then measures the length of core recovered per drill run (i.e. between consecutive core blocks). At the time of writing, measured core recoveries were available for a significant proportion of the total diamond drillholes. Approximately 542,061 m of diamond core metres have been assessed for recovery. Statistics of length-weighted core recoveries are shown in Figure 11.1-4; average core recovery is about 94.7%. Generally the core recoveries are acceptable.

Figure 11.1-4 Histogram of Length-Weighted Core Recoveries



Gold assays and core recoveries were composited into 1m intervals to allow direct comparison. Figure 11.1-5 shows a scatter plot of gold grades against core recoveries. There is no significant correlation between grade and core recovery.

Figure 11.1-5 Gold Grade vs Core Recovery



Core is marked in one-metre intervals and logged for rock mechanical properties, e.g. number of fractures, number of pieces greater than 10 cm length. The core is then photographed (both wet and dry).

At approximately 50 m intervals, a 10 cm to 15 cm length of core is selected for bulk density determination. The analysis of this bulk density information is provided in Section 11.4.

The core is then logged for lithology, alteration and mineralization features, with descriptors recorded for each sample interval. A sample number is then assigned to each interval that is to be assayed. Sample "from-to" intervals are recorded in a sample ticket book and two numbered tickets are inserted in a calico sample bag. The hole number, interval and sample number are also written on the calico bag by the geologist.

Core is sawn in half length-wise with a diamond saw under supervision of a geologist. Half core is bagged in numbered calico bags. Sample intervals are regularly checked against the intervals written on both the calico bag and on sample tickets contained in the bags. Intervals of core too broken to be sawn are sampled by taking every second piece of core in the sample interval.

11.1.5 Sample Quality

The production of high-quality samples has been a key aspect of the entire exploration programme and this philosophy has been applied to all aspects of sampling.

During rock chip sampling great care was taken to ensure that a sampling channel of consistent width and depth was collected.

Diamond core recoveries are routinely recorded as part of the standard geological logging practice. The vast majority of diamond core is highly competent and an average recovery of 96% has been achieved. Due to the consistently high core recoveries, the opportunity for over or under-reporting grades from diamond drill samples is considered minimal.

RC sampling weights have been recorded routinely allowing for calculation of a theoretical recovery. An average recovery of around 89% has been achieved and this is considered excellent. All RC drilling was carried out using face sampling hammers, employing rigs equipped with a large air capacity and pressure to effectively flush samples from the hole face through the rod line and hoses. The RC string and hole were cleaned after each drill rod was attached and prior to continuing drilling.

RC and diamond holes have been routinely sampled at one metre intervals and or to lithological contacts as decided by the logging geologist. This sampling interval is considered to be small compared to the typical width of the deposit and the spatial variation in gold grade within them. In addition, higher-grade zones are common within the deposits, and are generally thick enough to be defined by a series of adjacent one metre samples.

11.2 Sample Preparation

All diamond core and RC samples are prepared at the Sukari sample preparation facility.

11.2.1 Core Samples

The half core sample is put in an oven for two hours (110ºC). The dried material is initially passed through a jaw crusher to produce fragments less than 10 mm. The crushed material is then passed through a secondary crusher, to reduce the sample further (down to 4 mm) prior to pulverization. The sample is split into two equal halves, one half returned to the sample calico bag, using a riffle splitter. One of the equal sized splits is used for pulverizing and the other added to the calico bag for storage. The current procedure for cleaning the sample preparation equipment between samples is by high pressure air guns with dust extraction fans. Previous to this procedure (circa 2003) barren quartz wash was used.

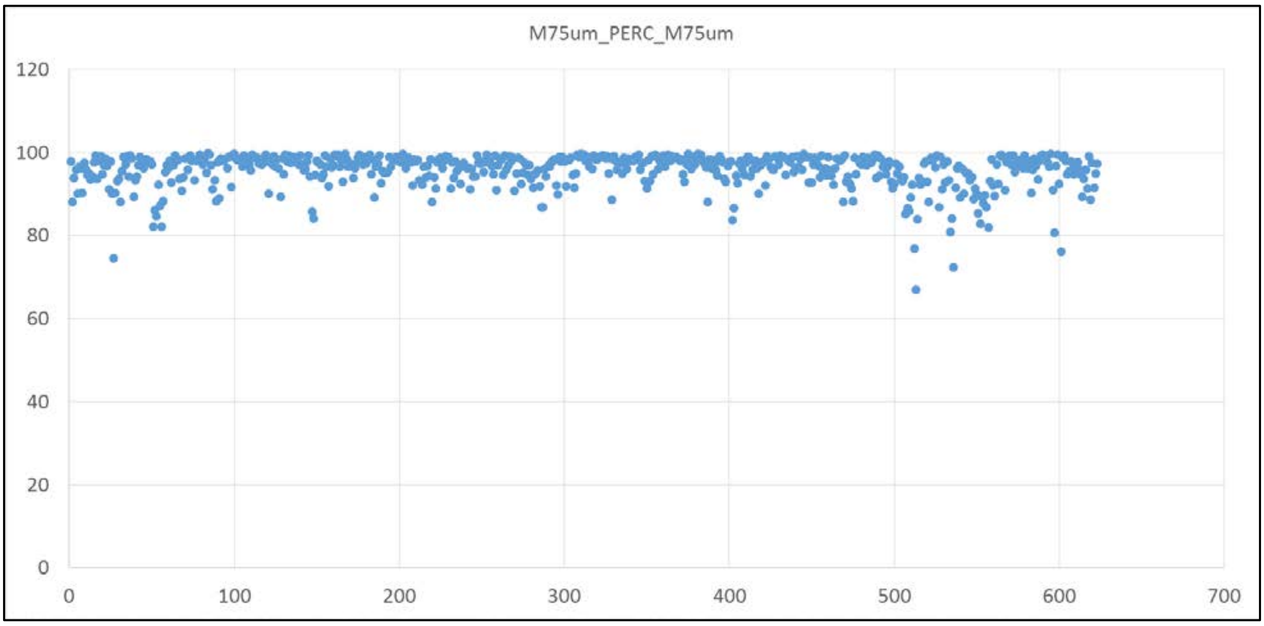
11.2.2 RC Samples

The RC samples are split upon receipt at the sample preparation facility to give an adequate size sample for the pulveriser.

All crushed material is pulverised as a single pass in a ring mill. The Sukari sample preparation facility has six Labtech Essa pulverisers (LM2) available. The pulverized sample is finger tested for powder consistency. If there is any roughness or coarse material, the sample is repulverized. Two approximately 250 g craft packets of sample are taken by dipping a small plastic scoop from several places into the pulverised material within the bowl once the puck has been removed. One sample is submitted to the contract assay laboratory and the other is submitted to the Egyptian Geological Mining and Survey Authority. Standards and blanks are submitted with the samples for assaying (Section 14).

Figure 11.2-1 shows the results of sieve analyses that is routinely undertaken by the analytical lab Bureau Veritas as part of the QAQC procedure. The plot shows that the Sukari sample preparation facility is performing well in terms of ensuring that pulverization of the sample is being undertaken according to design criteria, i.e. better than 90% samples passing 75 µm. The Sukari sample preparation facility is achieving approximately 96% of the samples passing 75 µm.

Figure 11.2-1 Percentage of Particles passing 75 µm



11.3 Assaying

All gold assays on samples collected by PGM/SGM have been conducted by either Minesite Reference Laboratories (earlier phases of drilling prior to 2005) or Ultra Trace (used during the 2005 drilling), both situated in Perth, Western Australia.

Upon receipt of samples, pulps are finger tested for fineness and if grinding is considered insufficient, individual sample pulps are selected for repulverizing.

Earlier assaying by Minesite Reference Laboratories (before recommencement of exploration activities in 2005) was conducted by Aqua-Regia digest (AR) of a 25 g portion selected by taking multiple dips from the sample pulp packet. Gold is extracted from the aqua-regia liquor into DIBK solvent and gold content determined by AAS. At least three duplicate assays are performed in every batch of samples, the samples for duplicate analyses being selected at random by the laboratory information management system (LIMS).

Ultra Trace has conducted all assaying of samples processed since the recommencement of exploration in 2005. Ultra Trace was acquired by the Bureau Veritas Group in 2007 and now operates as Bureau Veritas Minerals. Bureau Veritas is a globally-recognized and certified lab. Assaying by this laboratory has continued to be by AR, however, using a 40 g charge where the digest is diluted, mixed and an aliquot of the acid solution is taken and analyzed directly by ICP-MS for gold and other elements.

Of samples that return assays more than 1 g/t Au, a selection are subsequently analyzed by 30 g (Minesite Reference Laboratories) or 40 g (Ultra Trace) fire assay with AAS finish.

11.4 Rock Density Data

11.4.1 Site Measurements

Samples comprising intact pieces of core 10 to 15 cm length are taken at approximately 50 m downhole intervals. Bulk density (BD) is measured on site by two techniques:

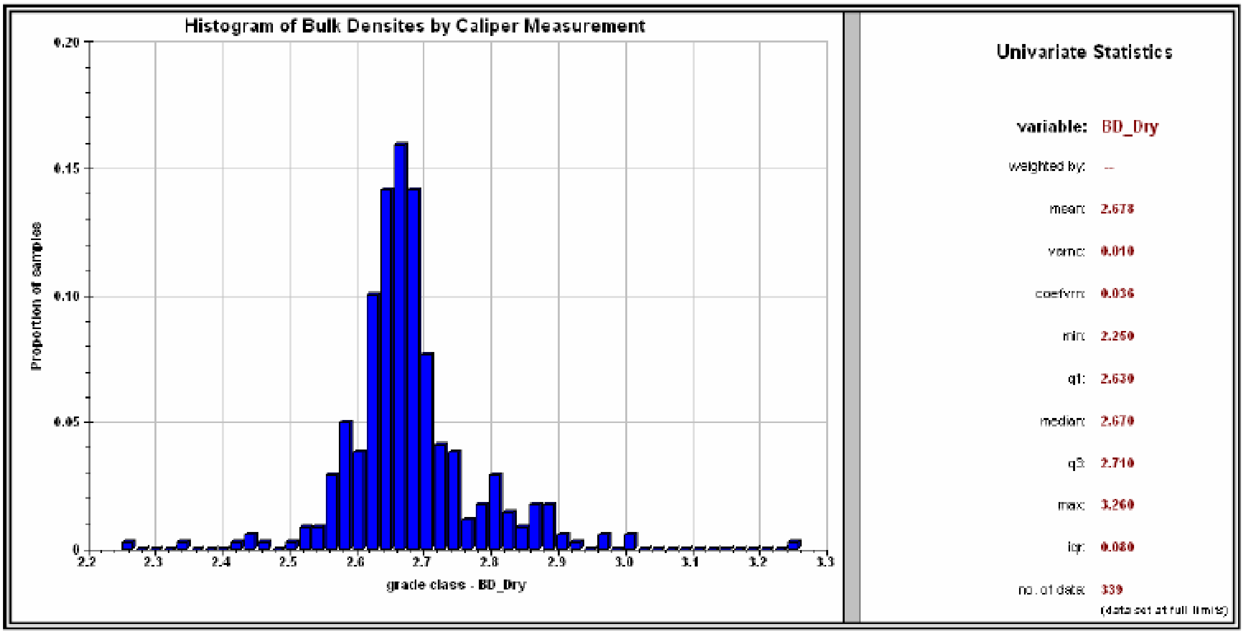
- Each sample is weighed twice on a laboratory electronic balance. Sample volume is estimated by measuring sample length (two measurements, rotating core each time) and core radius (six measurements, two at each end and two in the middle of the sample).

Radii are measured using a Vernier calliper. An effort is made to cut the ends of core samples flat in order to make length measurements more accurate.

- Weigh the sample as for the step immediately above, then immerse the sample in a measured volume of water in a graduated cylinder in order to measure sample volume by displacement. The graduated cylinder allows reading of volumes to about ± 0.2 cc.

Prior to 2005, a database of 339 BD measurements had been generated. Bulk densities calculated by the two methods rarely disagree by more than 5% and normally agree to within 2%. The mean of BD calculated from calliper measurements is 2.68 g/cc, compared to 2.69 g/cc for BD's using displacement volumes. Because samples have not been waxed prior to immersion for measurement of volume by displacement, H&S regarded the volumes calculated from calliper measurements likely to be more accurate. A histogram of BD's calculated from calliper measurements are shown in Figure 11.4-1.

Figure 11.4-1 Bulk Density Measurements Histogram



A bias has possibly been introduced into BD measurements by the process of sample selection. The methods used for bulk density measurements at site require solid lengths of drill core. In places, drilling has encountered broken and sheared ground. These zones almost certainly contain rock of lower bulk density but the volume of gold resources contained in such zones is thought to be small.

Bulk densities used in the current resource model are detailed Section 14.1.10. For the resource model BD's for the various modelling domains have been calculated. The BD data available is 3,718 measurements.

11.4.2 Independent Checks

Two composite metallurgical samples were subjected to bulk density determinations (Lakefield Oretest, 2000a), yielding 2.696 g/cc and 2.69 g/cc. In addition, bulk density measurements of 2.644 g/cc, 2.720 g/cc and 2.571 g/cc on lump rock samples used for comminution testwork (Lakefield Oretest, 2000b) support the drill core bulk density measurements. Samples for comminution testwork were taken from the old Sukari mine workings.

In August 2000, pulp material remaining from samples collected from site by H&S was submitted to Minesite Reference Laboratories for pulp density determinations. A Beck-man Model 930 vacuum pycnometer was used for measurements on 50 g pulps. Claimed accuracy for the instrument and method is ± 10%. Fifteen samples of quartz wash material reported bulk densities ranging from 2.59 g/cc to 2.74 g/cc, with a mean of 2.68 g/cc being very close to the expected bulk density of clean quartz (2.65 g/cc).

Pulp bulk densities measured by pycnometer average 2.74 g/cc. Bulk density determinations by pycnometer are, on average, two per cent higher than bulk densities measured on site. Measurements by pycnometer fail to account for in situ porosity and voids in sample material so the results are as expected. The bulk densities measured at site are considered reliable and adequate for the purpose of resource estimation.

11.5 QAQC Analysis

Centamin has instigated external QAQC processes to control the quality of the analytical data through systematic monitoring of the accuracy and precision (i.e. repeatability) within the drill sampling and assaying process, including checks for contamination. The types of check samples that have been introduced into the sample stream include blank samples (“blanks”), certified reference materials (“standards”), and field duplicate samples.

The QAQC procedures employed during the drilling programs have involved the routine insertion of control samples into the sample collection stream, consisting of Certified Reference Material (CRM), uncertified “blind” blank samples, and duplicate samples. Management and assessment of the analytical control data is undertaken by the site geologists to allow timely diagnostics of sample errors and identification of error sources.

In addition, the site based Centamin and external Ultra Trace analytical laboratories have their own internal quality performance processes which follow best practice guidelines required for qualification under International Organization for Standardization (ISO) standards. The standard QAQC protocols for the laboratories includes the insertion of CRMs, blanks, duplicates and repeat assaying to monitor the quality of the preparation and analytical processes of the laboratory. The results of the internal laboratory quality controls, where the internal lab is only used for sample preparation, are reported regularly to Centamin on a batch by batch basis, and are closely monitored by Centamin personnel.

The QAQC data available from 1 June 2010 to 30 June 2013 was previously reviewed and considered acceptable as part of the January 2014 Mineral Resource estimation (Smith et al., 2014). In order to assess the veracity of the sample data received after 30 June 2013 for inclusion in the mineral resource estimate, Cube independently reviewed the available QAQC sample data from 1 July 2013 to 10 March 2015.

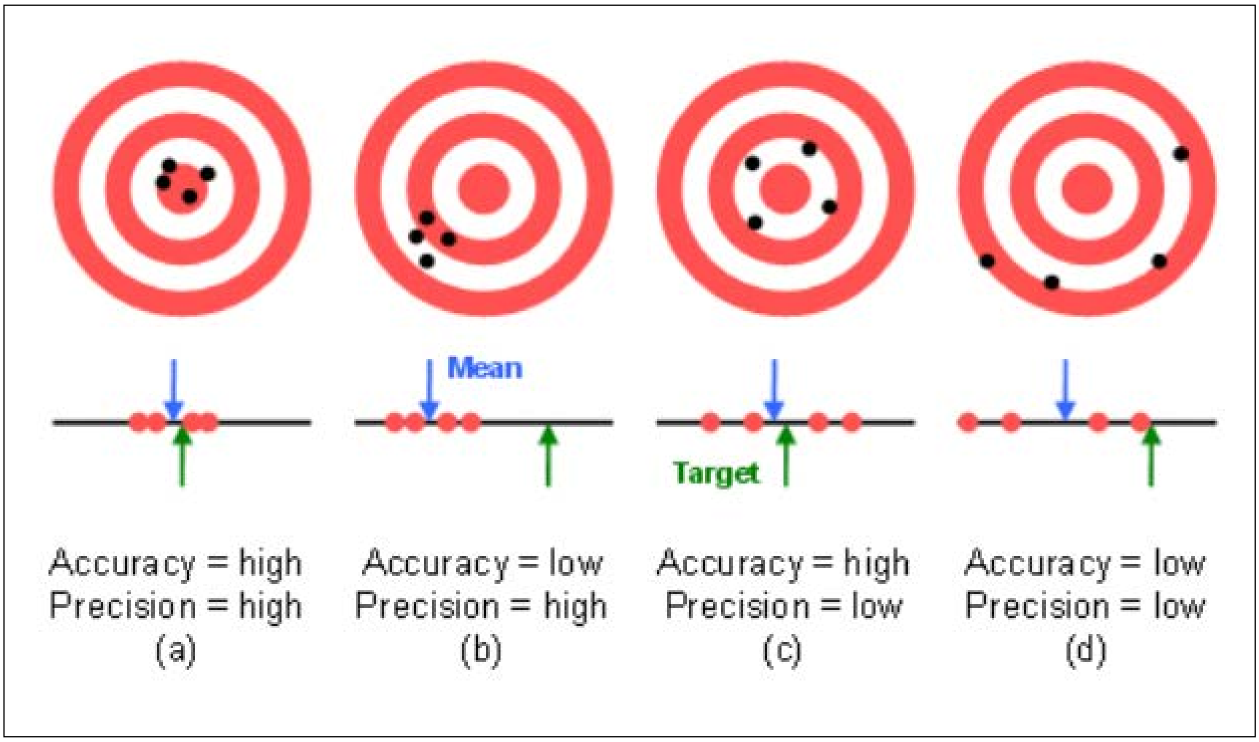
The QAQC processes in place at Sukari have only been applied to sample data associated with resource definition and has not been extended to include grade control drilling completed from underground.

11.5.1 Accuracy and Precision Concept

All control samples were assessed on the basis of accuracy and precision. The precision of the sample results is the measure of how closely the results can be repeated (repeatability). The accuracy of sample results relates to how similar the results are to the true, certified or expected value.

Clearly, it is possible to have good accuracy without good precision, and good precision without good accuracy as shown in Figure 11.5-1. Precision is measured by the use of duplicate and replicate assays, whereas accuracy is measured through the use of reference materials.

Figure 11.5-1 Accuracy and Precision Concept



11.5.2 Certified Reference Material (CRM)

The performance of the CRM sample data was assessed by plotting the replicate assay values of the CRMs against time on the control charts. Good quality analysis of the CRMs will be characterised by a random distribution of data points around the certified mean value, with 95% of the data points lying within two standard deviations of the mean (Abzalov, 2008). If more than 5% of the CRMs submitted are outside three standard deviations of the certified mean value, then corrective action should be taken. In addition, no trends or significant bias should be observed in the control charts and any obvious assay 'outliers' that are likely to be the result of sample mishandling or transcription errors, should be removed from the dataset prior to analysis to avoid any skewing of the dataset.

The CRM dataset was assessed using two statistical tests to demonstrate that the analytical accuracy and precision of the assays were comparable to the certified value of the CRM, and considered acceptable within the 95% confidence limit (Abzalov, 2008).

Accuracy Test – involved the comparison of the arithmetic mean of the replicate analysis of the CRM (*m*) against its certified mean (μ), and if the following condition is satisfied then the analytical results are considered acceptable with regard to accuracy:

$$|\mathrm{m} - \mathrm{\mu}| \leq 2\sigma_L$$

where σ_L is the standard deviation of the replicate analyses of the CRM.

Precision Test (*Chi Square*) – involves the comparison of the estimated standard deviation of the replicate assays against the CRM deviation, and if the following condition is satisfied then the analytical precision is considered acceptable:

$$\left(\frac{S_w}{\sigma_c}\right)^2 \leq \frac{X^2_{(n-1)0.95}}{n-1}$$

Where:

- s_w is the standard deviation of the replicate analyses of the CRM;
- σ_C is the certified value of the CRM standard deviation;
- $X^2_{(n-1)0.95}$ is the critical value of the 0.95 quartile of the X^2 distribution at $(n-1)$ degrees of freedom; and
- n is the number of replicate assays of the CRM.

A total of 451 CRM samples were inserted into the sample stream comprising approximately 1.6% of the total drill samples (28,713) submitted by Centamin since 1 July 2013. A summary of the results from the replicate CRM assays are detailed in Table 11.5-1.

Table 11.5-1 CRM Performance Summary – Sukari Underground Mineral Resource 2015

CRM	Analytical Method	Certified Value (ppm)	No. Assays	Accuracy Test	Precision Test	% Passing 3SD	% Bias	Misclassified Samples
ST01	AR	1.44	41	Passed	Passed	98%	2%	3 removed; 1 reclassified added
ST01	FA	1.46	41	Passed	Passed	100%	0%	1 removed
ST17	AR	0.78	57	Passed	Passed	100%	0%	2 removed
ST353	AR	0.51	43	Passed	Passed	100%	4%	2 removed; 1 reclassified added
ST355/451	FA	2.45	43	Passed	Passed	100%	-1%	OK
ST383	AR	7.26	27	Passed	Passed	100%	3%	1 removed; 2 reclassified added
ST383	FA	7.26	27	Passed	Passed	100%	2%	2 reclassified added
ST482	AR	1.90	62	Passed	Passed	100%	3%	4 removed; 1 reclassified added
ST482	FA	1.94	64	Passed	Passed	100%	0%	1 removed
ST530	AR	0.21	46	Passed	Passed	100%	0%	5 removed

Approximately 4% of the CRM results were identified as sample mishandling or transcription errors, which have been misclassified and were excluded from the analysis as outliers. It is recommended that these samples are located and corrected in the database for future analysis, and that the sample protocols for the insertion of CRMs be diligently applied onsite.

In general, the number of CRMs within three standard deviations was 100% and the bias was less than 5%. The overall accuracy and precision of the assay data relating to the CRMs is within the accepted tolerance limits, and no obvious trend or major bias is apparent within the primary assay data.

Figure 11.5-2 through Figure 11.5-11 show the control charts for each of the standards used (with ± 3 standard deviation control lines plotted either side of the recommended grade).

Figure 11.5-2 CRM ST01_AR Control Chart

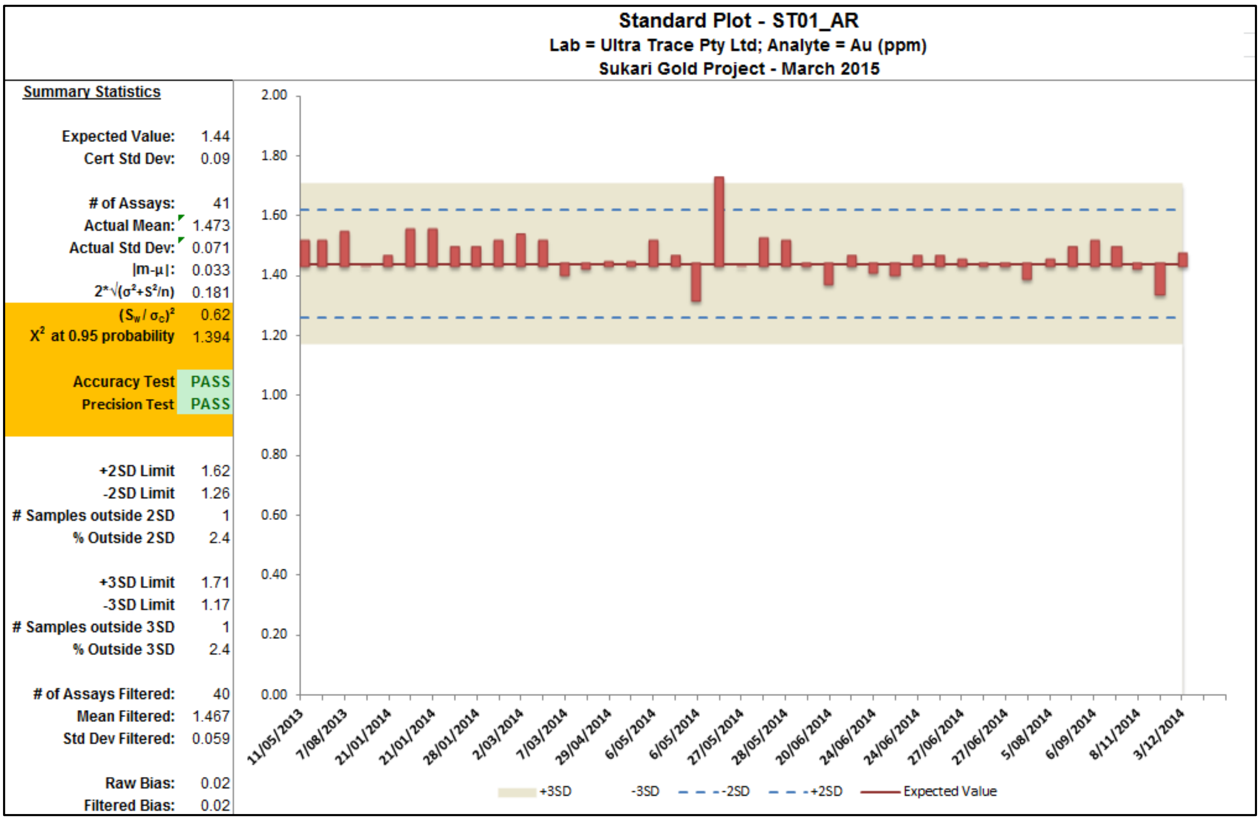


Figure 11.5-3 CRM ST01_FA Control Chart

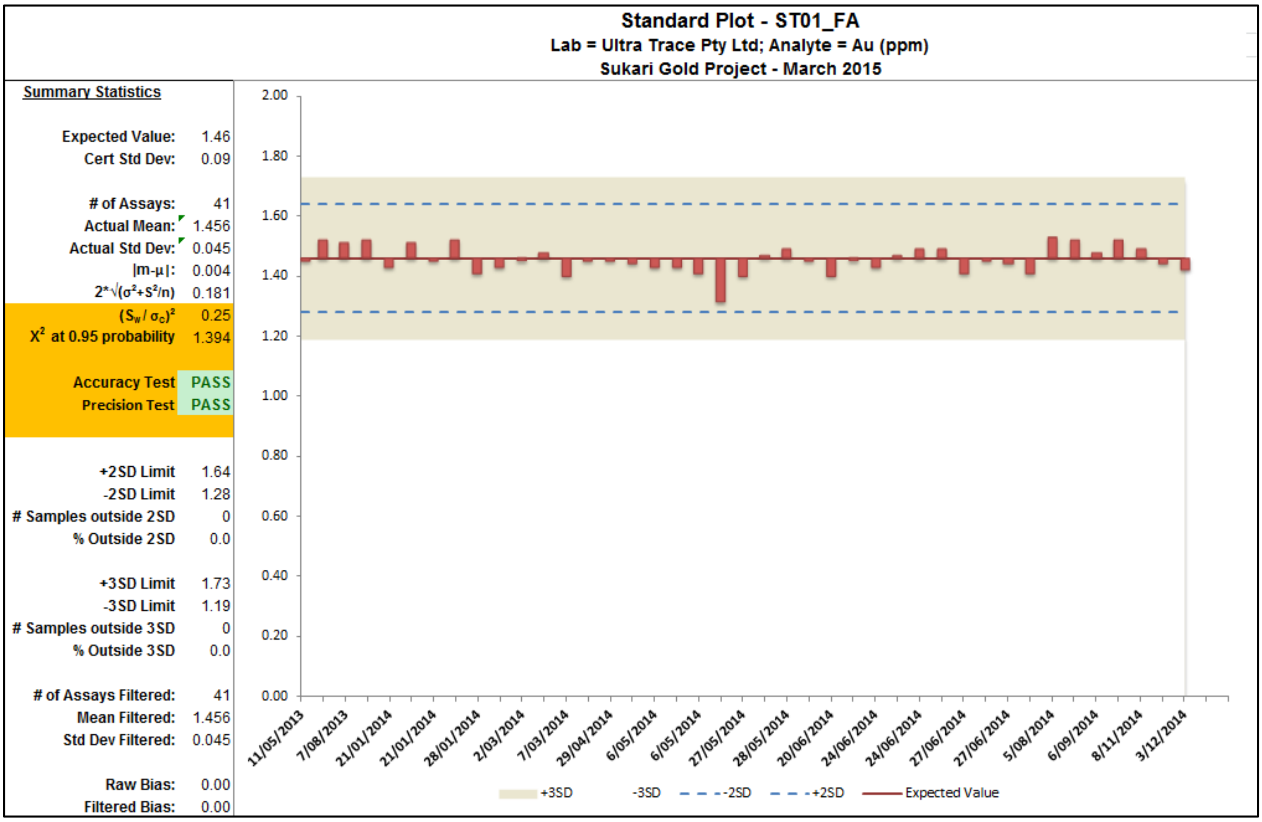


Figure 11.5-4 CRM ST17_AR Control Chart

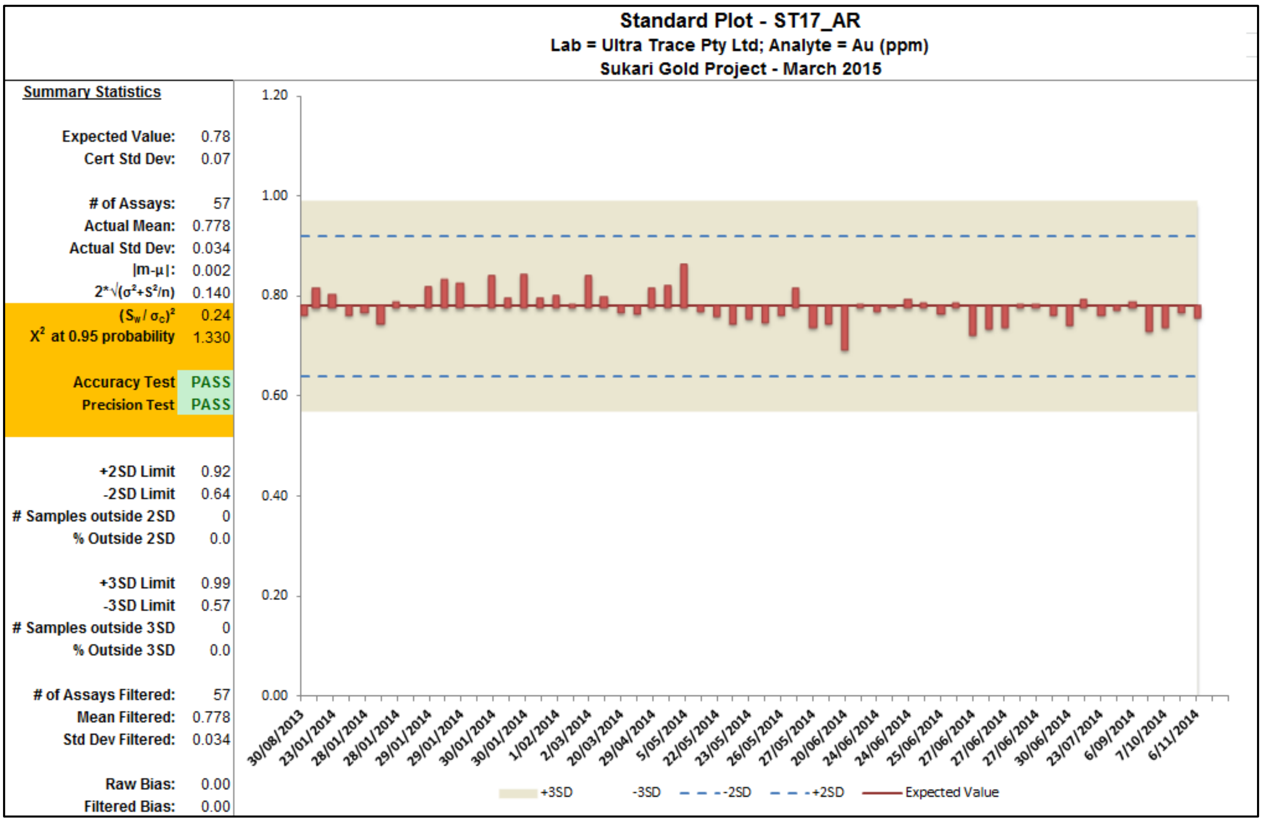


Figure 11.5-5 CRM ST353_AR Control Chart

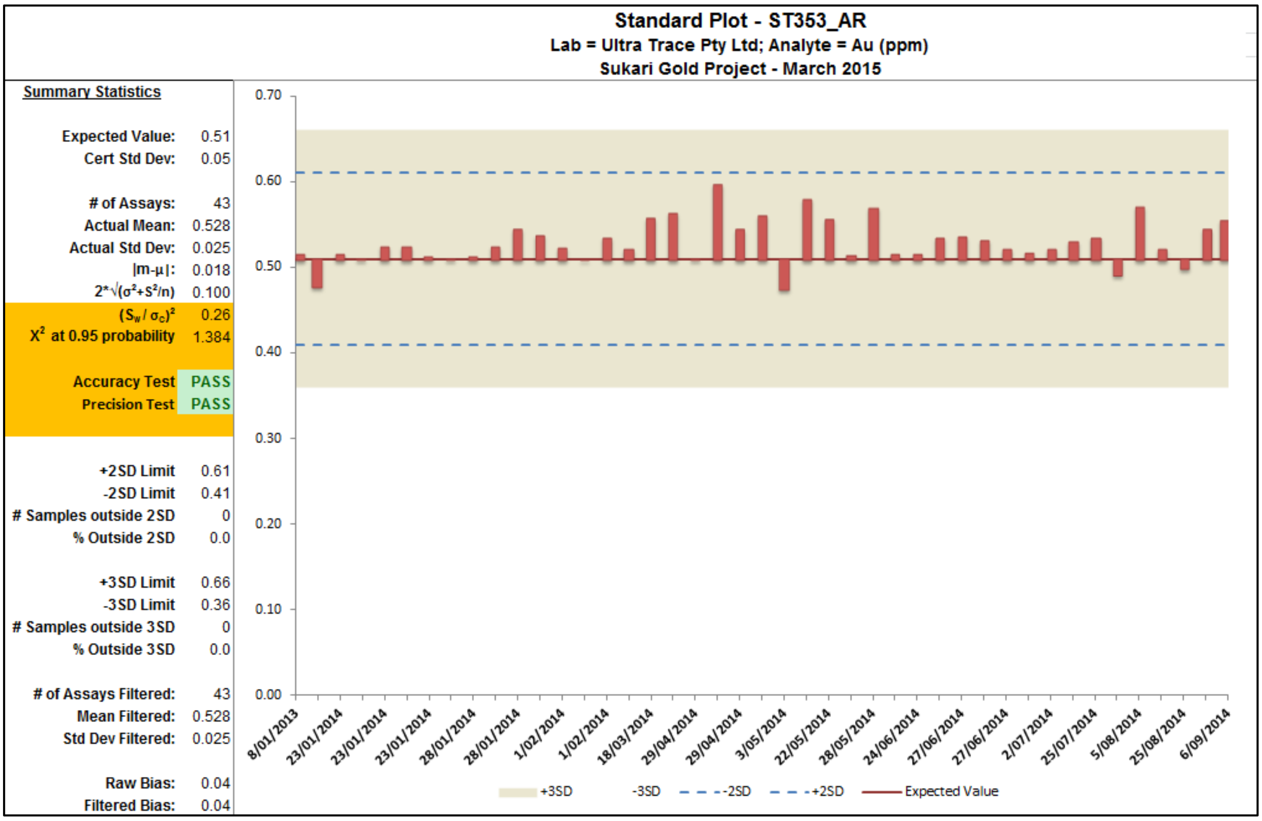


Figure 11.5-6 CRM ST355/451_FA Control Chart

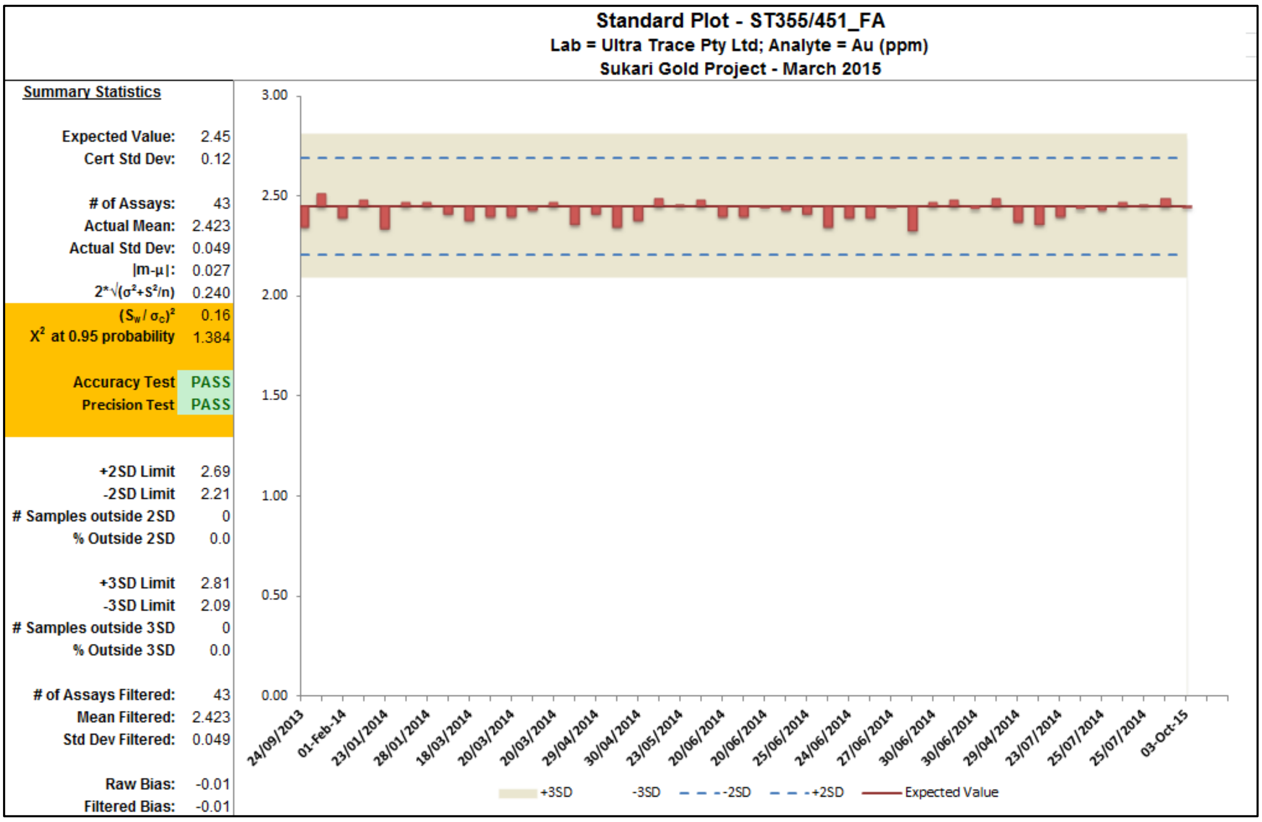


Figure 11.5-7 CRM ST383_AR Control Chart

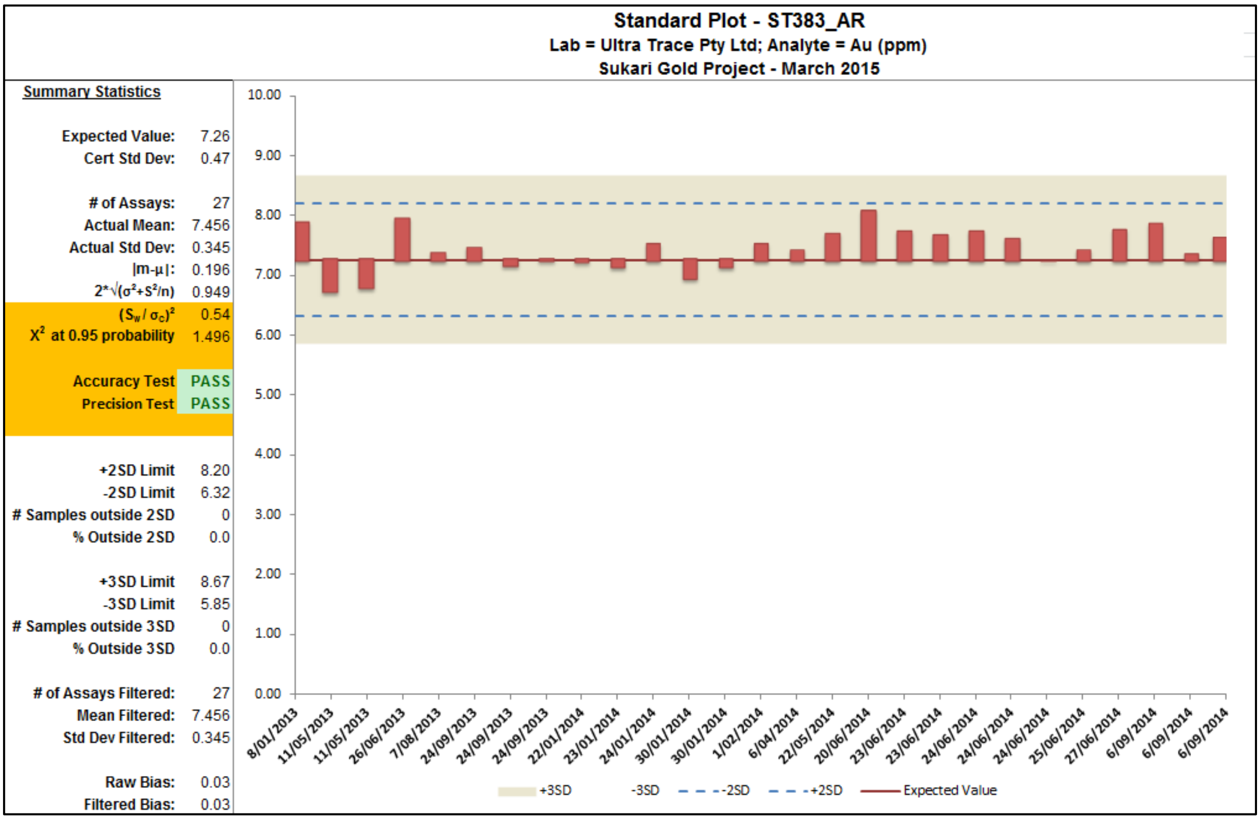


Figure 11.5-8 CRM ST383_FA Control Chart

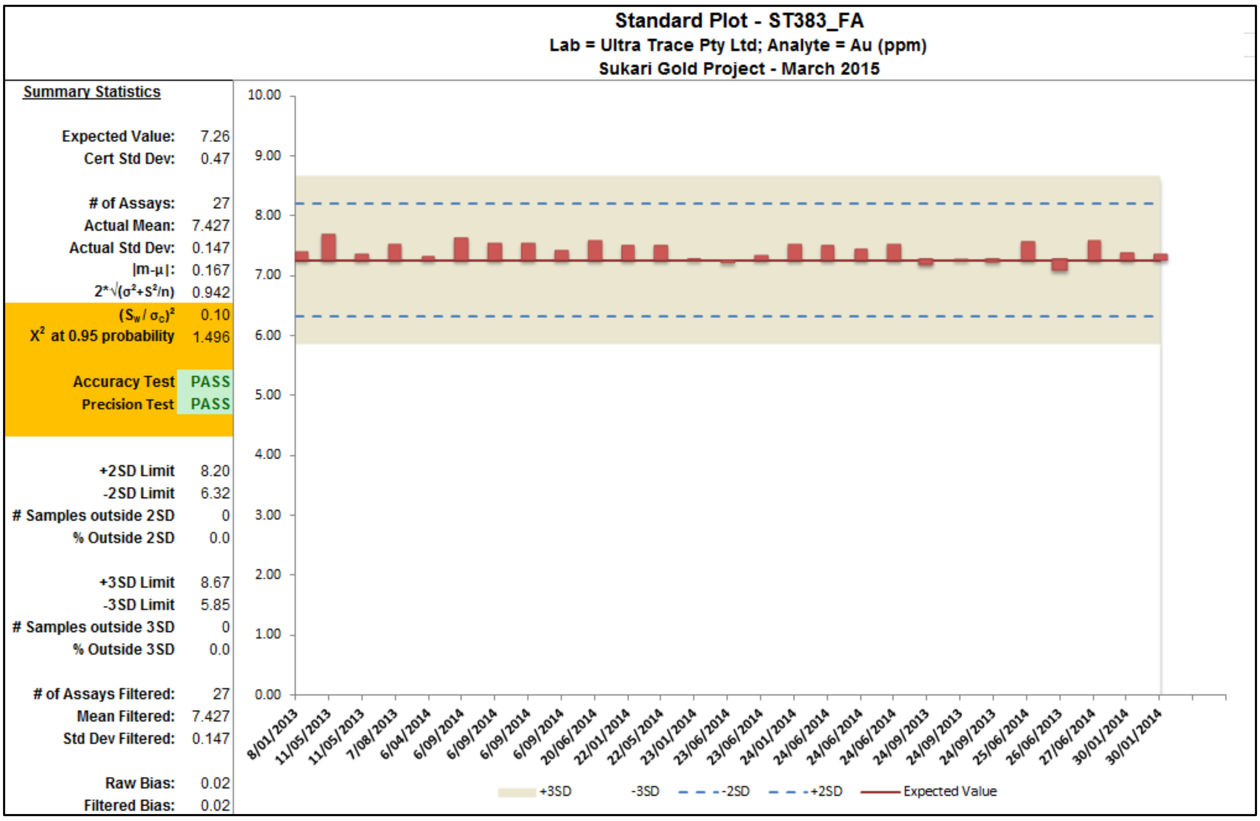


Figure 11.5-9 CRM ST482_AR Control Chart

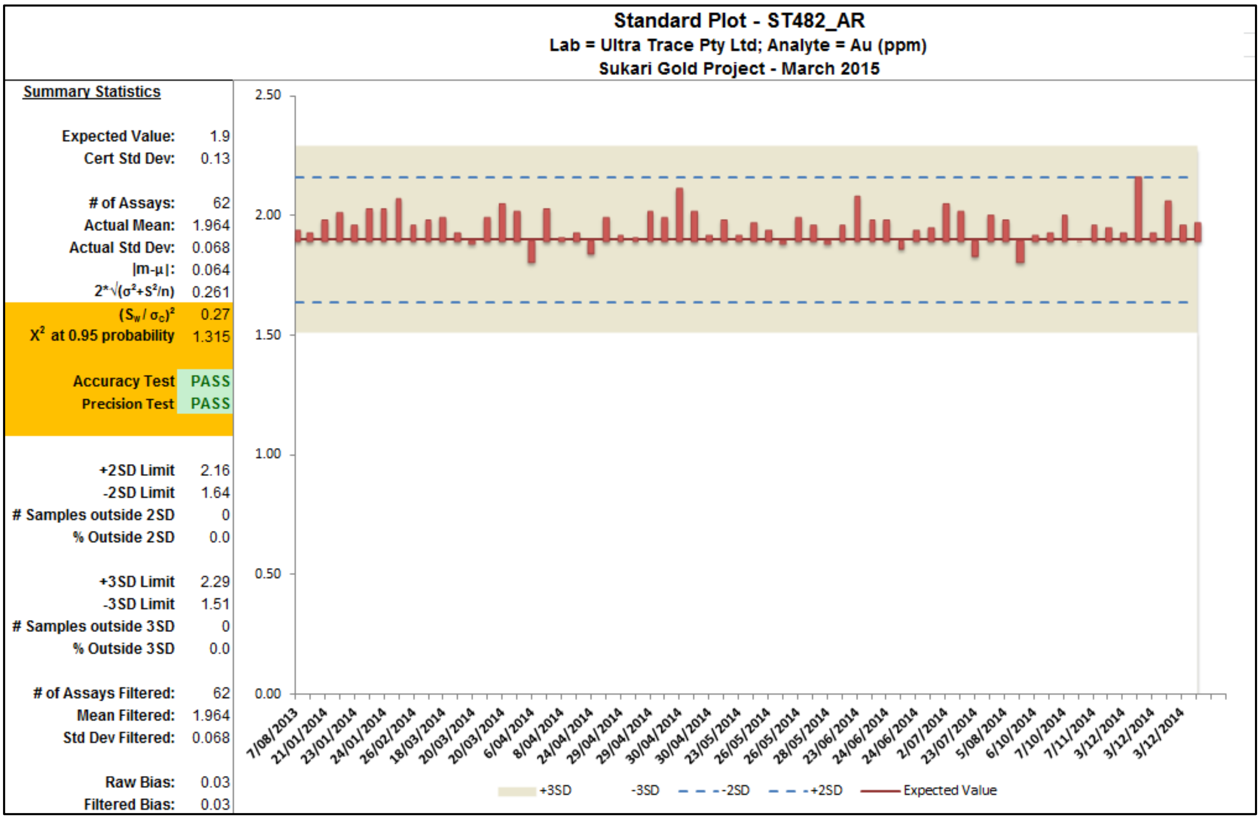


Figure 11.5-10 CRM ST482_FA Control Chart

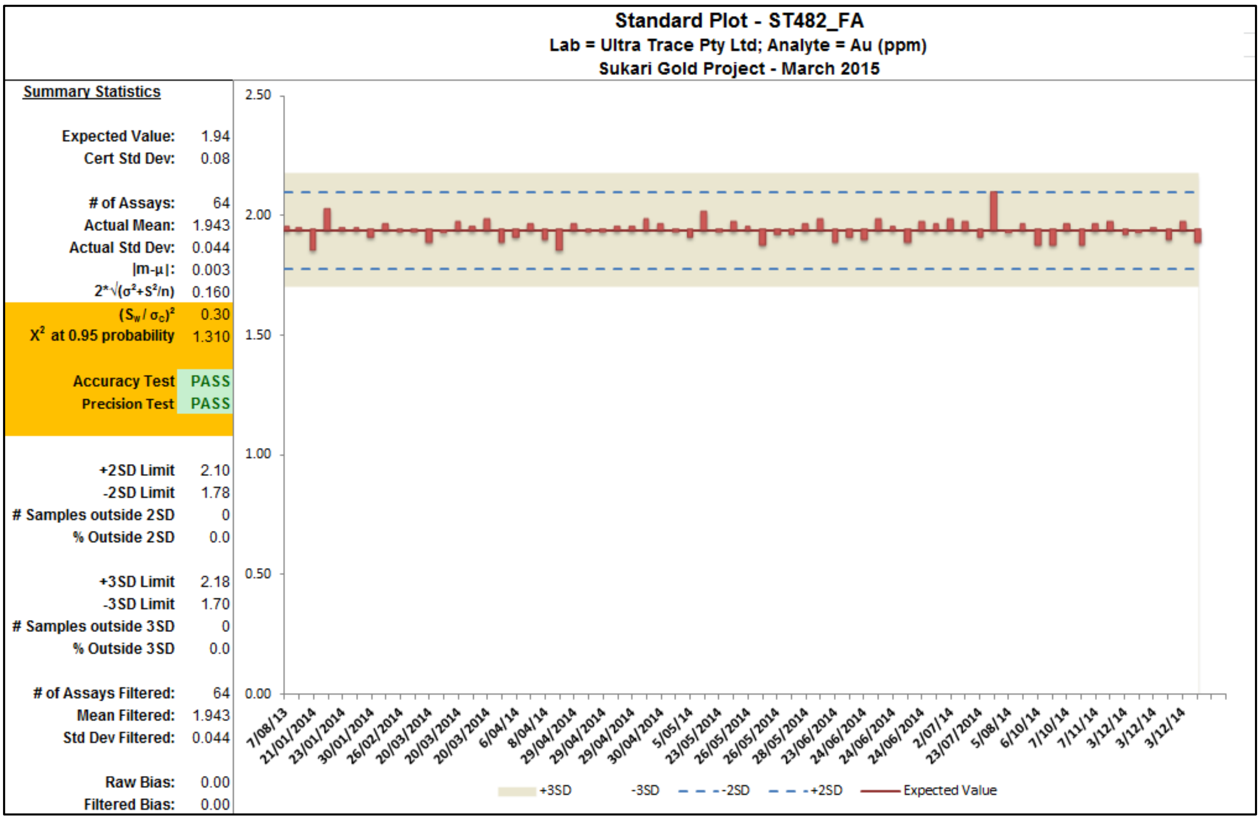
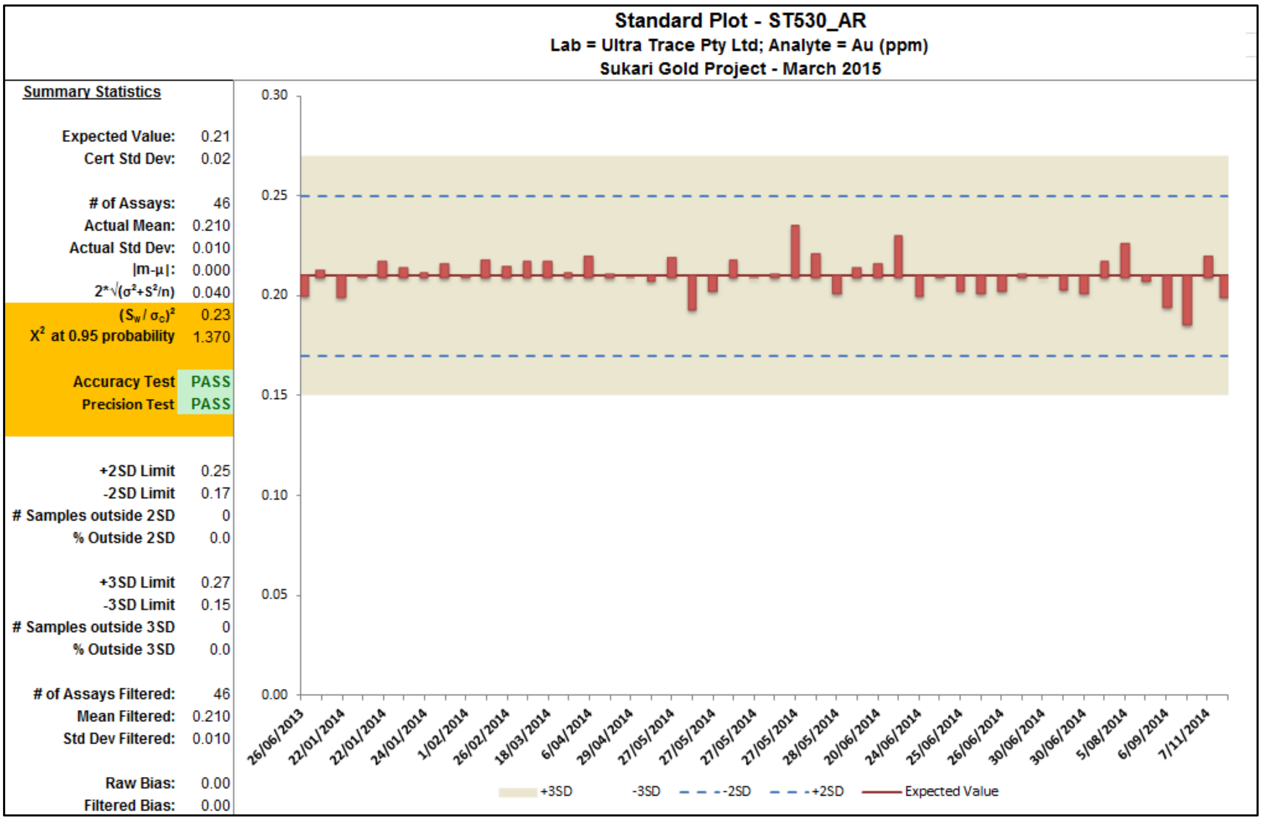


Figure 11.5-11 CRM ST530_AR Control Chart



11.5.3 Blank Reference Material

Blank material is inserted after a drill sample and before a standard to check for cross sample contamination at the pulverization stage of sample preparation. The blank material is sourced from a known barren outcrop (gabbro composition) within the Sukari project area.

Assays for the “in-house” uncertified coarse blanks were assessed by graphing the actual value and the maximum accepted value, which was assigned as 0.1 ppm Au. A maximum accepted value of 10 times the lower analytical detection limit has been used to remove the potential for bias, and precision issues which increase close to the assay method detection limit. Blanks should return a value less than 0.1 ppm Au at least 95% of the time.

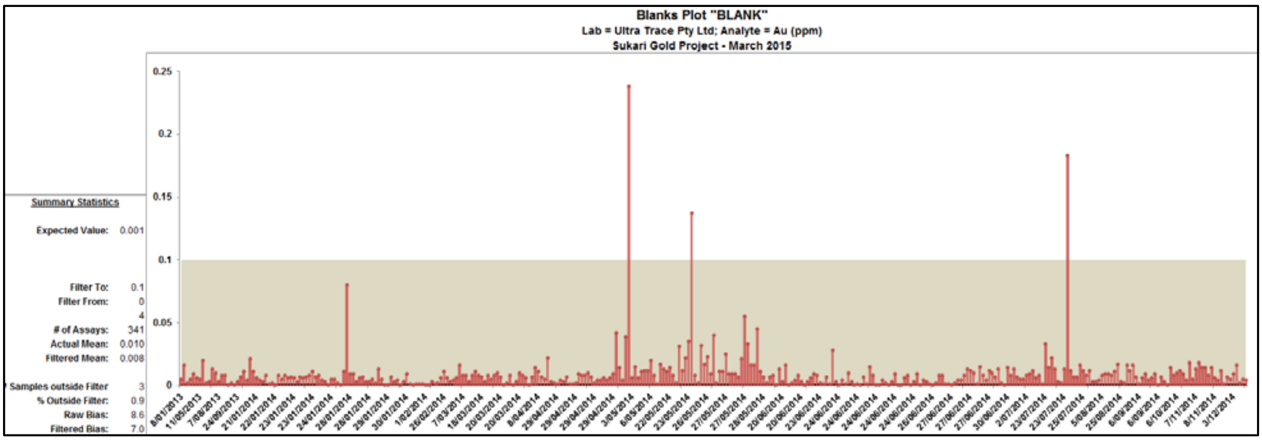
The results are summarized in Table 11.5-2 and Figure 11.5-12 and indicate that the assay blanks data are within acceptable limits. No obvious contamination issues are apparent within the primary assay data.

A total of 341 assay blank samples were inserted into the sample stream which comprises approximately 1.2% of the total drill samples (28,713) submitted by Centamin since 1 July 2013.

Table 11.5-2 Blank Performance Summary

No. of Samples	Blank Name	Max. Accepted Value (ppm)	No. Failing	% Passing	Misclassified Samples
1. 341	2. GLG907-1	3. 0.1	4. 3	5. 99%	6. None

Figure 11.5-12 Blank Control Chart



11.5.4 Fire Assay Checks

As a check of the repeatability of the pulp samples produced by the site sample preparation facility, a suite of duplicates from the pulverised sample have been assayed and are able to be compared to the original assay.

The results for pairs of duplicate samples (original and duplicate) are plotted as X/Y scatter plots and relative paired difference plots (RPD). Scatter plots allow for direct comparison of the data pairs and the assessment of general dispersion, data regression as well as the presence of any outliers. RPD plots evaluate the coefficient of variation for each pair (difference between pairs relative to the pair mean) and allow the measurement of the relative precision error between pairs based on the average coefficient of variation (ACV).

Approximate guidelines for assessing analytical quality allow for pulp duplicates to have an ACV value in the range of 10% to 20% (Abzalov, 2008). The RPD plots allow the visualization of any bias or trend differences between pairs.

A total of 20,768 pairs of pulp data were selected representing assaying by aqua regia (AR) at and fire assay (FA) at the Ultra Trace laboratory. In order to compare the two assay types, the duplicate paired assay data were filtered using only data above 0.1 ppm Au, which was considered the threshold level for mineralized material. The application of a threshold avoids the precision data being negatively biased by values at or near the detection limit. A total of 13,915 duplicate samples (filtered above 0.1 ppm) were reviewed and the analysis indicates that the sample precision was acceptable for this style of mineralization with an ACV of 15% for the umpire pulp duplicates, as indicated in Figure 11.5-13 and Figure 11.5-14.

Figure 11.5-13 Umpire Pulp Duplicates RMPD Plot – Original (AR) vs Duplicate (FA) – Ultra Trace

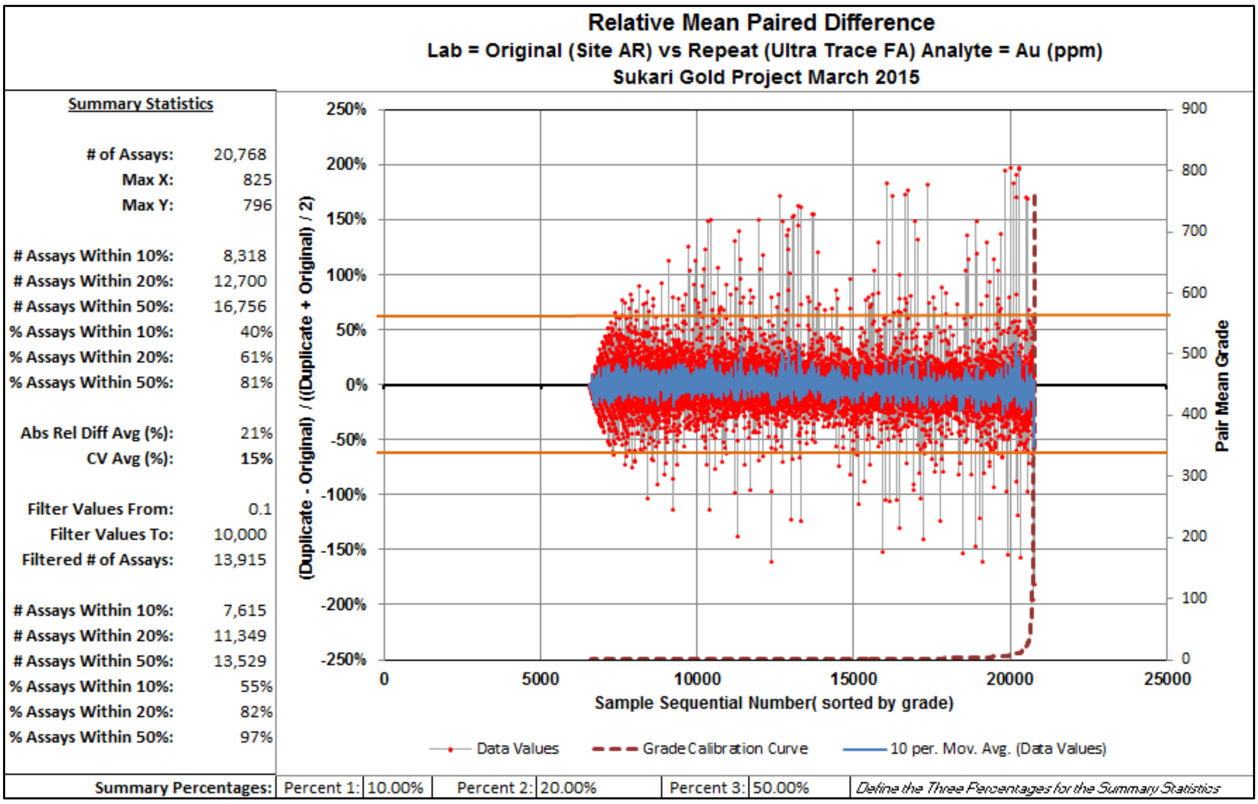
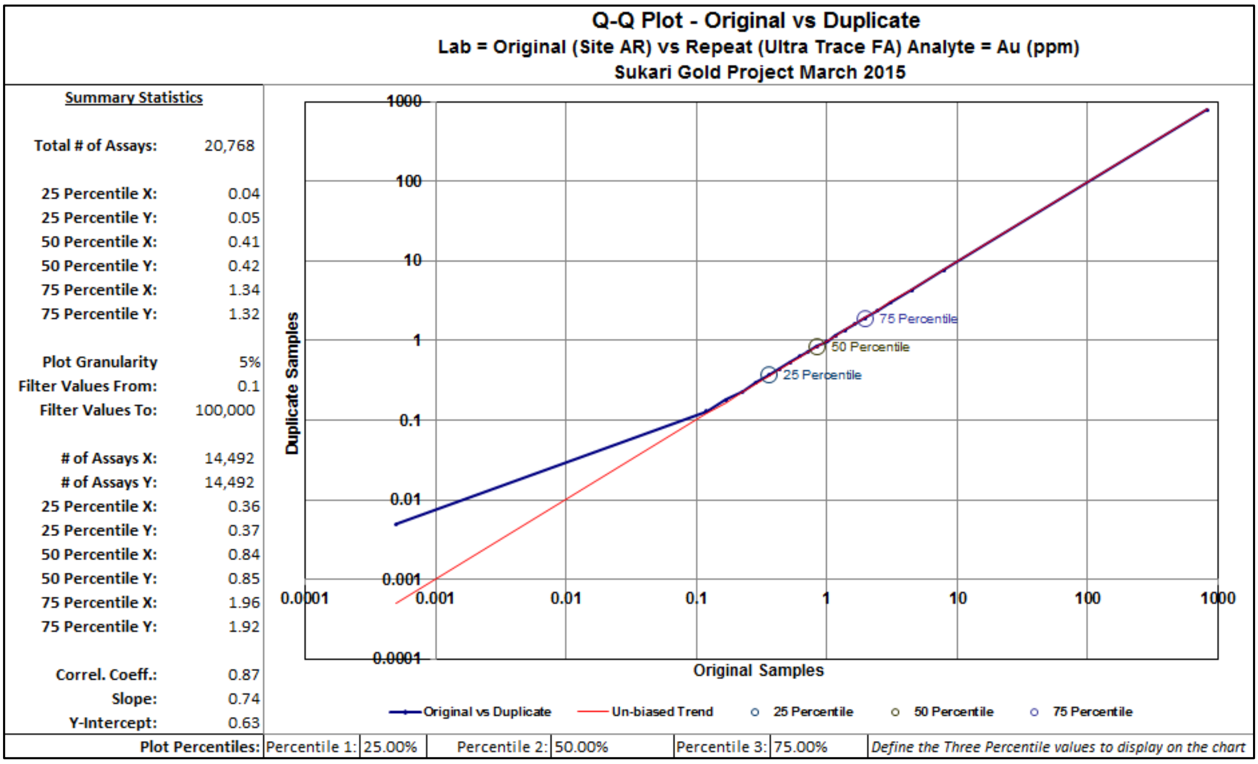


Figure 11.5-14 Umpire Pulp Duplicates QQ Plot – Original (AR) vs Duplicate (FA) – Ultra Trace



11.5.5 Summary

Overall, the QAQC analysis demonstrates that the analytical accuracy and precision for the resource definition drilling is acceptable and appropriate for the purpose of Mineral Resource estimation. Some minor precision errors are evident from the CRM replicate assays as a result of sample mishandling or transcription errors. The duplicate samples show analytical precision in line with acceptable practice.

12 DATA VERIFICATION

12.1 Property Inspection

A site visit was conducted by the QP, Mr Zammit, from 9 to 16 March 2015 to inspect the Sukari Gold Project in order to review the controls on mineralization, the geological interpretation and to review the data collection.

The site visit involved comprehensive data verification, inspections and reviews with site personnel of the following:

- Geology of the project area;
- Exploration model and strategy;
- Confirmation logging of selected diamond core intervals with digital core photos;
- Drilling data;
- Current drilling equipment and drilling conditions;
- QAQC procedures and control data;
- Sample handling and storage facilities on-site;
- Site analytical laboratory;
- Underground mapping and sampling; and
- Discussions regarding future work programs.

12.2 Drilling Database and Data Validation

The drillhole databases are managed on site by Centamin geological staff. Cube was provided with the final databases for the Amun and Ptah underground project areas dated 3 July 2015.

Cube completed validation checks prior to final interpretation and compositing for grade estimation. The validation checks included:

- Graphically check collar location with respect to topography and underground mine workings;
- Graphically check downhole survey;
- Check discrepancies in maximum depths between collar, assay, survey and geology records; and
- Check for overlapping and duplicate assay and geology records.

12.3 Independent Geological Logging

The digital photos for all available diamond drilling were supplied to the QP, Mark Zammit, by Centamin. The photographs for key drillholes were reviewed by the QP, Mark Zammit, to validate the assay results against the Centamin geology logging. The objective was to understand the style and paragenesis of the mineralization and to verify the mineralization boundaries.

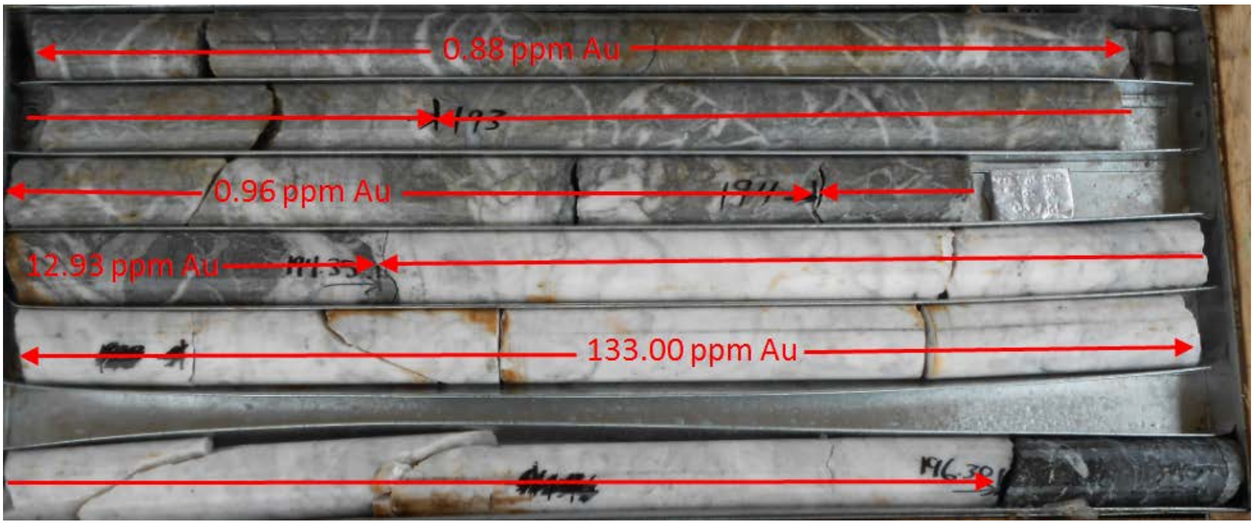
A selection of the mineralized intervals that were reviewed is detailed below in Table 12.3-1.

Table 12.3-1 Mineralized Intervals Verified by Inspection of Digital Photographs

Hole ID	Depth from (m)	Depth to (m)	Length (m)	Mineralized Domain No.
UGRSD0058	240	260	20	Amun – 13
D1339	360	400	40	Amun – 1 & 5
D1351	330	385	55	Amun – 1 & 3
D1345	340	500	60	Amun – 1, 3, 12 & 21
UGRSD0233	70	205	135	Amun – 4, 8, 10 & 12
D724	250	320	70	Ptah – 12 & 16
UGRSD0542	160	170	10	Ptah – 11
UGRSD0510	**170**	**188**	**18**	**Ptah – 11**
UGRSD0512	190	205	15	Ptah – 11
D1440	570	580	10	Ptah – 5
D1438	640	705	65	Ptah – 2 & 3

The independent logging by Cube has verified the Centamin geology logs and the assay tenor of the mineralized intercepts in the database. In addition, the key characteristics of the mineralization being visually identifiable, quartz veined zones with variable degrees of brecciation and alteration and relatively sharp contacts to the host rock was confirmed (Figure 12.3-1).

Figure 12.3-1 Visual Verification of High Grade Intersection – UGRSD0512 at 192 m to 196.5 m downhole



12.4 Qualified Person's Statement

The QP, Mark Zammit, has assessed the veracity of the drilling data for the Sukari Underground Project, with emphasis on the most recent drilling completed in 2014 and 2015. All logging, sampling and data QAQC procedures implemented by Centamin for the 2014 and 2015 drilling were undertaken to an acceptable industry standard. The record keeping and data management was considered adequate for an advanced project.

The QP's, Mark Zammit, site visit to the Project in March 2015 included two inspections of the active underground mine workings. All aspects of the collection, preparation and dispatch of drill samples carried out by Centamin personnel and its contractors were witnessed by representatives from Cube in March 2015.

A review of diamond core photography has confirmed the Centamin geology logs for selected core intervals, and confirmed the characteristics of the mineralization.

The QP, Mark Zammit, has independently reviewed all of the available quality control sample data relating to the resource definition diamond core drilling completed by Centamin at Sukari. Overall, the quality control samples are unbiased and have an acceptable level of precision, indicating that the sample data is of an acceptable standard and appropriate for the purpose of mineral resource estimation and the reporting of exploration results.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The metallurgical behaviour of Sukari ores has been established through various testwork programmes dating back to 2000 performed variously by Lakefield Oretest Pty Ltd, Independent Metallurgical Testing Laboratories Pty Ltd (IMl) and AMMTEC Ltd (AMMTEC), and subsequently the testwork results were combined with production data results. Definitive testwork was conducted, under Ausenco Limited (Ausenco) supervision, at the AMMTEC laboratory in Perth during 2006. Comminution, flotation and cyanidation of flotation concentrate and tailings were the dominant processes tested.

The ore is relatively hard and competent being hosted in porphyry and is suitable for SAG milling. The gold is fine and associated with pyrite which is readily floated at coarse grind size, to yield a sulphide preconcentrate at high gold recovery and ultra-fine grinding renders the gold in concentrates amenable to cyanidation.

Oxidized material from near surface yields acceptable gold recovery at coarse grind sizes, via direct cyanidation.

13.2 Metallurgical Sampling

Classification of ore types for the definitive AMMTEC program was based on the degree of weathering (that correlated with the degree of oxidation of sulphide mineralization), with highly weathered ore referred to as “Oxide ore” (M5) and primary, fresh ore as “Sulphide ore” (M1). This method of classification provided a suitable basis for the investigation and prediction of gold recovery.

The ore is hosted in porphyritic rock and gold tends to occur as fine inclusions in the sulphide minerals – mostly pyrite. Where weathering has occurred, the sulphide minerals have generally been broken down allowing access of cyanide solution to the contained gold and rendering the ore relatively free milling. The sulphur grade decreases from M1 to M5, consistent with the increasing proportion of oxide mineralization. The gold grade also decreases from M1 to M5, which reflects a general trend of lower grade ore closer to surface. Approximately 90% of the ore deposit has been classified as M1 or M2.

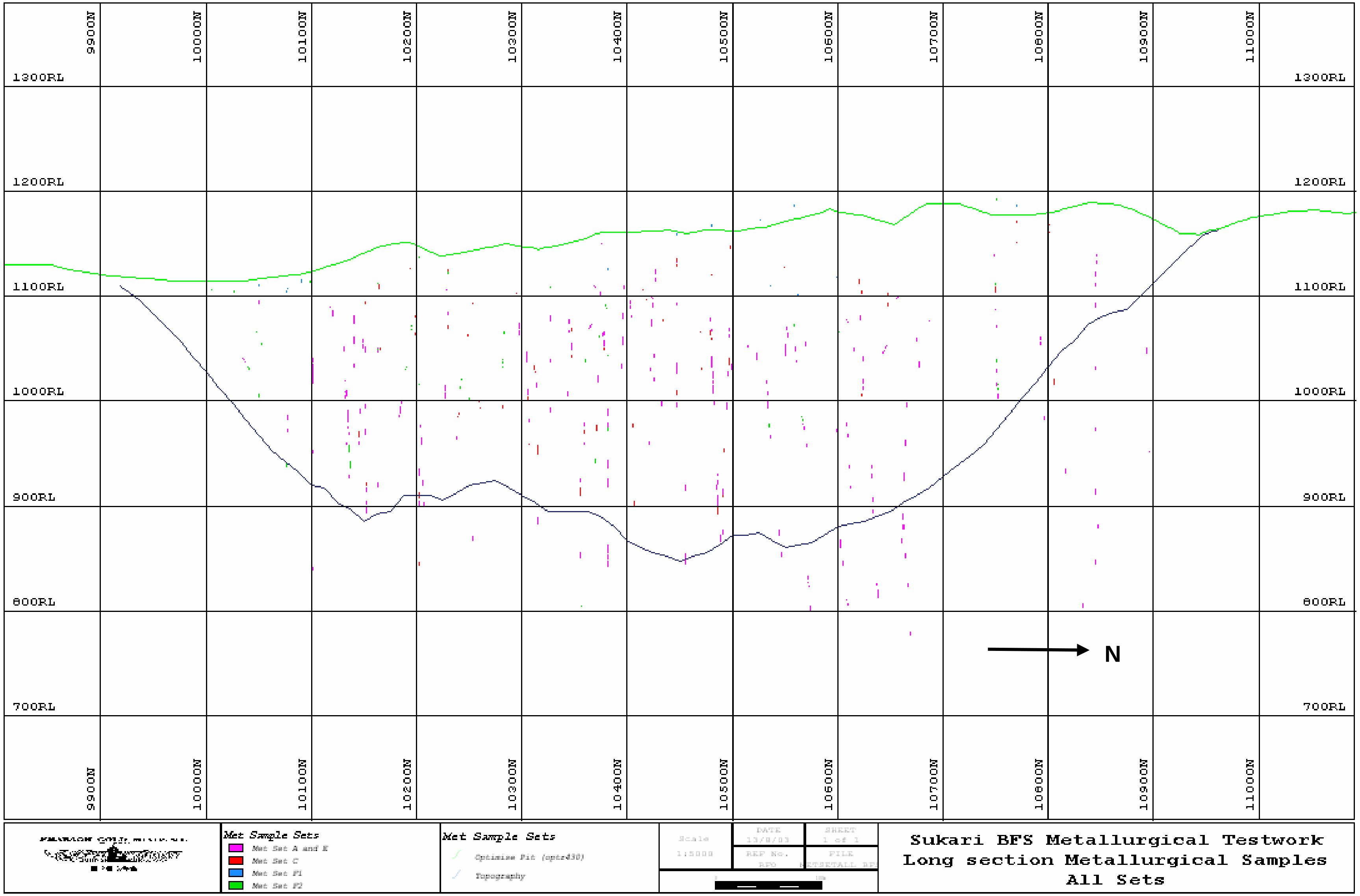
The definition and collection of samples for IML’s metallurgical testwork program was carried out in April and May 2004. Ausenco provided the requirements for the samples and the six sample sets were collected by PGM as summarized in Table 13.2-1.

Table 13.2-1 Metallurgical Testwork Samples

Set	Description	Weight (kg)	Grade (g/t)	Core Metres
A	Flotation and Cyanidation	736.4	1.73	1148.42
B	Comminution	149	N/A	N/A
C1	Weathered	66	1.57	156
C2	Hi Silica	158.5	1.63	224
C3	Brecciated	57	1.69	100
D1	Shore Seawater	50 litres	N/A	N/A
D2	Near Shelf Seawater	50 litres	N/A	N/A

Care was taken to ensure the samples taken were representative and over the full strike length, as is illustrated in Figure 13.2-1 of a long section of the optimized pit as it was then.

Figure 13.2-1 Long Section Displaying the Metallurgical Sample Intervals in relation to the Optimized Pit Shell (Blue) and Topography (Green)



Additional sample was obtained from site for the definitive AMMTEC program:

Weathering Type:

1	M1 – Sulphide Composite	13.5 kg
2	M2 – Mixed Composite	59.2 kg
3	M3 – Mixed Composite	27.1 kg
4	M4 – Mixed Composite	6.0 kg
5	M5 – Oxide Composite	182.7 kg

Variability Samples:

- By sulphur grade
- By gold grade
- By location in orebody

The sample classifications and head assays are summarized in Table 13.2-2.

Table 13.2-2 Mineralization Composite Head Assays

Sample Identification	Description	Laboratory	Gold Au g/t	Silver Ag g/t	Arsenic As ppm	Total Sulphur %S	Sulphide Sulphur %S	Total Carbon %C	Organic Carbon %C	Copper Cu g/t	Mercury Hg g/t	Iron % Fe
Met Set A	Met Composite (Fresh + Slightly Weathered)	IML	1.68	<0.5	988	1.10	1.08	0.79	0.067	10	<20	2.9
B	Comminution Testwork	IML	>0.5	–	–	–	–	–	–	–	–	–
C1	Weathered (Variability)	IML	1.57	–	–	–	–	–	–	–	–	–
C2	Highly Siliceous (Mostly Fresh)	IML	1.70	0.5	968	1.10	–	–	–	–	–	2.9
C3	Brecciated (Fresh & Weathered)	IML	1.69	–	–	–	–	–	–	–	–	–
D1 & 2	Rea Sea Water	IML		–	–	–	–	–	–	–	–	–
E1	Veined Mineralization (Fresh & Weathered)	IML	1.72	–	–	–	–	–	–	–	–	–
E2	Stockwork Mineralization (Fresh & Weathered)	IML	1.71	–	–	–	–	–	–	–	–	–
M1	Fresh, Sulphide	AMMTEC	1.74	<1	1080	0.91	0.85	0.39	0.04	–	2.63	–
M2	Mixed (>75% Sulphide)	AMMTEC	1.46	<1	544	0.62	0.57	0.34	<0.03	–	2.80	–
M3	Mixed (25% - 75% Sulphide)	AMMTEC	1.22	<1	376	0.43	0.35	0.19	<0.03	–	2.70	–
M4	Mixed (<25% Sulphide)	AMMTEC	1.28	<1	260	0.22	0.12	0.04	<0.03	–	2.70	–
M5	Oxidized	AMMTEC	1.13	<1	292	0.08	0.05	0.17	<0.03	–	2.70	–
Comp 1	Lakefield Oretest #8363	Lakefield	1.81	<5	2470	0.97	–	–	–	–	–	–
Comp 1		Lakefield	1.61	<2	–	1.27	–	–	–	–	–	–
Comp 2A		Lakefield	2.43	<2	–	1.56	–	–	–	–	–	–
Comp 2B		Lakefield	2.45	<2	–	1.24	–	–	–	–	–	–
Comp 3	Comminution	Lakefield	–		–	0.73	–	–	–	–	–	–
Comp 3B		Lakefield	2.15	<2	–	1.01	–	–	–	–	–	–

13.3 Metallurgical Testwork

13.3.1 Process Water

Due to the very low rainfall and the lack of reliable replenishment of ground water, water is pumped from the Red Sea, 25 km away, for use in the process plant. Perth sea water was used for the testwork programme after an analysis concluded that there was little difference in metallurgical performance. Perth sea water has lower Total Dissolved Solids (TDS), chloride, and calcium concentration compared to Red Sea water. Buffering tests on the Perth and Red Sea water samples indicate that there is little difference in the buffering behaviour of Perth sea water compared to Red Sea water, as would be expected from the similar magnesium levels. Buffering tends to occur at a pH of 10.3. It was recommended that cyanidation be conducted at a pH of 9.5 to 10 to avoid excessive lime consumption.

The results of comminution testwork are summarized in Table 13.3-1 and indicate that the ore is competent, resistant to breakage by impact and is abrasive. An assessment of the behaviour of the Sukari ore in the comminution circuit from Kori Kollo was undertaken by Dr Steve Morrell of SMCC Pty Ltd. It was concluded that the SAG mill was the throughput limiting item of equipment and that a pebble crusher was required. The ball mills have sufficient capacity to produce a finer grind or a similar grind at higher throughput if required.

Table 13.3-1 Comminution Test Results

Description	UCS MPa	Bond Rod Mill Work Index kWh/t	Bond Ball Mill Work Index kWh/t	Abrasion Index	Drop Weight Test
July 2000	16.2–93.1	–	19.7	0.42	–
October 2000	142–163	–	18.9	–	–
October 2001	64–105	18.1	17.4	0.49	–
Met Set B	–	22.4	17.6	0.683	A=77.26 b=0.49 t_a=0.26
Weathered C1	–	19.9	16.2	0.525	–
Siliceous 1 C2A	–	23.2	18.5	0.541	–
Siliceous 2 C2B	–	23.2	18.7	0.508	–
Brecciated C3	–	22.1	18.2	0.660	–
Barren Host Rock C4	–	23.9	19.3	0.566	–

The 85th percentile Bond ball mill work index of 19.1 kWh/t and 70th percentile Bond rod mill work index of 23.2 kWh/t were used as the basis for calculation of grinding circuit throughput. For oxide ore treatment, the weathered ore ball mill work index of 16.2 kWh/t was used. Note also that the results are highly consistent across the ore types and indicate that the ore body has a low variation in hardness and abrasion characteristics.

Desktop reviews and modelling of the comminution circuit efficiency were completed in 2010 by Ausenco (in conjunction with Dr. Steve Morrell of SMCC Pty Ltd), Orway Mineral Consultants and Senet Projects Ltd to identify the potential improvement in the grinding circuit throughput with the introduction of secondary crushing.

The reviews consistently concluded that the existing SAG circuit is capable of processing 5 Mtpa (625 tph) when provided with a secondary crushed feed (F_{80}) of 50 mm.

Table 13.3-2 Crushing Circuit – Test Results

Description	Circuit ID	Stage 1/2 Circuit Capacity 2nd Crush Stage 3
Throughput	Mt/a	5
Throughput	t/h	625
Crusher/s	#	1
SAG mill/s	#	1
HPGR	#	–
Ball mill/s	#	2
Bond baseline energy	kWh/t	14.5
Circuit efficiency factor	factor	1.22
Ausenco method (efficiency fact)	kWh/t	17.7
Morrell's Wt method	kWh/t	17.4
JKSimMet – method	kWh/t	18.1
Ave. circuit specific energy	kWh/t	17.7
Standard deviation	–	0.35
Required circuit pinion power	MW	11.1

13.3.2 Gravity Separation

Gravity gold recovery using centrifugal concentrators in the laboratory on Met Set A (IML) yielded gold recoveries of 11.2% to 18.6%. A sample of ore containing coarse gold and with a head grade of 20 g/t earlier gave a gravity recovery of 87% for gold and 62% for sulphur with a mass pull of 3.8%. This indicates that some of the gold is free and liberated in very high grade areas, however, the occurrence of high grade ore is rare and will occur at depth.

It was therefore concluded that space should be allowed for retrofit of a gravity section within the grind circuit as the pit develops to depth.

Subsequent flotation testwork on Met Set A (IML) was carried out on recombined gravity tailings material. The calculated overall gravity concentrate plus flotation concentrate gold recovery was 96% indicating that there is no apparent benefit by including the gravity step prior to flotation, compared with flotation only.

In October 2011, a testwork program was conducted by ALS AMMTEC on underground ore composites containing high-grade gravity recoverable gold to assess the amenability of the ore to a gravity recovery step prior to flotation. This testwork program also demonstrated that flotation only recovery was comparable to gravity + flotation recovery, indicating that there would be no apparent benefit with the inclusion of a gravity circuit prior to flotation.

13.3.3 Whole of Ore Cyanidation

A summary of whole of ore cyanidation gold recoveries is given in Table 13.3-3.

It was apparent from early testwork at IML that Sulphide ore types (Met Set A) would require fine grinding to achieve acceptable gold recovery. This led to the route of preconcentration of gold by flotation. However, direct cyanidation of Oxide ore (Weathered C1) yielded acceptable recovery at coarse grind sizes. Optimization work at AMMTEC on M5 Oxide material defined cyanide extraction at 92.9 % Au at the design grind size.

Reagent consumptions were 1.7 kg/t NaCN and 3.3 kg/t lime.

Table 13.3-3 Comparison of CIL and Direct Cyanidation Test Results

Sample	Laboratory	Test	Grind Size P_{80} µm	Calculated Head g/t Au	Gold Extraction %
Met Set A	IML	Direct CN	90	–	73.1
Met Set A	IML	Direct CN	75	–	79.5
Met Set A	IML	Direct CN	53	–	81.5
Met Set A	IML	CIL	75	1.67	76.6
Met Set A	IML	CIL	63	1.50	74.0
Met Set A	IML	CIL	53	1.69	75.5
Met Set A	AMMTEC	CIL	63	2.08	88.4
Met Set A	AMMTEC	Direct CN	63	2.02	87.3
Weathered C1	IML	CIL	150	1.19	85.7
Weathered C1	IML	CIL	75	1.18	83.9
Weathered C1	AMMTEC	Heap Leach	3350	–	88.6
M3	AMMTEC	Direct CN	106	–	85.6
M3	AMMTEC	Direct CN	150	–	86.5
M3	AMMTEC	Direct CN	250	–	84.7
M3	AMMTEC	Heap Leach	3350	–	66.3
M5 Oxide	AMMTEC	Direct CN	106	–	93.6
M5 Oxide	AMMTEC	Direct CN	150	–	92.9
M5 Oxide	AMMTEC	Direct CN	250	–	90.7
M5 Oxide	AMMTEC	Heap Leach	3350	–	83.9

13.3.4 Flotation

Flotation testwork results consistently showed a high gold recovery to flotation concentrate, at low concentrate mass pull, and at natural pH. Table 13.3-4 summarizes the test details and results. Gold recovery was consistent with mineralization type, with the highest recovery achieved for samples of type M1 (fresh sulphide) and the lowest recovery for type M5 (oxidized).

Table 13.3-4 Flotation Test Summary

Sample	Grind Size P_{80} µm	Head Grade g/t Au	Mass Pull %	Gold Recovery to Concentrate %	Sulphur Recovery %
IML Met Set A T5	212	1.68	7.2	96.8	–
IML Met Set A T2	175	1.68	7.1	96.3	–
IML Met Set A T3	150	1.68	7.8	97.4	–
IML Met Set A T4	125	1.68	7.5	97.8	–
IML C1A Highly Weathered	–	1.55	5.3	72.5	–
AMMTEC M1 Fresh	150	1.95	5.9	96.6	97.9
AMMTEC M2	150	1.16	4.1	80.6	93.4
AMMTEC M1+M2 50:50	150	1.39	4.2	76.6	95.5
AMMTEC M3	150	0.89	3.7	74.6	92.4
AMMTEC M4	150	1.06	3.1	66.5	68.5
AMMTEC M5 Oxide	150	0.96	3.6	65.4	34.9

The average mass pull for the tests was 5.0%. Ausenco determined that the 90^{th} percentile value for the sulphur head grade based on variability samples was 1.8% sulphur. The plant's concentrate circuit was therefore designed for 8% mass pull, to allow for short-term increases in the sulphur feed grade to the plant and/or subsequent plant throughput increases.

Ausenco conducted an economic evaluation of the anticipated revenues and selected an optimum grind of P_{80} = 175 µm for flotation. A grind size P_{80} of 150 µm was subsequently used for flotation testwork at AMMTEC and as the design basis as it was a standard-size fraction.

A test was carried out to compare a combined gravity then flotation flowsheet, with flotation only. The calculated overall gravity concentrate plus flotation concentrate gold recovery was 96% indicating that there was no apparent benefit by including the gravity step prior to flotation, compared with a flotation only process route.

The flotation testwork indicated that there was no benefit in addition of copper sulphate as an activator for the sulphides, however, given that residual cyanide may contaminate the process water, provision was made to retrofit a copper sulphate mixing, storage and distribution system. The optimum reagent additions were found to be 75 g/t potassium amyl xanthate (collector), 50 g/t A404 (collector/promoter particularly for gold), 20 g/t Interfroth IF20 (or MIBC) frother.

Sulphur recovery for all flotation optimization tests was consistently high at >96%. However, a low gold recovery of 88.8% was obtained from flotation of the lowest grade (0.5%) sulphur variability sample. Arsenic recovery ranged from 95% to 96%, while iron recovery ranged from 39% to 48%. These values support the recovery of gold associated predominantly with pyrite into flotation concentrate, together with arsenopyrite.

A lower gold recovery of 91.7% was obtained from flotation of the lowest-grade (0.75 g Au/t) gold sample. Other low-grade gold samples had average gold recoveries averaging 95.1%.

13.3.5 Cyanidation – Flotation Tailing

The gold assay of flotation tailings for primary Sulphide samples averaged less than 0.1g/t Au. Consequently, for Sulphide ore (M1), the flotation tail can be discarded directly to final tailing without incurring the costs associated with cyanidation. However, flotation tests on the Weathered (M2, M3 & M4) composite samples gave lower flotation recoveries with a flotation tailing grade of approximately 0.4 g Au/t, which was considered sufficiently high to consider tailings cyanidation. The Oxidized (M5) composite gave superior gold recovery through direct cyanidation, rather than flotation with separate cyanidation of float concentrates and float tail.

AMMTEC performed direct cyanidation tests on the flotation tailings from the mineralization composite flotation tests. The flotation tailings gold extraction results, at 12 hours and 24 hours, are listed in Table 13.3-5.

Table 13.3-5 Flotation Tails Cyanidation Results

Test	Sample	Extraction at 12 hours %	Extraction at 24 hours %
RG7494	M1	67.0	67.0
RG7490	M2	73.0	77.4
RG7491	M3	72.5	72.5
RG7492	M4	61.4	64.2
RG7493	M5	70.2	73.4

The M2, M4 and M5 samples showed a slight increase in gold extraction between 12 and 24 hours. However, based on this insignificant benefit, a tails cyanidation residence time of 12 hours was selected by Ausenco.

The average lime consumption for the mineralization composite and variability flotation tailings cyanidation tests was 0.85 kg/t of hydrated lime with 84% $Ca(OH)_2$ (or 64% CaO), equivalent to 0.72 kg/t for quicklime.

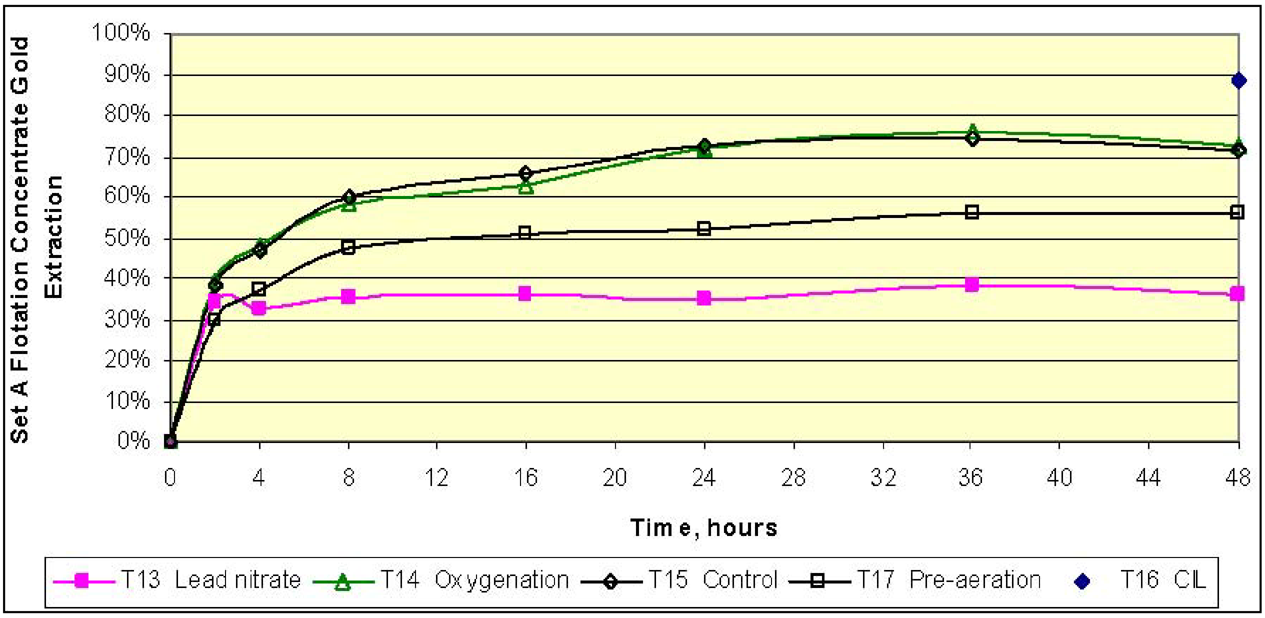
The average cyanide consumption in the flotation tailings cyanidation tests was 0.37 kg/t. A residual cyanide concentration in flotation tailings leach tails was chosen as 150 ppm to calculate the total cyanide addition. Overall cyanide addition was then calculated from these two values as 0.60 kg NaCN/t.

13.3.6 Cyanidation – Flotation Concentrate

Most of the sulphur in the flotation concentrate is present as sulphide. Gold, silver, iron, copper, arsenic, and sulphur grades have been concentrated by flotation.

Results of the initial optimization for flotation concentrate leaching are shown in Figure 13.3-1 with the CIL test having the highest gold extraction of 88.7% at 48 hours leach time.

Figure 13.3-1 Concentrate Cyanidation Optimization



A slight reduction in gold extraction from 36 to 48 hours in the non-CIL tests was attributed to mild "pregnant-robbing" (the readsorption of dissolved gold and silver onto a particular component of the leached solids) by Ausenco. This factor is also seen in the ultimate extraction of + 90% Au only being achieved in the CIL tests (c/f T15 versus T16 above). In practice, any tendency to Pregnant Robbing will be negated by the use of CIL leaching.

The results of grind optimization tests using CIL showed a strong trend toward higher gold extraction at finer grinds.

Variability sample flotation concentrate CIL gold extraction ranged from 85.5% to 99.3% with an average of 95.1% for all samples. Gold extraction was consistent with mineralization type, with the highest gold extraction found on M5 (oxide) samples and the lowest gold extraction found on the M1 (sulphide) samples.

The average lime consumption for the IML and AMMTEC concentrate cyanidation tests was 4.77 kg/(t concentrate), based on hydrated lime with 84% $Ca(OH)_2$ content, as used in the laboratory. This is equivalent to 4.04 kg/(t concentrate), based on quicklime having 75% contained CaO and 0.32 kg/(t feed) based on a design concentrate mass pull of 8%.

The average cyanide consumption for the IML and AMMTEC concentrate cyanidation tests was 2.79 kg/(t concentrate). A residual cyanide concentration in concentrate leach tails was chosen as 300 ppm to calculate the cyanide addition rate. The overall cyanide addition was then calculated from these two values as 3.35kg NaCN/(t concentrate) or 0.27 kg/(t feed) based on a design concentrate mass pull of 8%.

13.4 Testwork Summary

Definitive testwork by AMMTEC on the five ore classifications provided a basis for prediction of gold recovery based on flotation response and resulted in allowance for three processing routes:

- Circuit 1 - Whole ore direct cyanidation of “oxide ore”. (M5).
- Circuit 2 – Flotation of “mixed ore” with separate cyanidation of reground concentrate and flotation tail (M2, M3 and M4).
- Circuit 3 – Flotation of “sulphide ore” with cyanidation of reground concentrate and discarding flotation tail. (M1).

Gold recoveries for the three process circuits were derived from the metallurgical testwork, with adjustment for plant losses, and are shown in Table 13.4-1.

Table 13.4-1 Recoveries by Ore Type and Processing Circuit

Ore Type	Circuit 3 Float + Conc CIL	Circuit 2 Float + Conc CIL + Float Tails CIL	Circuit 1 Direct CIL
M1 - Sulphide	**89.7%**	92.2%	70%
M2, M3, M4 - Mixed	69.4%	**87.4%**	81.6%
M5 - Oxide	62.8%	88.8%	**90.8%**

14 MINERAL RESOURCE ESTIMATES

14.1 Open-pit Mineral Resource

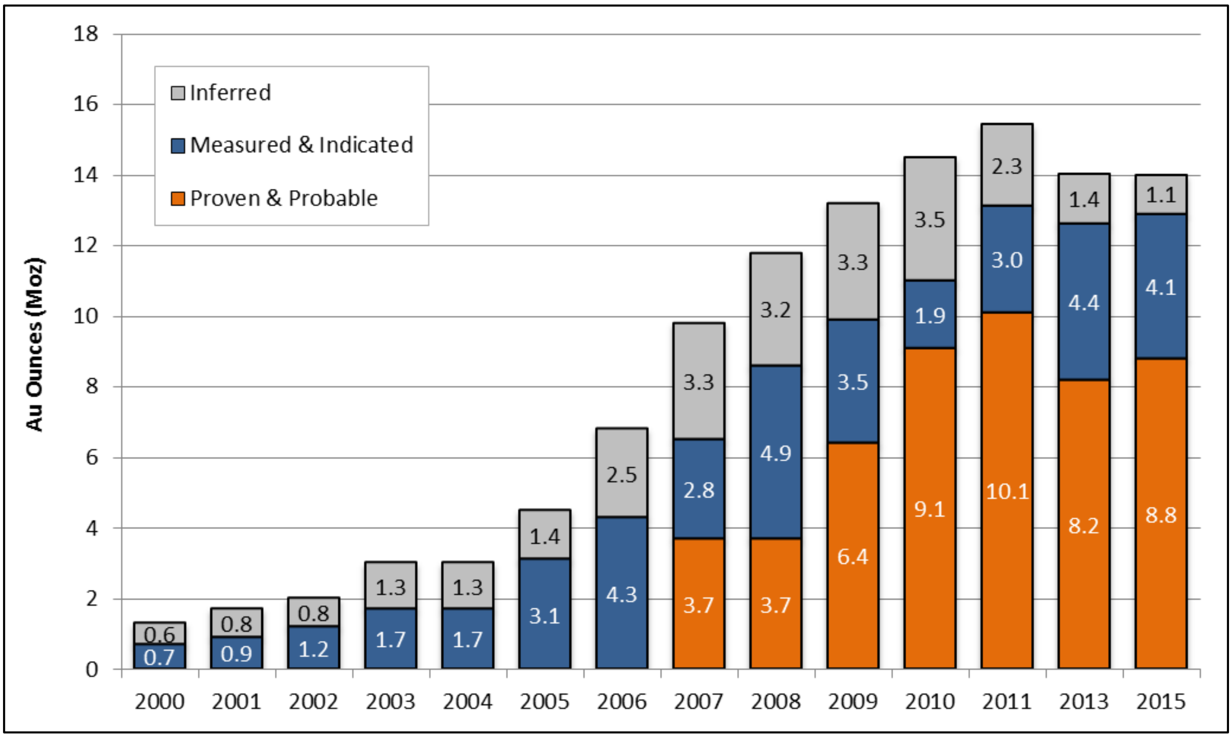
MPR Geological Consultants Pty Ltd (MPR) was retained by Centamin to update the Sukari open-pit resource model based on additional drilling as at the effective date of 30 June 2015, with immaterial timing differences as noted in Section 2.2. Estimates were prepared with reference to the Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards (2005) and CIM Best Practice Guidelines (2003) for preparing mineral resources and mineral reserves.

Under NI 43-101 reporting requirements and CIM standards and guidelines, Nicolas James Johnson, a member of the Australian Institute of Geoscientists, with more than five years' experience in the use of geostatistics for estimation of mineral resources in gold deposits using recoverable-MIK method, is the QP for the purposes of this work. Mr Johnson has visited the mine on several occasions, most recently in July 2013.

The recoverable resource models were built using GS3©, the MIK software developed by FSSI Consultants (Aust.) Pty Ltd, and are suitable for use in open-pit optimization studies.

Since the first resource model was built by Hellman and Schofield Pty Ltd and reported in 2000, the resource model has been updated on an annual or bi-annual basis. The consequent resource results in terms on contained gold ounces are shown in Figure 14.1-1 (with the component of the mineral resources that have converted to total mineral reserves, open-pit and underground, shown in orange).

Figure 14.1-1 Growth in Contained Ounces at each Resource Estimate Update



Notes to figure :

- 2000 – 2010: Mineral Resource COG = 0.5 g/t Au
- 2011 – 2015: Mineral Resource COG = 0.3 g/t Au, with depletion from mining activities.
- 2007 – 2010: Total Mineral Reserve includes open-pit Mineral Reserve only
- 2011 – 2015 Total Mineral Reserve includes open-pit and underground Mineral Reserves

14.1.1 Introduction

Stockwork gold deposits such as Sukari do not readily lend themselves to resource estimation by polygonal methods, nor by interpolation of block grades by methods such as ordinary kriging or inverse distance weighting into polygons (or wireframes) interpreted around drillhole assays. Experience has demonstrated that such methods normally under-estimate recoverable ore tonnages and overestimate recoverable grades.

The QP, Mr Nicolas Johnson, has for some years used a method of Multiple Indicator Kriging (MIK) to estimate recoverable resources in gold and polymetallic deposits, with positive results. The method is particularly suited to estimation of resources in deposits such as Sukari.

It must be emphasized that recovery of resources modelled in the approach taken for Sukari relies upon the use of an optimum method of ore selection during mining. Use of a polygonal method to select ore and waste in mining at Sukari will almost certainly result in a shortfall of ore tonnes against the MIK model. MPR recommends the use of the MP3© method for ore selection in mining (Schofield & Rolley, 1997).

14.1.2 Indicator Kriging for Recoverable Resource Estimation

The MIK method was developed in the early 1980s with a view toward addressing some of the problems associated with estimation of resources in mineral deposits. These problems arise where sample grades show the property of extreme variation and consequently where estimates of grade show extreme sensitivity to a small number of very high grades. These characteristics are typical of many gold deposits, where the coefficient of variation in samples normally exceeds 2.0. MIK is one of a number of methods which can be used to provide better estimates than the more traditional methods such as ordinary kriging and inverse distance weighting.

It is fundamental to the estimation of resources that the estimation error is inversely related to the size of the volume being estimated. To take the extreme case, the estimate of the average grade of a deposit generated from a weighted average grade of the entire sample dataset is much more reliable than the estimate of the average grade of a small block of material within the deposit generated from a local neighbourhood of data.

Another fundamental notion relevant to the optimization of resources to develop an open-pit mine and schedule is that the optimization algorithm does not require the resource be defined on extremely small blocks relative to data spacing. Small blocks cannot provide the basis for reliable estimates of recoverable resources.

The basic unit of an MIK block model is a panel that normally has the dimensions of the average drillhole spacing in the horizontal plane. At Sukari, the average drillhole spacing is 20m in the grid east direction and 25 m in the grid north direction. The panel should be large enough to contain a reasonable number of blocks, or Selective Mining Units (SMUs; about 15). The SMU is the smallest volume of rock that can be mined separately as ore or waste and is usually defined by a minimum mining width. At Sukari, the dimensions of this block are assumed to be in the order of 5 mE x 8 mN x 10 mRL.

The goal of MIK is to estimate the tonnage and grade of ore that would be recovered from each panel if the panel were mined using the SMU as the minimum selection criteria to distinguish between ore and waste. To achieve this goal, the following steps are performed:-

1. Estimate the proportion of each domain within each panel. This estimation can be achieved by kriging of indicators of domain classifications of sample data points or by passing the model panels through wireframe interpretations of domains and calculating domain proportions directly from the wireframes. Wireframes have been used for the assigning domain proportions into panels for the Sukari resource model.

2. Estimate the histogram of grades of sample-sized units within each domain within each panel using MIK. MIK actually estimates the probability of the grade within each panel being less than a series of indicator threshold grades. These probabilities are interpreted as panel proportions.
3. For each domain, and for each panel that receives an estimated grade greater than 0.0g/t Au, implement a block support correction (variance adjustment) on the estimated histogram of sample grades in order to achieve a histogram of grades for SMU-sized blocks. This step incorporates an explicit adjustment for Information Effect.
4. Calculate the proportion of each panel estimated to exceed a set of selected cut-off grades, and the grades of those proportions.
5. Apply to each panel, or portion of a panel below surface, a bulk density to achieve estimates of recoverable tonnages and grades for each panel.

Apart from considerations of resource confidence classification (Section 17.1.11), Step 5 completes construction of the resource model. The estimates of recoverable resources for each panel may be combined to provide an estimate of global recoverable resources for the deposit.

14.1.3 The Resource Dataset

The resource dataset has been addressed in Sections 10 and 11 and has an effective date of 30 June 2015, with immaterial timing differences as noted in Section 2.3. The primarily one metre sample assays were composited to 2 m downhole lengths with residual intervals less than 2.0 m length being discarded. After trimming to the resource study area i.e. discarding sterilization drilling and other peripheral drillholes not relevant to the current estimate, the final resource dataset contained 252,449 composites (including surface rock chip samples).

14.1.4 Geological Interpretation and Domaining of Data

Throughout the area drilled to date, the limits of the main Sukari porphyry body effectively delineate the limits of gold mineralization. Samples in footwall and hangingwall rocks rarely assay greater than detection limit (0.02 g/t Au) whereas most samples within the porphyry report some detectable level of gold. An exception is gold mineralization in porphyry dykes to the east of the main porphyry body, intersected by drilling on sections between 10,275 mN to 10,650 mN and 10,900 mN to 11,025 mN. Drill coverage is insufficient to define the trend of these subsidiary porphyries but surface mapping indicates them to be discrete dykes, up to 20 metres wide, striking grid north-south.

For the purpose of resource estimation, eight domains were interpreted:

- **Domain 1:** A footwall waste domain, west of the main porphyry body.
- **Domain 2*:*** Oxide zone of the Southern Sukari porphyry, combined with the sulphide zone (Domain 3) hosts the bulk of mineralization defined by drilling. The western and eastern domain limits of the main porphyry were normally digitised at logged geological contacts however, where mineralization extends for short distances (rarely more than 2 m to 3 m) into footwall and hangingwall rocks, domain limits were expanded to include this mineralization. The northern limit to the southern porphyry domains are defined by an approximately 50 degree dipping plane extending north from 10,950 mN.
- **Domain 3:** Sulphide zone of the Southern Sukari porphyry.
- **Domain 4:** Oxide zone of the Northern Sukari porphyry, lower grade, domain in the Sukari porphyry, north of section 10,950 mN. The justification for separating the main porphyry body into northern and southern modelling domains is based on a combination of observations made when viewing sections and plans through the resource data and differences seen in the univariate statistics (Section 17.1.6) between the two domains.
- **Domain 5:** Sulphide zone of the Northern Sukari porphyry.

- **Domain 6:** A domain encompassing mineralization interpreted to relate to the Horus Zone.
- **Domain 7:** A domain encompassing mineralized intercepts in subsidiary porphyry dykes positioned in the hanging wall of the main porphyry body.
- **Domain 8:** Contains all remaining resource not contained in Domains 1 to 7.

Interpretation of domain boundaries utilised surface mapping and drillhole information. Domain boundaries were digitised on cross-sections, snapped to drillhole traces where appropriate, then wireframed into three-dimensional solids. Views of domains are shown in Figure 14.1-2 to Figure 14.1-7.

Figure 14.1-2 Domain 1, Footwall Waste

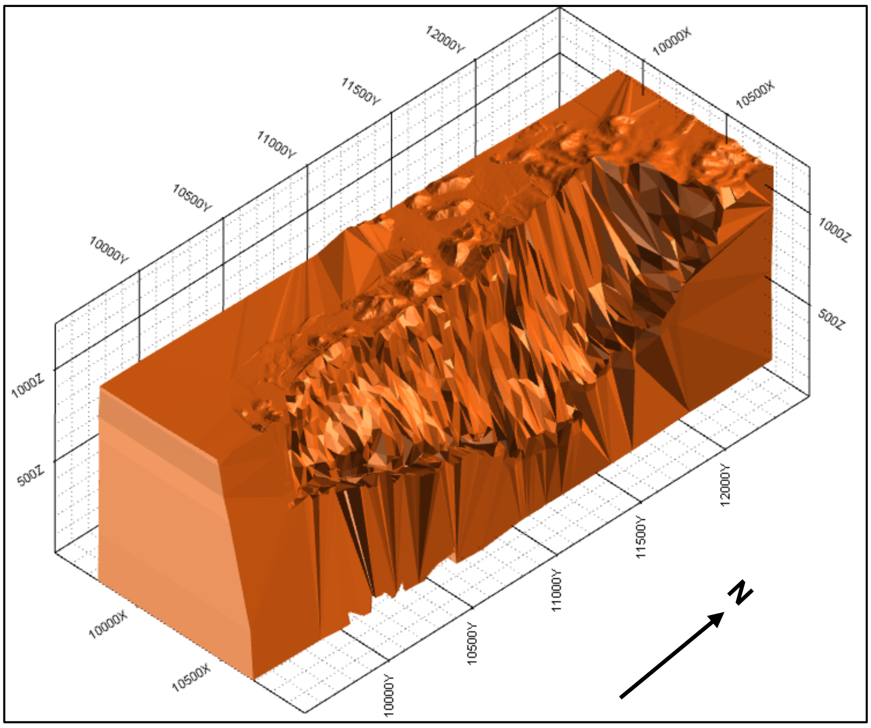


Figure 14.1-3 Domain 2, Oxide Zone Sukari Porphyry South

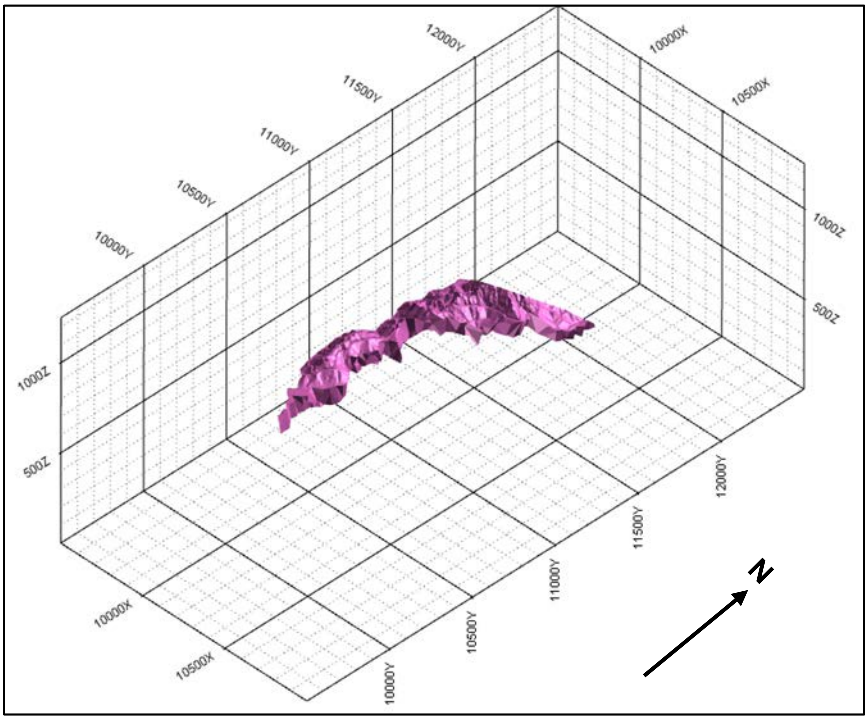


Figure 14.1-4 Domain 3, Sulphide Zone Sukari Porphyry South

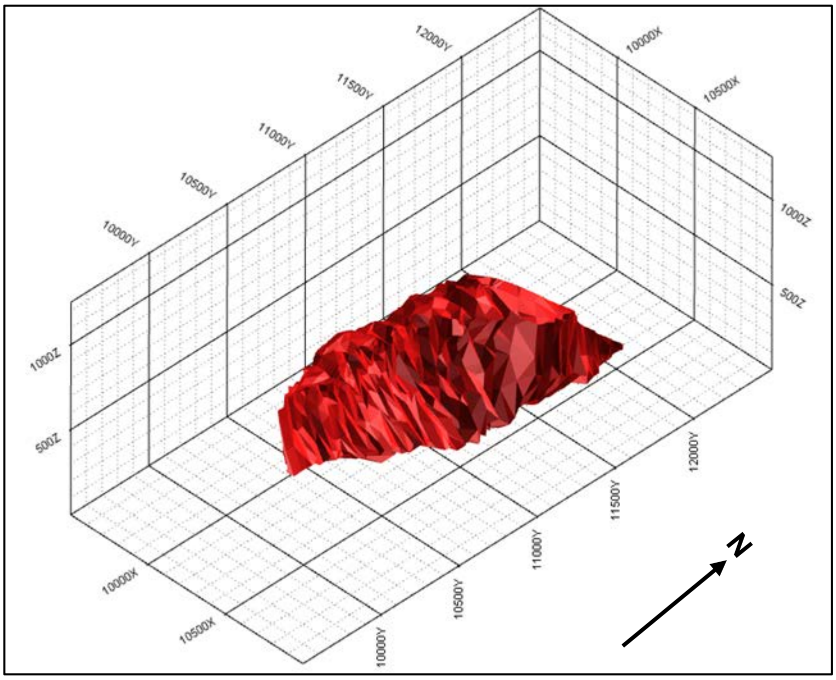


Figure 14.1-5 Domain 4, Oxide Zone Sukari Porphyry North

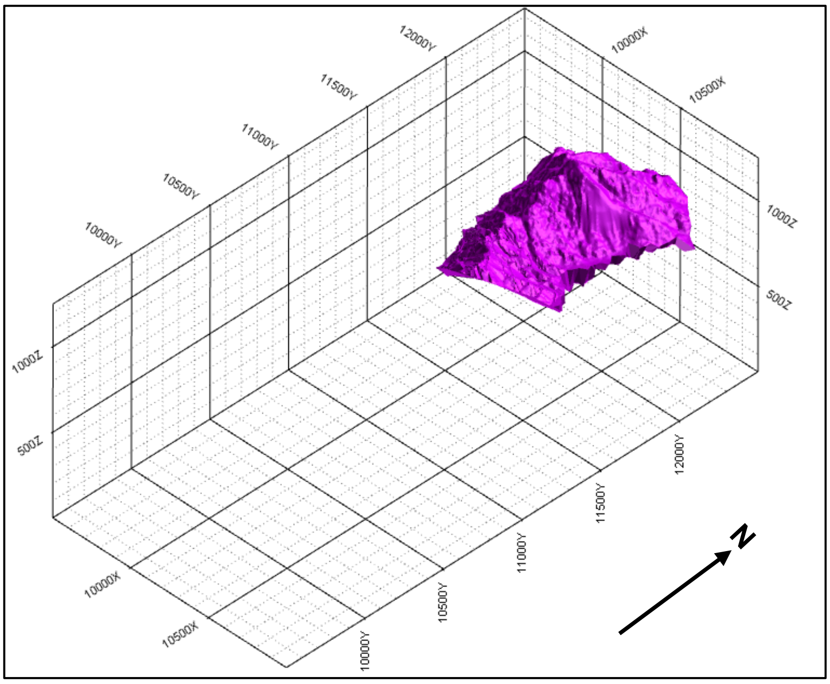


Figure 14.1-6 Domain 5, Sulphide Zone Sukari Porphyry North

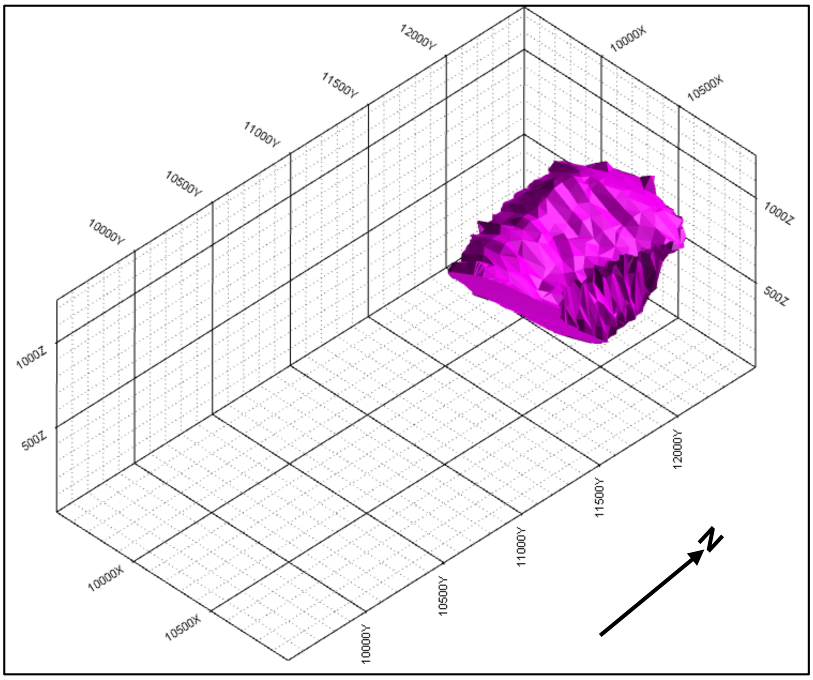


Figure 14.1-7 Domain 6, Horus Zone

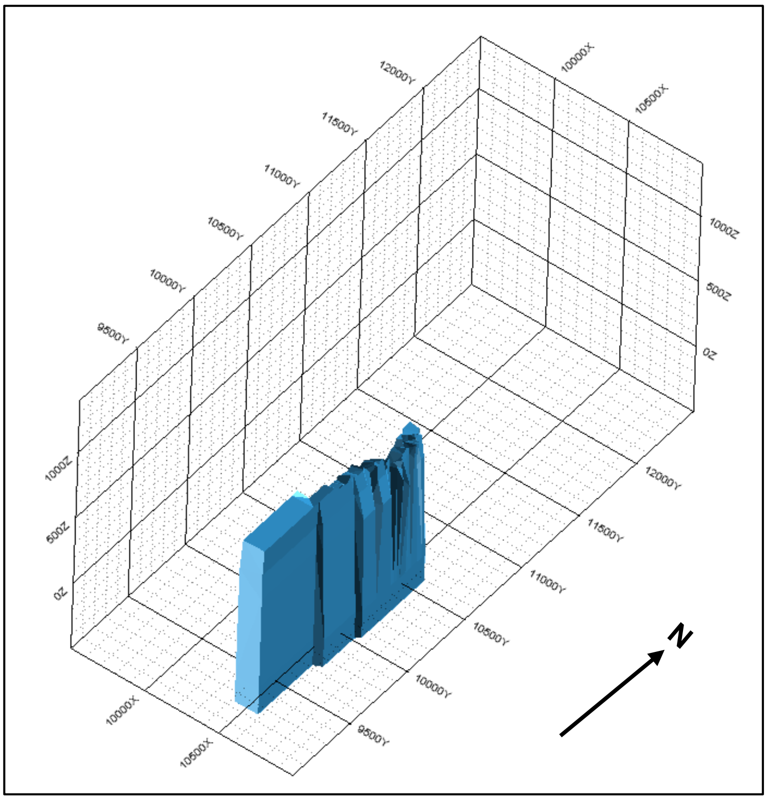
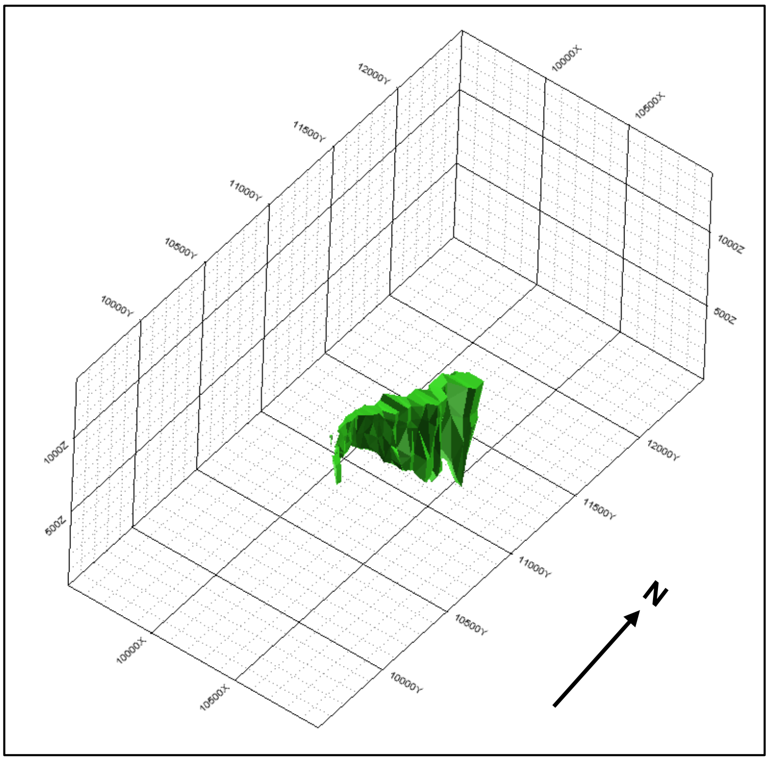


Figure 14.1-8 Domain 7, Hangingwall Porphyry Mineralized Zones



Drillhole composites and surface rock chip samples were flagged by domain code and combined into one file of de-surveyed located samples.

A panel template model was established, proportions of panels within each domain being calculated by passing the model through the domain wireframes.

14.1.5 Oxidation Interpretation

14.1.5.1 Oxidation

Drill sections at 50 m intervals, extending from 9,900 mN through to 11,200 mN, and for the most part at 50 m intervals to 12,200 mN, have been logged using the oxidation criteria shown in Table 14.1-1. The sectional interpretations developed from the oxidation logging produce four transitions of M1–M2, M2–M3, M3–M4 and M4–M5 on sections south of 11,200 mN, while north of this section, due to the majority of drilling being RC, only transitions of M1-M2 and M4–M5 were able to be interpreted. The sectional interpretations were used to develop DTMs of the transitions.

The DTMs have been used to provide the basis for partitioning the final resources into the oxidation types (M1 through to M5). Note however that only the M1 surface has been used in the grade modelling process and this DTM has been modified slightly in the current study due to additional information provided by the mine grade control drilling. The close-spaced mine drilling has shown that the depth of 100% sulphide mineralization is commonly encountered lower in the mine than was interpreted from the broad spaced exploration drilling. To account for this in the current resource model the sulphide DTM has been moved down 15 metres.

The sulphide DTM used to identify top of sulphide has been used to create the wireframes defining the domains representing the oxide and sulphide primary domains of the main Sukari porphyry and assigning the sub-domain coding (oxide or sulphide) of resource composites in the remaining model domains.

Table 14.1-1 Sukari Oxidation Logging Codes

M1	Fresh rock, comprises 100% sulphide mineralization
M2	Mixed sulphide and oxide, comprises >75% sulphide mineralization
M3	Mixed sulphide and oxide, comprises >25% to <75% sulphide mineralization
M4	Mixed sulphide and oxide, comprises <25% sulphide mineralization, >75% secondary hematite
M5	Fully oxidized, comprises 100% oxide (secondary hematite) mineralization

14.1.6 Exploratory Data Analysis

Table 14.1-2 shows univariate statistics of gold for composites coded as Domains 2 through 5, the main Sukari porphyry mineralization. The summary univariate statistics shown are calculated from the entire dataset. Typical of many gold deposits, all populations of gold grades show strong positive skewness with coefficient of variation (CV) ranging from 2.97 to 14.11.

The CVs are very high for the southern porphyry (Domains 2 and 3) indicating that it will be difficult to maintain a high degree of selectivity in mining at Sukari. The very high CV's shown for Domain 2 and 3, being 6.992 and 11.700, respectively, are being affected by a few very high gold grades. Excluding the two highest composite grades in Domain 2 (276.44 and 95.92 g/t Au) reduces the CV to 2.995. The recalculated CV for Domain 3 with the eight highest grades excluded reduces to 5.300 (effectively an upper grade cut of 300 g/t Au). Given the effect of these rare individual grades on the domain statistics, particularly the mean grade of the highest indicator bin, MPR is of the opinion that it is appropriate to exclude these sample grades when calculating the indicator statistics for use in the MIK model.

The CV of 3.095 and 4.075, Domain 4 and 5, respectively, for gold in the north porphyry domains are lower, albeit still extreme, compared to the southern porphyry domains. The difference in the CVs, together with the differences seen in the mean grades and the presence of a significant population of high composite grades in Domain 2 and 3, support the separation of the main porphyry into the north and south modelling regions.

Table 14.1-2 Univariate statistics of Gold in composites Domain 2, 3, 4, and 5

Statistic	Domain 2	Domain 3	Domain 4	Domain 5
No. Data	18,043	74,011	25,158	59,516
Mean (Au g/t)	0.396	1.024	0.265	0.211
Variance	5.992	143.570	0.672	0.741
CV	6.186	11.700	3.095	4.075
Minimum (Au g/t)	0.000	0.000	0.000	0.001
Q1 (Au g/t)	0.018	0.025	0.010	0.007
Median (Au g/t)	0.081	0.220	0.040	0.036
Q3 (Au g/t)	0.325	0.792	0.211	0.169
Maximum (Au g/t)	276.440	2,749.500	44.166	107.860

The histogram of gold in samples in Domain 6, the domain capturing the interpretable mineralized intercepts in Horus Zone, is shown in Table 14.1-3. This domain only exists at depth, beneath Domain 3, so only the primary (sulphide) sub-domain is modelled for this domain. The mean grade and CV for this domain are similar to those seen for the north Sukari porphyry sulphide domain (Domain 5). The recalculated CV for Domain 6 with the single highest grade excluded reduces to 2.992. Given the effect of the highest grade on the domain statistics, particularly the mean grade of the highest indicator bin, MPR is of the opinion that it is appropriate to exclude this sample grade when calculating the indicator statistics for use in the MIK model.

Table 14.1-3 Univariate statistics of Gold in composites Domain 6

Statistic	Domain 6
No. Data	3,154
Mean (Au g/t)	0.284
Variance	1.543
CV	4.379
Minimum (Au g/t)	0.000
Q1 (Au g/t)	0.004
Median (Au g/t)	0.040
Q3 (Au g/t)	0.255
Maximum (Au g/t)	53.750

The histogram of gold in samples in Domain 7, the domain capturing the interpretable mineralized intercepts in subsidiary porphyry dykes in the hangingwall, is shown in Table 14.1-4. Table 14.1-5 shows the histograms of gold in samples for the, essentially barren, footwall and hangingwall domains.

Table 14.1-4 Univariate Statistics of Gold in Composites Domain 7

Statistic	Oxide	Sulphide
No. Data	2,320	7,053
Mean	0.143	0.204
Variance	0.162	0.532
CV	2.821	3.577
Minimum	0.000	0.001
Q 1	0.002	0.002
Median	0.010	0.005
Q3	0.062	0.055
Maximum	6.120	21.896

Table 14.1-5 Univariate statistics of Gold in composites Domain 1 and 8

Statistic	Domain 1, Oxide	Domain 1, Sulphide	Domain 8, Oxide	Domain 8, Sulphide
No. Data	1,633	14,365	14,018	33,178
Mean	0.032	0.044	0.022	0.023
Variance	0.095	1.241	0.024	0.942
CV	9.523	25.271	7.156	42.588
Minimum	0.001	0.000	0.000	0.001
Q 1	0.002	0.001	0.001	0.002
Median	0.010	0.002	0.002	0.003
Q3	0.015	0.005	0.009	0.003
Maximum	12.154	102.886	11.520	120.514

Conditional statistics calculated for data within each domain are listed in Table 14.1-7 to Table 14.1-12 shown later in this section. In Domain 1, the footwall position to the main porphyry, 99% of samples grade less than 0.033 g/t Au and 0.014 g/t Au, oxide and sulphide, respectively. Clearly this domain will make very little, if any, contribution to potentially economic gold resources.

14.1.7 Spatial Continuity Analysis

14.1.7.1 Measures of Spatial Continuity

Most resource estimation methods use a measure of spatial continuity to estimate the grade of blocks in a resource model. In some methods, the measure is implicit; for example a polygonal method assumes that the grade is perfectly continuous from the sample to its surrounding polygon boundary. Geostatistical methods like Ordinary Kriging and Indicator Kriging are among those methods for which the continuity measure is explicit and is customised to the dataset being studied. This measure in its many forms is usually called the variogram.

Geostatistics provides several measures for describing spatial continuity: the variogram, the covariance, the correlogram and many others. All are valid descriptions but not all provide a basis for constructing kriging and simulation models of mineralization. Whatever the method of description used, it is common to use the term variogram in a generic sense to describe contour plots and directional plots of spatial continuity measures.

Throughout the present work, the maps and directional variograms used are all based on the correlogram measure. Directional correlograms are displayed inverted so as to resemble familiar variogram plots. The use of the correlogram as a robust and reliable measure of spatial continuity is proposed by Srivastava & Parker (1988) and Isaaks & Srivastava (1989). The correlogram measure has the advantages of being standardised to a sill of 1.0 and being robust with respect to clustering in the sample data. Models of the sample correlogram can be used directly in Ordinary Kriging and Indicator Kriging.

The various parameters of the variogram model, such as the nugget effect and ranges in different directions, describe properties of the statistical continuity of metal grades. For example, a variogram with high nugget may indicate that there is a high level of error in the sample grades being used to construct the variograms or that there is a high degree of variability in the grade over very short distances in the mineralization. A different range in one direction compared to another is likely to be indicating that grade is more continuous in one direction than another.

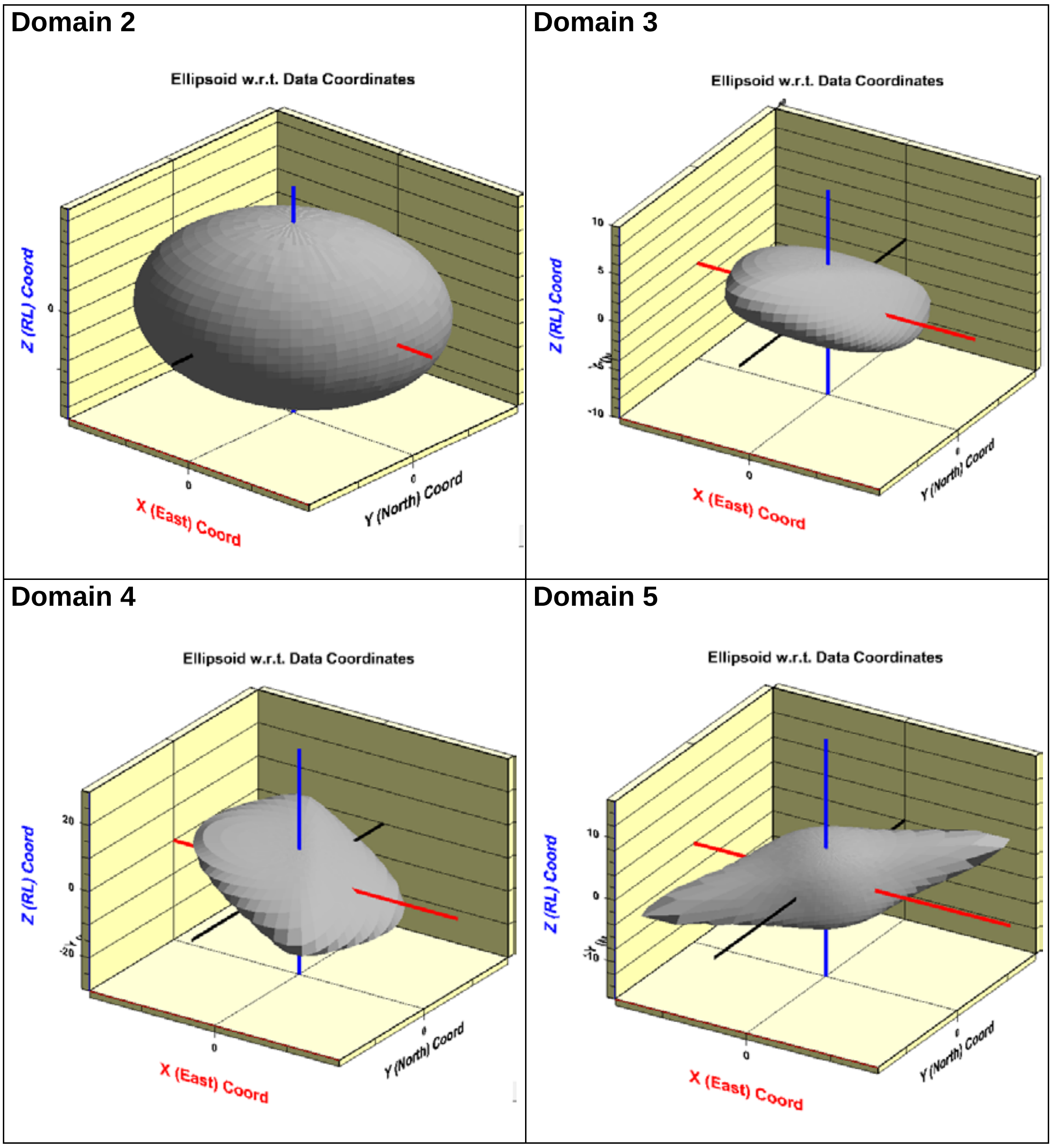
14.1.7.2 Sukari Indicator Variogram Models

Directional variograms were calculated for each indicator threshold, using Domain 2, 3, 4, and 5 sample composites only, for a series of directions as dictated by the average drill hole orientations. These domains contain the bulk of gold mineralization at Sukari and are the only domains containing sufficient mineralized samples, at reasonable spacing, to allow calculation of meaningful gold and indicator variograms. Indicator transforms were undertaken with probability thresholds 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97, and 0.99 for data in each data subset.

The indictor and gold variograms for Domain 2 have been used for modelling the gold mineralization within the oxide sub-domain for Domain 1 and 7. Domain 3 indictor and gold variograms have been used for modelling the gold mineralization within the remaining domains and sub-domains.

The number of variograms modelled in the present study was very large (75 for each domain). The spatial continuity observed in the variograms is consistent with the known geological interpretation. The fitted models generally have a fairly large short range structure and a smaller long range structure consistent with the strike and plunge of dominant mineralized vein orientation. Figure 14.1-9 is presented as a summary of the modelled variograms at Sukari. The plots shown on the figure are 3D-variogram surface maps for the median indicator variogram for each of the mineralized domains modelled. The viewing angle is generally looking north and down and the modelled variogram ranges at approximately 50% sill are used for the contouring interval.

Figure 14.1-9 Three-Dimensional Variogram Plots



14.1.8 Indicator Kriging Parameters

The input parameters to Indicator Kriging of the Sukari mineralization include:

- Indicator variogram models describing the spatial continuity of indicator variables within each domain at each indicator threshold.
- Variograms describing the spatial continuity of gold grades within each domain.
- Mean gold grades of each of the indicator classes within each domain.

Table 14.1-6 shows the grid framework and kriging parameters used in the indicator kriging model. A description of the categorization of resources is described in Section 14.1.11.

Table 14.1-6 Model Framework and Kriging Parameters

Panel Model Extents			
	m East	**m North**	**m Elevation**
Panel origin (centroid)	10,200	9,625	5
Panel Dimensions	20	25	10
No. of panels	51	104	135
Panel Discretization	5	5	5
Kriging Parameters (all domains)			
Criteria	**Measured**	**Indicated**	**Inferred**
Min no. of data	16	16	8
Max no. of data per octant	8	8	8
Min no. of octants with data	4	4	2
X (east) search radius (metres)	30	45	45
Y (north) search radius (metres)	30	45	45
Z (elevation) search radius (metres)	20	30	30
Search Rotations			
	Rotation Axis	**Rotation**	**Grid Direction**
All Domains 1	x (east)	-20	-20/360
2	y (north)	20	-20/090

Table 14.1-7 to Table 14.1-12 show the indicator thresholds and class means for each domain used in modelling gold at Sukari. Note that class means for Domain 1 for both oxide and sulphide domains have been assigned 0.00 g/t Au in the MIK model. Grade control drilling is limited to not extending into the footwall domain, therefore, the minor gold mineralization in this domain is not modelled and excluded from the Mineral Resource estimate.

Table 14.1-7 Indicator Class Means Domain 1

Probability Threshold	Domain 1			
	Oxide		Sulphide	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.001	0.000	0.001	0.000
0.2	0.001	0.000	0.001	0.000
0.3	0.002	0.000	0.001	0.000
0.4	0.005	0.000	0.002	0.000
0.5	0.010	0.000	0.002	0.000
0.6	0.010	0.000	0.002	0.000
0.7	0.012	0.000	0.003	0.000
0.75	0.015	0.000	0.005	0.000
0.8	0.024	0.000	0.008	0.000
0.85	0.035	0.000	0.010	0.000
0.9	0.055	0.000	0.017	0.000
0.95	0.110	0.000	0.051	0.000
0.97	0.170	0.000	0.097	0.000
0.99	0.280	0.000	0.408	0.000
Max	12.154	0.000	102.886	0.000

Table 14.1-8 Indicator Class Means Domain 2 and 3

Probability Threshold	Domain 2		Domain 3	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.010	0.006	0.004	0.002
0.2	0.010	0.010	0.014	0.009
0.3	0.025	0.018	0.043	0.026
0.4	0.047	0.035	0.110	0.073
0.5	0.081	0.063	0.220	0.161
0.6	0.140	0.108	0.380	0.295
0.7	0.242	0.186	0.622	0.491
0.75	0.324	0.279	0.791	0.703
0.8	0.436	0.377	1.022	0.900
0.85	0.610	0.519	1.358	1.178
0.9	0.900	0.739	1.920	1.610
0.95	1.556	1.172	3.152	2.430
0.97	2.220	1.850	4.325	3.679
0.99	4.070	2.908	9.405	6.023
Max	37.725	8.160	289.865	30.741

Table 14.1-9 Indicator Class Means Domains 4 and 5

Probability Threshold	Domain 4		Domain 5	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.003	0.002	0.003	0.002
0.2	0.007	0.005	0.005	0.004
0.3	0.011	0.009	0.010	0.008
0.4	0.021	0.016	0.019	0.014
0.5	0.040	0.030	0.036	0.027
0.6	0.080	0.058	0.066	0.049
0.7	0.150	0.110	0.124	0.091
0.75	0.212	0.180	0.169	0.145
0.8	0.303	0.254	0.236	0.201
0.85	0.442	0.367	0.339	0.284
0.9	0.682	0.552	0.516	0.419
0.95	1.211	0.911	0.909	0.680
0.97	1.700	1.418	1.280	1.071
0.99	3.058	2.237	2.488	1.728
Max	44.166	5.572	107.860	4.927

Table 14.1-10 Indicator Class Means Domain 6

Probability Threshold	Domain 6	
	Grade Threshold	Class Mean
0.1	0.001	0.001
0.2	0.003	0.002
0.3	0.005	0.004
0.4	0.012	0.008
0.5	0.040	0.024
0.6	0.100	0.068
0.7	0.188	0.143
0.75	0.255	0.218
0.8	0.334	0.294
0.85	0.434	0.385
0.9	0.620	0.521
0.95	1.026	0.784
0.97	1.603	1.283
0.99	3.383	2.247
Max	22.750	6.027

Table 14.1-11 Indicator Class Means Domain 7

Probability Threshold	Domain 7			
	Oxide		Sulphide	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.001	0.001	0.001	0.001
0.2	0.002	0.002	0.001	0.001
0.3	0.003	0.003	0.002	0.002
0.4	0.006	0.005	0.003	0.003
0.5	0.010	0.009	0.005	0.004
0.6	0.015	0.011	0.010	0.008
0.7	0.034	0.022	0.025	0.015
0.75	0.060	0.045	0.055	0.037
0.8	0.120	0.087	0.125	0.088
0.85	0.220	0.162	0.253	0.183
0.9	0.404	0.295	0.518	0.374
0.95	0.827	0.570	1.159	0.789
0.97	1.156	0.995	1.717	1.405
0.99	1.970	1.501	3.085	2.304
Max	6.120	2.860	21.896	5.250

Table 14.1-12 Indicator Class Means Domain 8

Probability Threshold	Domain 8			
	Oxide		Sulphide	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.001	0.001	0.001	0.001
0.2	0.001	0.001	0.002	0.001
0.3	0.001	0.001	0.002	0.002
0.4	0.002	0.001	0.002	0.002
0.5	0.002	0.002	0.002	0.002
0.6	0.003	0.003	0.003	0.002
0.7	0.006	0.004	0.003	0.003
0.75	0.009	0.007	0.003	0.003
0.8	0.010	0.010	0.004	0.004
0.85	0.015	0.011	0.006	0.005
0.9	0.027	0.020	0.010	0.009
0.95	0.065	0.041	0.023	0.014
0.97	0.127	0.089	0.045	0.032
0.99	0.395	0.220	0.170	0.083
Max	11.520	0.977	15.455	0.852

In addition to reviewing the effect of high grades on the indicator statistics, further checks have been made to test the effect on the estimated resources to the use of hard or soft boundaries (particularly contact between the mineralized porphyry and barren footwall domain) and this showed that the calculated resources are insensitive to the choice of boundary option. It is however geologically sensible for the footwall contact between the porphyry (Domains 2, 3, 4 and 5) and footwall domain (Domain 1) to be treated as hard for grade estimation. All other domain transitions have been treated as soft boundaries.

14.1.9 Block Support Adjustment (Variance Adjustment)

14.1.9.1 General

The block support adjustment is one of the most important properties of a recoverable resource model based on non-linear estimation methods like MIK. It is an essential part of the model and involves important assumptions about the nature of the block grade distribution within each panel of the model.

Indicator Kriging provides a direct and reliable estimate of the histogram of grades of sample-sized units within each panel of the model provided the panel dimensions are of an appropriate size. However, ore is not selected on sample-sized units during mining; it is selected by shovels that have a minimum mining width and loaded into trucks that are despatched to deliver either ore or waste. The flexibility of digging equipment and the size of the trucking equipment provide an indication of the size of the smallest block of rock that will be mined as ore or waste. To estimate with some accuracy the resources in a deposit that will be recovered with a certain set of mining equipment, the histogram of grades of sample-sized units in a panel provided by MIK must be adjusted to account for the size of the mining block.

There are a number of adjustment methods that can be used and most of these are described well in Journel & Huijbregts (1978) or Isaaks & Srivastava (1989). These methods make three reasonable assumptions:

- The average grade of sample-sized units and blocks within the panel is the same and is equal to the estimated average grade of the panel.

- The variance, or spread, of the block grades within the panel is less than the variance of grades of sample-sized units within the panel and the change of variance from sample-sized units to blocks can be calculated from the variogram of gold grades.
- The approximate shape of the histogram of block grades can be reasonably predicted by some appropriate assumptions.

14.1.9.2 The Variance Adjustment

The size of the variance adjustment needed to obtain the variance of the block grade distribution within the panel can be calculated using the rule of additivity of variances, which in the case of block support adjustment is often called Krige's Relationship:-

Var(samples in a panel) = Var(samples in a block) + Var (blocks in a panel)

The variance of sample grades in a panel and the variance of samples within a block can be directly calculated from the variogram of gold grade for the particular domain. The ratio of **Var(blocks in panel) to Var(samples in panel)** is that required to implement the block support adjustment.

14.1.9.3 Shape of the Block grade Distribution

There are a number of rules of thumb that are useful when making judgements about the shape of the block grade distribution within each panel and they relate to the size of the variance adjustment ratio:

- If the variance adjustment ratio is greater than 0.7, it may be useful to assume that the shape of the histogram of block grades is similar to that of the histogram of grades of sample-sized units. This is known as the Affine Correction method. Its application to gold deposits is usually inappropriate.
- If the variance adjustment ratio is between 0.3 and 0.7 and the information adjustment is negligible, then the Indirect Lognormal Correction method of Isaaks & Srivastava (1989) can be useful. This is a rule of thumb based on the experience of the authors.
- If the variance adjustment ratio is less than 0.3, a high degree of symmetrization in the block grade histogram will occur and a direct lognormal assumption (Journel & Huijbregts, 1978, page 481) for the shape of the block histogram is an appropriate choice. This model is well supported by reconciliation studies of resource and grade control models.

14.1.9.4 The Information Effect

The variance adjustment described above is only part of the adjustment required in many gold deposits because the short scale variation in gold grades is extreme, as is the case at Sukari. This variance adjustment provides an estimate of the variance of *true* block grades under the assumption that grade control selection will operate with knowledge of the true block grades. While this assumption is never absolutely true, it can be a reasonable assumption in some deposits where the short scale variability is small and the grade control sampling density is high. In many deposits, however, an additional variance adjustment must be undertaken to account for the "Information Effect".

In the absence of production information or grade control sampling, the Information Effect ratio is based on the variograms of gold grade and on the grade control sample spacing expected to be used during mining.

14.1.9.5 Variance Adjustments Applied to the Sukari Model

For Sukari, total variance adjustment ratios ranging from 0.019 to 0.071 have been applied in estimating recoverable gold resources (Table 14.1-13). These ratios have been applied using the Direct Lognormal Correction method (i.e. incorporating symmetrization of block grade distributions). SMU dimensions of 5 mE x 8 mN x 10 mRL have been assumed.

Table 14.1-13 Variance Adjustments

Domain	1, 2 and 7	3 and 8	4	5	6
Panel to Block Adjustment	0.078	0.049	0.085	0.090	0.049
Information Effect Total Ratio	0.522	0.322	0.588	0.789	0.322
Total Adjustment	0.041	0.016	0.050	0.071	0.016

14.1.10 Bulk Densities

The bulk densities (BD) used in calculating tonnages in the Sukari resource model were obtained from the site generated BD database and are shown in Table 14.1-14.

Table 14.1-14 Bulk Densities

Rock Type	Modelling Domain	Bulk Density
Porphyry, Weathered	Domains 2, 4 and 6, 7, 8 Oxide sub-domains	2.63
Porphyry, Primary	Domains 3, 5, and 6, 7, 8 Sulphide sub-domains	2.66
Footwall, Weathered	Domains 1, Oxide	2.74
Footwall, Primary	Domains 1, Sulphide	2.78

14.1.11 Resource Classification

Panels in the resource model have been allocated a confidence category based on the number and location of samples used to estimate proportions and grade of each panel. The approach is based on the principle that larger numbers of samples, which are more evenly distributed throughout the search neighbourhood, will provide a more reliable estimate. The number of samples and the particular geographic configurations that may qualify the panel as Measured rather than Indicated or Inferred are essentially the domain of the Qualified Person. The search parameters used to decide the classification of a panel resource in this study are:

- <u>Minimum number of samples found in the search neighbourhood.</u>

  For Measured and Indicated categories, this parameter is set to sixteen. For Inferred category, a minimum of eight samples is required. This parameter ensures that the panel estimate is generated from a reasonable number of sample data.

- <u>Minimum number of spatial octants informed.</u>

  The space around the centre of a panel being estimated is divided into eight octants by the axial planes of the data search ellipsoid. This parameter ensures that the samples informing an estimate are relatively evenly spread around the panel and do not all come from one drill hole. For Measured and Indicated categories, at least four octants must contain at least one sample. For Inferred panels, at least two octants must contain data.

- <u>The distance to informing data.</u>

  The search radii define how far the kriging program may look in any direction to find samples to include in the estimation of resources in a panel. Panel dimensions and the sampling density in various directions usually influence the length of these radii. It is essential that the search radii be kept as short as possible while still achieving the degree of resolution required in the model. For Measured category, the easting, northing and elevation search radii are set to 30 m, 30 m and 20 m respectively. For Indicated and Inferred categories, these radii are each expanded by 50%.

The number of samples and the particular geographic configurations that may qualify the panel as Measured rather than Indicated or Inferred may be a somewhat subjective decision. The confidence classification is essentially the domain of the Qualified Person and, in MPR's experience, the strategy adopted here results in a geologically sensible classification whereby Measured and Indicated panels are surrounded by data in close proximity. Inferred panels may occur on the peripheries of drilling but are still related to drilling data within reasonable distances. Downgrading of these confidence categories may result from a consideration of other factors such as QAQC or areas of inadequate drillhole sampling, etc.

MPR is of the view that the modelled categorization used in the current study is appropriate for resource classification at Sukari.

14.1.12 Mineral Resource Estimates

Total Measured and Indicated Mineral Resource estimate of 13 Moz contained gold is reported as an open-pit Mineral Resource at a 0.3 g/t cut-off grade. This total is inclusive of the estimated 1.0 Moz underground Mineral Resource.

Table 14.1-15 lists estimated mineral resources by confidence categories as described in the previous section.

The resource estimate is reported below as at 30 June 2015 open-pit mined surface and reduced by the tonnage mined by underground development and stoping as at 30 June 2015. The resources extend to 0 mRL (a maximum depth of 1,350 m from surface).

Table 14.1-15 Total Mineral Resource for Sukari

Cut-off	Measured		Indicated		Total Measured & Indicated			Inferred		
	Tonnes (Mt)	Grade (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)
0.3	**198**	**1.05**	**188**	**1.02**	**386**	**1.03**	**12.9**	**33**	**1.0**	**1.1**
0.4	160	1.22	152	1.18	312	1.20	12.0	26	1.2	1.0
0.5	133	1.38	124	1.34	257	1.36	11.2	21	1.3	0.9
0.7	95	1.69	87	1.66	182	1.68	9.8	15	1.7	0.8
1.0	62	2.14	56	2.12	118	2.13	8.1	9	2.1	0.6

Notes to table:

- The Mineral Resource estimate is based on the open-pit mined surface as at 30 June 2015 and depleted for underground mine workings as at 30 June 2015.
- All available assays as at February 2015.
- Resource dataset comprises 252,449 two metre down hole composites and surface rock chip samples.
- Mineral Resource is reported inclusive of those resources converted to Proven and Probable Mineral Reserves.
- The resource is an estimate of recoverable tonnes and grades using Multiple Indicator Kriging with block support correction.
- Measured Resources lie in areas where drilling is available at a nominal 25 x 25 metre spacing, Indicated Resources occur in areas drilled at approximately 25 x 50 metre spacing and Inferred Resources exist in areas of broader spaced drilling.
- The resource model extends from 9,700 mN to 12,200 mN and to a maximum depth of 0 mRL (a maximum depth of approximately 1,000 metres below wadi level).

This Mineral Resource estimate has been classified and reported in accordance with NI 43-101 and the classifications adopted by CIM Council in November 2004. Furthermore, the resource classifications are also consistent with the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" of 2012 (JORC, 2012) as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia (JORC).

The reporting of resource classification under the JORC Code and NI 43-101 are essentially identical, the notable difference being the requirement to report Inferred Mineral Resources separate from the totalled Measured and Indicated Mineral Resources under NI 43-101.

The QP responsible for the Mineral Resource is Nicolas Johnson who is a full-time employee of MPR Geological Consultants Pty Ltd and a member of the Australian Institute of Geoscientists. Mr Johnson has visited the Sukari project site on numerous occasions, most recently in July 2013.

14.1.13 Plots of the Sukari Resource Model

Figure 14.1-10 and Figure 14.1-11 show representative cross-sections. Each figure shows resource panels scaled and coloured by the estimated resource tonnes above 0.3 g/t Au cut-off grade and coloured by the average panel grade (Note: the estimated proportion at 0.3 g/t cut-off grade has been used to scale the east dimensions of the panels). The model blocks in all figures are only shown below the end of December 2015 (blue polyline) mined surface.

It should be noted that when viewing these plots through the resource model there are situations where the panels appear to be un-correlated to the mineralized intercepts in the neighbouring drillholes. This is occurring because of the way the resource panels have been presented. The panels plotted are only those that contain an estimated resource above 0.3 g/t Au cut-off and the proportion of ore has been used to scale the east dimension of the panel for presentation purposes. The scaling occurs about the panel centroid coordinate and therefore introduces the apparent miss-match between data and the resource panels.

Figure 14.1-10 Section 10,325 mN Panel Estimated Proportion and Grades at 0.3 g/t Au cut-off

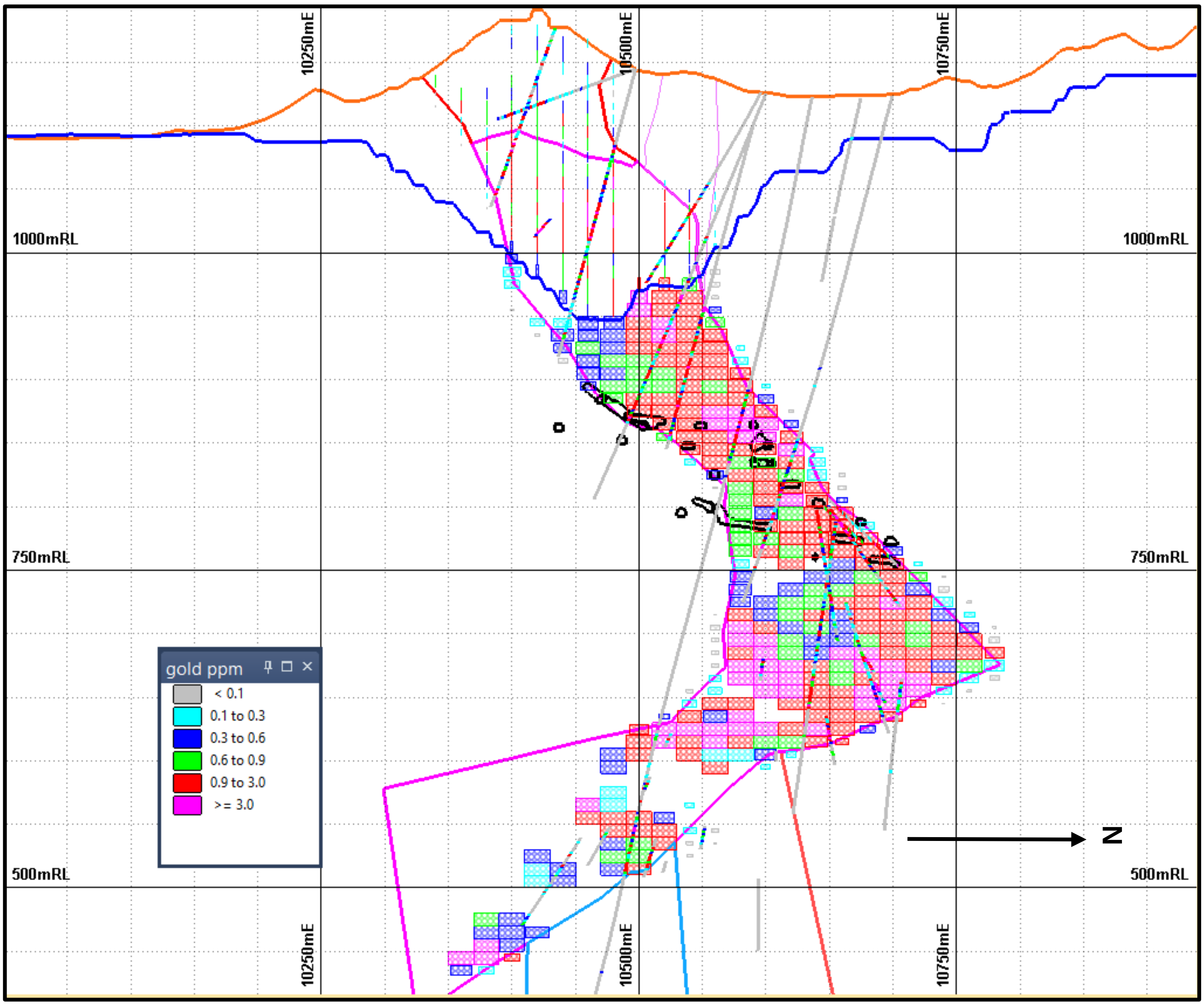
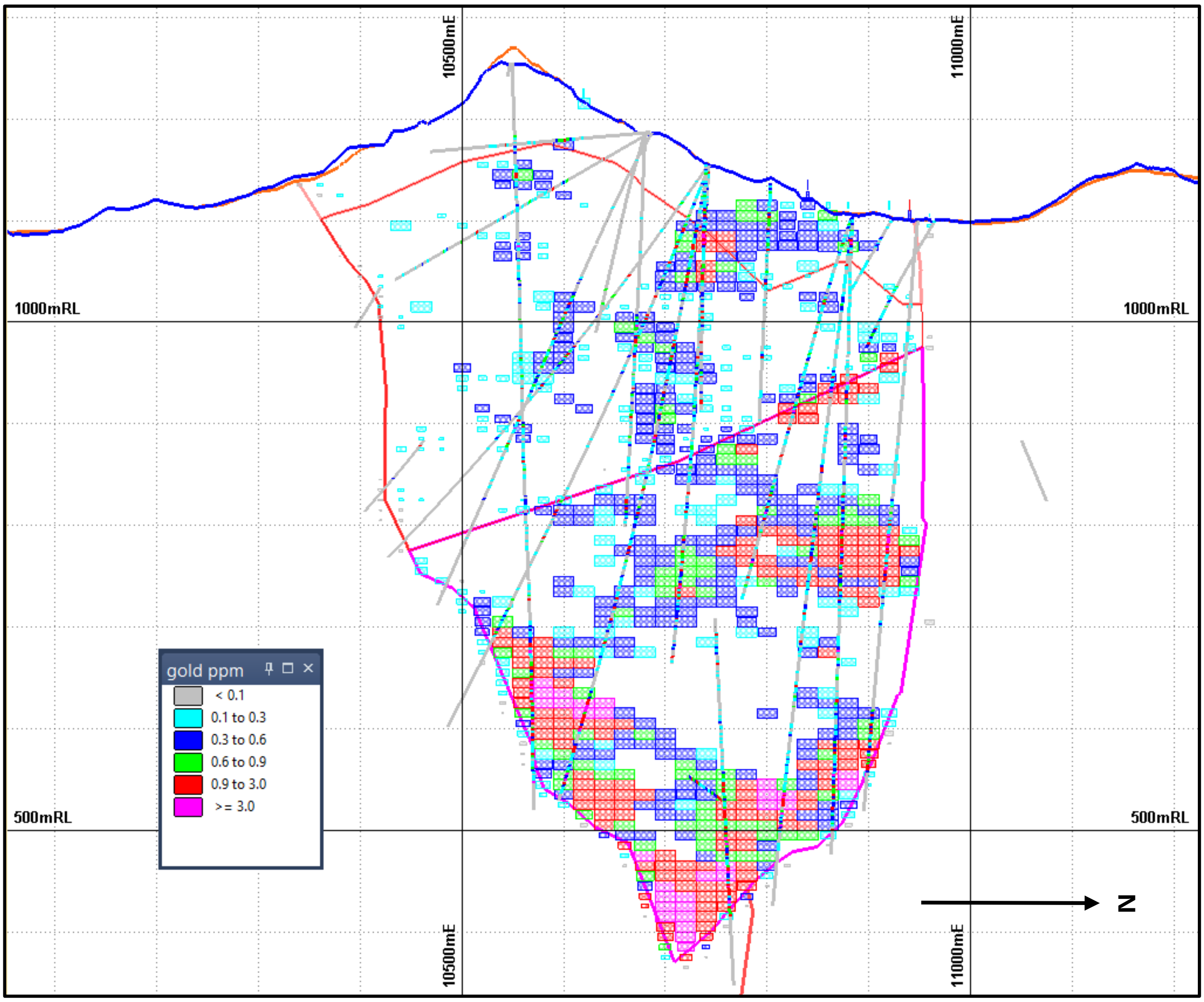


Figure 14.1-11 Section 11,300 mN Panel Estimated Proportion and Grades at 0.3 g/t Au cut-off



14.2 Underground Mineral Resource

14.2.1 Introduction

The updated Mineral Resource estimate for the Underground Sukari Mineral Resource was completed by Cube in July 2015. This estimate represents an update of the Mineral Resource previously reported in the NI 43-101 dated January 2014.

The update was completed under the supervision of Mark Zammit BSc (Hons) GradCertGeostats GradDipBus MAIG.

The aims of the work were to:

- Update the gold mineralization interpretation for the higher grade Sukari domains for exploitation by underground mining.
- Update the estimation of these higher-grade Sukari domains; and
- Update the associated Mineral Resource classification.

All estimation work was carried out using SURPAC mining software and Isatis geostatistical software. Grade interpolation for gold has used an Indicator weighting method of 1 m downhole composite drill data.

During 2014 and 2015, Centamin continued a program of resource delineation drilling to infill, extend and verify the historical resources at Sukari. The data gathered by the drill programs and the additional exposure from underground mining, has formed the basis for this July 2015 Mineral Resource estimate update undertaken by Cube.

14.2.2 Previous Resource Estimate

The previous Mineral Resource estimate for the Underground Resource at Sukari reported in the NI 43-101 technical report dated January 2014 contained 844,000 oz Au in the Measured and Indicated category and 489,000 oz Au in the Inferred category. These Mineral Resources were modelled using a 2 g/t cut-off to determine the resources outlines.

Table 14.2-1 Historic Sukari Underground Mineral Resource Estimate – January 2014

Resource Classification	Tonnes (Mt)	Grade (g/t)	Contained Gold (koz)
Measured Mineral Resource	0.537	12.8	222
Indicated Mineral Resource	3.805	5.1	622
Total Measured and Indicated	**4.342**	**6.1**	**844**
Inferred Mineral Resource	2.925	5.2	489

14.2.3 Data Supplied

Data supplied to Cube by Centamin included:

- Drillhole data for the Amun project area in the form of a Microsoft Access file
- Drillhole data for the Ptah project area in the form of a Microsoft Access file
- Digital photography of available diamond drillholes
- Working interpretations for the mineralization models for the Amun and Ptah project areas
- Lithological model for the Sukari porphyry
- Underground mine development and stoping as Surpac 3DMs (three-dimensional solids; and
- Digital scans of available underground development face mapping.

14.2.4 Geology and Mineralization Modelling

14.2.4.1 Lithology

The majority of the gold mineralization at Sukari is hosted in the Sukari Porphyry. Some minor mineralization is also present directly adjacent to the porphyry in fault-related structures. The limits of the Sukari Porphyry have been interpreted previously by the Centamin geological staff and this was supplied to Cube in the form of a valid 3DM solid. Cube sliced the 3DM solid on 50 m spaced east-west sections and modified the porphyry interpretation to include all available drilling. The updated slice polygons were wireframed and used as a guide to the structural controls and mineralization interpretation.

Andesite dykes are present with variable widths from centimetres up to approximately 20 m which postdate the mineralization. The larger and more continuous dykes were interpreted by Cube on 25 m to 50 m spaced east-west sections and valid wireframes constructed.

14.2.4.2 Weathering

The area represented by the Underground Mineral Resource is typically more than 200 m below surface. This area is dominated by the presence of sulphide mineralization and the absence of oxidation. Therefore the use of oxidation surfaces has not been included as part of this Mineral Resource estimate and all mineralization has been defined as fresh rock.

14.2.4.3 Mineralization Domains

Mineralization relating to the Sukari Underground Mineral Resource is separated into a northern and southern area referred to as Ptah and Amun respectively. The separation between the two areas occurs approximately at 10,700 mN with the Amun Decline accessing the southern mineralization and a second decline known as the Ptah Decline branching off the Amun Decline and accessing the northern mineralization. The initial mineralization interpretations provided to Cube by Centamin geological staff, were based on local geological knowledge and typically focussed on grades greater than 2 g/t Au.

The initial mineralization domains were subsequently added to and modified by Cube in order to complete the Mineral Resource estimate. The modifications made by Cube included allowing the interpretation to include some intersections which displayed the same geological characteristics but were lower grade. The aim was to produce a more robust interpretation with better domain continuity down dip and along strike, with some lower grade material being included as internal dilution to preserve the overall continuity of the mineralized zones. The final interpretations were completed on variable spaced sections depending on the density of available data. Amun mineralization was based on a combination of 10 m and 25 m east-west sections. The Ptah mineralization is typically based on 12.5 m and 25 m east-west sections south of 11,000 mN and 50 m spaced sections north of 11,000 mN.

Cube completed a boundary analysis for the larger domains and graphs for the Amun Domain 1 and Ptah Domain 107 are shown in Figure 14.2-1 and Figure 14.2-2 respectively. Both graphs clearly show an abrupt grade change at the interpreted boundary position (position 0 on the horizontal axis) with average grades inside the domain boundary significantly higher than the average grades outside. This suggests the absence of any obvious diffuse grade boundary relationship and confirms the interpreted boundary position and wireframe approach as being appropriate.

Figure 14.2-1 Boundary Analysis – Amun Domain 1

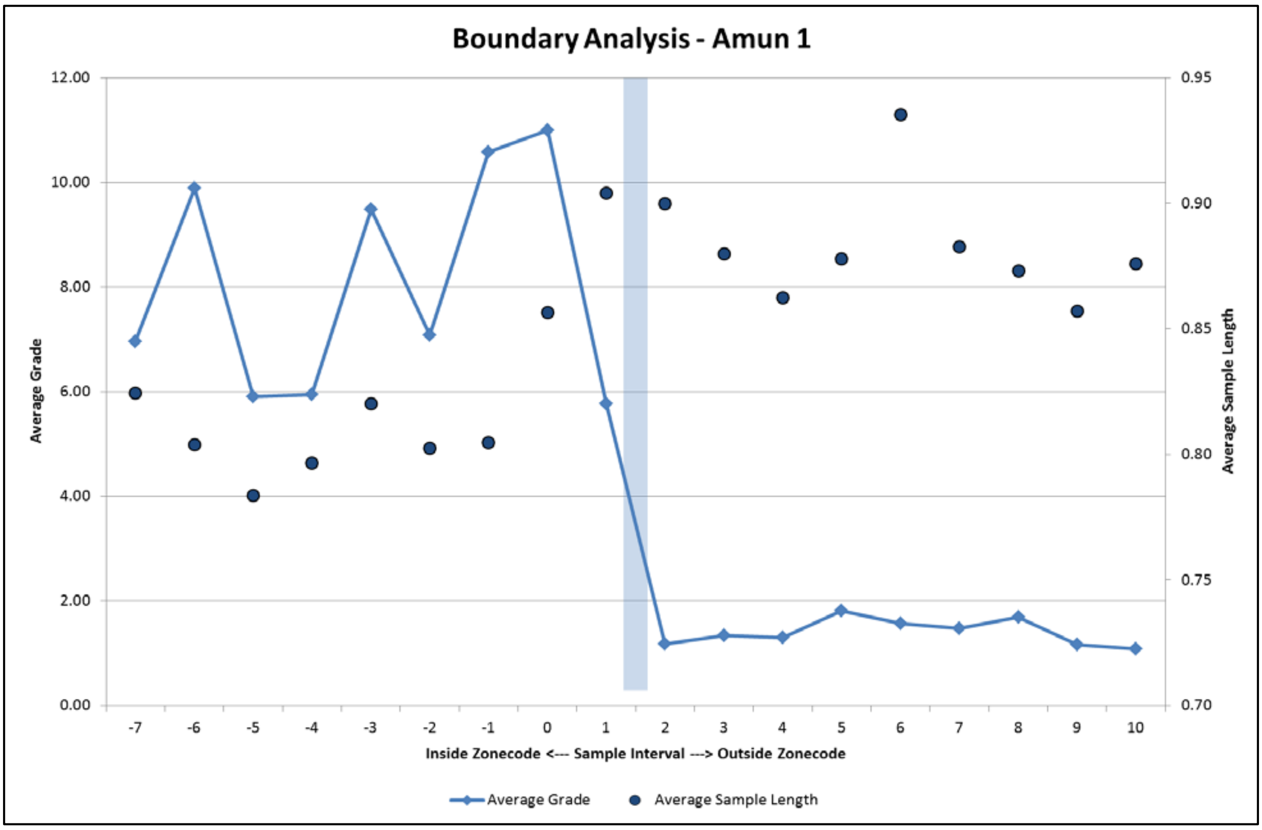
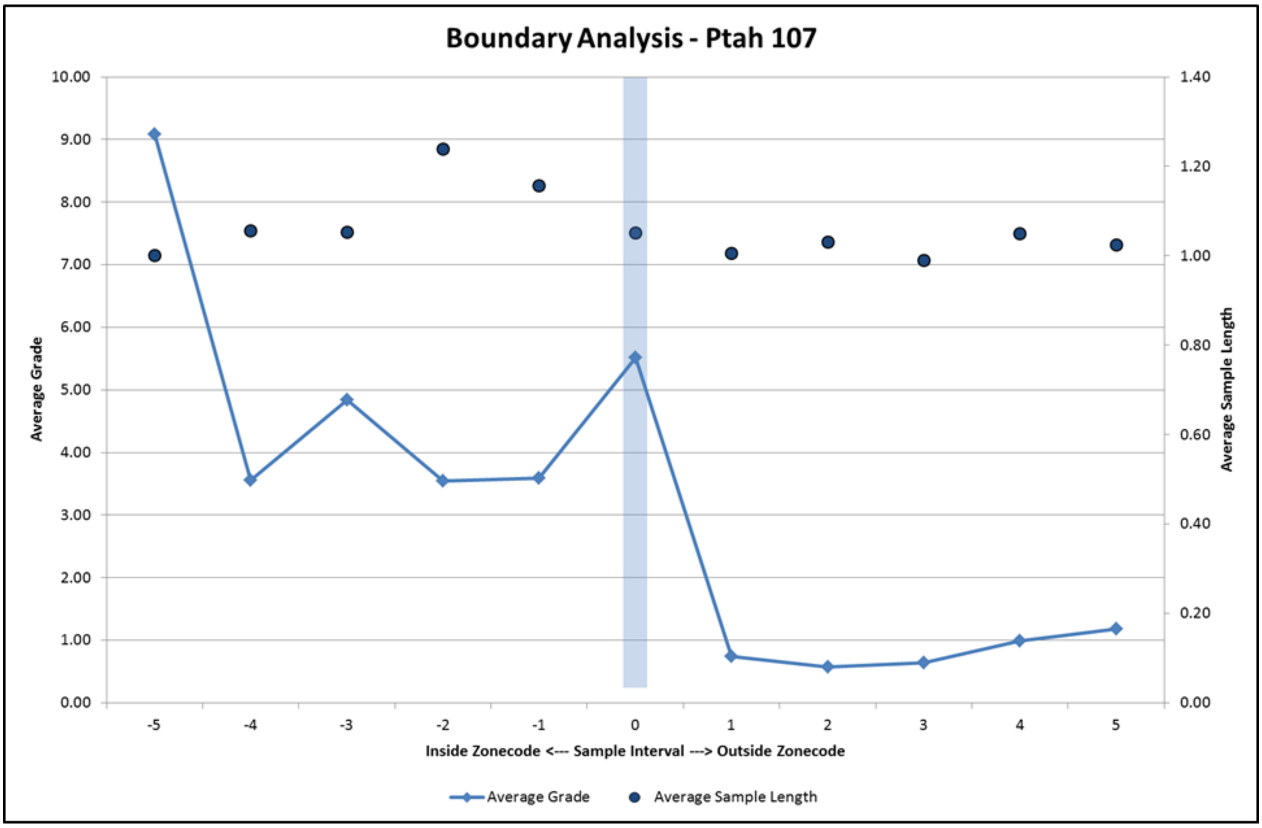


Figure 14.2-2 Boundary Analysis – Ptah Domain 107



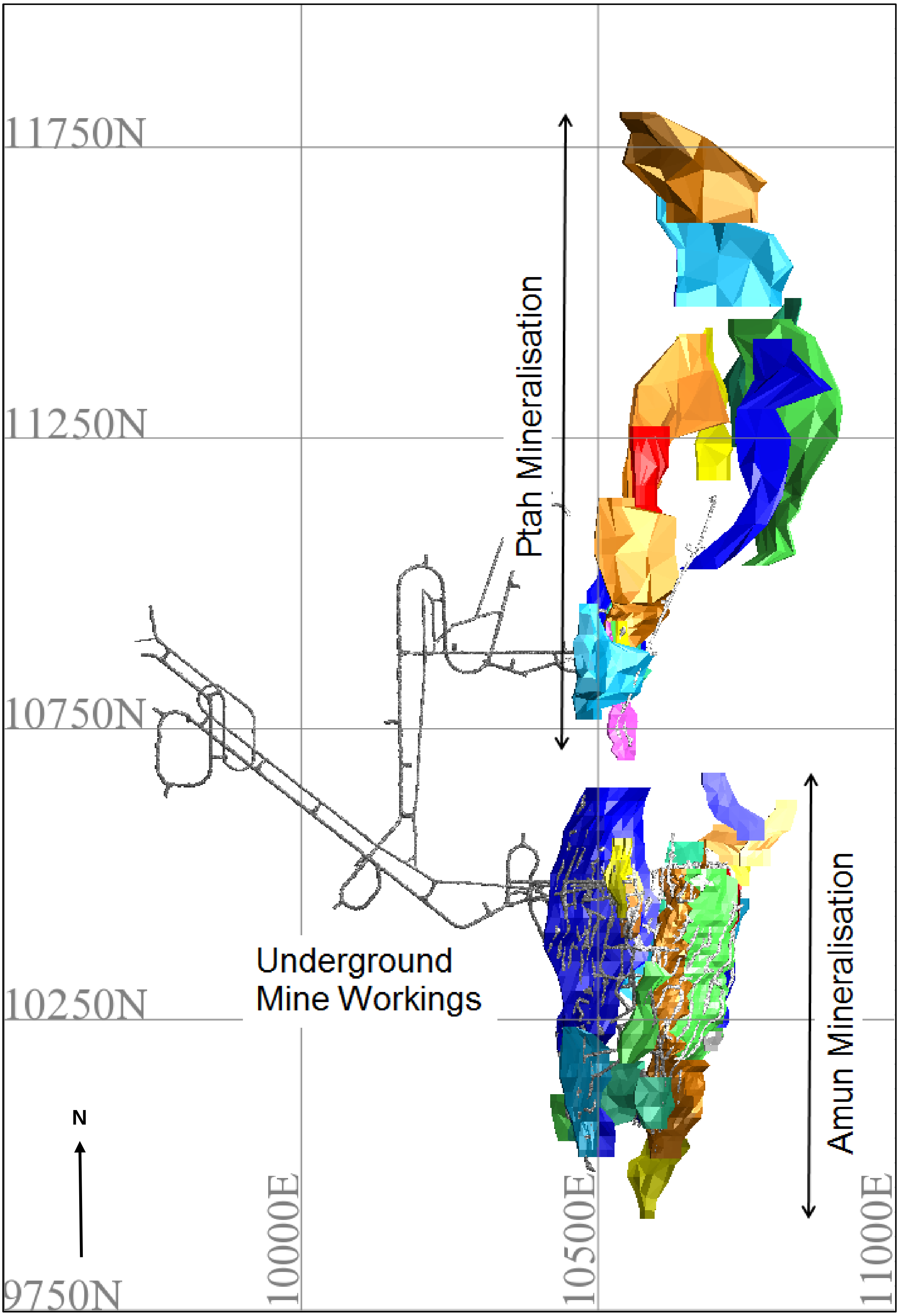
On each section, the interpreted polylines were snapped to the drillhole sample positions where possible. There are instances where snapping of the interpretation to drilling is not practical, particularly in areas where there are a large number of drillholes within the interpretation. A minimum downhole length of 2 m was used with the interpreted domains. For each section the interpretation was not typically extended more than 15 m along strike or down-dip past the last drillhole intersection. However, sectional interpretations were extended further than 15 m on some sections in some sparsely sampled areas of the better defined and continuous domains.

The final interpretation being 27 domains for the Amun area and 19 domains for Ptah area, was reviewed by Centamin staff. The 46 domains which collectively make up the Sukari Underground Mineral Resource are summarized below in Table 14.2-2 with a brief description of the general mineralization orientation and displayed in Figure 14.2-3.

Table 14.2-2 Mineralization Sub-Domains

Area	Domain Number	General Orientation
Amun	3, 16, 18	Steep east dipping (~700º)
	1, 2, 4, 5, 8, 9, 10, 19, 31, 32, 33, 34, 35	Moderate east dipping (~400º – 500º)
	6, 7, 11, 13, 14, 17, 20, 21, 36	Flat to shallow west dipping (~00º – 100º)
	12, 15, 16, 17	Moderate to shallow west dipping (~100º – 250º)
Ptah	106, 107, 108, 109, 110, 111, 114, 118	Moderate to steep east dipping (~600º – 700º)
	112, 113, 115	Moderate east dipping (~350º – 500º)
	112, 116, 120	Moderate to shallow east dipping (~400º – 500º)
	101, 102, 103	Flat to shallow east dipping (~00º – 100º)
	104, 105, 121	Moderate west dipping (~350º – 500º)

Figure 14.2-3 Mineralized Amun and Ptah Domains – Plan View



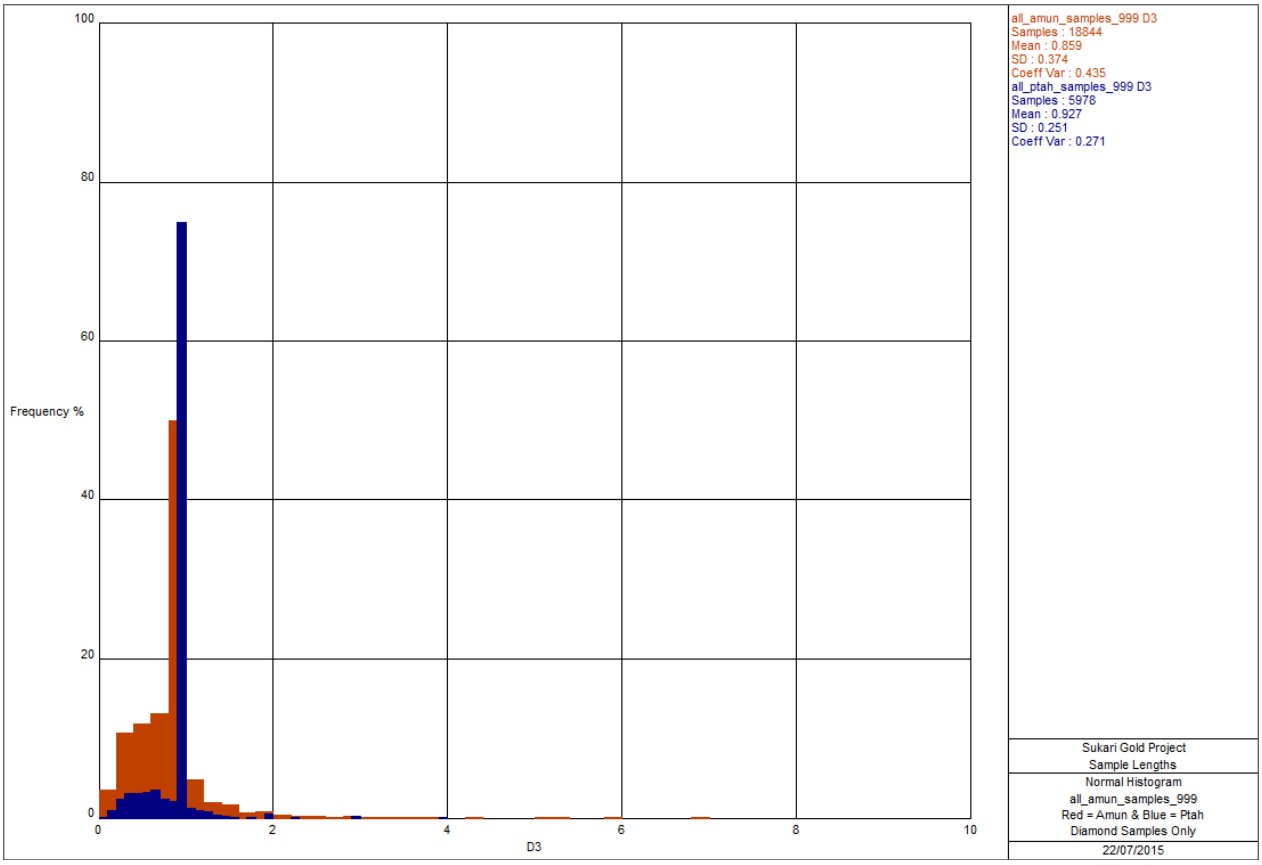
14.2.5 Compositing

In the drillhole databases specific to Amun and Ptah both contain sample data relating to drilling (diamond and RC) and face sampling. The method for taking the face samples typically involves dividing the development face into 5 equal areas and taking chip samples that represent a composite sample for each area. The five discrete samples are then converted into the equivalent of a channel sample and stored in the drillhole database. Given the discrete face samples represent different sample support to the continuous linear sampling represented by the diamond drilling, only the diamond drill samples were used for grade estimation in this Mineral Resource update.

In the Amun and Ptah drillhole databases, a unique code for drill intercepts within each of the mineralized domains was added to the database table "*zonecode*_amun" and "*zonecode_ptah*". The process of coding the database was carried out by manually identifying the appropriate downhole interval to be coded and assigning the unique code to the domain wireframe. This coded interval was used to control the compositing process whilst extracting sample and composite data combinations for statistical analysis and subsequent estimation.

Diamond drilling for the mineralized domains was generally sampled on 1 m intervals with 55% of the 8,421 Amun mineralized samples and 78% of the 4,324 Ptah mineralized samples having a sample length of 1 m. Sample lengths varied from 0.1 m to 7 m with a mean length of 0.84 m and 0.94 m for Amun and Ptah respectively. For the combined project area, 63% of the diamond drilling samples had a length of 1 m as shown in Figure 14.2-4. In addition, a total of 7,372 face samples are included within the combined mineralized domains of which 7,205 (98%) of the samples are 1 m in length.

Figure 14.2-4 Raw Sample Length Histogram for Mineralized Amun and Ptah Domains



Four main criteria were considered when determining the most appropriate compositing approach:

1. Sample length statistics
2. Additivity of variables
3. Homogeneity of composited zones
4. Proposed block estimate and SMU size.

After an examination of the above criteria, 1 m downhole composites were deemed as being an appropriate composite length to be used for the Sukari Underground mineralized domains. The downhole compositing process used a "best-fit" approach resulting in composites of slightly variable but equal length within a mineralized domain. This ensures that the assay sample composite length is as close as possible to the nominated 1 m composite length whilst reducing the amount of residual lengths normally created during fixed length compositing.

For Amun, the gold composites were extracted from the "*au*" field within the "*Assay*" table of the "*amun_ug_db_030415.mdb*" database. For Ptah, the gold composites were extracted from the "*Au*" field within the "*Assay*" table of the "*Ptah_UG_DB_030415.mdb*" database. The "*zonecode_amun*" and "*zonecode_ptah"* interval tables were used from the respective databases to ensure only mineralized coded sample intervals were composited. In both databases, a "hole_type" field was used to control the compositing process and to ensure the grade control face sampling and diamond drillholes were composited separately. Using this approach a total of 15,987 diamond composites for Amun was extracted and 5,382 diamond composites for Ptah.

The structure for composite files is summarized in Table 14.2-3.

Table 14.2-3 Composite File Data Fields

Field	Description
D1	Au ppm – uncut interval composite
D2	Hole ID
D3	Interval from Depth
D4	Interval to Depth
D6	Downhole Composite Interval Length
D9	DH (Diamond) or FS (Face Sample)
D11	Au ppm – cut interval composite

14.2.6 Exploratory Data Analysis

A statistical and spatial analysis of the extracted 1 m downhole composites was undertaken separately for each of the 46 mineralized domains. The basic statistics of the 1 m diamond drill hole composites for Amun and Ptah only are shown below in Table 14.2-4 and Table 14.2-5 respectively with the log probability plots shown in Figure 14.2-5 and Figure 14.2-6.

An investigation of the sample statistics indicates a change in assay population at approximately 5 g/t Au. Using Isatis software, a data selection for each composite file was set up to identify individual composites that were above (referred to as BZ) or below (referred to MZ) the 5 g/t Au cut-off.

Table 14.2-4 Statistics for Amun 1 m Diamond Drillhole Composite Data (g/t Au)

Domain	No. Data	Min Value g/t Au	Max Value g/t Au	Mean g/t Au	Median g/t Au	Std Dev	CV
1	4,086	0.0	1,835.5	8.2	2.4	38.4	4.7
2	268	0.0	444.0	5.7	2.3	28.2	5.0
3	200	0.1	120.5	5.6	2.4	13.3	2.4
4	316	0.0	297.2	10.9	2.5	26.8	2.5
5	672	0.0	979.0	14.0	3.4	51.8	3.7
6	44	0.0	5,420.0	141.0	4.3	815.9	5.8
7	35	0.2	21.4	3.9	2.6	3.9	1.0
8	113	0.0	210.5	7.5	2.1	22.5	3.0
9	63	0.3	62.2	6.8	2.8	11.4	1.7
10	44	0.0	271.4	27.6	8.2	46.9	1.7
11	32	0.1	104.0	12.2	4.0	22.7	1.9
12	227	0.0	191.0	12.2	3.4	28.5	2.3
13	79	0.0	128.0	8.6	3.2	19.5	2.3
14	20	0.7	112.9	10.4	2.5	24.8	2.4
15	69	0.1	119.0	7.4	2.6	19.3	2.6
16	126	0.0	87.0	5.6	2.5	12.1	2.2
17	247	0.0	484.4	16.2	2.6	53.9	3.3
18	59	0.5	30.7	4.4	2.6	5.7	1.3
19	20	0.0	1,040.0	74.9	4.6	235.3	3.1
20	35	0.0	73.4	5.2	0.6	13.6	2.6
21	62	0.1	105.6	11.4	4.9	17.8	1.6
31	4,700	0.0	701.8	7.9	1.7	28.5	3.6
32	1,807	0.0	1,350.2	3.7	1.8	32.1	8.8
33	508	0.0	53.2	4.1	1.6	6.1	1.5
34	760	0.0	49.2	3.1	1.9	4.3	1.4
35	1,343	0.0	144.1	3.9	1.6	9.5	2.5
36	52	0.0	81.9	6.9	0.7	17.6	2.6

Table 14.2-5 Statistics for Ptah 1 m Diamond Drillhole Composite Data (g/t Au)

Domain	No. Data	Min Value g/t Au	Max Value g/t Au	Mean g/t Au	Median g/t Au	Std Dev	CV
101	188	0.0	56.0	3.3	1.3	6.7	2.0
102	175	0.0	139.0	3.7	1.6	11.6	3.1
103	300	0.0	22.8	2.2	1.3	2.8	1.3
104	192	0.1	41.9	5.2	2.8	7.1	1.4
105	531	0.0	745.0	10.3	1.8	50.3	4.9
106	269	0.0	31.0	3.7	2.1	4.5	1.2
107	330	0.0	410.0	4.9	2.4	23.2	4.7
108	281	0.0	80.1	5.0	2.6	9.7	1.9
109	574	0.0	45.2	2.6	1.5	4.0	1.5
110	42	0.0	151.0	19.4	2.3	36.4	1.9
111	128	0.0	418.0	41.7	7.5	74.0	1.8
112	311	0.0	34.9	3.4	2.3	4.3	1.3
113	203	0.0	354.6	11.4	2.8	34.7	3.0
114	177	0.0	18.3	2.1	1.3	2.5	1.2
115	429	0.0	16.8	2.2	1.6	2.2	1.0
116	228	0.0	23.3	2.5	1.6	3.1	1.2
118	81	0.4	14.0	3.5	2.4	3.0	0.9
120	805	0.0	189.0	3.8	2.2	9.3	2.5
121	138	0.1	288.0	7.0	1.2	27.0	3.8

Figure 14.2-5 Log Probability Plot for Amun 1 m Diamond Drillhole Composite Data (g/t Au) by Domain

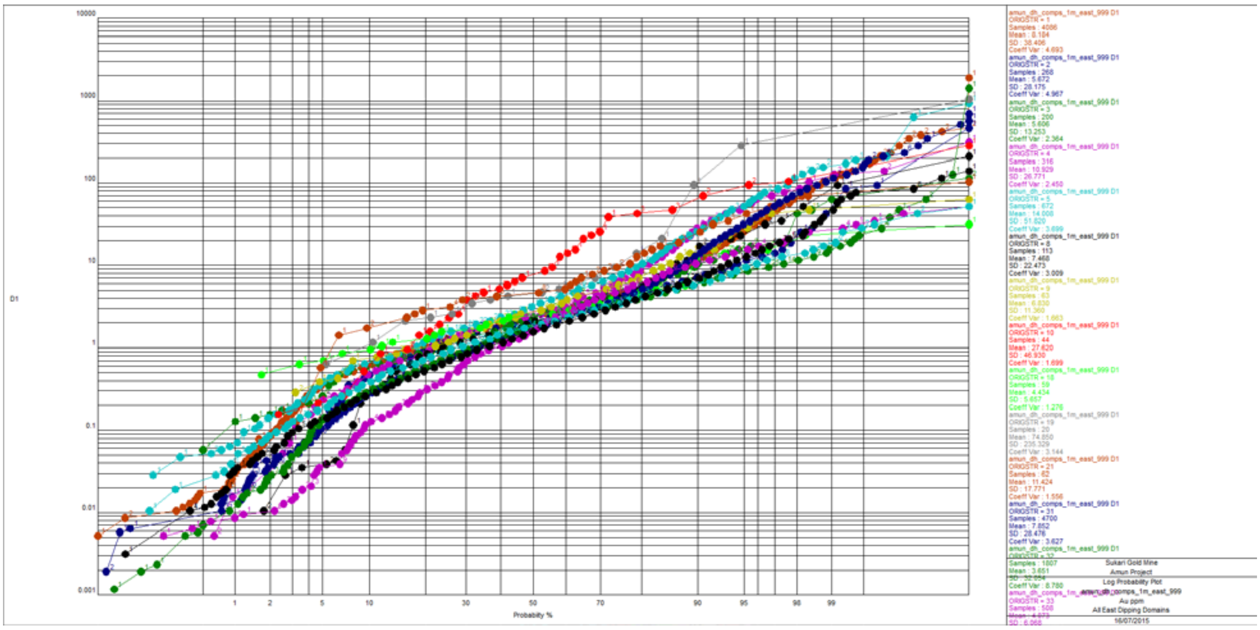
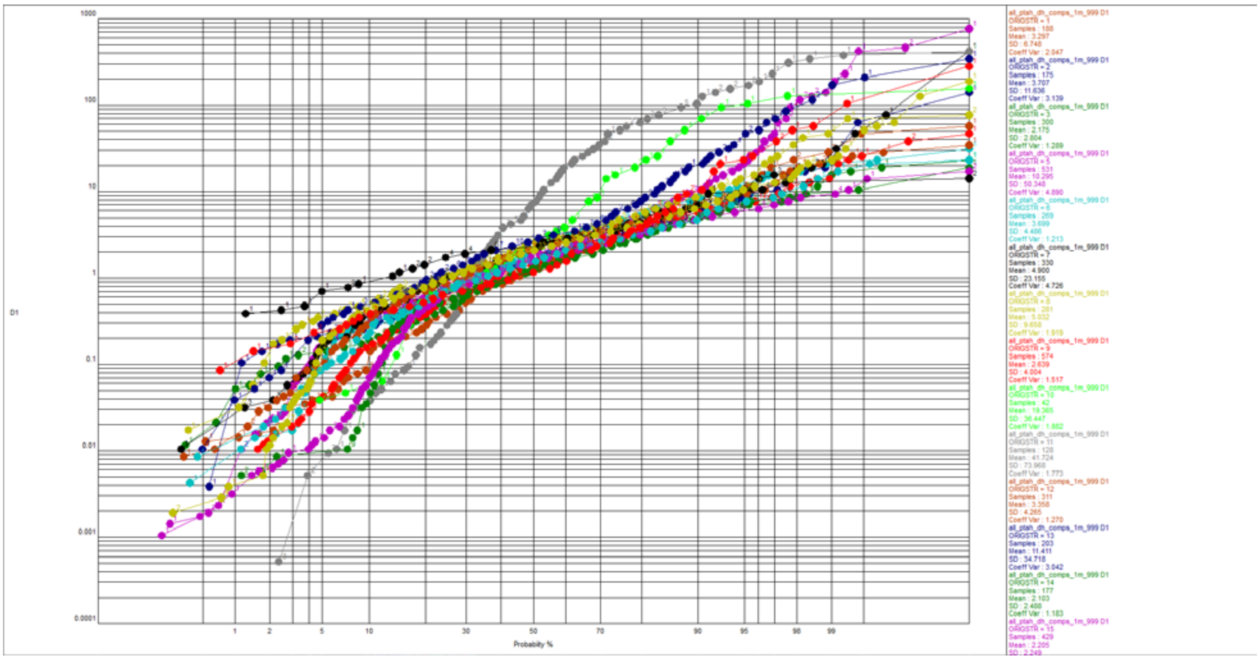


Figure 14.2-6 Log Probability Plot for Ptah 1 m Diamond Drillhole Composite Data (g/t Au) by Domain



14.2.6.1 Grade Outliers and Top Cuts

Cube reviewed the statistics of the composites to check for outlier composite grades prior to estimation. The composite data was reviewed globally and for each individual domain using histograms, log-histograms, log-probability plots and high grade metal sensitivity analysis, combined with spatial inspection of the grade distribution and outlier locations.

Appropriate high-grade cuts were applied as required on an individual domain basis. For some domains high grade cuts were not required where the grade variability relative to the mean was acceptable and spatial analysis of the high composite gold values did not indicate that they were outliers.

The summary of the descriptive statistics for all the Amun and Ptah domains is listed in Table 14.2-6 and Table 14.2-7. The number of composites influenced by the high-grade cut is shown in parenthesis next to the top cut value. Included in the summary statistics for the top cut composites is the declustered mean based on a 50 m x 50 m x 50 m cell size.

Table 14.2-6 Statistics for Amun 1 m Diamond Drillhole – Cut Composite Data (Cut g/t Au)

Domain	No. Data	Mean g/t Au	Au Top Cut	Cut Mean g/t Au	Decl. Cut Mean g/t Au	Cut CV
1	4,086	8.18	400	7.79	7.34	3.3
2	268	5.67	100	4.39	4.87	2.3
3	200	5.61	–	5.61	6.08	2.4
4	316	10.93	150	10.46	10.46	2.2
5	672	14.01	200	12.15	12	2.3
6	44	141.02	100	14.95	13.31	1.7
7	35	3.86	–	3.86	3.27	1.0
8	113	7.47	100	6.49	6.48	2.2
9	63	6.83	–	6.83	5.68	1.7
10	44	27.62	100	23.70	24.64	1.3
11	32	12.19	50	10.09	10.94	1.6
12	227	12.22	–	12.22	14	2.3
13	79	8.64	50	6.72	7.7	1.5
14	20	10.43	50	7.29	7.8	1.6
15	69	7.43	50	5.50	5.48	1.8
16	126	5.60	50	4.98	4.64	1.7
17	247	16.24	200	13.31	11.8	2.5
18	59	4.43	–	4.43	4.05	1.3
19	20	74.85	100	19.35	17.12	1.8
20	35	5.18	50	4.51	3.98	2.3
21	62	11.42	–	11.42	10.11	1.6
31	4,700	7.85	250	7.47	6.43	3.0
32	1,807	3.65	100	2.95	2.73	1.7
33	508	4.07	–	4.07	3.84	1.5
34	760	3.09	–	3.09	3.38	1.4
35	1,343	3.87	–	3.87	3.63	2.5
36	52	6.88	–	6.88	7.48	2.6

Table 14.2-7 Statistics for Ptah 1 m Diamond Drillhole – Cut Composite Data (Cut g/t Au)

Domain	No. Data	Mean g/t Au	Au Top Cut	Cut Mean g/t Au	Decl. Cut Mean g/t Au	Cut CV
1	188	3.30	–	3.30	4.76	2.05
2	175	3.71	30	2.89	3.1	1.42
3	300	2.18	–	2.18	2.53	1.29
4	192	5.17	–	5.17	4.36	1.38
5	531	10.30	200	7.81	6.68	3.29
6	300	3.70	–	3.53	3.11	1.22
7	330	4.90	100	3.96	4.53	2.02
8	281	5.03	–	5.03	4.29	1.92
9	574	2.64	–	2.64	3.36	1.52
10	42	19.37	–	19.37	26.56	1.88
11	128	41.72	300	39.99	52.35	1.66
12	311	3.36	–	3.36	3.7	1.27
13	203	11.41	150	9.93	8.38	2.37
14	190	2.10	–	2.12	2.11	1.14
15	416	2.21	–	2.20	2.4	1.03
16	228	2.49	–	2.49	2.56	1.24
18	81	3.50	–	3.50	3.47	0.85
20	805	3.76	80	3.58	4.35	1.78
21	138	7.04	100	5.66	5.55	2.58

14.2.7 Variography

Variography has been used to analyze the spatial continuity within the mineralized zones and to determine appropriate estimation inputs to the interpolation process. The variogram modelling process followed by Cube was undertaken using Isatis software and involves the following steps:

- Calculate and model the omni-directional or downhole variogram to characterise the Nugget Effect.
- Systematically calculate orientated variograms in three dimensions to identify the plane of greatest continuity.
- Calculate a fan of variograms within the plane of greatest continuity to identify the direction of maximum continuity within the plane. Model the variogram in the direction of maximum continuity and the orthogonal directions.

Variography was undertaken on the 1 m cut composite data for the diamond drilling only and modelled for four variables which included:

- Cut Au (All Data)
- Indicator for Au >= 5
- Cut Au for Au >=5 (Selection = BZ)
- Cut Au for Au <5 (Selection = MZ)

In some instances, variography was undertaken on Gaussian transformed 1 m downhole high assay cut composite data. The Gaussian transformation was modelled in Isatis on the declustered 1m composite data. The Gaussian variogram model was back transformed and modelled to obtain the appropriate variogram model for interpolation of raw composite data.

Mineralization at both Amun and Ptah is generally moderate dipping to the east with a lesser proportion of flat dipping domains. Variograms were modelled for the most sampled domains at both project areas for the two dominant mineralization styles. At Amun, variograms were modelled for Domains 1 and 31 to represent dipping mineralization and Domain 17 for flat dipping mineralization. At Ptah, variograms were modelled for Domain 120 representing moderate to steep-dipping mineralization and Domain 103 for flat-dipping mineralization. Variogram modelling for the more sparsely sampled domains was difficult and not considered appropriate for use as the number of composite samples was often limited. Cube adopted the modelled variogram parameters for the remaining mineralized domains.

During grade estimation, the rotation of the variogram models was adjusted to follow the orientation of the search ellipsoid and better fit the orientation of each individual mineralized domain. This was achieved by estimating the dip and strike based on the orientation of a single DTM plane representing the general trend for each domain. Anisotropy in the minor direction was introduced in the variogram models at a ratio of typically 1:4.

Table 14.2-8 to Table 14.2-15 below summarize the variogram parameters with the data for Domain 1 displayed graphically in Figure 14.2 7 to Figure 14.2 12.

Table 14.2-8 Absolute Variogram Parameters – Cut Au

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	455	155	8	8	3.5	43	32	32	8	–	–	–	–
17	925	460	4.5	4.5	4.5	185	12	12	12	–	–	–	–
31	300	145	10	10	4	37	30	30	10	–	–	–	–
103	5.91	2.37	5	5	5	1.52	22	22	22	–	–	–	–
120	50.9	11.8	6	6	6	4.7	35	35	35	–	–	–	–

Table 14.2-9 Relative Variogram Parameters – Cut Au

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	0.697	0.237	8	8	3.5	0.066	32	32	8	–	–	–	–
17	0.589	0.293	4.5	4.5	4.5	0.118	12	12	12	–	–	–	–
31	0.622	0.301	10	10	4	0.077	30	30	10	–	–	–	–
103	0.603	0.242	5	5	5	0.155	22	22	22	–	–	–	–
120	0.755	0.175	6	6	6	0.070	35	35	35	–	–	–	–

Table 14.2-10 Absolute Variogram Parameters – BZ Indicator (Cut Au >=5)

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	0.07	0.085	10	10	3	0.04	60	60	7	–	–	–	–
17	0.1	0.1	12	12	12	0.06	25	25	25	–	–	–	–
31	0.06	0.057	8	8	4	0.04	35	35	8	–	–	–	–
103	0.065	0.02	3	3	3	0.015	15	15	15	–	–	–	–
120	0.08	0.03	4	4	4	0.034	10	10	10	–	–	–	–

Table 14.2-11 Relative Variogram Parameters – BZ Indicator (Cut Au >=5)

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	0.359	0.436	10	10	3	0.205	60	60	7	–	–	–	–
17	0.385	0.385	12	12	12	0.231	25	25	25	–	–	–	–
31	0.382	0.363	8	8	4	0.255	35	35	8	–	–	–	–
103	0.650	0.200	3	3	3	0.150	15	15	15	–	–	–	–
120	0.556	0.208	4	4	4	0.236	10	10	10	–	–	–	–

Table 14.2-12 Absolute Variogram Parameters – MZ Cut Au (Cut Au < 5)

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	0.6	0.5	10	10	5	0.3	25	25	10	0.14	100	100	12
17	0.75	0.15	4	4	4	0.4	12	12	12	–	–	–	–
31	0.4	0.6	10	10	5	0.5	30	30	10	–	–	–	–
103	0.7	0.3	4	4	4	0.55	22	22	22	–	–	–	–
120	0.8	0.3	4	4	4	0.54	18	18	18	–	–	–	–

Table 14.2-13 Relative Variogram Parameters – MZ Cut Au (Cut Au < 5)

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	0.390	0.325	10	10	5	0.195	25	25	10	0.091	100	100	12
17	0.577	0.115	4	4	4	0.308	12	12	12	–	–	–	–
31	0.267	0.400	10	10	5	0.333	30	30	10	–	–	–	–
103	0.452	0.194	4	4	4	0.355	22	22	22	–	–	–	–
120	0.488	0.183	4	4	4	0.329	18	18	18	–	–	–	–

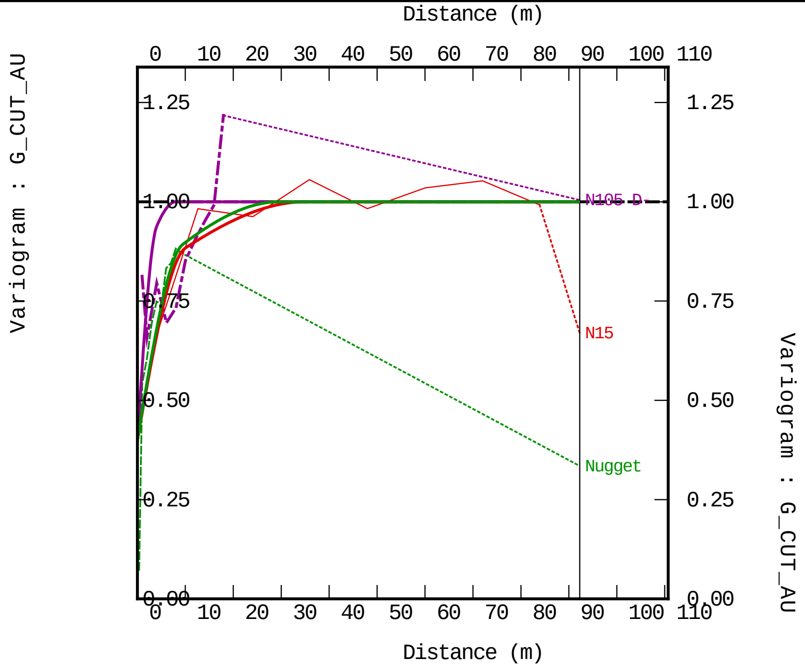
Table 14.2-14 Absolute Variogram Parameters – BZ Cut Au (Cut Au >=5)

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	1620	470	8	8	5	190	80	80	10	–	–	–	–
17	3085	285	4	4	4	450	30	30	30	–	–	–	–
31	1165	870	10	10	5	260	55	55	10	–	–	–	–
103	12.66	3.05	4	4	4	2.58	10	10	10	–	–	–	–
120	135.1	23.2	5	5	5	30.8	28	28	28	–	–	–	–

Table 14.2-15 Relative Variogram Parameters – BZ Cut Au (Cut Au >=5)

Domain	Nugget	Spherical 1				Spherical 2				Spherical 3			
		Sill	Major	Semi	Minor	Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	0.711	0.206	8	8	5	0.083	80	80	10	–	–	–	–
17	0.808	0.075	4	4	4	0.118	30	30	30	–	–	–	–
31	0.508	0.379	10	10	5	0.113	55	55	10	–	–	–	–
103	0.692	0.167	4	4	4	0.141	10	10	10	–	–	–	–
120	0.714	0.123	5	5	5	0.163	28	28	28	–	–	–	–

Figure 14.2-7 Domain 1 Variogram – Gaussian Transformed 1 m DH Composites – Cut Au




```
Isatis
Data/amun_dh_comps_1m_1
- Variable #1 : G_CUT_AU
Variogram : in 3 direction(s)
D1  : N15
      Angular tolerance =  90.00
      Lag =  12.000m, Count = 8 lags, Tolerance = 50.00%
      Vertical Slicing =  3.000m
D2  : N105 D-55
      Angular tolerance =  40.00
      Lag =  2.000m, Count = 10 lags, Tolerance = 50.00%
D3  : Nugget
      Angular tolerance =  90.00
      Lag =  1.000m, Count = 10 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N15.00 Dip=35.00 (Geologist Plane)
S1  - Nugget effect, Sill =        0.4
S2  - Spherical - Range = 4.000m, Sill =       0.4
      Directional Scales = (      10.000m,      10.000m,       4.000m)
S3  - Spherical - Range = 8.000m, Sill =       0.2
      Directional Scales = (      35.000m,      35.000m,       8.000m)
```

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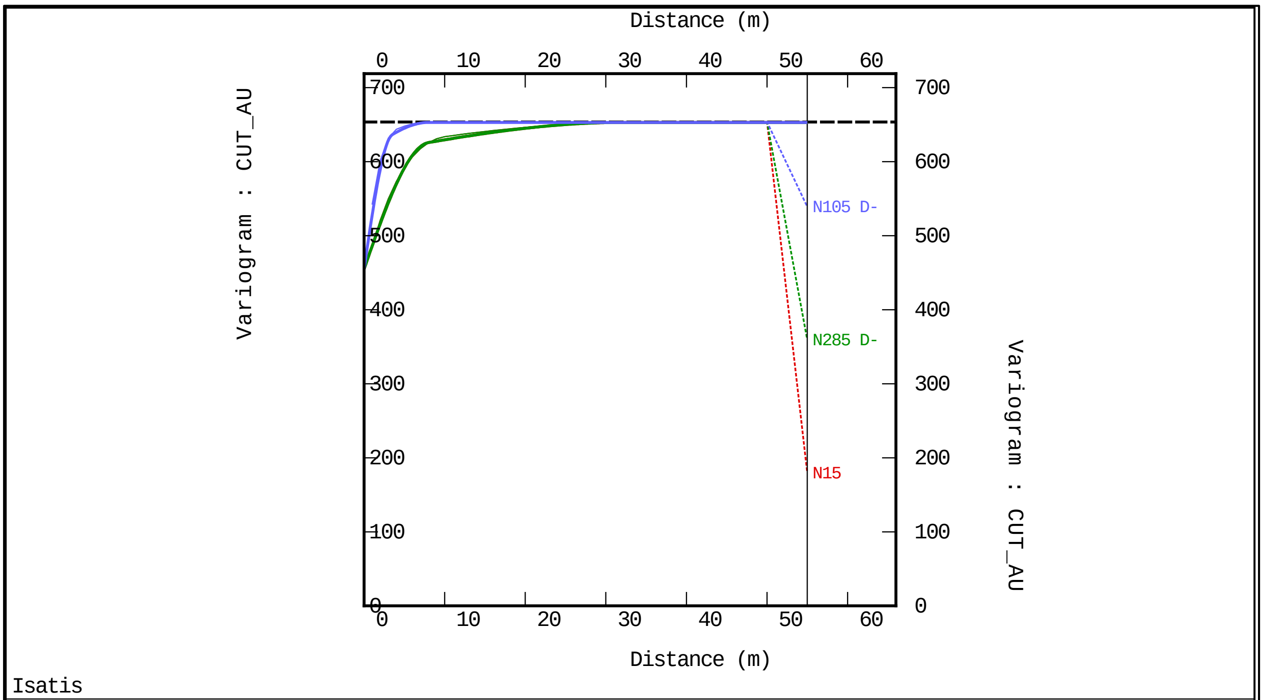
Sukari_2015_02

Figure 14.2-8 Domain 1 Variogram – Back Transformed 1 m DH Composites – Cut Au



Data/amun_dh_comps_1m_1
- Variable #1 : CUT_AU
Variogram : in 3 direction(s)
D1 : N15
Lag = 1.000m, Count = 50 lags
D2 : N285 D-35
Lag = 1.000m, Count = 50 lags
D3 : N105 D-55
Lag = 1.000m, Count = 50 lags
Model : 3 basic structure(s)
Global rotation = Azimuth=N15.00 Dip=35.00 (Geologist Plane)
S1 - Nugget effect, Sill = 455
S2 - Spherical - Range = 3.500m, Sill = 155
Directional Scales = (8.000m, 8.000m, 3.500m)
S3 - Spherical - Range = 8.000m, Sill = 43
Directional Scales = (32.000m, 32.000m, 8.000m)

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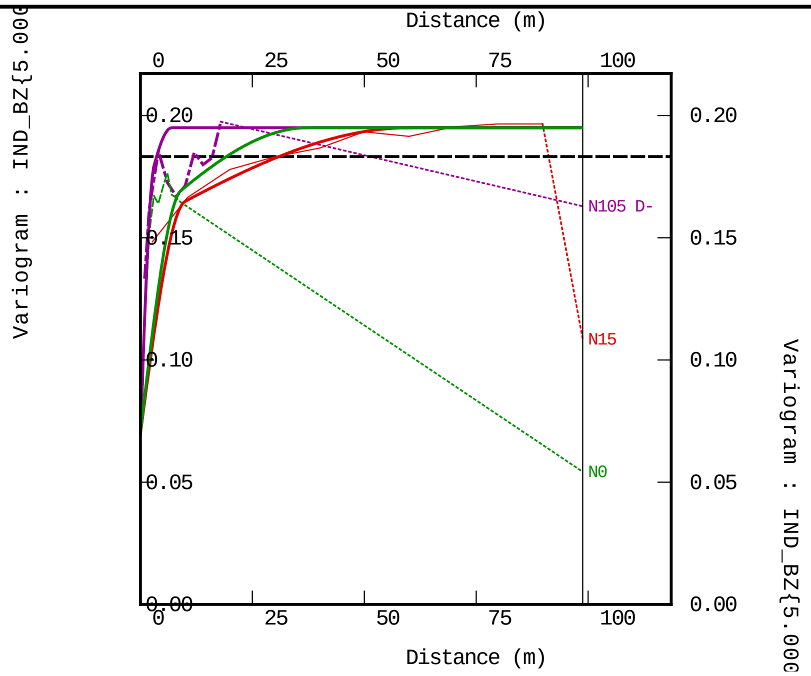
Sukari_2015_02

Figure 14.2-9 Domain 1 Variogram – Raw 1 m DH Composites – BZ Indicator (Cut Au > 5)

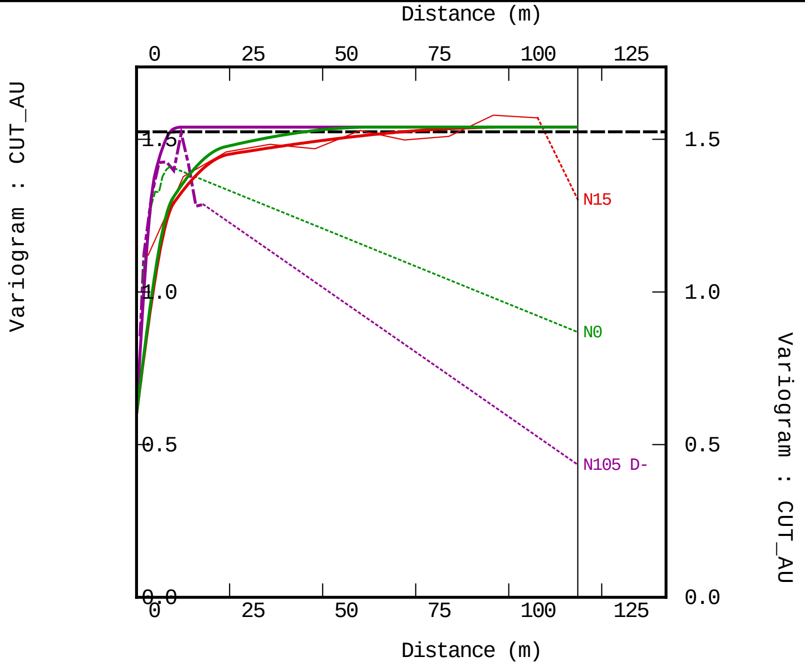


Isatis

Data/amun_dh_comps_1m_1
- Variable #1 : IND_BZ{5.000000}
Variogram : in 3 direction(s)
D1 : N15
Angular tolerance = 90.00
Lag = 10.000m, Count = 10 lags, Tolerance = 50.00%
Vertical Slicing = 3.000m
D2 : N105 D-55
Angular tolerance = 40.00
Lag = 2.000m, Count = 10 lags, Tolerance = 50.00%
D3 : N0
Angular tolerance = 90.00
Lag = 1.000m, Count = 10 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N15.00 Dip=35.00 (Geologist Plane)
S1 - Nugget effect, Sill = 0.07
S2 - Spherical - Range = 3.000m, Sill = 0.085
Directional Scales = (10.000m, 10.000m, 3.000m)
S3 - Spherical - Range = 7.000m, Sill = 0.04
Directional Scales = (60.000m, 60.000m, 7.000m)

mikem
Jul 16 2015
Sukari_2015_02

Figure 14.2-10 Domain 1 Variogram – Raw 1 m DH Composites – MZ Cut Au (Cut Au < 5)

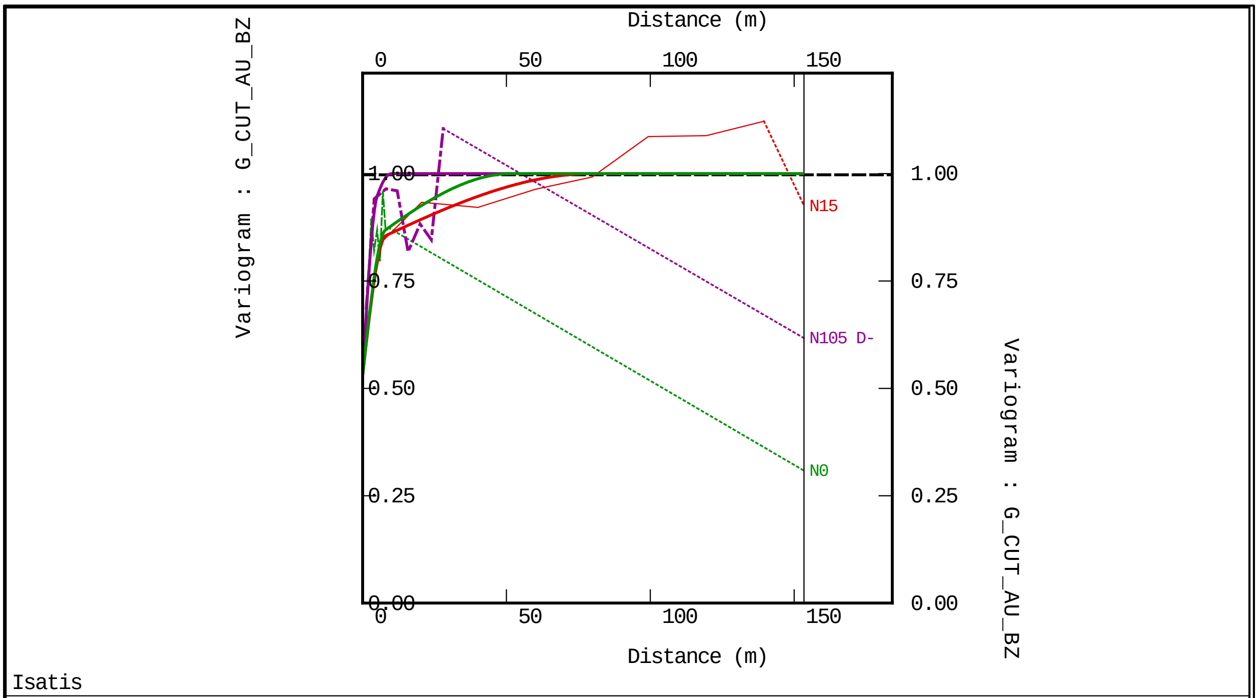


Isatis

Data/amun_dh_comps_1m_1(sel MZ)
- Variable #1 : CUT_AU
Variogram : in 3 direction(s)
D1 : N15
Angular tolerance = 90.00
Lag = 12.000m, Count = 10 lags, Tolerance = 50.00%
Vertical Slicing = 3.000m
D2 : N105 D-55
Angular tolerance = 40.00
Lag = 2.000m, Count = 10 lags, Tolerance = 50.00%
D3 : N0
Angular tolerance = 90.00
Lag = 1.000m, Count = 10 lags, Tolerance = 50.00%
Model : 4 basic structure(s)
Global rotation = Azimuth=N15.00 Dip=35.00 (Geologist Plane)
S1 - Nugget effect, Sill = 0.6
S2 - Spherical - Range = 5.000m, Sill = 0.5
Directional Scales = (10.000m, 10.000m, 5.000m)
S3 - Spherical - Range = 10.000m, Sill = 0.3
Directional Scales = (25.000m, 25.000m, 10.000m)
S4 - Spherical - Range = 12.000m, Sill = 0.14
Directional Scales = (100.000m, 100.000m, 12.000m)

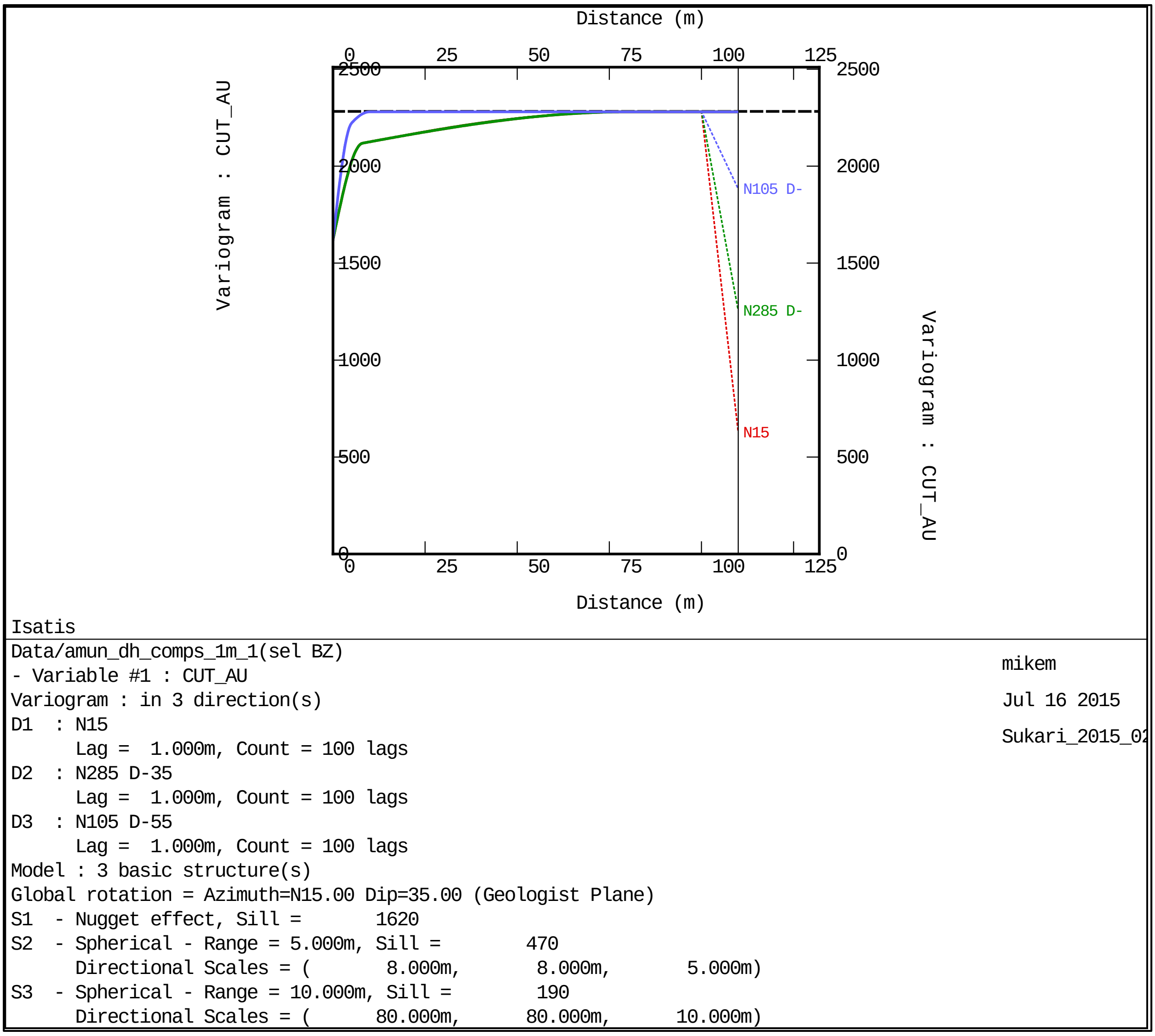
mikem
Jul 16 2015
Sukari_2015_02

Figure 14.2-11 Domain 1 Variogram – Gaussian Transformed 1 m DH Composites – BZ Cut Au (Cut Au > 5)




```
Data/amun_dh_comps_1m_1(sel BZ)                                             mikem
- Variable #1 : G_CUT_AU_BZ
Variogram : in 3 direction(s)                                               Jul 16 2015
D1  : N15
      Angular tolerance =  90.00                                            Sukari_2015_0
      Lag =  20.000m, Count = 8 lags, Tolerance = 50.00%
      Vertical Slicing =  3.000m
D2  : N105 D-55
      Angular tolerance =  40.00
      Lag =  4.000m, Count = 8 lags, Tolerance = 50.00%
D3  : N0
      Angular tolerance =  90.00
      Lag =  1.000m, Count = 10 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N15.00 Dip=35.00 (Geologist Plane)
S1  - Nugget effect, Sill =       0.53
S2  - Spherical - Range = 5.000m, Sill =       0.3
      Directional Scales = (       8.000m,       8.000m,       5.000m)
S3  - Spherical - Range = 10.000m, Sill =      0.17
      Directional Scales = (      80.000m,      80.000m,      10.000m)
```

Figure 14.2-12 Domain 1 Variogram – Back Transformed 1 m DH Composites – BZ Cut Au (Cut Au > 5)



14.2.8 Block Model Definition

A number of criteria including data spacing, geometry of mineralized domains and volume fill were the primary considerations taken into account when selecting an appropriate estimation block size. Data spacing within the mineralized domains varies from approximately 12.5 m to 50 m spaced sections.

Cube considers it good geostatistical practice to use an estimation parent cell size that approaches the data spacing where possible, whilst at the same time being mindful of potential mine design and selectivity implications. Cube reviewed the data spacing and conceptual SMU relative to the mineralized zones and determined that an estimation block size of 8 mN x 8 mE x 8 mRL as being appropriate. The estimation parent block was sub-blocked to 1 mN x 1 mE x 1 mRL, to improve the volume representation of the block model within the narrow mineralized zones.

The definitions for the block models *"suk_amun_july2015.mdl"* and *"suk_ptah_july2015.mdl"* are summarized Table 14.2-16 and Table 14.2-17 respectively. The block model attributes and descriptions are summarized in Table 14.2-18.

Table 14.2-16 Block Model Definition – suk_amun_july2015.mdl

	Minimum	Maximum	Model Extent (m)
Easting	10,300	11,004	704
Northing	9,800	10,704	904
RL	200	1,000	800
Parent Cell X m	8	**Min Sub-Cell X m**	1.0
Parent Cell Y m	8	**Min Sub-Cell Y m**	1.0
Parent Cell Z m	8	**Min Sub-Cell Z m**	1.0

Table 14.2-17 Block Model Definition – suk_ptah_ july 2015.mdl

	Minimum	Maximum	Model Extent (m)
Easting	10,300	11,004	704
Northing	10,624	11,904	1,280
RL	200	1,000	800
Parent Cell X m	8	**Min Sub-Cell X m**	1.0
Parent Cell Y m	8	**Min Sub-Cell Y m**	1.0
Parent Cell Z m	8	**Min Sub-Cell Z m**	1.0

Table 14.2-18 Block Model Attributes

Name	Type	Background	Description
x	–	–	X Block Centroid
y	–	–	Y Block Centroid
z	–	–	Z Block Centroid
au	float	–99	Final Au grade for reporting based on “au_orig” (g/t)
au_orig	float	–99	Original Au grade for estimated using a single Indicator weighting method (g/t)
au_ok	float	–99	Au check estimate using Ordinary Kriging
au_idw2	float	–99	Au check estimate using Inverse Distance Squared
au_nn	float	–99	Au check estimate using Nearest Neighbour
bz_ppn	float	–99	BZ Indicator proportion estimated by Ordinary Kriging
density	float	2.66	Assigned Insitu Bulk Density
depletion	integer	1	0=UG mined, 1=Insitu, 2=Pillar
weathering	integer	4	0=Air, 1=Overburden, 2=Saprolite, 3=Transitional, 4=Fresh
lithology	character	BKGR	Rock Lithology
classification	integer	4	1=Measured 2=Indicated 3=Inferred 4=Unclassified
domain	integer	0	Mineralization Domain Code
pass	float	0	Estimation search pass
avd	float	0	Estimation quality – Average distance to composites
dns	float	0	Estimation quality – Distance to nearest composite
kv	float	0	Estimation quality – Kriging Variance
ns	float	0	Estimation quality – Number of composites
sor	float	0	Estimation quality – Slope of regression

14.2.9 Grade Interpolation

Cube utilized a single Indicator weighted Kriging method (IK) to estimate gold for each of the mineralization domains within the given Amun (suk_amun_july2015) and Ptah (suk_ptah_july2015) block models. Variogram models and search neighbourhoods were used to interpolate 1 m composite data. All block estimates were based on interpolation into

8 mN x 8 mE x 8 mRL parent cells with sub-celling to 1 mN x 1 mE x 1 mRL. Block discretization points were set to 4 (Y) x 4 (X) x 4 (Z) points. Estimation was constrained to within the modelled mineralization domains.

Cube utilized histograms, log-transformed probability plots, percentile analysis and visual graphical analysis to select the grade indicator of 5 g/t for gold to determine two populations above (BZ) and below (MZ) the indicator grade for each domain.

The IK method utilized for the estimations, involved the following steps:

- Flag the composited data into indicator values of "1" (BZ population for data >= 5 g/t) or "0" (MZ population for data < 5 g/t).
- Perform Ordinary Kriging of the indicator variable (BZ_Ind) to produce a value between 1 and 0 representing the proportion above 5 g/t.
- Perform Ordinary Kriging of the filtered composite data >= 5 g/t to estimate the BZ population gold grade (BZ_Cut_Au).
- Perform Ordinary Kriging of the filtered composite data < 5 g/t to estimate the MZ population gold grade (MZ_Cut_Au).
- Calculate the final single indicator weighted gold grade.

During estimation, a local rotation was applied to both the variogram model and search ellipsoid. The orientation of this local rotation was controlled by the trend of individual DTM surfaces modelled to reflect the general trend of each domain. The rotations were interpolated into the volume intermediate to and beyond the controlling surfaces for use in the grade interpolation. The local rotations were used to orient both the variogram model and search neighbourhood.

The IK method was selected in order to produce a less smoothed result than if standard Ordinary Kriging (OK) was used. Localized higher grade domains are clearly present from exposure underground and IK has been used to help honour these observations.

Ordinary Kriging, Inverse Distance Squared and Nearest Neighbour check estimates were estimated for comparison and validation of the IK estimates.

Table 14.2-19 Block Model Interpolation Parameters

Attribute	Discretization			Min Samp	Max Samp	Search Radii (m)			Isatis Rotation		
	x	y	z			major	semi	minor	A	+X	-Z
BZ_Ind.	4	4	4	8	16	100	100	25	Dynamic Local Rotation with Major Axis horizontal and Parallel to Strike of Controlling Surfaces		
MZ_Cut_Au	4	4	4	4	24	100	100	25			
BZ_Cut_Au	4	4	4	10	28	160	160	40			

Note that for the BZ_Cut_Au grade estimation, the uncut 1 m composite data was within a distance of 5 m from the composite point and beyond this the cut value was used (Section 14.2.6.1).

14.2.9.1 Search Neighbourhood Analysis

Cube attempted to characterise the spatial relationship of the data using variography and has sought to implement search strategies aimed at producing a robust block estimate whilst at the same time minimising estimation error and conditional biases. Cube optimizes search neighbourhoods by undertaking Kriging Neighbourhood Analysis (KNA), analysing estimation quality data such as Slope of Regression and Kriging weights for various search neighbourhoods, in combination with other primary considerations such as data spacing, geometry of mineralized domains and variogram models.

As data spacing at the Sukari deposit was variable throughout the mineralized domains, KNA was undertaken on blocks representing poor and well informed neighbourhoods. The aim of these tests is to optimize the kriging search neighbourhood and maximise the quality of the kriging when dealing with a non-exhaustive dataset.

14.2.10 Bulk Density

The bulk density for all mineralization associated with the Underground Mineral Resource was assigned as 2.66 t/m^3. This is based on rock types defined as primary porphyry sulphide sub-domains in the January 2014 Mineral Resource estimation (Smith et al., 2014).

14.2.11 Model Validation

The Amun and Ptah models were validated statistically and graphically for all estimated domains. Spatial validation of the block models was also undertaken, comparing the block estimate values against composite data on a section by section basis. Additional check estimates were undertaken using Ordinary Kriging, Inverse Distance Squared (IDW2) and Nearest Neighbour. In summary, statistical, graphical and spatial validations of the Amun and Ptah models demonstrated robust model outcomes.

The reported mean of interpolated gold grades for each domain is tabulated against the declustered mean composite grade and the mean grade for the OK, IDW2 and NN check estimates in Table 14.2-20 and Table 14.2-21. Although the comparison between the estimated and composite mean is not strictly comparable due to data clustering and volume influences, they do provide a useful validation tool in detecting any major biases requiring further spatial investigation, whilst providing global comparison of input composite grade and the estimated block grade.

Figure 14.2-13 Cross-section at 10400 N Looking North

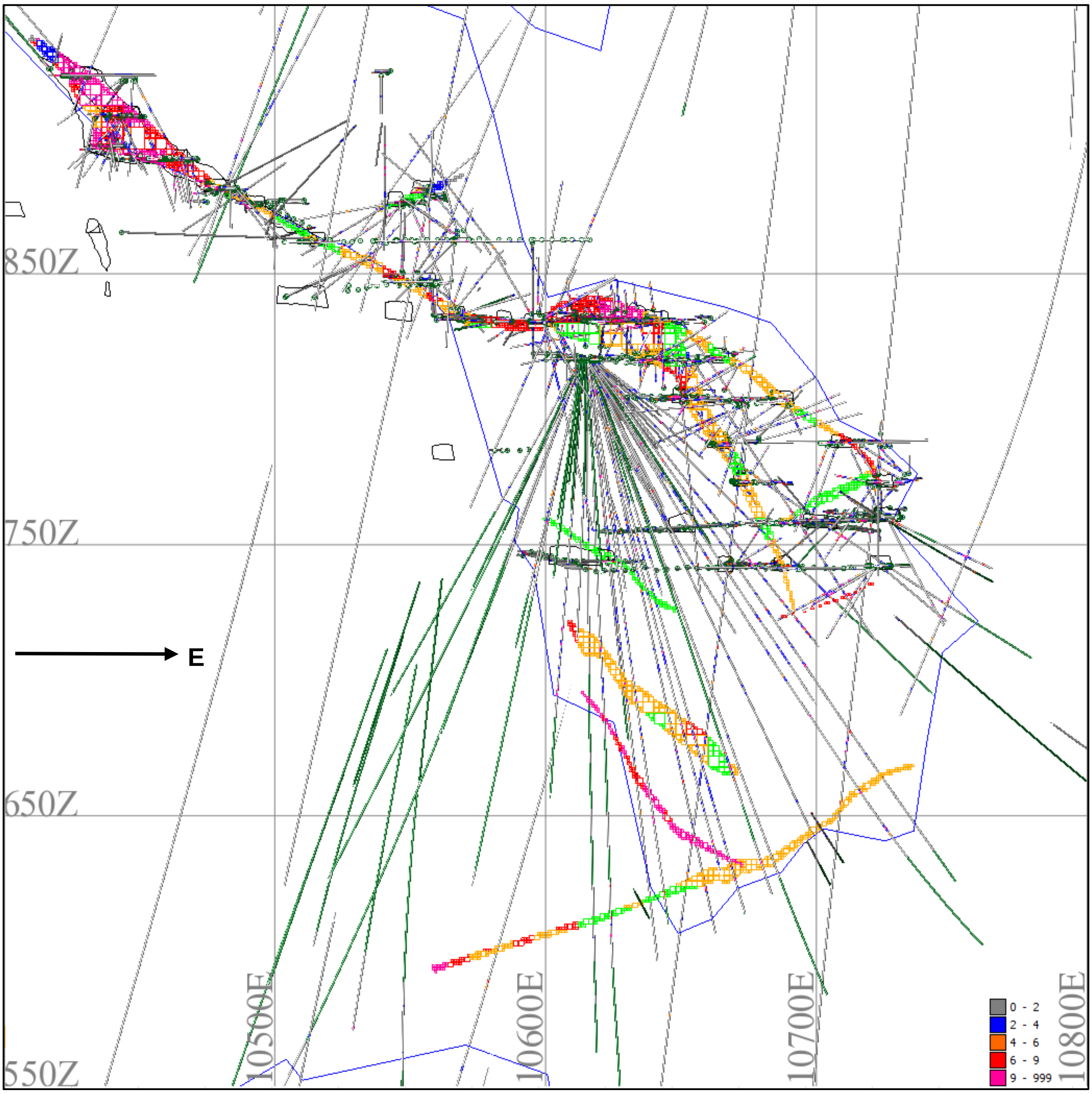


Table 14.2-20 Amun – Estimation Method Comparison

Domain	Au Final Estimate by IK	Declustered Composite Au	Au Check Estimate by OK	Au Check Estimate by IDW2	Au Check Estimate by NN
1	8.06	7.34	8.29	8.23	8.10
2	4.52	4.87	4.25	4.66	4.23
3	6.38	6.08	6.90	7.00	6.60
4	10.35	10.46	10.68	10.00	11.54
5	10.75	12.00	11.22	11.36	14.96
6	21.83	13.31	16.29	16.20	21.60
7	3.52	3.27	3.60	3.88	3.73
8	7.14	6.48	5.83	6.21	8.43
9	6.37	5.68	5.99	6.42	9.68
10	22.51	24.64	24.26	20.67	18.48
11	11.19	10.94	10.00	10.03	11.10
12	13.05	14.00	14.01	15.57	14.26
13	6.87	7.70	6.64	6.64	6.26
14	7.59	7.80	7.31	7.12	8.25
15	6.59	5.48	6.21	5.90	7.43
16	5.55	4.64	6.34	6.00	6.28
17	15.62	11.80	15.51	13.80	16.93
18	4.69	4.05	4.60	4.40	5.31
19	20.17	17.12	19.73	19.29	32.36
20	4.67	3.98	3.23	4.01	6.49
21	11.48	10.11	10.57	11.07	10.70
31	6.70	6.43	6.60	6.93	7.50
32	3.06	2.73	2.94	3.01	2.92
33	3.94	3.84	3.82	4.12	3.26
34	3.20	3.38	3.21	3.12	3.26
35	3.69	3.63	3.77	3.84	3.76
36	6.82	7.48	8.81	8.18	9.03

Table 14.2-21 Ptah – Estimation Method Comparison

Domain	Au Final Estimate by IK	Declustered Composite Au	Au Check Estimate by OK	Au Check Estimate by IDW2	Au Check Estimate by NN
101	3.31	4.76	3.52	2.83	3.44
102	3.18	3.10	3.04	3.36	3.59
103	2.39	2.53	2.46	2.38	2.58
104	4.74	4.36	4.46	4.99	5.65
105	7.50	6.68	7.51	6.54	9.96
106	3.07	3.11	3.04	3.13	3.85
107	3.79	4.53	3.89	3.77	4.14
108	4.85	4.29	4.74	4.87	5.80
109	2.98	3.36	2.99	2.92	3.34
110	25.20	26.56	25.84	23.23	20.93
111	44.37	52.35	45.86	43.50	37.50
112	3.58	3.70	3.49	3.45	3.90
113	9.63	8.38	9.59	9.92	9.30
114	2.03	2.11	1.98	2.00	1.62
115	2.35	2.40	2.28	2.26	2.52
116	2.59	2.56	2.59	2.65	2.88
118	3.53	3.47	3.43	3.56	3.05
120	4.05	4.35	4.04	3.96	3.92
121	5.56	5.55	6.51	5.36	5.81

Swath plots (grade trend profiles) showing the estimated tonnes, grade, number of composites and mean cut composite grade (tabulated by Northing) were created for all domains. The limitations of this comparison should be kept in mind when drawing conclusions; however there is generally good correlation between the block estimate and the declustered composite mean. As expected, the estimated grade is more smoothed compared to the often variable composite mean grades. The greatest differences occur in poorly sampled areas and where the composites display high degrees of local variation. Validation swath plots for the three largest domains at Amun and Ptah are presented in Figure 14.2-14 to Figure 14.2-19.

Figure 14.2-14 Swath Plot by Northing – Amun Domain 1

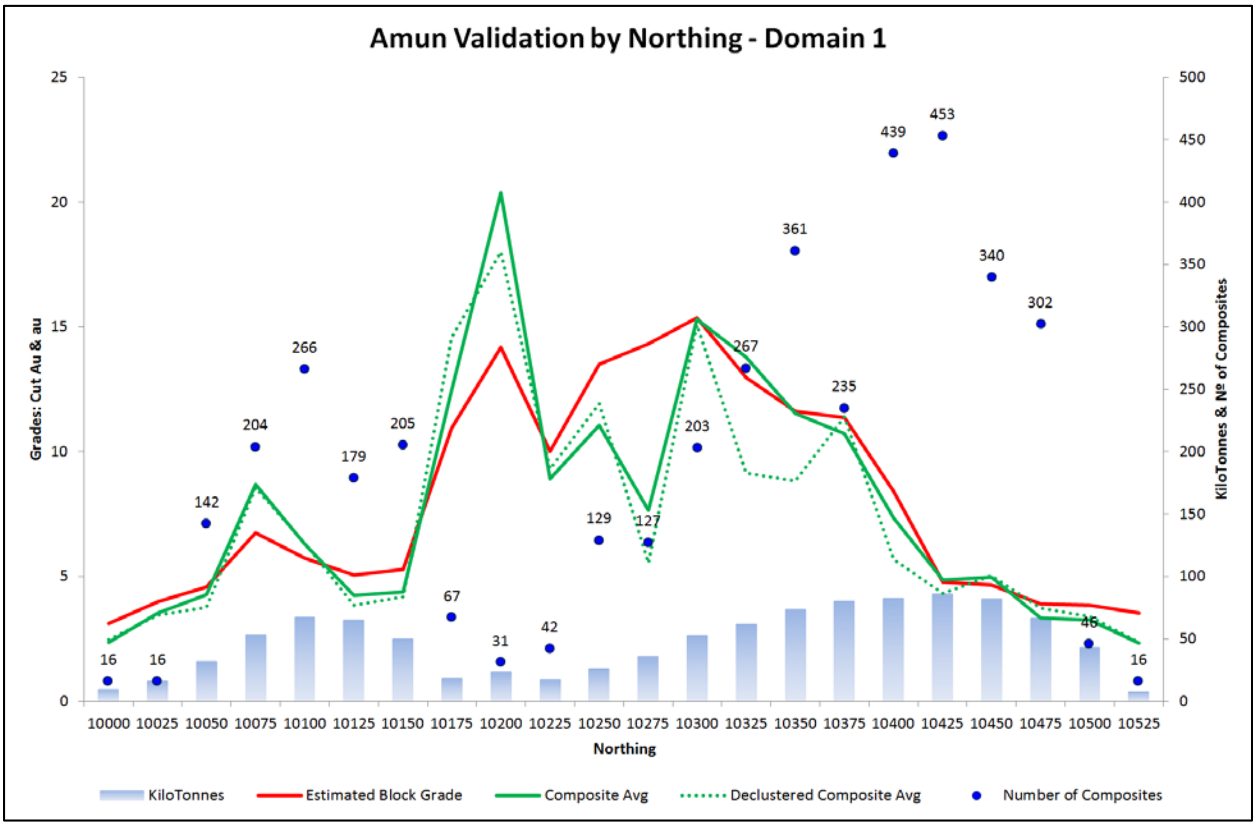


Figure 14.2-15 Swath Plot by Northing – Amun Domain 31

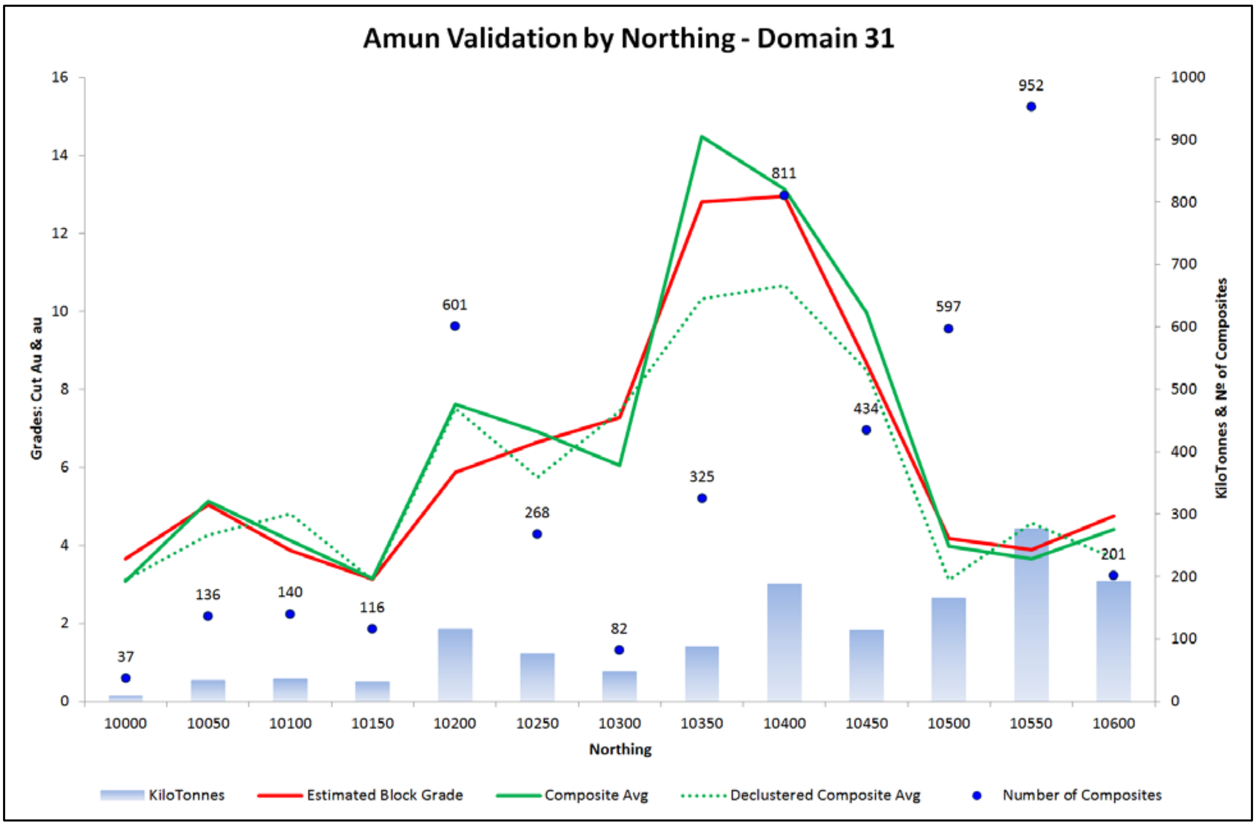


Figure 14.2-16 Swath Plot by Northing – Amun Domain 32

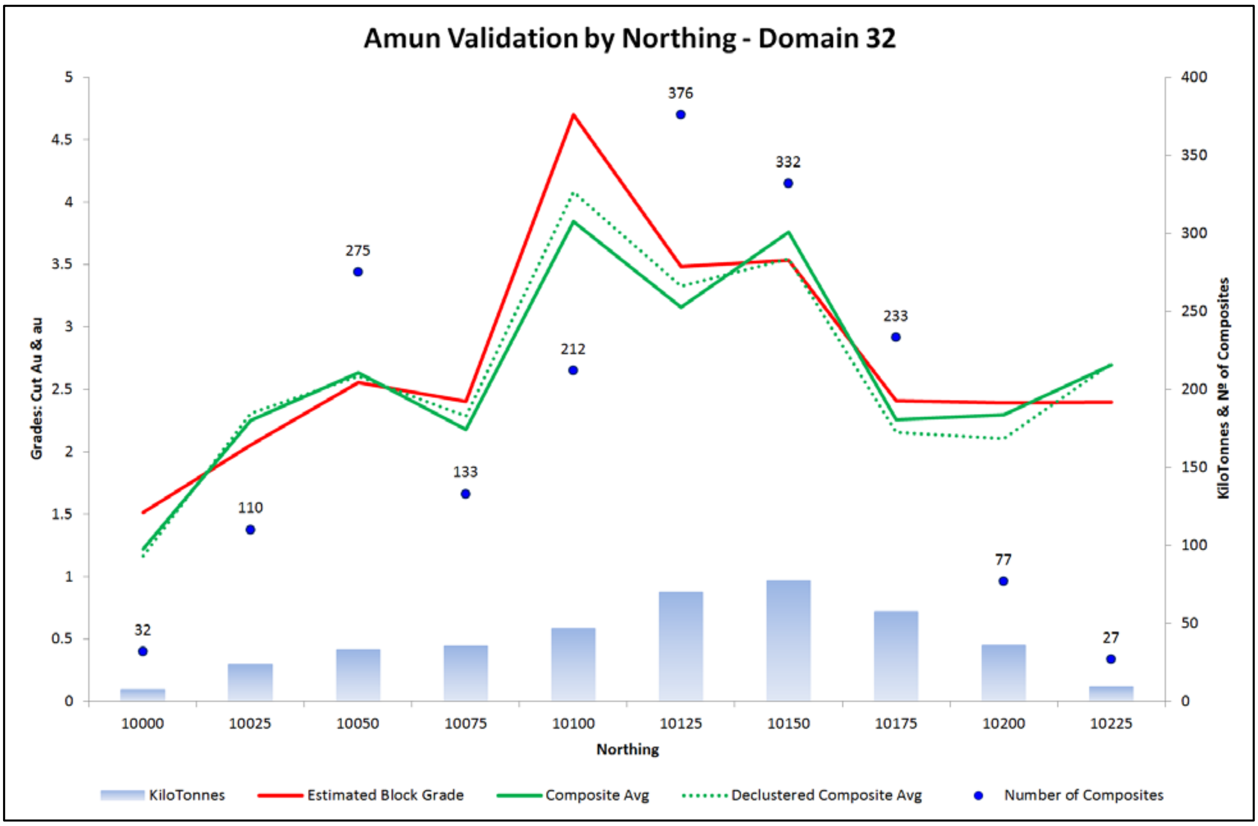


Figure 14.2-17 Swath Plot by Northing – Ptah Domain 102

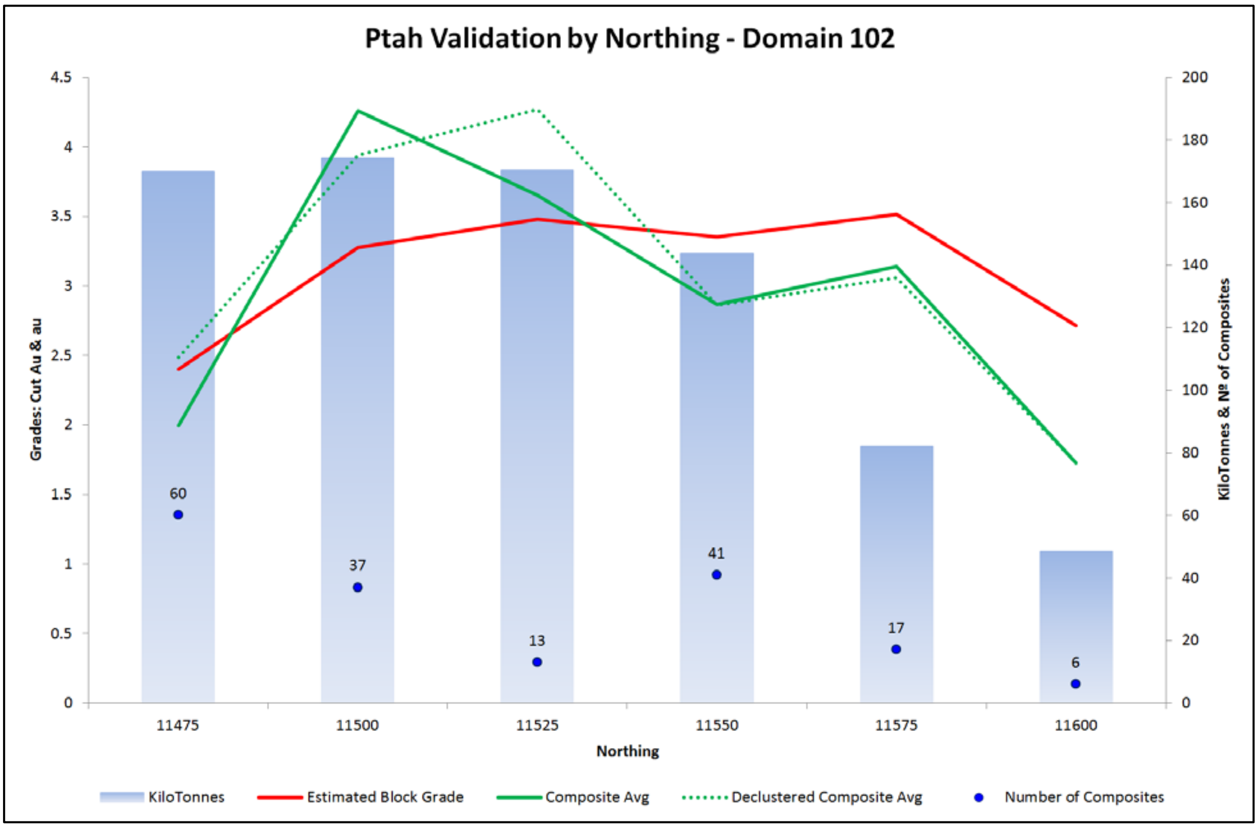


Figure 14.2-18 Swath Plot by Northing – Ptah Domain 103

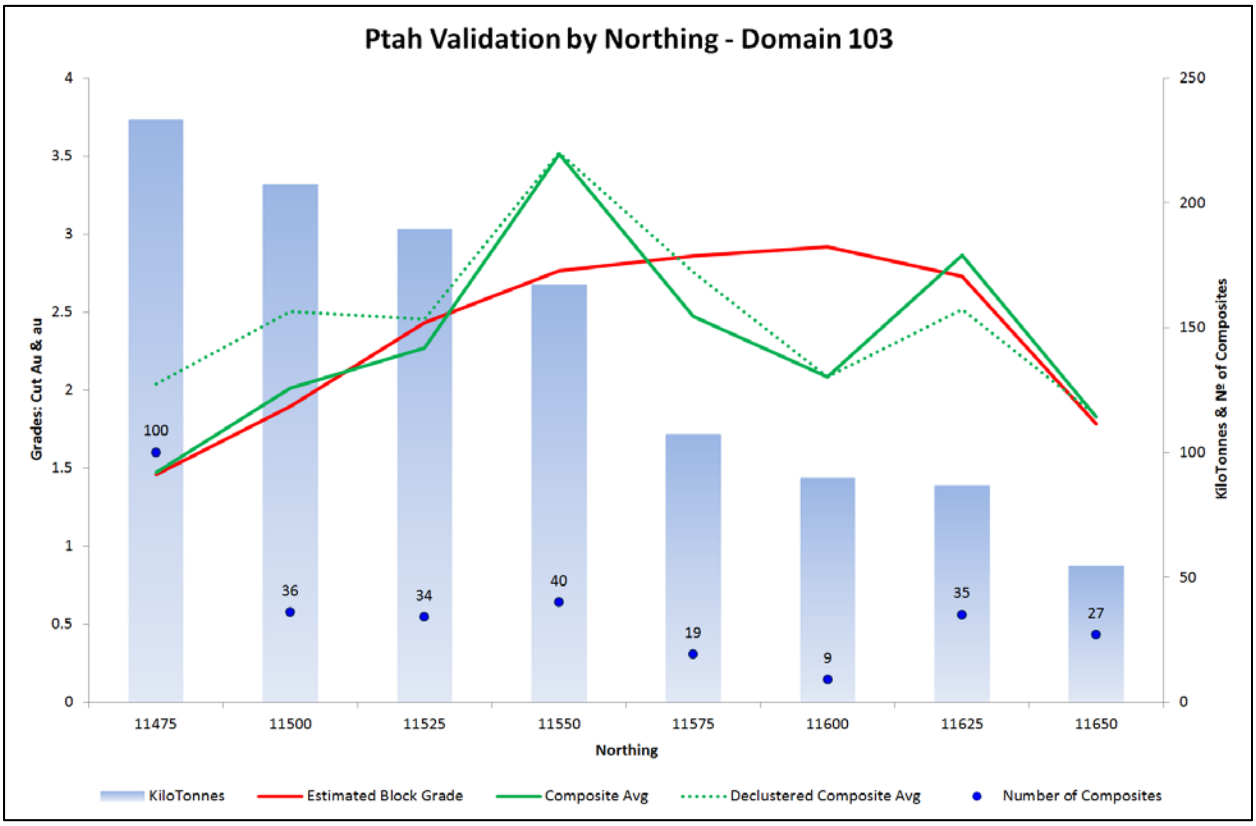
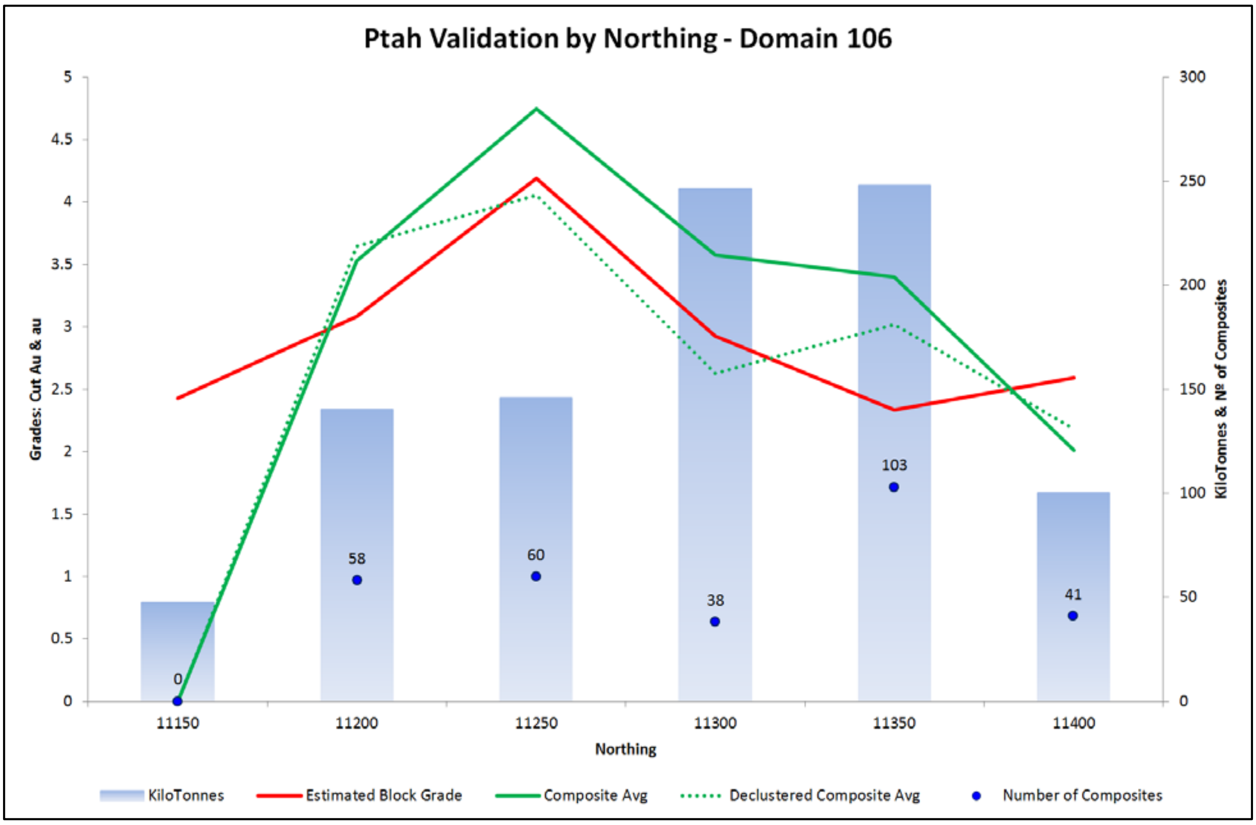


Figure 14.2-19 Swath Plot by Northing – Ptah Domain 106



14.2.12 Mining Depletion

Underground mine workings were supplied to Cube as three-dimensional (3D) wireframes for the level development and stopes current to 30 June 2015. The stope wireframes have been generated using "Cavity Monitoring System" equipment and the output wireframes are not typically valid for use as a block model constraint. Therefore the stope 3D wireframes have been sliced and the polygon outlines rewireframed. The resultant wireframes are therefore easily validated and honour the original stope volume. All 3D wireframes were combined into the file "*all_dev_june2015.dtm*" and all stope 3D wireframes combined into the file "*all_stope_june2015.dtm*".

The final 3D wireframes were used to flag the block model. The block model attribute "*depletion*" was flagged as "*1*" for insitu and "*0*" for mined. The block model attribute "*depletion_type*" was flagged with the mining type, which was either "*insitu*", "*development*" or "*stope*".

Domains 31, 32 and 35 have extensive stope depletion with a small proportion of mineralized blocks that remain in situ but are fringing the stope voids. These blocks denote material that is not going to be accessible from underground mining or represent slight local disparities between the mineralization interpretation and true mineralized boundaries. Either way, two-dimensional (2D) long section polygon outlines have been used to flag the block model attribute "*depletion*" as "*2*" for inaccessible pillar. This resource update has not taken into account any blocks flagged as "2" nor any remaining broken stocks or backfill at the time of reporting.

14.2.13 Porphyry Depletion

A 3D andesite porphyry wireframe was created by Cube for the Ptah resource area (ptah_an_interp_2015.dtm). This wireframe was used to flag the block model attribute "*lithology*" as "*AN*". The porphyries are barren and are interpreted to postdate the mineralization and therefore the block model gold grade (Au) was reset to 0.01 g/t within the flagged blocks. The block model field "*au_orig*" retains the original estimated values prior to porphyry flagging and resetting.

The andesite porphyry encountered within the Amun area has not been modelled. The porphyry appears to occur as thin swarms and do not materially impact the mineralization. Where they are coincident to the mineralization, they have been included as internal dilution.

14.2.14 Mineral Resource Classification

The mineralized domains defining the underground Mineral Resource at Sukari are of sufficient grade, geological continuity and drill density to support the classification criteria of Measured, Indicated and Inferred Mineral Resources in accordance with the NI 43-101 guidelines. Cube assessed the confidence levels on a range of criteria when determining the appropriate category and included:

- Geological, grade and volume continuity
- Drill data density, spacing and quality
- Estimation methodology
- Kriging quality (Slope of Regression, local estimation bias)
- Reliability of supplied depletion surfaces

As with any non-rigidly defined classification there will always be some blocks within categories that depart from the defined criteria. It is Cube's view that the final outcome must reflect a practical combination of both geological knowledge and estimation quality parameters that may be more numerical in nature. This approach to classification aims to avoid creating a complex numerically based "mosaic" distribution of classified blocks.

The primary criterion for Measured Mineral Resources is defined by a drill spacing of at least 20 m x 20 m. In addition, Measured Mineral Resources were confined to the interpreted mineralization defined by underground mine development and therefore has the least amount of risk associated with geological interpretation and continuity.

Indicated Mineral Resources are defined as areas outside the Measured Mineral Resource and defined by approximately 20 m x 20 m drill spacing. Inferred Mineral Resources include all remaining estimated mineralization defined by a drill spacing of approximately 50 m x 50 m.

Figure 14.2-20 and 14.2-12 show Mineral Resource classifications for Amun and Ptah, respectively.

Figure 14.2-20 Amun Mineral Resource Classification – Long Section Looking East

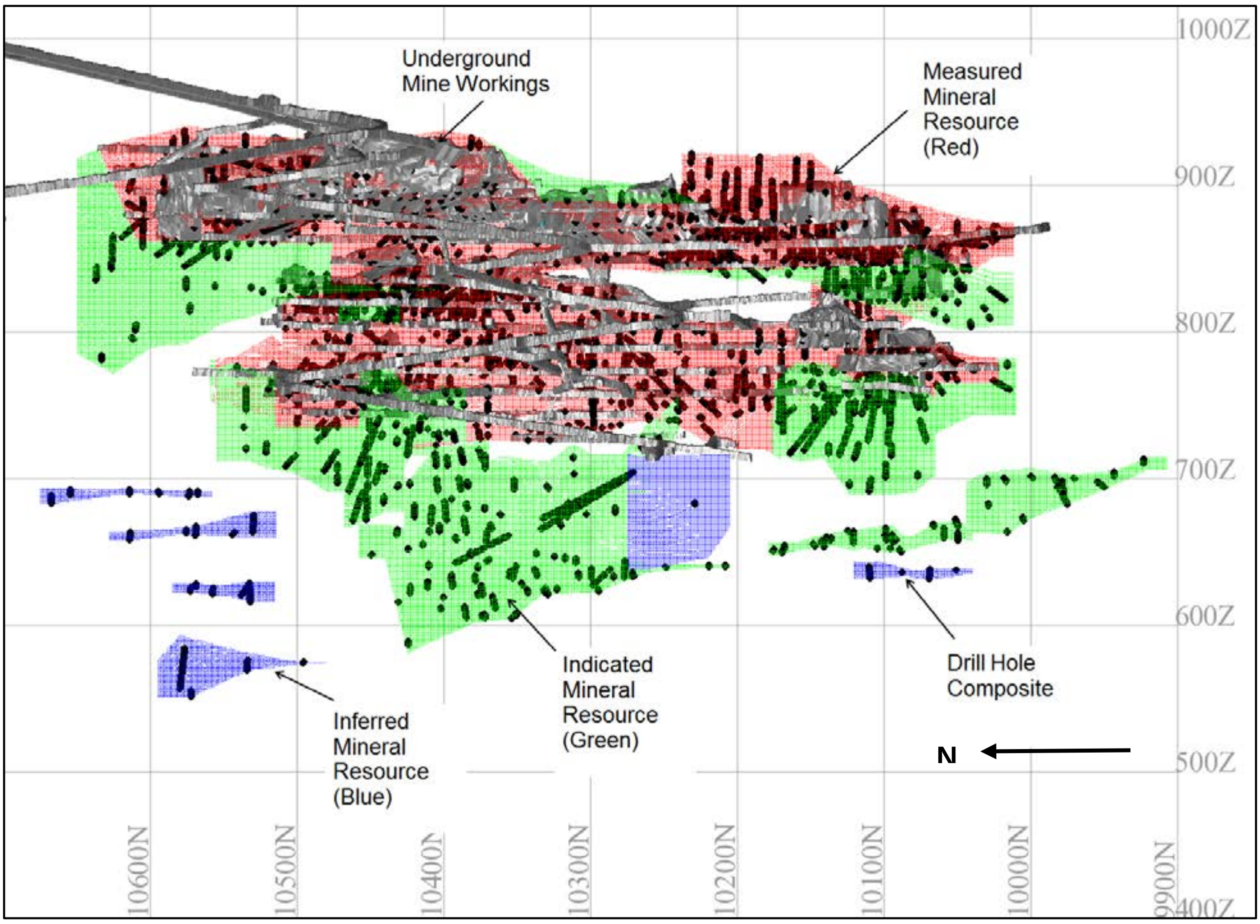
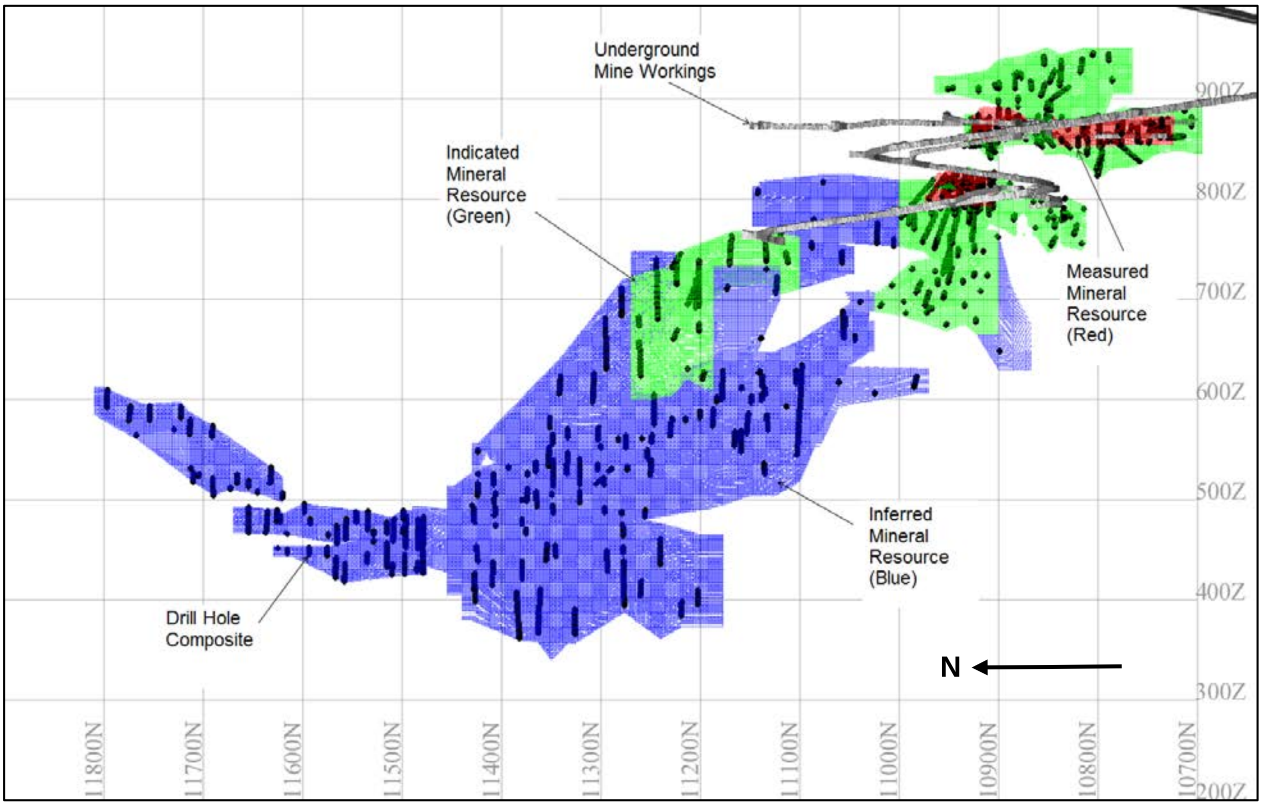


Figure 14.2-21 Ptah Mineral Resource Classification – Long Section Looking East



14.2.15 Mineral Resource Statement

The estimate of underground Mineral Resources is current as of 30 June 2015. The total underground Sukari Mineral Resource is tabulated in Table 14.2-22. Reported above 2 g/t cut-off and within the interpreted mineralized domains, the total is also sub-tabulated for the two underground zones, Amun and Ptah, in Table 14.2-23 and Table 14.2-24, respectively.

Table 14.2-22 Total Underground Sukari Mineral Resource at a 2 g/t Au Cut-off

Resource Classification	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	1.85	6.5	0.39
Indicated	2.82	7.0	0.63
Total Measured and Indicated	4.67	6.8	1.02
Inferred	6.97	5.6	1.24

Notes to table:

- Totals may not equal the sum of the components due to rounding adjustments.
- Mineral Resources are reported inclusive of those resources converted to Proven and Probable Mineral Reserves.

Table 14.2-23 Updated Amun Mineral Resource at a 2 g/t Au Cut-off

Resource Classification	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	1.68	6.8	0.36
Indicated	1.40	7.9	0.36
Total Measured and Indicated	3.08	7.3	0.72
Inferred	0.32	11.6	0.12

Notes to table:

- Totals may not equal the sum of the components due to rounding adjustments.
- Mineral Resources are reported inclusive of those resources converted to Proven and Probable Mineral Reserves.

Table 14.2-24 Updated Ptah Mineral Resource at a 2 g/t Au Cut-off

Resource Classification	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	0.17	4.0	0.02
Indicated	1.42	6.1	0.28
Total Measured and Indicated	1.59	5.9	0.30
Inferred	6.65	5.3	1.13

Notes to table:

- Totals may not equal the sum of the components due to rounding adjustments.
- Mineral Resources are reported inclusive of those resources converted to Proven and Probable Mineral Reserves.

The Mineral Resource has demonstrated economic viability. All tonnage, grade and ounces have been rounded to reflect the relative uncertainty and the approximate quality of the estimate.

Comparison of the Mineral Resource to the historical resource (January 2014) at a 2.0 g/t Au cut-off is detailed in Table 14.2-1. The total Measured and Indicated gold ounces have increased moderately as a result of the closer spaced drilling and exposure from underground mining confirming confidence in the geology and grade continuity. The Inferred tonnes have increased significantly, mainly due to the inclusion of the Ptah mineralization with the Sukari Underground Mineral Resources.

15 MINERAL RESERVE ESTIMATES

15.1 Total Mineral Reserve

The Mineral Reserve for Sukari was estimated by AMC (for the open-pit) and Crosscut (for the underground).

The open-pit Mineral Reserve estimate is based upon the 2015 open-pit resource model prepared by MPR and is reported as at 30 June 2015. The underground Mineral Reserve estimate is based upon the underground resource model prepared by Cube in July 2015 and is reported as at 30 June 2015.

The total Mineral Reserve estimate, open-pit and underground, is tabulated in Table15.1-1.

Table 15.1-1 Total Combined (Open-pit and Underground) Mineral Reserve for Sukari

	Proven		Probable		Total		
	Tonnes (Mt)	Gold (g/t)	Tonnes (Mt)	Gold (g/t)	Tonnes (Mt)	Gold (g/t)	Contained Gold (Moz)
New Mineral Reserve (1-3)	**152**	**1.05**	**101**	**1.15**	**253**	**1.09**	**8.8**
Previous Mineral Reserve (4)	119	1.06	111	1.17	230	1.11	8.2

Notes to table:

Totals may not equal the sum of the components due to rounding adjustments.

(1) Based on a metal price of US$1,300/oz Au and includes:
Open-pit reserve totalling 229 Mt @ 1.09g/t for 8.0 Moz
Underground reserve totalling 2.7 Mt @ 6.0g/t for 0.5 Moz
Surface stockpiles totalling 21 Mt @ 0.42g/t for 0.3 Moz

(2) Based on open-pit mined surfaces as at 30 June 2015 and underground workings as at 30 June 2015

(3) Final open-pit design has a waste to ore ratio of 5.9:1 (including the in-pit dump leach ore, but not stockpiles)

(4) As at 30 September 2013 using US$1,300/oz Au

The total combined (open-pit and underground) Mineral Reserve at Sukari was estimated at 253 Mt of ore at an average grade of 1.09 g/t Au for 8.8 Moz of contained gold.

The Mineral Reserve is based on a gold price of US$1,300/oz and open-pit gold cut-off grades of 0.08 g/t Au for oxide (including dump leach material) and 0.42 g/t for both transition and sulphide material. The underground cut-off grade is 3.0 g/t. The reference point for the Mineral Reserve estimate is the mill feed, reported as mined ore delivered to the plant or dump leach processing facilities.

This Mineral Reserve estimate is classified and reported in accordance to National Instrument 43-101 (NI 43-101) *Standards of Disclosure for Mineral Projects*, Form 43-101F1 *Technical Report*, and Companion Policy 43-101CP which came into force on 30 June 2011.

Furthermore, the reserve classifications are also consistent with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves of 2012 (the JORC Code) as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia JORC (2012), with the minor exception that the Code refers to Ore Reserves while NI 43-101 refers to Mineral Reserves.

The Qualified Person responsible for the open-pit Mineral Reserves estimate is Patrick Smith, a full-time employee of AMC Consultants Pty Ltd. Mr Smith has the appropriate relevant qualifications and experience to be considered a Qualified Person as defined in NI 43-101.

The Qualified Person responsible for the underground Mineral Reserve estimate is Declan Franzmann, an independent mining engineering consultant trading as Crosscut Consulting. Mr Franzmann has the appropriate relevant qualifications and experience to be considered a Qualified Person as defined in NI 43-101.

15.2 Open-pit Mineral Reserve

The open-pit Mineral Reserve estimate is based on the open-pit resource model produced by MPR Geological Consultants in July 2015.

The open-pit resource model contains an allowance for dilution within the limits of the chosen SMU. The QP, Patrick Smith, notes that the open-pit resource model contains inbuilt allowances for dilution and mining losses, and that it is reasonable, at this stage, to not include any further allowance for these effects in the Mineral Reserve estimate.

The Sukari pit will produce ores of varying stages of weathering. The processing costs and metallurgical recoveries are modelled to vary with the extent of the weathering, and hence the cut-off grade applied also varies with the weathering state. The processing plant cut-off grades for the oxide, transition and sulphide rock types were 0.40 g/t, 0.42 g/t and 0.42 g/t, respectively. The dump leach cut-off grade for the oxide rock type was 0.08 g/t.

The Sukari pit will be developed in a number of open-pit mining stages, as shown in Figure 16.1-4.

The mining and processing schedule for the open-pit uses an elevated cut-off grade through the early years of production to increase the head grade to the processing plant.

The Mineral Reserve for the open-pit, including stockpiles and dump leach ore, were estimated to be 250 Mt of ore at an average grade of 1.03 g/t Au, containing 8.3 Moz gold. The open-pit Mineral Reserve is summarized by category in Table 15.2-1. The open-pit Mineral Reserve is contained within designed and scheduled open-pits which were based upon the results of Lerchs Grossman pit optimizations of Measured and Indicated Resources. The Inferred Resources which occur within the pit design are treated as mineralized waste in the production schedule and contribute no value to the economic evaluation of the Mineral Reserve.

Table 15.2-1 Open-pit Mineral Reserve for Sukari

Mineral Reserve Category	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Proven – in pit	130	1.11	4.6
Probable – in pit	99	1.07	3.4
Proven – Stockpile	21	0.42	0.3
Total Mineral Reserve	**250**	**1.03**	**8.3**

The estimated split of the open-pit Mineral Reserve by processing method is set out in Table 15.2-2.

Table 15.2-2 Estimated Split of Mineral Reserve by Processing Method and Category

Mineral Reserve Classification	Tonnes (Mt)	Au (g/t)	In situ Au (Moz)
CIL Circuit			
Proven – in pit	114	1.23	4.5
Probable –in pit	88	1.18	3.3
Proven - Stockpile (non-oxide)	10	0.53	0.2
Total CIL Circuit	**212**	**1.18**	**8.0**
Dump Leach			
Proven – in pit	16	0.20	0.1
Probable – in pit	11	0.19	0.1
Proven - Stockpile (oxide)	11	0.31	0.1
Total Dump Leach	**38**	**0.23**	**0.3**

15.3 Underground Mineral Reserve

The Mineral Reserve for the Sukari underground mine are based on the underground resource model produced by Cube Consulting in July 2015. All resources are in a fresh state of weathering.

Dilution and loss have been applied based on the type of stoping being undertaken (detailed in Section 16.2.3). The cut-off grade used for the underground Mineral Reserve is 3.0 g/t, using the same gold price assumptions and gold recoveries as used in the open-pit Mineral Reserve. The underground Mineral Reserve is based on designed stopes and development within the resource model (detailed in Sections 16.2.2 and 16.2.4), and costed using current costs and assumption of continuation of the current production rate of 1 Mtpa from combination of both stoping and development.

The Mineral Reserve for the Sukari underground mine is estimated to be 2.7 Mt of ore at an average grade of 6.0 g/t Au, containing 520 koz of gold. The Mineral Reserve is summarized by category in Table 15.3-1.

Table 15.3-1 Underground Mineral Reserve for Sukari

Mineral Reserve Category	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Proven	1.02	6.1	0.20
Probable	1.70	5.9	0.32
Total Mineral Reserve	**2.72**	**6.0**	**0.52**

Notes to Table:

- Totals may not equal the sum of the components due to rounding adjustments.
- Mineral Resources are reported inclusive of those resources converted to Proven and Probable Mineral Reserves.

The underground Mineral Reserve estimate includes broken stocks (at 30 June 2015) of 111 kt of ore at an average grade of 9.5 g/t Au and containing 34 koz of gold. The balance of the underground Mineral Reserve was estimated from development and stope designs accessed via the Amun and Ptah Declines.

15.3.1 Underground Mineral Reserve by area

Table 15.3-2 and Table 15.3-3 shows the Mineral Reserve for each are within the underground mine.

Table 15.3-2 Underground Mineral Reserve at Ptah

Mineral Reserve Category	Tonnes (Mt)	Grade (g/t)	Contained Gold (koz)
Proven	0.02	4.1	3
Probable	0.29	9.0	84
Total Mineral Reserve	0.31	8.7	87

Table 15.3-3 Underground Mineral Reserve at Amun

Mineral Reserve Category	Tonnes (Mt)	Grade (g/t)	Contained Gold (koz)
Proven	1.00	6.1	200
Probable	1.41	5.3	240
Total Mineral Reserve	2.41	5.6	440

Notes to Table:

- Totals may not equal the sum of the components due to rounding adjustments.
- Mineral Resources are reported inclusive of those resources converted to Proven and Probable Mineral Reserves.

16 MINING METHODS

16.1 Open-pit Mining

SGM is conducting owner-mining open-pit operations at Sukari and utilizing Barminco Egypt as a contractor for underground mining. SGM adopted this approach in the absence of an established local mining contracting industry in Egypt and in recognition of the long project life that would allow full utilization of the open-pit mining fleet and the development of in-house capabilities.

16.1.1 Mining Method

16.1.1.1 Load and Haul

All ore and waste is being mined using open-pit gold mining methods. The mine currently uses four (4) CAT 6040 face-shovels (or equivalent), four (4) O&K RH120 backhoe excavators and forty-eight (48) CAT785C haul trucks to carry out the bulk of the ore and waste.

The operation is selective in terms of separating ore and waste. In the opinion of the QP, Patrick Smith, the degree of selectivity is appropriate; for the scale of mining equipment, the proposed grade control method, and the nature of the mineralization. A SMU of 5 m (east-west), 8 m (north-south), and 10 m (vertical) was adopted for the block support adjustment employed in developing the Multiple Indicator Kriged (MIK) resource model.

The mining fleet includes the requisite ancillary equipment (track and wheel dozers, motor graders, front-end wheel loaders, service trucks, and water trucks) to maintain the pit haul roads, loading and tipping areas, for ROM pad operations, and a projects fleet for pioneering work and TSF construction.

Ore is hauled to the ROM pad adjacent to the primary crusher. The majority of ore is direct tipped into the crusher, with provision for ore to be stockpiled for reclaim by a front-end-loader operated as part of the crushing and processing operation. Ore is not expected to always be visually distinct from waste, and hence ore and waste segregation relies on RC drilling, sampling, and assays for definition of ore blocks.

Waste is used for construction or is hauled to the waste dumps on the east and south sides of the pit. Waste is used to construct ramps to provide access on the hillside and to provide fill for the TSF. The haul roads from the pit to the ROM pad and waste dumps were constructed with waste rock sourced from the pit.

Due to the exceedingly dry climate, in the opinion of the QP, Patrick Smith, acid generation from waste is not anticipated to be an issue, and hence no consideration of acid-forming waste has been included in the waste dump design or schedule. Sub-grade ore, when selectively mined and stockpiled, is placed near the low-grade ore stockpile, and will be encapsulated under a waste dump should economics not allow treatment of this material towards the end of the mine life.

The load and haul costs are based upon actual mining costs and cost inputs developed by SGM personnel using haul profiles derived from pit designs. Mining benches are named according to the bench toe elevation in mRL.

16.1.1.2 Grade Control

Grade control (GC) samples are collected through RC drilling campaigns. The drilling patterns drilled are appropriate for this style of gold mineralization and is based on recommendations by MPR. GC holes are drilled vertically on multiples of 10 m, to match mining bench levels. The holes are usually drilled to 40 m deep, with a spacing of 8 m east-west by 12 m north-south.

The GC patterns target the main mineralized structures and the associated Sukari stockwork mineralization. The hanging wall porphyries are drilled on the same pattern size as the stockwork mineralization (8 m east-west x 12 m north-south x 40 m downhole depth).

GC drilling is conducted by a contract drilling company. Drilling takes place on dayshift and nightshift over two 12-hour shifts.

Personnel in the SGM Mining-Geology department design the GC patterns. The GC holes are also utilized as advance probe drilling for voids around the current underground mine. SGM samplers sample drillhole cuttings. One sampler is allocated to each RC drill rig. Samples are collected as 2.5 m composites downhole, to give sufficient sample density for ore delineation.

Sample preparation and analysis of GC samples occurs at the onsite mine laboratory. The samples are analyzed using a Fire Assay technique. Quality control samples are submitted within the sequence of GC samples. These quality control samples consist of blanks, standards, field resplits, pulps, and umpire laboratory duplicates. Sample recoveries are recorded at the drill rig.

Geological and positional data are electronically recorded into a digital data logger at the rig and imported directly into the GC database, which is a subset of the central geological database. The recorded geological and metallurgical properties for each hole are used for ore type interpretation and delineation. Conditional simulation is used to model ore blocks from GC data, and the AMC considers that it is suitable for the style of mineralization at Sukari.

16.1.1.3 Mining Dilution, Ore Loss and Reconciliation

The MIK resource model has parent cell dimensions that are 20 m x 25 m x 10 m in the east-west, north-south, and vertical directions respectively. An SMU size of 5 mE x 8 mN x 10 mRL was adopted by MPR to predict the grade and tonnage at elevated cut-off grades. As such, each parent cell can be considered to represent 12.5 SMUs.

SGM undertakes a series of monthly reconciliation tasks to reconcile the published mined ore tonnage and grade to the processing plant gold production results. Four specific reconciliations are undertaken:

- Resource model to the grade control model,
- Grade control model to the mining ore blocks,
- Mining ore blocks to the trucked ore, and
- Trucked ore to the reconciled processing grade.

At the reserve cut-off grade SGM has observed reconciliation fluctuations on contained gold, both positive and negative, between the resource model and the grade control model at different periods over the past five years. This reconciliation is counter-acted and complicated by an observed assaying bias. Overall, for production from 2009 to end of June 2015, the resource model and grade control model appear to reconcile to within a few percent, once adjusted for the assaying bias.

The QP, Patrick Smith, recommends a site-wide reconciliation process be developed and implemented to analyse all aspects of the production chain. With the major ramp-up in mining and processing rates at Sukari in recent years this process is needed to allow robust analysis of resource, reserve, and grade control models.

The open-pit resource model contains an allowance for dilution within the limits of the chosen SMU. The QP, Patrick Smith, notes that the open-pit resource model contains inbuilt allowances for dilution and mining losses, and that it is reasonable, at this stage, to not include any further allowance for these effects in the Mineral Reserve estimate.

16.1.1.4 Drill and Blast

All in situ ore and waste requires blasting with no "free dig" material. However, there are some small areas of transported material residing at the foot of the slopes of the hills and in valley (wadi) floors that does not require blasting.

The average powder factor for ore and waste is 0.63 kg/bcm. Drilling is conducted over 20 m bench heights, with 165 mm and 203 mm diameter holes drilled for production, and 140 mm diameter holes for pre-splits. However, there are variations to pattern size, hole diameter, and powder factor, depending on rock type, oxidation state, and structure, to ensure optimal fragmentation of the rock mass for mining operations.

16.1.1.5 Other Mining Activities

All production mining activities at Sukari operate on 12-hour day and night shifts. Haul roads are maintained by SGM personnel. Waste dump slopes will be progressively battered down to their final profiles during construction.

Portable lighting towers, trailer-mounted diesel generator sets with banks of halogen floodlights mounted on an easily erected tower, are used to illuminate the working areas in the open-pit at night. Typically, lighting towers are used at the excavating face, dumping face, other locations around the pit perimeter to give overall illumination of working areas, and ramp intersections. Lighting towers are also required for night shift drilling crews. Permanent lighting for night time operation is installed at fixed locations close to mains power, such as the ROM pad.

16.1.2 Geotechnical Considerations

The pit slope designs were based upon advice from George, Orr and Associates (Australia) Pty Ltd (GOA, 2013; Orr, 2013) for the 2013 updated estimate of open-pit Mineral Reserves, with no changes to pit slopes since that time. GOA (2013) has made recommendations for the pit slopes in the initial stages of the pit, including stages 3A and 3B currently being mined. The geotechnical recommendations are summarized as follows:

- West wall at overall slope of 37 degrees
- North and south walls at overall slope of 40 degrees
- East wall at overall slope of 40 degrees

Additionally, in-pit geotechnical data and observations are undertaken by the Geology-Mining department daily, through mapping, structural analysis, and prism monitoring. This information is used to refine the slope designs in later pit stages.

For the purposes of the mineral reserve estimate, the slope designs recommended for the initial stages have been applied to all stages.

16.1.3 Hydrogeology and Hydrology Input

The Sukari region is generally quite dry. Rainfall events are very infrequent, but of high intensity, and subsequently there is a risk of high volumes of water shedding from the wadi into the pit. In-pit groundwater inflows are expected to be low volume and localized.

KP developed a surface water management plan for Sukari, in consultation with the stakeholders.

Historical records of rainfall events were limited; however, KP used the records of a high-intensity rainfall event in November 2001 to develop the plan. An accurate watershed flow path during a heavy rainfall event was difficult to establish due to the flat undefined nature of the

Wadi valleys. KP inferred the natural watershed, based on topographical information supplied by PGM. Conservative assumptions were made to determine the critical flow paths.

The KP plan sets out a system to control surface water using drainage channels and diversion bunds. Topographical features and gravity flow were utilized, such that pumping would not be required for the surface water system. KP recommends that three major diversion bunds be built, in addition to the pit diversion system. Diversion bunds are constructed to protect the plant site, pit, waste dump, and haul road network.

The pit diversion system comprises a series of small steep catchments and bunds, constructed concurrently with pit development that would collect surface flows and discharge the water away from the pit. The diversion drains would be excavated in competent rock, such that the water is discharged away from the pit with minimal water seeping back towards the pit, potentially energizing an in-pit structure.

In-pit water inflows and rainfall captured within the pit will be managed with in-pit sumps and pumps. Booster pumps would be required with increasing pit depth. In-pit water collected in sumps, would be pumped to a storage dam on the surface and be used as process water in the plant or haul road dust suppression.

16.1.4 Pit Optimization

16.1.4.1 Model Preparation

AMC was engaged by SGM to complete an estimate of the open-pit Mineral Reserves for Sukari.

The 2015 resource model was supplied by MPR as an MIK model. The model contains parent cells with dimensions of 25 m x 20 m x 5 m, representing the north-south, east-west and vertical directions respectively. The resource model has grades estimated for “bins” at the following cut-off grades 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.8, 0.9, and 1.0 g/t Au for each parent cell.

Each of the grade ranges in the MIK resource model can be considered to be individual parcels of ore within each parent cell.

The resource model was imported to Datamine Studio 3 mining software and the grade bins were converted to subcells to create a mine planning model for ease of processing and visualization. Converting the grade bins to subcells within the Datamine model may create a false impression of the spatial location of metal within a parent cell. However, the pit design process does not use this spatial distribution within the parent cell.

The mine planning model was compared with MPR’s evaluation of the resource model at a range of cut-off grades, and as the differences between the two models were within AMC’s tolerance range (<0.1% difference), the mine planning model was accepted.

The resource model also contained descriptions of material according to weathering state, ore type, and resource classification. These descriptions were preserved in the mine planning model.

There are no physical constraints to mining within the lease area. No property, infrastructure, or environmental issues are known to AMC, which may limit the extent of mining. The pit limit optimization process was restricted to apply only to Measured and Indicated Mineral Resources.

Pit wall slope angles were set consistent with the slopes angles as discussed in Section 16.1.2.

To define the economic pit limits, the geological model was imported into Whittle Four-X pit optimization software (W4X). Other key inputs to W4X were:

- Estimates of the mining costs
- Estimates of the processing costs
- Process plant and dump leach metal recoveries
- Geotechnical design criteria
- Gold selling price and royalty charges

The operating cost and sales price assumptions are provided in Section 16.1.4.2. The metallurgical recovery assumptions are outlined in Section 16.1.4.3.

16.1.4.2 Costs and Revenue Factors

The mine operating costs were derived from actual production and cost performance data and projected costs based on the current life-of-mine plan and budget estimates.

When compared to the September 2013 mineral reserve estimate, the processing costs decreased from US$15.50/t to US$14.53/t for transitional and sulphide ore due to a decrease in power costs. This was based on the price assumption for diesel fuel at US$0.70/litre compared to the assumed price of US$0.84/litre for the 2013 mineral reserve.

Mining operating costs decreased from US$1.55/t to US$1.47/t at the 1,100 mRL reference point, also based on the change in diesel price.

The input costs are summarized in Table 16.1-1.

Table 16.1-1 Pit Optimization Input Costs

Item	Units	Rate
Mining base cost (1,100 mRL)	US$/tonne	1.47
Mining incremental cost (above 1,100 mRL)	US$/tonne/10 metre bench	0.014
Mining incremental cost (below 1,100 mRL)	US$/tonne/10 metre bench	0.043
Processing - oxide	US$/tonne	5.90
Processing - transitional	US$/tonne	14.53
Processing - sulphide	US$/tonne	14.53
Dump leach - oxide	US$/tonne	1.85
General and administration overheads	US$/tonne	0.80
Refining cost / treatment charges and refining costs	US$/oz	1.20
Royalty cost (3% of net sale revenue)	US$/oz	39.00

The Government of Egypt receives a royalty of 3% of net sale revenue, payable in cash each calendar half-year, which was treated as a pit optimisation input cost of US$39/oz.

The long-term gold price for Mineral Reserve estimation was assumed at US$1,300/oz based on market price forecasts and historical prices.

16.1.4.3 Metallurgical Recoveries

The metallurgical recoveries were based on current process plant performance and projections for future performance by SGM. The metallurgical recovery by processing method and ore type is set out in Table 16.1-2.

Table 16.1-2 Metallurgical Recoveries

Processing Method and Ore Type	Metallurgical Recovery
Processing - oxide	90%
Processing - transition	90%
Processing - sulphide	91%
Dump leach - oxide	60%

16.1.4.4 Optimization Results

Using the operating costs, metallurgical recoveries, and geotechnical parameters, a pit optimization of the resource model was conducted. Only Measured and Indicated Resources were used in the pit optimization.

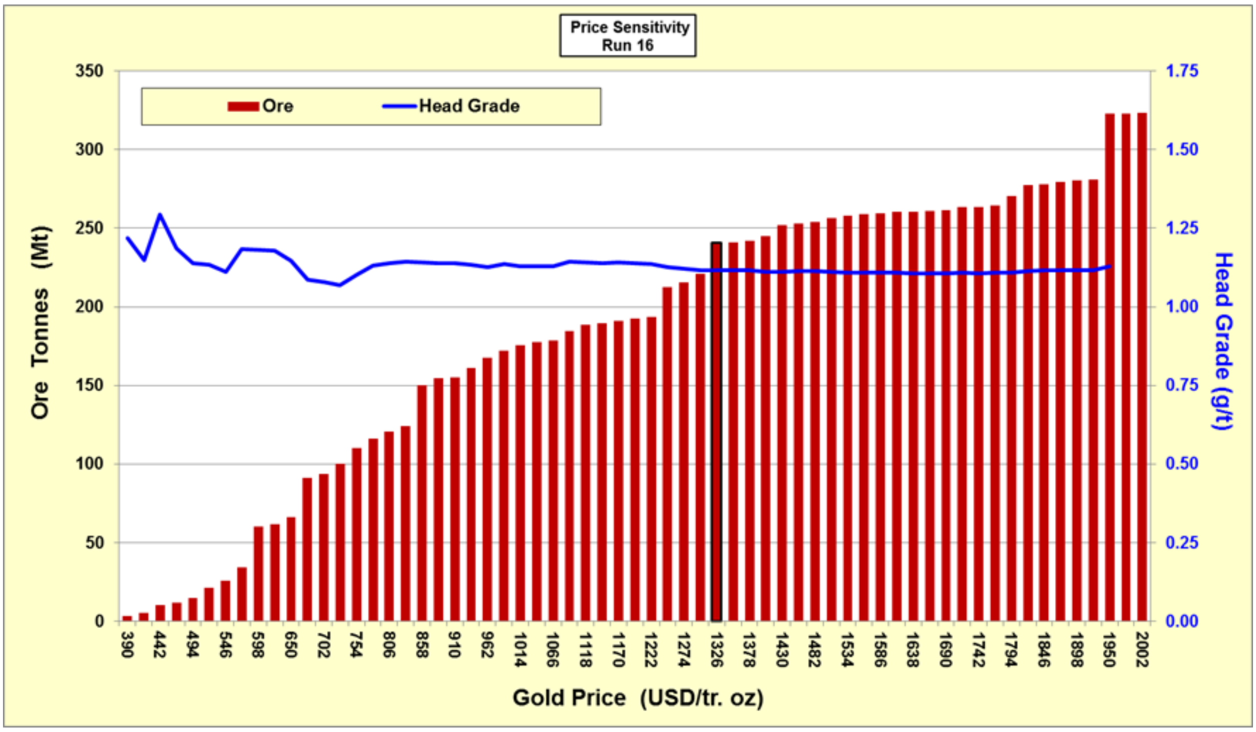
Pit optimizations were initially conducted without consideration of the influence of future underground mining. Current underground operations and underground mineral reserves are located well above the final pit limit. A subsequent pit optimization tested this assumption, and the resource model depleted by future underground mining had no effect upon the final pit limits and final pit selection.

The Mineral Reserve was classified based on the confidence of the Mineral Resource and the level of detail in the mine planning. The Mineral Resource within the pit shell that was identified as Measured was classified as Proven for reporting a Mineral Reserve. The Mineral Resource within the pit shell that was identified as Indicated has been classified as Probable for reporting a Mineral Reserve. No Inferred Resources have been used in this estimate.

The pit optimization results were examined and the selected pit shell validated against an incremental strip ratio and marginal economics analysis.

The W4X pit optimization defined a series of nested pit shells, based on a gold price of US$390/oz to US$2,600/oz in increments of US$26/oz. The results, as illustrated in Figure 16.1-1, indicated that while the average grade is fairly constant over the range of pit shells, the mining inventory (tonnage) was quite sensitive to gold price. Pit Shell 37 (Revenue Factor 1.02) from optimization Run 16 was selected for detailed pit design and for estimation of the Mineral Reserves.

Figure 16.1-1 Open-pit Optimization Results



The project is currently operating with established infrastructure and has successfully demonstrated its ability, through annual reporting, to operate at a profit. Regular updates to the life-of-mine (LOM) plans confirm the practical and economic extraction of ore.

There is a low project operational risk. The cost inputs used to estimate the open-pit Mineral Reserves are based on historical costs and forecasts to account for future operations. Operating cost estimates were based on a delivered price of diesel at US$0.70/litre, which is above the average actual price over the past operating year.

The open-pit Mineral Reserve, split by processing methods, is shown in Table 16.1-3. The reserve and project economics are most sensitive to movements in the gold price. A material fall in gold price may potentially reduce the quantity of mineable ore, and equally an increase in gold price would lower the cut-off grade and increase the quantity of mineable ore.

Table 16.1-3 Estimated Open-pit Reserve by Processing Method

Mineral Reserve Classification	Tonnes (Mt)	Au (g/t)	In situ Au (Moz)
CIL Circuit			
Proven – in pit	114	1.23	4.5
Probable –in pit	88	1.18	3.3
Proven - Stockpile (non-oxide)	10	0.53	0.2
Total CIL Circuit	**212**	**1.18**	**8.0**
Dump Leach			
Proven – in pit	16	0.20	0.1
Probable – in pit	11	0.19	0.1
Proven - Stockpile (oxide)	11	0.31	0.1
Total Dump Leach	**38**	**0.23**	**0.3**

16.1.5 Pit Designs

16.1.5.1 Design Basis

Pit shell 37 from optimization Run 16 was selected by SGM as the basis for the final pit design. This was the Revenue Factor 1.02 shell. The pit design based on this shell has a total rock content of 1,579 Mt, comprising 1,350 Mt waste rock and 229 Mt ore at 1.09 g/t Au, yielding a strip ratio of 5.9:1 (waste to ore, with the latter including the dump leach ore).

The estimate of underground Mineral Reserve is 2.7 Mt at 6.0 g/t for 0.5 Moz Au. Underground mining is planned to source ore from stopes located within the designed final open-pit, and well above the final pit limits. A volume adjustment representing the stoping component of this 2.7 Mt was made to the MIK resource model to account for the volume to be removed by the underground mineral reserve in advance of later open-pit mining.

The open-pit is to be mined in stages. Stage 1 and 2 are complete. The mining sequence progresses through Stages 3, 4, 5, and 6, with Stage 7 comprising the final pit design. Sub-staging is used by the site when appropriate to meet short-term scheduling objectives with Stage 3 currently being mined as two sub-stages, Stage 3A and 3B.

16.1.5.2 Haul Road Designs

The design specifications of the final pit in-pit haul roads are set out in Table 16.1-4.

Table 16.1-4 Final Pit In-Pit Ramp Design Specifications

	Dual Lane	Single Lane
RL (Top)	1,100 mRL	700 mRL
RL (Bottom)	700 mRL	560 mRL
Ramp Gradient	10%	10%
Ramp Width	32 m running width	20 m running width

Note: In lower levels of pit stages the width ramp reduces to single-lane, one way traffic

16.1.5.3 Pit Slopes

The final pit design overall slope angles (from pit floor to the pit crest at natural surface) are described in Section 16.1.2. The batter angle (or face angle) is 80° for all pit walls. Pit slope design parameters are set out in Table 16.1-5.

Table 16.1-5 Pit Slope Design Parameters

	Foot Wall (West)	Hangingwall (East) and North and South Walls
Batter height	10 m	20 m
Batter angle	80°	80°
Berm width	10 m	10 m
Wide berm width	20 m every 60 m	15 m to 25 m variable every 40 m as required to meet overall slope angle after accounting for intersected in-pit ramps
Overall slope	37°	40°

The completed design pit, as shown in Figure 16.1-2, is 2,600 m long and 1,400 m wide with a maximum depth of 540 m, measured from the wadi at a nominal surface elevation of 1,100 mRL. Figure 16.1-3 shows the footprint of this pit superimposed on the end June 2015 surface topography.

Figure 16.1-2 Open-pit Final Design

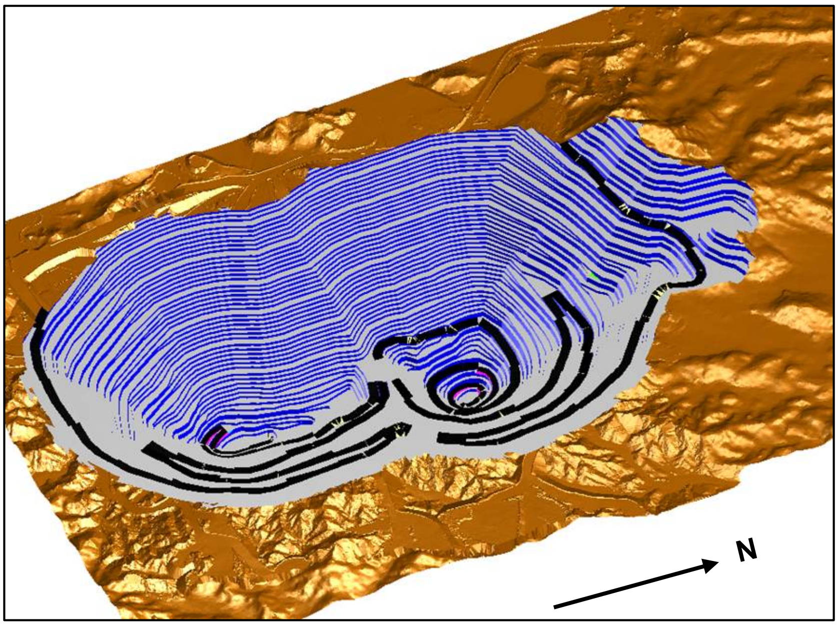


Figure 16.1-3 Open-pit Final Design Footprint

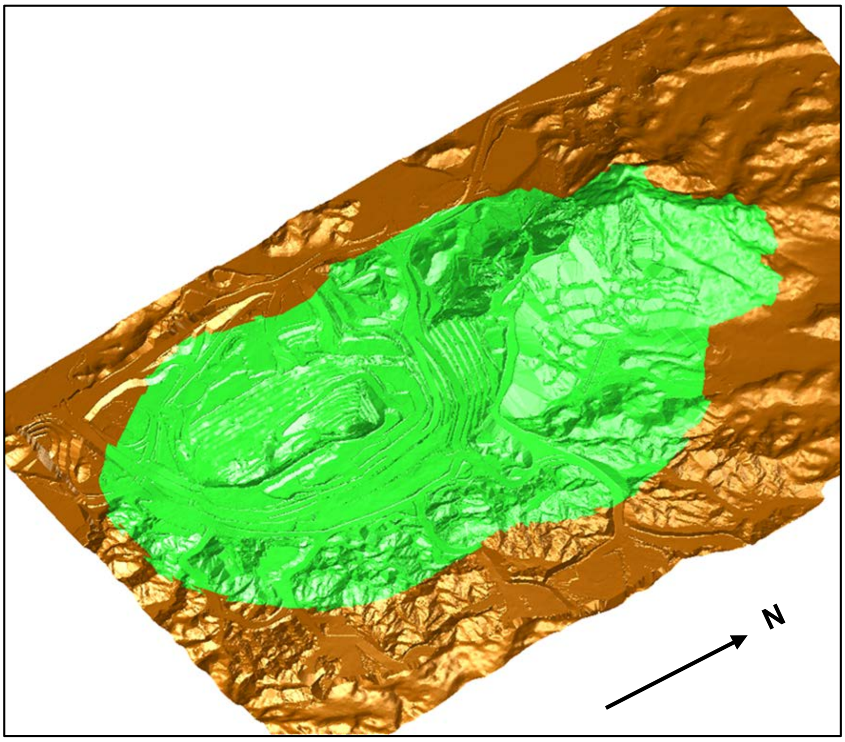
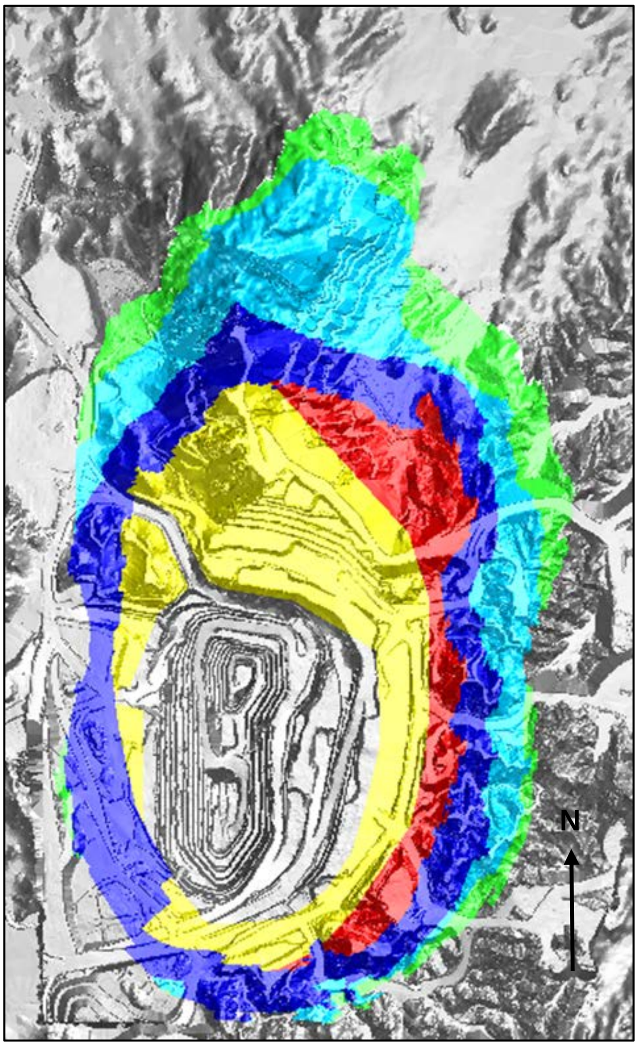


Figure 16.1-4 Open-pit Stage Design Footprints



16.1.6 Waste Dumps

A total of 1,350 Mt or 485 Mbcm of waste is to be mined from the final pit. A swell factor of 25% was used, which equates to a waste dumping requirement of 606 million cubic metres. Waste rock will be hauled to and placed in the south, east or north waste dumps, as well as being used to construct the TSF stages (lifts). The dump design volumes are sufficient to contain the planned mining volume.

A ring road has been constructed on the east side of the pit that links the east dump to the northeast and southeast pit ramps, providing SGM with haul-route options to optimize the waste haulage cycles.

In addition to the waste dumps adjacent to the open-pit, a series of ore stockpiles (sub-grade, low-grade, and ROM feed grade) are designed as close as practicable to the plant site.

16.1.7 Mining and Treatment Schedule

Centamin has, in 18 May 2015, published a five-year production forecast from 2015 to 2019 (CEY Forecast). This CEY Forecast of future production includes an assumption for the open-pit total annual material movement of 66 Mtpa, and process plant feed of 11 Mtpa – comprising approximately 10 Mtpa sourced from the open-pit and 1 Mtpa from the underground for the full five years of the forecast schedule timeframe.

Centamin notes that the potential quantity and grade of the forecast underground production in the CEY Forecast is conceptual in nature, and there has been insufficient exploration to define an underground mineral resource to support the expected life of mine.

For the purposes of ensuring that the open-pit mineral reserve are practical and economic, the QP, Patrick Smith, has reviewed an indicative open-pit LOM mining and processing schedule (Indicative Schedule) that was prepared to validate that the extraction of the open-pit Mineral Reserve can be reasonably justified. Developed for the purposes of determining the practicality and economics of the Mineral Reserve, the validation is conducted using Mineral Reserve only, and excludes conceptual tonnage such as material classified as Inferred Mineral Resources.

Ore is assumed to be processed via the CIL and flotation processing plant, or the dump leach facility. Ore processing parameters for each processing method were applied in the schedule.

Processing throughput rate assumptions include:

- Processing plant throughput capacity of 11 Mtpa until end of mine life.
- Underground ore production rate of 1 Mtpa until depletion of the current underground Mineral Reserve of 2.7 Mt.
- The plant is fed preferentially with high-grade ore sourced direct from the open-pit, and from the underground (until estimate of underground Mineral Reserve is depleted), and supplemented by low-grade ore reclaimed from stockpiles.
- Low-grade oxide ore will be placed on the dump leach pad directly from the pit.

The Indicative Schedule was based on the 2015 open-pit final design and a combination of historical and revised staged pit designs. Mining has been completed in Stages 1 and 2, and mining is currently underway in Stages 3A and 3B.

The open-pit mining assumptions used to develop the indicative schedule are.

- The maximum vertical advance is 90 metres per annum (except for very small tonnage benches on Sukari Hill, which AMC considers can be mined faster).
- Open-pit production rate commences at 66 Mtpa (current mining rate) and then progressively increases, to nominal rates of 80 Mtpa and 100 Mtpa, as required to maintain processing plant throughput capacity without contribution from underground and to minimize the use of low grade (<0.6 g/t) sulphide ore.
- Current supply of blasting product is sufficient to meet the current production rates, and additional supplies of blasting product can be secured to meet increased production rates.
- Smaller-scale project fleet to undertake pioneering pre-strip work.

The indicative schedule strategy is to feed the highest grade of ore to the mill at the maximum mill throughput, within the constraint of the total period movement. This will be achieved by:

- Continue mining stage 3A until completed in early 2017 and stage 3B until completed in 2018.
- Continue pioneer development of stages 4 to 7 (bringing Sukari Hill down to an accessible level)

The Indicative Schedule of the mining of each open-pit stage is shown as a progression in Figure 16.1-5 with the completion of mining occurring in 2033.

The indicative schedule of open-pit sourced mill feed tonnes and grade to the processing plant is shown in Figure 16.1-6 with the completion of processing in 2035. The underground Mineral Reserve of 2.7 Mt is assumed to be mined and processed at 1 Mtpa from mid-2015 to 2018.

Figure 16.1-5 Open-pit Stage Mining Progression

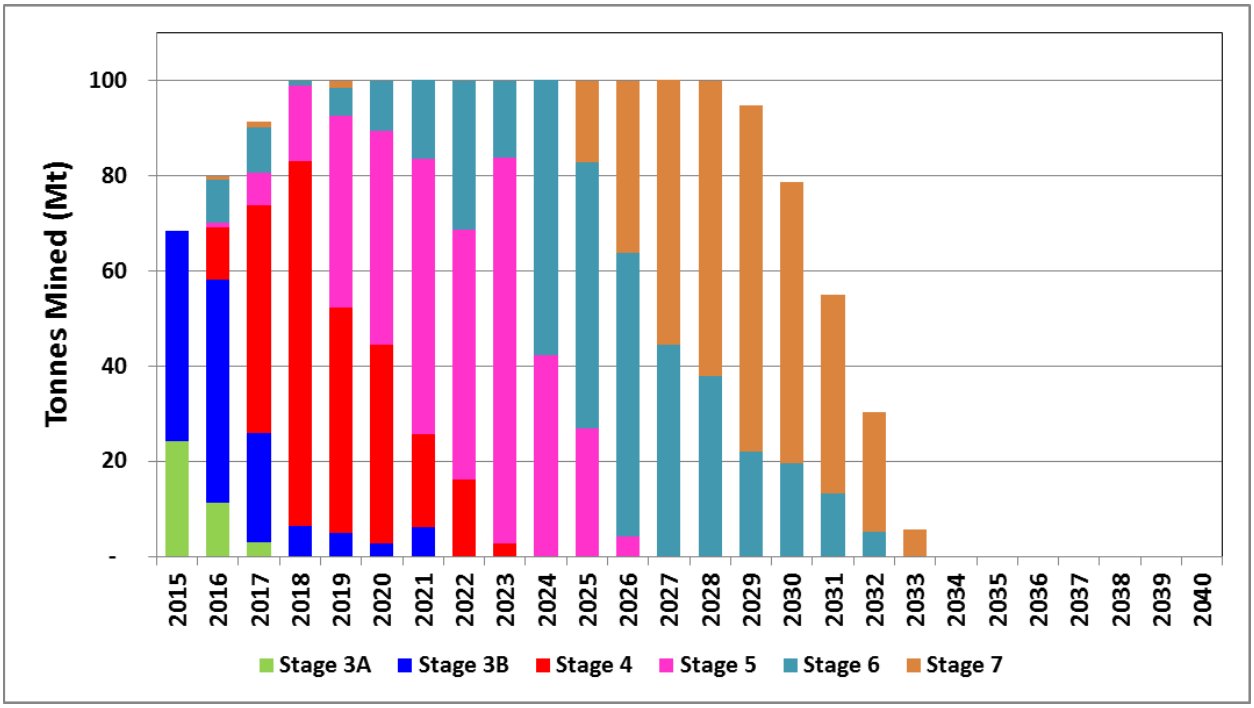
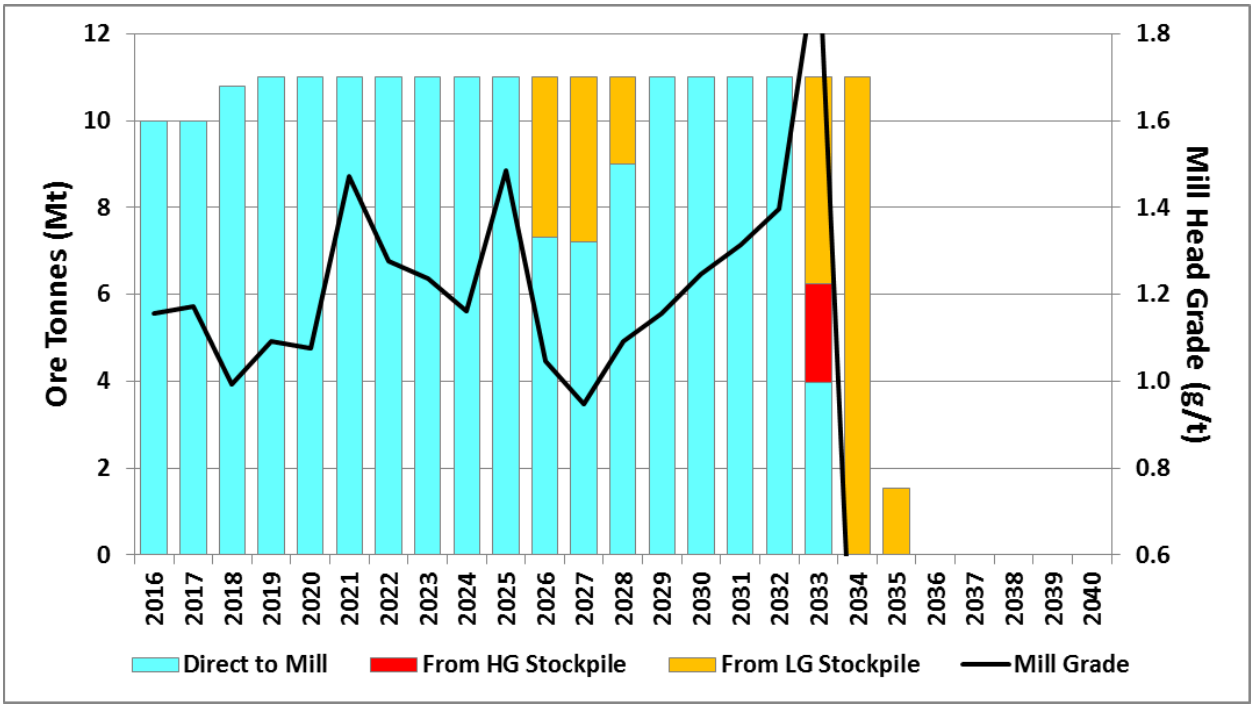


Figure 16.1-6 Mill Feed Schedule for Modelled Strategy – Open-pit Only



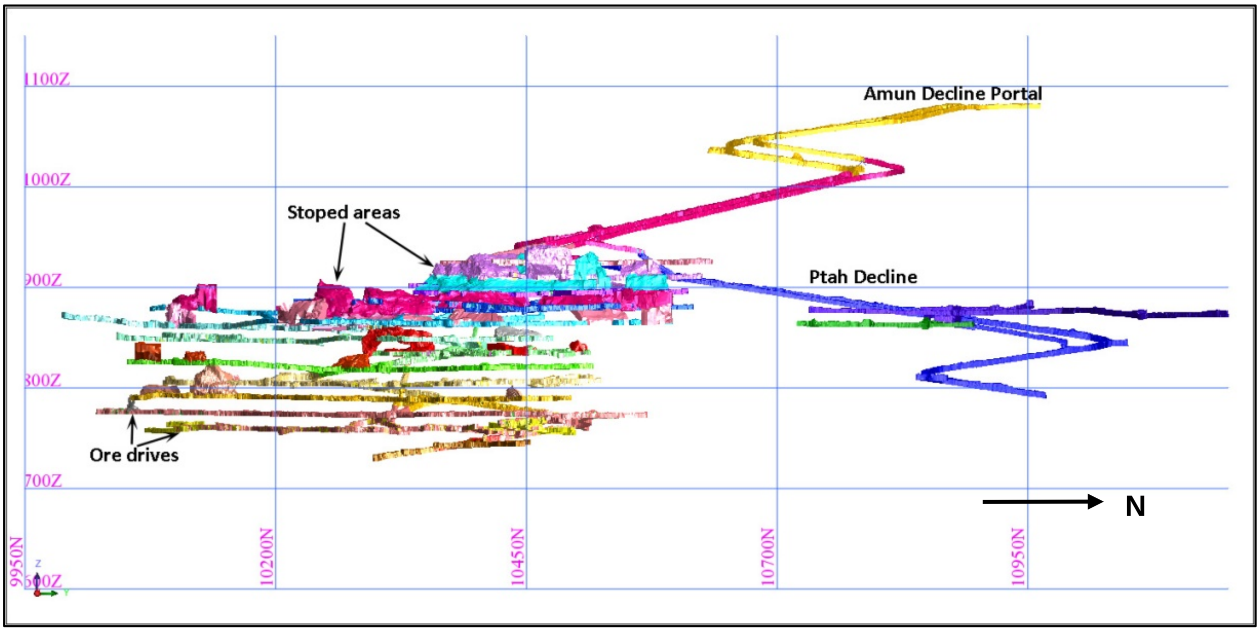
16.2 Underground Mining

The Sukari underground mine is a trackless diesel mine, with all equipment rubber tyred and self-powered. The workings are accessed via a ramp system declining at a gradient of 1 in 7. Levels are typically developed at 15 metre vertical intervals, although this is dependent on orebody geometry and mining method.

Access into mine is via the Amun Decline with the portal located at 1,075 mRL. A second decline ramp, the Ptah Decline, has been developed from the 950 mRL level of the Amun Decline accessing the northern resource area.

Figure 16.2-1 shows a long section of the current mine development and stope voids as at 30 June 2015.

Figure 16.2-1 Long Section Looking North



16.2.1 Mining Method

Underground mining utilizes a fleet of conventional 50 tonne trucks and Elphinstone R2900 loaders for material movement. Ore is sourced from jumbo drill development headings and production stoping areas that utilize a number of different stoping methods. Ore is hauled to the surface ROM pad and used for blending with ore from the open-pit operation.

Waste rock is generated primarily from the access declines, and comprises around 15% of the total material movement. Waste is hauled to a surface waste dump adjacent to the Amun Decline portal.

A fleet of two boom drill jumbos are used for lateral development and ground support. Stope blast holes are drilled using a conventional long-hole drill with a rod handling carousel, which is capable of drilling 89 mm diameter holes, up to 50 m in length. The boom rig configuration rather than horse shoe style allows large fly angles to be used in the flatter stopes, maximizing recovery.

The layout of the mine incorporates two declines. The Amun decline provides access to the Hapi, Amun Deeps, Osiris and Horus zones. The Ptah decline accesses the northern portion of the resource.

Ventilation and a second means of egress is provided by a second decline system which parallels the access declines. The mine has a number of four-man and sixteen-man mobile refuge chambers. Fixed permanent fresh air bases are also in place or planned. An emergency set of services runs through the exhaust system, providing an independent source of compressed air and fire-fighting water. Radio communications are available throughout the mine, and a backup conventional telephone system is also in place.

16.2.2 Economic Parameters for Stope Design

The underlying parameters used for stope designs are detailed in Table 16.2-1.

Table 16.2-1 Stope Design Parameters

Item	Units	Rate
Underground Mining Cost	US$/tonne	58.04
Processing Cost	US$/tonne	14.53
General and Administration Overheads	US$/tonne	0.80
Total Cost	US$/tonne	73.37
Processing Recovery	%	91
Government Royalty	%	3
Gold Price	US$/oz Au	1,300
Breakeven Grade	Au g/t	2.0

Whilst the calculated break-even grade to support all mining and processing costs was 2.0 g/t Au, the hurdle grade used for inclusion in the underground mineral reserve estimate was 3 g/t Au. The broad, low grade nature of the Sukari mineralization, together with the fact that eventually the open-pit will mine through the completed underground workings, allows the use of a higher cut-off grade to improve the operating margin of the underground mine.

There is considerable flexibility in the selection of the hurdle grade used to target production. The aim is to provide sufficient underground feed at high grade to improve the overall cash flow of the operation.

16.2.3 Mining Dilution and Ore Loss

The stoping blocks fall into three broad categories:

- Steeply dipping veins with an angle of between 55 and 75 degrees from horizontal.
- Shallow dipping veins, with a dip angle from 35 to 48 degrees.
- Flatly-dipping veins of sufficient grade to make development and stoping economic. Typically these veins dip at between 6 and 10 degrees and may be associated with multiple veins or a vein swarm.

Modifying factors generally used for the estimation of reserve vary with the differing stoping methods, and are summarized in Table 16.2-2.

Table 16.2-2 Modifying factors for underground mineral reserve estimation

Stope Method Type	Unplanned Dilution		Mining Losses (%)	Mineral Reserve Contained Gold by Stope Method (%)
	Quantity (%)	Grade (g/t)		
Flat dipping – room and pillar	10%	0.8	40%	18%
Shallow dipping – long hole stope	15%	0.4	50%	1%
Steeply dipping – long hole stope	15%	0.4	10%	58%
Development headings	5%	0.8	0%	23%

The modifying factors vary depending on the physical characteristics each stope. The dilution and losses used for each individual stope accounted for the following:

- Stope geometry and location of ore drives
- Dip of stope
- Width of stope
- The use of backfilling materials

Two dilution grades were used in the reserve estimation.

A grade of 0.8 g/t Au was used where the use of backfill was not anticipated, such as in mine development and in the application room and pillar mining method. Given the extensive and varied types of mineralization around the high-grade veins targeted by the underground mine, the use of a dilution grade is warranted.

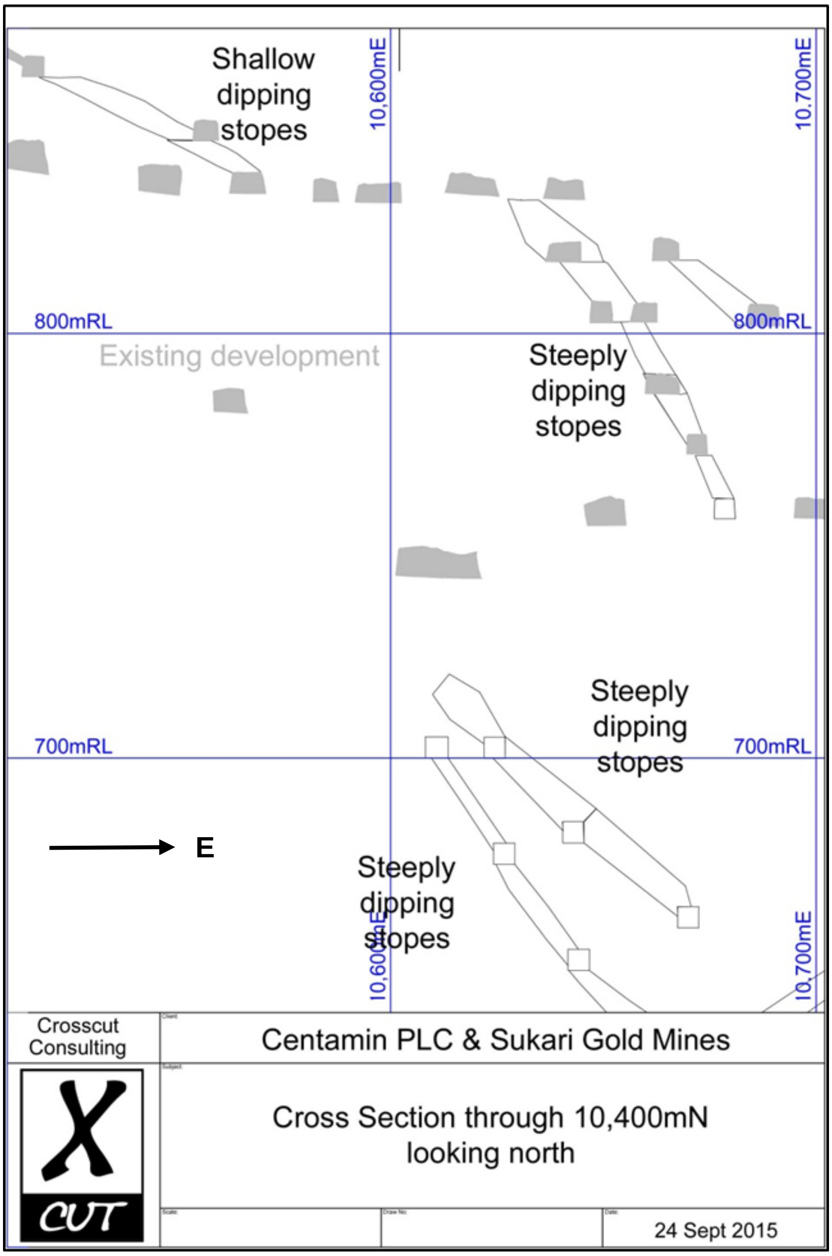
In areas where a long-hole stoping method was applied, the assumption that waste rock will be used as a fill medium was included, and that this would form half of the dilution material. Consequently a dilution grade of 0.4 g/t was used in these areas.

16.2.4 Stope Design

Development and stope designs were based on the July 2015 underground resource model prepared by Cube, and the “as built” development and stope voids current at the end of June 2015. Stope designs were typically constructed on 10 metre sections. For the long-hole stoping method in undeveloped areas, ore drive and stope designs were created to maximize the recovery of the resource. In areas with ore drive access already developed, only the stope was designed.

Stope designs also allowed for practical geometry and mining widths, which results in dilution of the underlying resource in the design process, otherwise described as planned dilution. The dilution described in the previous section (Section 16.2.3) deal with the unplanned component of dilution. A typical section through existing development and reserve designs is shown in Figure 16.2-2.

Figure 16.2-2 Typical section (10,400 mN)



Several areas of flat-lying mineralization were also included in the Mineral Reserve. The design of these areas included a minimum mining height of 5.0 m. For all mining methods, if the average grade of the stopes was less than 3 g/t, it was excluded from the Mineral Reserve.

Figure 16.2-3 illustrates the location of Mineral Reserve development and stope designs while Figure 16.2-4 and Figure 16.2-5 show a long section looking west at Amun and Ptah resource areas.

Figure 16.2-3 Plan View of Workings Mineral Reserve Designs

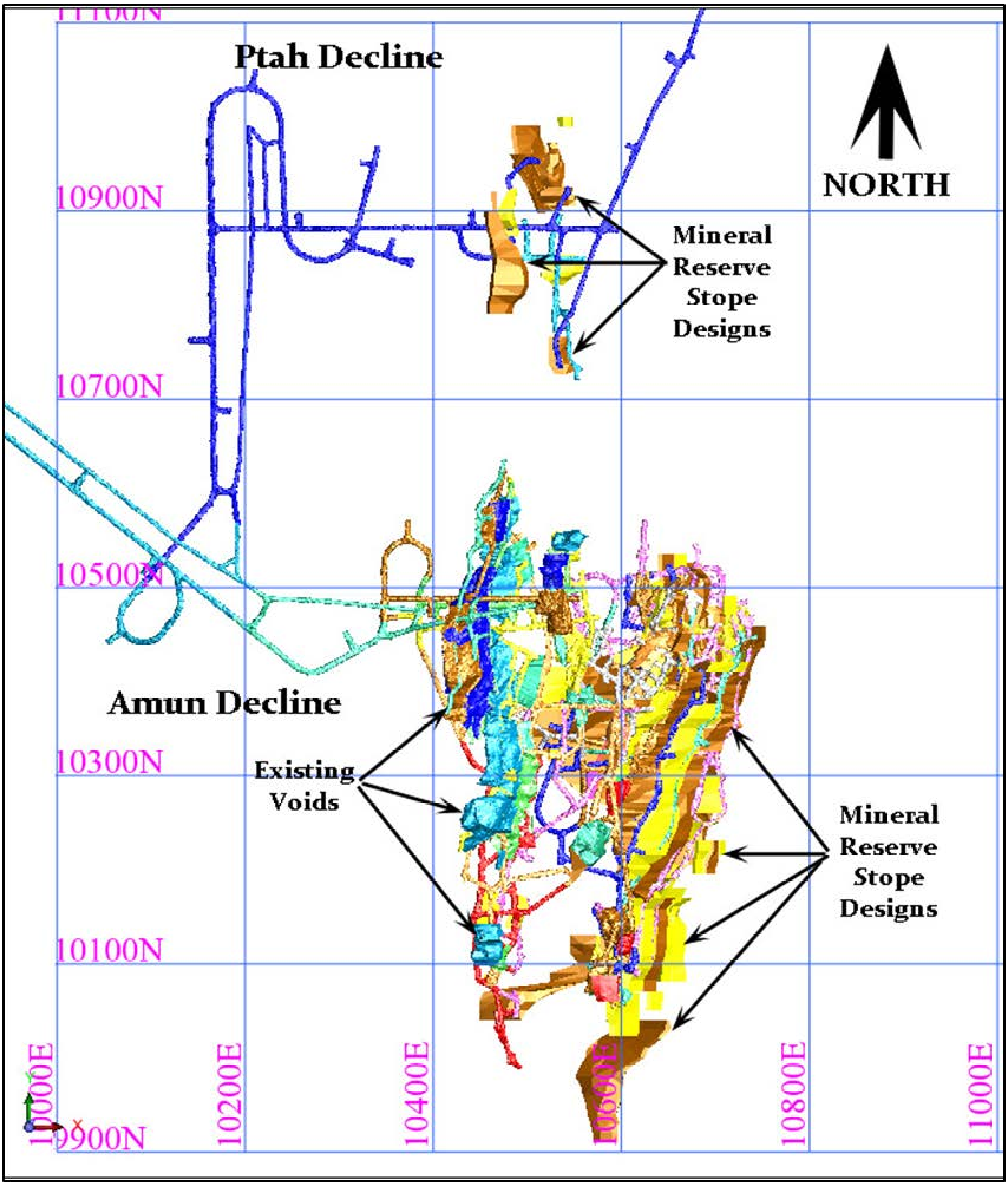


Figure 16.2-4 Long-Section Looking West at Amun

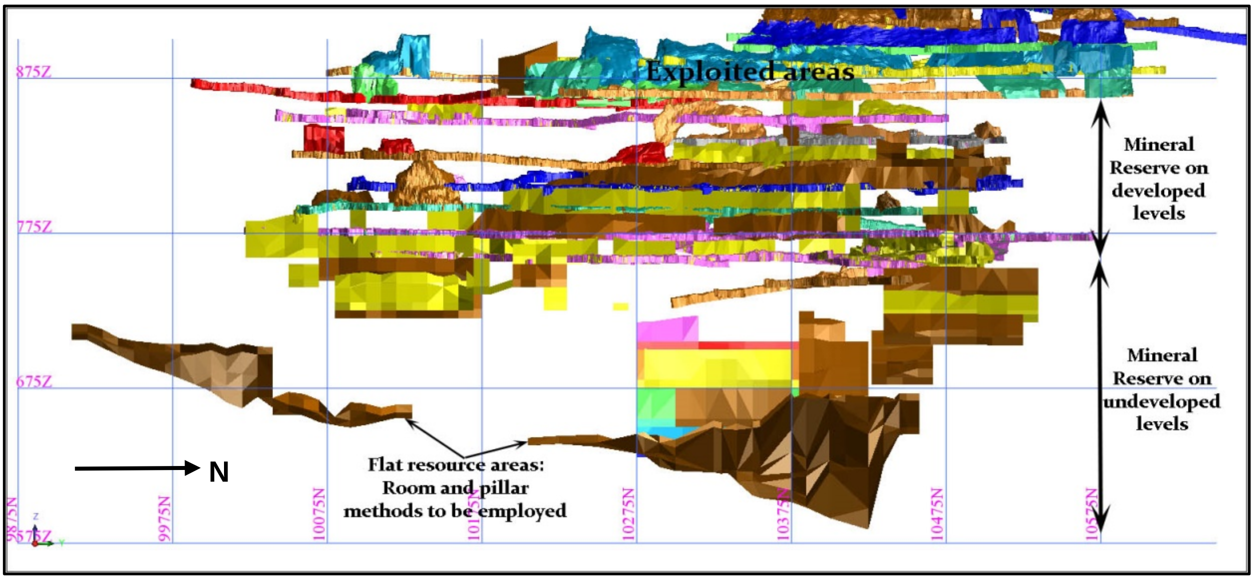
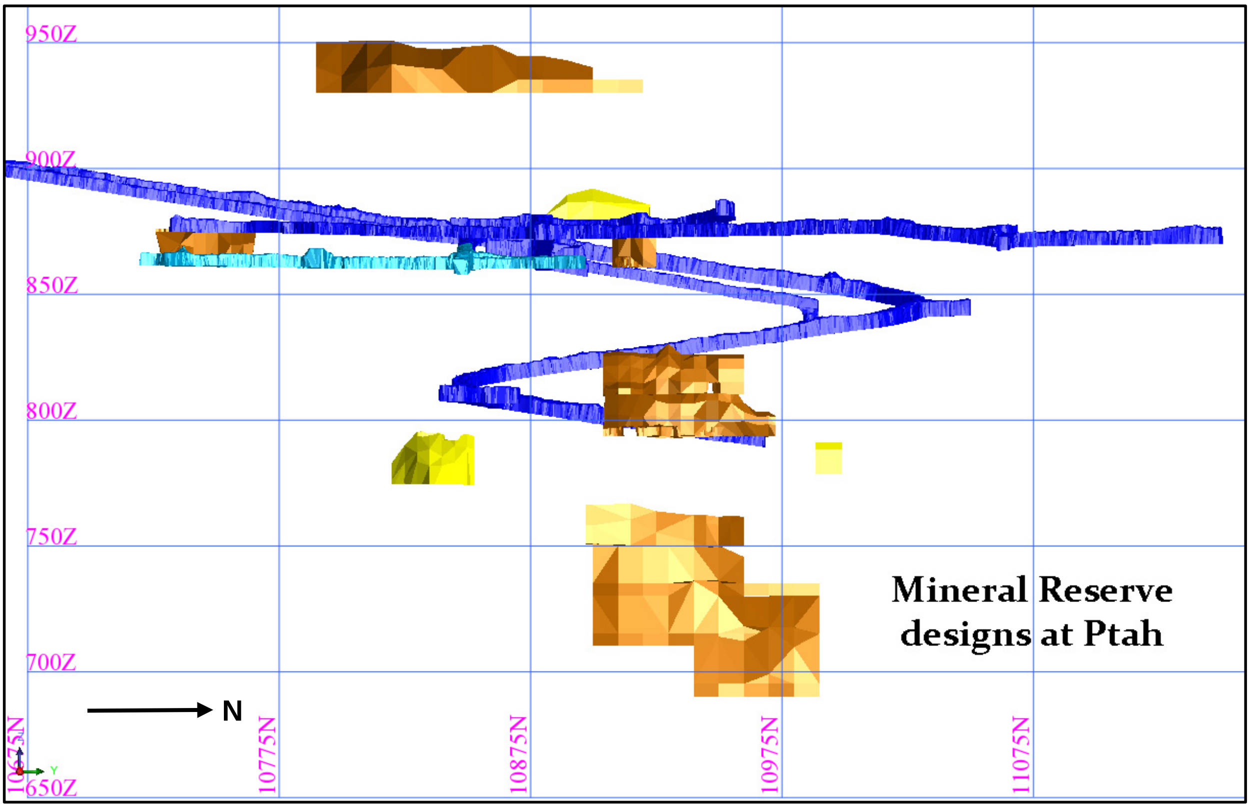


Figure 16.2-5 Long-Section Looking West at Ptah



16.2.5 Geotechnical Considerations

Geotechnical conditions are considered good in the sediments to excellent in the porphyry, although some areas of intense shearing are evident with resulting poor rock mass conditions. These areas of shearing are typically constrained and whilst they will impact on stope performance, they only influence a small proportion of the mine.

A recent independent geotechnical review was completed in March 2015 by Cameron Tucker of Ground Control Engineering Pty Ltd. It was found that the ground control practices and support regimes implemented at the mine were of an acceptable standard.

Stope designs have been based on the body of knowledge that has been gained to date in the development and production of the Amun Decline. Stope strike lengths can be greater than 50 m where consistent mineralization is encountered, and either allowances for support pillars or the use of backfill have been made in the estimation of the Mineral Reserves. Work is underway examining the backfilling options available, including the use of a hydraulic fill utilizing deslimed tailings from the processing plant. The use of this kind of engineering fill medium will likely reduce the reliance on crown and rib pillars, allowing for a higher mining recovery.

17 RECOVERY METHODS

17.1 Process Plant

17.1.1 Process Route

The process route encompasses:

- Crushing
- Stockpiling crushed ore
- Grinding
- Flotation of a bulk sulphide concentrate
- Thickening of the concentrate and the flotation tailings
- Fine milling of the concentrate
- Leaching the precious metals from the concentrate and/or flotation tailings in a dilute cyanide solution
- Adsorbing the precious metals onto activated carbon
- Eluting (or desorbing) the precious metals from the carbon
- Recovering the precious metals as gold doré
- Combining the flotation tailings and leached concentrate and pumping to the TSF

17.1.2 Plant Construction Stages

The process plant has been constructed and operated over a number of staged iterations.

Stage 1 comprised of the coarse ore stockpile, milling and CIL circuits relocated from Kori Kollo combined with a refurbished primary crusher and new elution/gold room facility for the treatment of oxide ore, using whole ore direct cyanidation at the commencement of the operation.

Stage 2 comprised of the addition of a new flotation, thickening, regrind, concentrate CIL and concentrate elution circuits for the treatment of sulphide bearing ore at a design rate of 4 Mtpa of mill feed. The Stage 2 sulphide ore processing circuits were commissioned in April 2010.

Stage 3 comprised of the addition of a secondary crushing circuit designed to reduce the ore feed size to the SAG mill from the initial design of P80 of 105 mm to a P80 of 50mm to allow an increase in the plant throughput rate from the initial design of 500 tph (4 Mtpa) to a new nominal 625 tph (5 Mtpa) since commencing the operation of Stage 3 the mill throughput has increased to 6 Mtpa (680 tph).

The secondary crushing circuit was designed and built for a nominal rate of 10 Mtpa to allow for future, Stage 4, plant expansion. The Stage 3 secondary crushing circuit was commissioned in July 2011. Since commencing operation of the secondary crushing circuit with only one crusher high than designed throughputs have been achieved with only one crusher in operation. As part of the Stage 4 expansion a third secondary crusher has been added to ensure 11 Mtpa capacity is achieved.

Stage 4 comprised of the addition of a new primary crusher, a third secondary crusher, and second train of milling, flotation and thickening, as well as upgrading the existing regrind circuit. The plant expansion was designed to increase the nominal capacity of the process plant from 5 Mtpa to 10 Mtpa. In addition, a new regeneration kiln designed specifically for highly fouled carbon and sized for the gold loadings on carbon at 10 Mtpa will be installed.

It is expected that the ore treated through the new Stage 4 flotation circuit will be predominantly sulphide based ore, amenable to recovery by flotation. Any ore that may be oxide or transitional in nature will be treated through the existing processing circuit by adjustment of the crushed ore product splits to each of the crushed ore stockpiles.

The Stage 4 expansion project has successfully been commissioned and is operating at above the 10 Mtpa design rate.

Table 17.1-1 provides a summary of the design criteria for each stage from which equipment sizing and operating costs are derived.

Table 17.1-1 Summary of Process Plant Design Criteria

Criteria		Units	Stage 1 Oxide	Stage 2 Sulphide	Stage 3 Sec Crush	Stage 4 Sulphide	Source
Ore Throughput		Mt/a	4.6	4.0	5.0	10.0	PGM/SGM
		t/h	580	500	625	1,250	Calculated
Plant Availability	Design	%	91.3	91.3	91.3	91.3	Engineer
Gold Grade	Design	g Au/t	1.55	1.83	1.83	1.50	PGM
Physical Characteristics	BWI	kWh/t	16.2	19.1	19.1	19.1	Testwork
	RWI	kWh/t	23.2	23.2	23.2	23.2	Testwork
	CWI	kWh/t	6.9-35.4	6.9-35.4	6.9-35.4	6.9-35.4	Testwork
Flotation Grind Size	P_{80}	µm	150	150	150	150	Testwork
Concentrate Regrind Size	P_{80}	µm	N/A	12	12	12	Testwork
Overall Plant Recovery	Gold	%	90.8	89.7	89.7	90.9	Calculated
Flotation Tails CIL Residence Time		hours	24	32.4	25.9	23.0	Calculated
Flotation Conc CIL Residence Time		hours	N/A	43.4	43.4	13.1	Calculated
Collector Addition		g/t	N/A	75	75	75	Calculated
Frother Addition		g/t	N/A	20	20	20	Testwork
Flocculant Addition		g/t	18	18	18	18	Engineer
Max. Cyanide Consumption		kg/t	1.74	2.96	2.96	2.96	Testwork
Max. Lime Consumption		kg/t	3.3	3.36	3.36	3.36	Testwork

Major equipment items selected for the plant unit operations are listed in Table 17.1-2.

Table 17.1-2 Major Equipment Items

Description	Type	Source
	Stage 1	
Primary crusher	Gyratory, 54" – 74"	New
Milling circuit	SABC	Kori Kollo
SAG mill	8.53 m Ø x 4.27 m EGL, 5.6 MW	Kori Kollo
Ball mill	5.03 m Ø x 9.29 m EGL, 4.1 MW	Kori Kollo
Recycle crusher	Cone crusher HP500	New
Flotation. Tail CIL Tanks	8 x 3250 m^3	Kori Kollo
Elution circuit	1 x 8 t batch capacity pressure Zadra	New
Tails thickener	23m diameter	Kori Kollo
	Stage 2	
Flotation cells	6 x 100 m^3 tank cells	New
Flotation Tails thickener	23 m diameter	New
Concentrate thickener	14 m diameter	New
Regrind mills	1 x Vertimill 1250 HP, 2 x SMD 355 kW	New
Concentrate CIL tanks: (Stage 2 & 3)	9 x 285 m^3	New
Elution circuit	1 x 2 t batch capacity pressure Zadra	New
	Stage 3	
Secondary crusher	3 x Cone crusher CH870C	New
Vibrating screens	2 x Double deck screens 6.1 x 7.0 m	New
	Stage 4	
Primary crusher	Gyratory TSU, 1400 x 2100	New
Milling circuit	SABC	New
SAG mill	8.53 m Ø x 4.65 m EGL, 7.0 MW	New
Ball mill	6.10 m Ø x 9.92 m EGL, 7.0 MW	New
Recycle crusher	Cone crusher XL300	New
Flotation cells	6 x 120 m^3 tank cells	New
Flotation Tails thickener	25 m diameter	New
Concentrate thickener	15 m diameter	New
Regrind mills (additional)	6 x SMD 355 kW	New
Concentrate CIL Preleach Tank	1 x 285 m^3	New
Regeneration Kiln	750 kg/hr	New

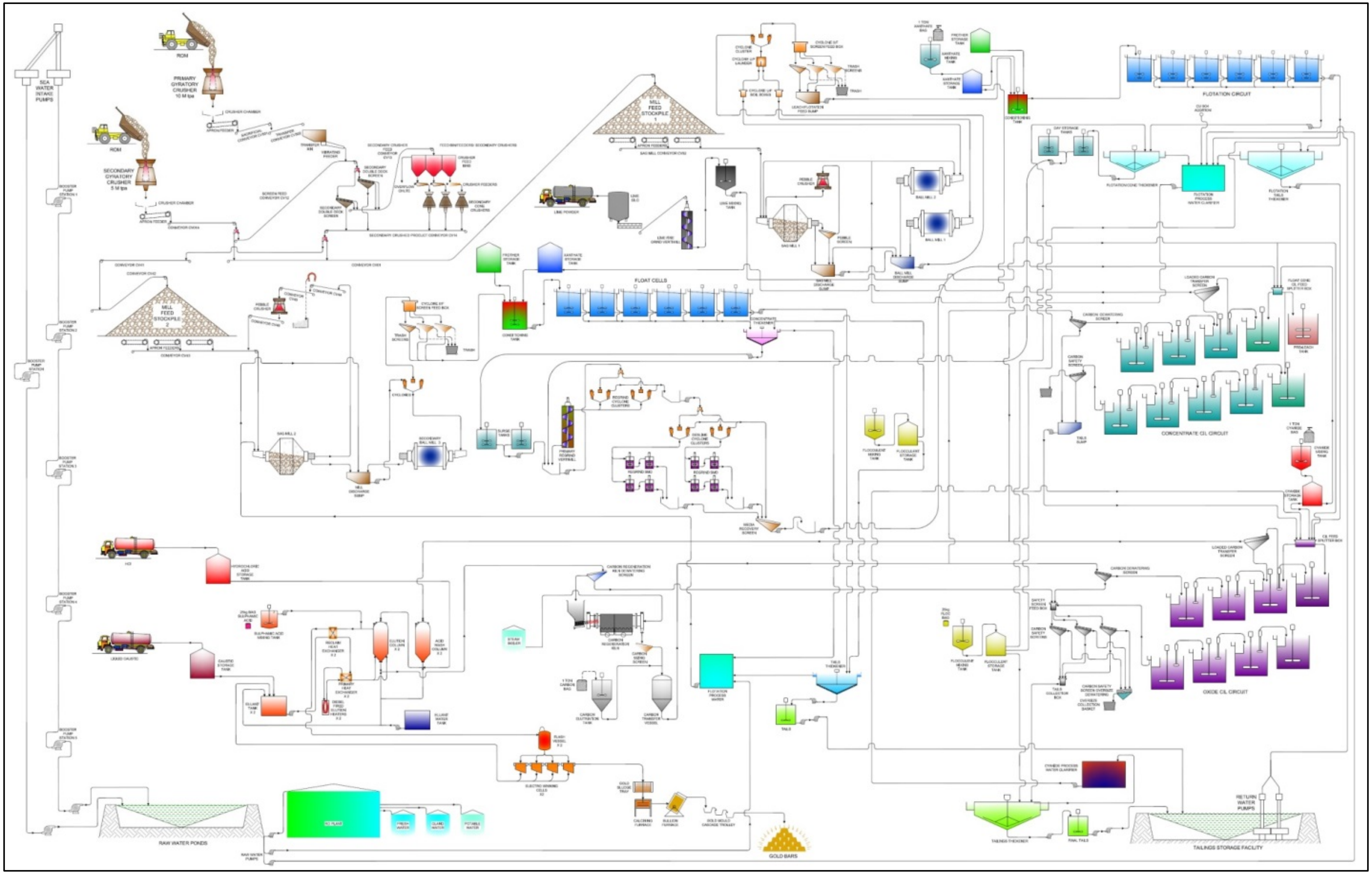
Air compressors are included to provide low and high pressure air for flotation, CIL air lifts, plant utility and instrument air.

Plant controls are through a PLC and SCADA system. Belt weighers and magnetic flowmeters together with process samplers are provided for metallurgical accounting.

Power is distributed to area sub-stations at 11 kV where it is transformed to 4,160 V for the mill drives and 400 V for the balance of drives. Electrical power systems are standard at 400 V and 50 Hz which coincides with the electrical system standards of the Kori Kollo plant.

A process flow diagram depicting the processing route incorporating the Stage 4 expansion is shown as Figure 17.1-1.

Figure 17.1-1 Process Flow Diagram



17.1.3 Primary Crushing and Stockpiling

A new primary crusher capable of crushing 11 Mtpa has been added to the start of the crushing circuit.

ROM ore is dumped directly into the 500 t capacity dual-tip crusher pocket by haul trucks or stockpiled on the ROM pad. The tipping station is designed to handle Caterpillar 789C, or equivalent open-pit haul trucks. Ore from the dump pocket is crushed in a 1,400 x 2,100 mm gyratory crusher, fitted with a 600 kW motor and operating at a nominal 165 mm open side setting. Oversize is broken using a hydraulic rock breaker.

Crushed rock (with a P100 of 229 mm and a P80 of 140 mm), which falls into the primary crusher discharge pocket, is withdrawn by a variable speed apron feeder which, in turn, feeds a short sacrificial conveyor. Chute work under the apron feeder directs fines spillage from the apron feeder onto the sacrificial conveyor.

This conveyor discharges onto a transfer conveyor that discharges onto the screen feed conveyor which feeds two double deck vibrating screens, operating in parallel, fitted with 100 mm and 50 mm screen panels. Screen undersize material (-50 mm) is discharged to the secondary crusher product conveyor. Screen oversize material (+50 mm) is discharged to the secondary crusher feed conveyor and fed into the crusher feed bin, situated above the three secondary crushers. The oversize material passes through two of three CH870C cone crushers (two on duty, one on standby). The cone crushers operate in open circuit and reduce the oversize material to a P80 of 35 mm, prior to discharging on to the secondary crusher product conveyor.

The secondary crusher product conveyor discharges to the stockpile splitter chute diverter gate that splits the crushed product proportionally, according to operator input, to each mill feed stockpile conveyor

Dust suppression is provided by high-pressure water sprays in the tipping area and all material transfer points.

Crushed ore is stored on a 15,000 t live capacity mill feed stockpile for each grinding circuit. If the crushing circuit is off-line for an extended period, the dead portion of each pile can be reclaimed by bulldozer, excavator or front end loader.

For the Stage 1/2 grinding circuit, crushed ore is drawn from the Stage 1/2 mill feed stockpile at a controlled rate by a combination of three 1.2 m wide x 6.0 m long variable speed apron feeders discharging onto the SAG 1 mill feed conveyor. The design capacity of each of the apron feeders is 580 tph and any one or more feeders may be operated at the same time.

Dust suppression is provided in the form of a dust collector on all feeder discharge points.

The SAG 1 mill feed conveyor is fitted with a dual idler weightometer to measure the plant feed rate for plant metallurgical accounting and control of the SAG mill.

For the Stage 4 grinding circuit, crushed ore is drawn from the Stage 4 mill feed stockpile at a controlled rate by a combination of three 1.3 m wide x 5.5 m long variable speed apron feeders discharging onto the SAG 2 mill feed conveyor. The design capacity of each of the apron feeders is 750 tph and any one or more feeders may be operated at the same time.

Dust suppression is provided in the form of a dust collector on all feeder discharge points.

The SAG 2 mill feed conveyor is fitted with a dual idler weightometer to measure the plant feed rate for plant metallurgical accounting and control of the SAG mill.

Secondary crushed ore with a P80 of 50 mm is transferred to a second crushed ore stockpile prior to grinding through the new milling circuit. The new milling circuit is a two-stage circuit, consisting of a SAG mill and ball mill, with hydrocyclone classification and pebble crushing facility. The circuit has been designed to operate 24 hours day, seven days a week with a utilization of 91.3% and a nominal throughput rate of 625 tph.

17.1.4 Grinding and Classification

Two grinding circuits are installed and designed to be operated in parallel. Each grinding circuit is a two-stage milling circuit with hydrocyclone classification and pebble crushing facility. Each circuit is designed to operate 24 hours a day, seven days a week with a utilization of 91.3%.

Due to the lack of a local source of fresh water in the Sukari area, seawater is used for process water and is pumped a distance of 25 km from the Red Sea. Storage capacity at the plant is provided in two lined ponds of 61,000 m^3 total capacity equivalent to two days at maximum demand. A Reverse Osmosis (RO) plant is provided to produce fresh water for the elution process and for domestic uses on site. The reject stream from the RO plant reports back to the seawater storage ponds.

A seven-axis mill liner handler is included in the plant equipment to facilitate removal and installation of SAG mill liner plates and lifter bars. The liners are positioned on the concrete work platform with a mobile crane so that the liner handler can lift them from the transport pallet and position them in the mill.

Stage 1/2 grinding circuit

The Stage 1/2 grinding circuit throughput rate is 700 tph treating secondary crushed ore with a F_{80} size of 35 mm and producing a ground cyclone overflow with a product size P_{80} of 150 µm, suitable for flotation feed.

The SAG mill is an 8.53 m diameter by 4.27 m long (3.81 m effective grinding length) grate discharge mill with a 5,595 kW synchronous motor installed and an air clutch for starting the mill. The mill operates at 76% critical speed with a ball charge of 15% to draw 4,540 kW at the pinion. The mill is fitted with a steel lining system with 40 mm grate apertures and 60 mm pebble ports. The make-up ball size is 125 mm.

Water is added to the SAG mill feed to attain a desired SAG mill discharge density (approximately 75% solids by weight). A flow metre and proportional flow control valve in the mill feed water line allows this to be controlled automatically to a feed rate ratio or to a set point flow rate utilising a PID loop controller in the plant control system.

Ground slurry from the SAG mill is screened by a 2.5 m diameter by 3.7 m diameter steel discharge trommel, manufactured with 12 mm by 100 mm slots. Trommel screen undersize reports to the SAG mill discharge hopper and is pumped by one of two (one on duty, one on standby) variable speed SAG mill discharge pumps to the ball mill discharge hopper.

An ultrasonic level detector is fitted over the SAG mill discharge hopper and provides continuous level indication on the plant control system. The SAG mill discharge pump speed is varied by a PID feedback loop controller to maintain a set point hopper level. Trommel screen oversize reports to the pebble crushing circuit for size reduction before being returned to the SAG mill for further comminution.

Pebbles from the SAG mill are conveyed to the HP500 pebble crusher (cone crusher), operating with a closed side setting of 12 mm. The design crusher feed rate is 150 tph or 30% of new feed to the mill. Crushed pebbles are returned to the SAG mill via the SAG mill feed conveyor.

Trommel screen undersize from the SAG mill reports to the ball mill discharge hopper where it combines with the ball mill discharge streams and is pumped by one of two (one on duty, one on standby) variable speed cyclone feed pumps to a cluster of 12 Cavex 500CVX10 (500 mm diameter) cyclones. Eight cyclones are operating with four spare. Cyclone feed pump speed is varied by a PID feedback loop controller to maintain a set point hopper level.

Cyclone overflow is flotation feed and gravitates to the trash screen feed box, which feeds three trash screens operating in parallel. Cyclone underflow is split into two boil boxes to feed each ball mill.

Cyclone underflow reports to two 5.03 m diameter by 9.29 m long overflow ball mills with rubber linings and fitted with 4,100 kW synchronous motors and air clutches for starting. The ball mills are operated with a 34% ball charge and a make-up ball size of 60 mm. Each mill is fitted with a 2.2 m diameter by 2.4 m steel discharge trommel, manufactured with 20 mm by 100 mm slots. Trommel oversize reports to a scats bay for later collection.

Water is added to the mill feed chutes and to the common mill discharge hopper to attain the desired densities for optimum milling performance.

Stage 4 grinding circuit

The Stage 4 grinding circuit design throughput rate is 625 tph treating secondary crushed ore with a design F_{80} size of 50 mm and producing a ground cyclone overflow with a product size P_{80} of 150 µm, suitable for flotation feed. With the crushing circuit producing SAG feed material with a F_{80} size of 35 mm, it has been shown that the realized throughput is higher than design.

The SAG mill is an 8.53 m diameter by 5.50 m long (4.65 m effective grinding length) grate discharge mill with a 7,000 kW variable speed induction motor incorporating slip energy recovery and a resistance chopper drive for starting the mill. The mill is designed to operate at 75% critical speed with a ball charge of 10% to draw 5,335 kW at the pinion. The mill is fitted with a steel lining system with 25 mm grate apertures and 60 mm pebble ports. The design make up ball size is 125 mm.

Water is added to the SAG mill feed to attain a desired SAG mill discharge density (approximately 70% solids by weight). A flow metre and proportional flow control valve in the mill feed water line allows this to be controlled automatically to a feed rate ratio or to a set point flow rate utilising a PID loop controller in the plant control system.

Ground slurry from the SAG mill is screened by a 2.5 m diameter by 3.7 m diameter steel discharge trommel, manufactured with 12 mm by 100 mm slots. Trommel screen undersize reports to the mill discharge pump box and is pumped by one of two (one on duty, one on standby) variable speed cyclone feed pumps to a cluster of 12 Cavex 500CVX10 (500 mm diameter) cyclones. Eight cyclones are designed to operate with four spare. Cyclone feed pump speed is varied by a PID feedback loop controller to maintain a set point hopper level.

Trommel screen oversize reports to the pebble crushing circuit for size reduction before being returned to the SAG mill for further comminution.

Pebbles from the SAG mill are conveyed to the pebble crusher feed bin and are withdrawn at a controlled rate using a belt feeder to one of two (one on duty, one on standby) XL300 pebble crushers (cone crushers), operating with a closed side setting of 12 mm. The belt feeder speed is controlled by a level indicator situated above each pebble crusher and a PID feedback loop controller to maintain a choke feeding arrangement. Crushed pebbles are conveyed to the pebble crushing circuit surge bin and are withdrawn at a controlled rate using a belt feeder for return to the SAG mill via the SAG mill feed conveyor.

Allowance has been made in the design to retro fit a secondary pebble crushing stage and conveyor feeding crushed pebbles directly to the ball mill if required for additional throughput.

Cyclone overflow is flotation feed and gravitates to the trash screen feed box which feeds three trash screens operating in parallel.

Cyclone underflow reports to the 6.10 m diameter by 9.76 m long overflow ball mill with rubber linings and fitted with 7,000 kW induction motor and resistance chopper drive for starting. The ball mill is designed to be operated with a 33% ball charge and a make-up ball size of 60 mm to 80 mm. The mill is fitted with a 2.2 m diameter by 2.4 m steel discharge trommel, manufactured with 20 mm by 100 mm slots. Trommel oversize reports to a scats bay for later collection and trommel undersize reports to the mill discharge pump box.

The ball mill and mill liner handler has been designed to allow steel lining to be fitted to the ball mill in the future. The ball mill currently has a rubber lining system.

Water is added to the mill feed chutes and to the common mill discharge hopper to attain the desired densities for optimum milling performance.

17.1.5 Flotation

Two flotation circuits are installed and designed to be operated in parallel. The flotation circuits are designed to recover a bulk sulphide concentrate for further comminution and subsequent leaching. The circuits are designed to operate 24 hours a day, seven days a week with a utilization of 91.3%.

Each flotation circuit incorporates a mechanically agitated flotation feed conditioning tank, six mechanically agitated flotation tank cells operating in series, flotation concentrate and flotation tails pumping equipment.

Cyclone overflow material from each grinding circuit reports to the respective conditioning tank of each flotation circuit to feed the flotation cells.

The Stage 1/2 flotation circuit comprises of six by 100 m^3 mechanically agitated flotation tank cells and the Stage 4 flotation circuit comprises of six by 120 m^3 mechanically agitated flotation tank cells.

Air to the flotation cells is provided by two of three (two on duty, one on standby) 0.5 bar blowers. The air volumes and froth depths of each individual flotation cell are controlled by utilising PID loop controllers in the plant control system.

The flotation concentrate from the Stage 1/2 flotation cells is combined in a launder and gravitates to the rougher concentrate pump box and is then pumped to the Stage 1/2 flotation concentrate thickener. Flotation tailings report to the flotation tailings pump box and are pumped to the Stage 1/2 flotation tails thickener.

The flotation concentrate from the Stage 4 flotation cells is combined in a launder and gravitates to the rougher concentrate pump box and is then pumped to the Stage 4 flotation concentrate thickener. Flotation tailings report to the flotation tailings pump box and are pumped to the Stage 4 tails thickener.

17.1.6 Concentrate Thickening and Regrind

The flotation concentrate is thickened and ground to a product size P80 of 12 μm through the regrind circuit before gold recovery treatment in the concentrate CIL circuit.

Two concentrate thickening circuits (one for each flotation circuit) are installed and designed to be operated in parallel. The concentrate thickeners are designed to thicken the diluted flotation concentrate from 20% solids to 60% solids to be suitable as feed to the regrind circuit.

Concentrate from each flotation circuit is pumped into the feed box of the respective concentrate thickener for each circuit. Flocculent and dilution water is added to the concentrate stream as it enters the thickener to assist settling.

The Stage 1/2 concentrate thickener is a 14 m diameter high rate thickener designed to process 25 tph of concentrate and the Stage 4 concentrate thickener is a 15 m diameter high rate thickener designed to process 50 tph of concentrate.

Two mechanically agitated concentrate surge tanks are installed in each circuit (a total of four concentrate surge tanks) and each pair of surge tanks receives the concentrate thickener underflow from each concentrate thickener. The surge tanks have been designed to allow for process interruptions and maintenance in the regrind circuit.

The concentrate overflow from each thickener gravity flows to a process water tank installed in each circuit.

Concentrate slurry from each circuit is pumped from the surge tanks into the regrind cyclone feed pump feed, where the two streams are combined.

The regrind circuit is designed to produce a ground product size P_{80} of 12 microns, sufficient to liberate the fine gold particles for subsequent leaching. The circuit is designed to operate 24 hours a day seven days a week with a utilization of 91.3% and to treat 75 tph of flotation concentrate.

The upgraded regrind circuit consists of a Metso VTM1250 Vertimill in closed circuit with 100 mm cyclones. Cyclone overflow is pumped to additional cyclones for desliming (separation of the fines fraction), with the deslime cyclone underflow being ground further in eight 355 kW stirred media detritor mills (SMDs). The SMDs are arranged in four parallel trains, with each train consisting of two SMDs in series.

The final product, at a p80 size of 12 microns, is a combination of SMD product and deslime cyclone overflow which is then pumped to the concentrate CIL circuit.

17.1.7 Leaching and Carbon in Leach

The CIL circuits are designed to leach gold (and silver) from the ground ore and recover it onto carbon for subsequent recovery by elution electrowinning and smelting. There are two leaching circuits included in the plant. The flotation concentrate CIL circuit recovers gold from the reground concentrate and the flotation tails CIL circuit recovers non-sulphide related gold from the Stage 1/2 flotation tailings and any remaining gold from the concentrate CIL tailings by providing additional residence time for the concentrate stream.

The circuits are designed to operate 24 hours a day, seven days a week with a utilization of 91.3%. The combined design throughput rate is 625 tph treating secondary crushed ore. Viscosity characteristics of the pulps allow use of conventional open tank, moderate power dual impeller agitators to achieve evenly suspended solids and sufficient shear action to achieve satisfactory carbon adsorption kinetics.

17.1.8 Flotation Concentrate CIL

Reground flotation concentrate reports to the concentrate leach feed box and then to the concentrate pre-leach tank. The design throughput rate is 75 tph of flotation concentrate.

The concentrate CIL circuit consists of ten mechanically agitated leach/adsorption tanks in series interconnected with launders to allow the slurry to flow by gravity through the tank train. The first tank is a 265 m^3 pre-leach tank with oxygen injection through an oxygen shear injector to pre-oxidize the concentrate prior to the introduction of cyanide.

Oxidized concentrate reports to the 265 m^3 leach tank followed by eight 247 m^3 (active) CIL tanks. Cyanide is dosed into the leach tank and provision has been made for further stage dosing of cyanide in the other CIL tanks if required.

Slaked lime slurry is added to the pre-leach tank for pH control using lime control valves and PID loop controllers in the plant control system.

A bulk oxygen tank has been installed to provide oxygen to the CIL circuits for pre-oxidation of sulphide concentrates and increased leaching kinetics throughout the CIL circuits. Oxygen is sparged in the leach tank and a number of the CIL tanks to enhance leaching kinetics.

The launder system of the circuit allows individual tanks to be taken off line for maintenance without unduly compromising recovery.

Carbon is advanced counter current to the pulp flow through the CIL tanks using airlifts to maintain the required carbon concentrations for high gold adsorption efficiencies and a low solution tail. Each CIL tank is fitted with a mechanically swept, wedge wire inter-tank screen to retain carbon. Loaded carbon is recovered from the first CIL tank by pumping to a vibrating loaded carbon screen and barren carbon is returned to the final CIL tank in the circuit after stripping and carbon regeneration.

The concentrate CIL tailings are directed to the feed of the flotation tails CIL circuit for additional recovery.

Leach tails gravitate to the carbon safety screen. Screen oversize is collected in drums and returned to the circuit. Screen undersize reports to the variable speed concentrate tailings pump and is directed to the feed of the Stage 1 CIL circuit.

17.1.9 Flotation Tails CIL

The flotation tails are thickened to recover the maximum amount of uncontaminated water and then report to the flotation tails CIL circuit, where the pulp is diluted to the target density using cyanide process water, for the recovery of non-sulphide associated gold.

Tailings from the Stage 1/2 flotation circuit are pumped to the Stage 1/2 flotation tailings thickener collection box and from there gravitate into the 23 m diameter high rate flotation tailings thickener. Flocculant and dilution water is added to assist in settling of the solid.

Thickener underflow is pumped to the leach feed distribution box by one of two (one operating, one standby) variable speed flotation tailings leach feed pumps where it is combined with the concentrate CIL tailings. Thickener overflow gravitates to the process water tank.

The CIL circuit consists of eight mechanically agitated leach/adsorption tanks in series interconnected with launders to allow the slurry to flow by gravity through the tank train. The first tank is a 3,250 m^3 leach tank followed by seven 3,050 m^3 (active volume) CIL tanks. Eight tanks in series were selected to maximize leach recovery. A single leach stage has been utilized to attain higher gold in solution tenor in the first stage of adsorption thereby allowing higher gold on carbon loadings.

Cyanide is dosed in to the leach tank and provision has been made for further stage dosing of cyanide in the other CIL tanks if required.

Slaked lime slurry is added to the leach tank for pH control using lime control valves and PID loop controllers in the plant control system.

Oxygen is sparged in the leach tank and a number of the CIL tanks to enhance leaching kinetics.

The launder system allows individual tanks to be taken off line for maintenance without unduly compromising recovery.

Carbon is advanced counter current to the pulp flow through the CIL tanks using airlifts to maintain the required carbon concentrations for high gold adsorption efficiencies and a low solution tail. Each CIL tank is fitted with a mechanically swept, wedge wire inter-tank screen to retain carbon. Loaded carbon is recovered from the first CIL tank by pumping to a vibrating loaded carbon screen and barren carbon is returned to the final CIL tank in the circuit after stripping and carbon regeneration.

Leach tails gravitate to the carbon safety screen feed boil box and then to three vibrating carbon safety screens.

The combined flotation tails and CIL circuit tailings are thickened and discharged to the tailings storage facility.

17.1.10 Tailings Thickening and Disposal

The tailings thickening and disposal circuits are designed to thicken the tailings stream to recover water for reuse and minimize the volume of the supernatant lake in the tailings storage facility.

The tailings thickening and disposal circuits are designed to operate 24 hours a day, seven days a week with a utilization of 91.3%. Tailings from the Stage 1/2 flotation tailings CIL plant is screened and reports to the 23 m diameter high rate tails thickener and flotation tailings from the Stage 4 flotation circuit report to the 25 m diameter high rate tails thickener.

Flocculent and dilution water is added to each thickener to assist in settling of the solid.

Thickened tailings residue from each thickener is pumped to the TSF utilising separate tailings pumps and tailings discharge line for each circuit.

Supernatant liquor from the TSF is recovered by a decant system and returned to the plant in either the cyanide process water or the process water tank. Due to the low concentration of cyanide in the TSF return water, copper sulphate is added to the process water tank to prevent the contained cyanide species from depressing the sulphide minerals in flotation.

17.1.11 Gold Recovery

There are two separate elution and electrowinning circuits due to the large difference in elution plant capacity required for the different stages of the operation.

Both circuits are pressure Zadra type circuits. Pressure Zadra circuits were selected because of their lower requirement for high quality water. A two-tonne carbon capacity circuit is installed for the flotation concentrate CIL carbon elution duty. An eight tonne carbon capacity circuit is installed for the flotation tail CIL carbon elution duty. The operation of both circuits is essentially the same with the exceptions that the smaller circuit has a separate acid wash column and an integrated pressure electro winning cell.

The following operations are carried out in the elution and gold room areas:

- Acid washing of carbon
- Stripping of gold and silver from loaded carbon using a Zadra elution circuit
- Electro-winning of gold and silver from pregnant solution
- Smelting of electro-won products

The elution and gold room area operate up to seven days a week, with the loaded carbon recovery on night shift and the majority of the elution occurring during day shift.

17.1.12 Carbon Regeneration

Upon completion of the elution process, the barren carbon is transferred from the elution columns to the carbon regeneration facility. The carbon regeneration facility has been designed to operate 24 hours a day, seven days a week with a utilization of 91.3%.

Barren carbon reports to the sand removal column for separation of sand and grit from the carbon and the carbon then gravitates to the carbon dewatering screen.

Dewatered carbon reports to one of two regeneration kiln hoppers via a diverter chute. The kiln hoppers are designed in such a way that one hopper will receive fresh batches of carbon whilst the other is feeding the kiln. Dewatering feeder screens at the base of each hopper provide additional carbon dewatering and feed the kiln at a controlled rate.

The kiln is a horizontal carbon regeneration kiln designed to heat 750 kg/h of carbon up to 850°C and hold it at that temperature for 20 minutes in a steam atmosphere as required to remove flotation reagents from the carbon and reactivate the carbon.

Steam is supplied to the regeneration kiln by a boiler fitted with automated burner management and steam control.

Regenerated carbon is cooled by passing through a fluidized bed cooler and passes over the discharge carbon screen to remove carbon fines generated during the regeneration and carbon cooling process.

The screened carbon is wetted and conditioned in a conditioning tank and is continuously pumped back to the dewatering screens on either CIL circuit.

17.1.13 Reagents

The main processing reagents are:

- Collector (Potassium Amyl Xanthate)
- Frother (MIBC)
- Quicklime (CaO)
- Sodium Cyanide (NaCN)
- Sodium Hydroxide (Caustic Soda, NaOH)
- Hydrochloric Acid (HCl)
- Copper Sulphate ($CuSO_4$)
- Grinding Media
- Flocculent
- Activated Carbon
- Dump Leach

A dump leach operation is operated in parallel with the process plant for the treatment of low-grade oxide ore of less than 0.59 g/t Au.

Ore is dumped by dump trucks onto a lined pad. The leach pad is constructed with a compacted synthetic membrane liner over an HDPE lined compacted base. A 0.5 m thick layer of scats is placed onto the pad before dumping commences.

Leaching takes place by applying a dilute cyanide solution to the top of the ore stacked on each dump. Leach solution is applied by means of a sprinkler system to ensure maximum coverage of the ore.

The leach pad is surrounded by a berm to contain the leach solution and the leach solution is collected in lined channels to direct it to the HDPE lined pregnant liquor pond. Overflow from the pregnant liquor pond reports to the HDPE lined intermediate pond.

Pregnant solution is pumped from the pregnant liquor pond either to a series of carbon columns or to the processing plant Stage 1 CIL circuit for recovery of gold onto carbon.

The dump leach intermediate pond provides the cyanide solution feed for pumping to the top of the dump leach. Make-up water is supplied from the TSF return water. Cyanide solution is piped from the process plant to the dump leach intermediate pond and is transferred as required to maintain the setpoint cyanide concentration. Quicklime is added to the pond to maintain the required pH.

An additional HDPE lined pond is connected to the intermediate pond by means of an overflow. This pond functions as an emergency containment pond for excess leach solution gravity flowing from the dump in case of a major pumping failure.

18 PROJECT INFRASTRUCTURE

18.1 Plant Site Geotechnical Investigations

A report has been prepared by KP (Tailings Storage Facility Design and Plant Site Foundation Assessment, Feasibility Study PE 401 – 00015/5, July 2006) that provides details of the plant site foundations at Sukari.

The process plant site is located on a flat area with a gentle downward slope to the north west of 1 m vertical:100 m horizontal. The hill on which the Primary Crusher is located (at the south east end of the plant site) has a slope of approximately 1 m vertical:5 m horizontal.

The ground surface at the plant site consists of alluvial wadi gravel. The wadi material generally consists of sub-angular sandy gravel that commonly contains cobbles and boulders. The fines content of the gravel is generally less than 10%, and the gravel, cobbles and boulders are composed of metavolcanics/tuff, porphyry/granitoid, quartz and metasediments lithologies. The main difference in near-surface ground conditions across the plant site is the depth of the unconsolidated wadi gravel, sand and cobbles/boulders observed in the test pits and drillholes.

The bedrock in the vicinity of the plant site is from the calc-alkaline volcanic and subvolcanic group. The drilling program at the plant site identified gabbrodiorite bedrock at the Primary Crusher location and lapilli tuff and agglomerate bedrock to the north west of the Primary Crusher. The gabbro-diorite bedrock is medium to high strength and the lapilli tuff and agglomerate is low to medium strength. Strength of both rocks generally increases with depth. Both rock types are generally distinctly weathered.

Groundwater levels were monitored in three of the five drill holes for approximately one week after the holes were drilled. Water levels continued to fall during that time with the lowest depths of the water table being between 15 m and 21 m below grade, suggesting that the water levels recorded do not represent static water levels.

The foundation conditions for the constructed process plant were deemed suitable for stability of the installed equipment.

There are no significant geotechnical issues at the process plant or crusher locations.

The plant site is located in a major wadi channel. Surface runoff during the infrequent precipitation events may have some potential to mobilize and erode the sand and gravels from around footings. Risks have been mitigated with surface water diversion channels and bunds.

18.2 Tailings Storage Facility

18.2.1 TSF Overview

Knight Piésold Ltd (KP) was engaged by PGM to design the TSF for the Sukari mine site.

Site surveys and geotechnical investigations were carried out at alternate locations and a site was chosen immediately adjacent to the proposed plant site.

To avoid tailings and strongly saline water penetrating the wadi groundwater, the TSF basin is lined with 1.5 mm thick HDPE material.

Tailings is deposited by sub-aerial techniques and decant water is pumped from a floating barge back to the plant process water tank.

The KP design carried out was for a total of 48 Mt tailings capacity and SRK Consulting (SRK) was engaged by SGM in 2013 to review the design and further design a southern tailings compartment and subsequent upstream wall embankment lifts to increase the total tailings capacity to 68 Mt.

The most recent design for the TSF is based on the following design parameters:-

- Total storage is 68 million tonnes.
- Throughput is 11 million tonnes per annum.
- Tailings slurry percent solids are 45%.
- Flotation tailings and leached CIL tailings are stored in a single facility.

18.2.2 Geotechnical Investigation

A geotechnical investigation of the site was undertaken. In general the site is characterized by two distinct surface condition types, with a well-defined boundary separating these two.

Rock outcrops are found where the topography is steep. Their degree of weathering varies with the topography, making them difficult to utilize for infrastructure development. They are usually quite fractured and vary locally between low and high strength.

The other typical geotechnical features at the site are the large, broad alluvial in-filled valleys that span between the hills. These areas are sometimes described as “wadi” sands and gravels. At the TSF site there is typically ~1.7 m of transported non-cohesive soil over the fractured bedrock.

18.2.3 Tailings Characteristics

Three samples of tailings were generated from the metallurgical bench scale testwork. The samples are described as follows:

1. Flotation tailings – waste product of flotation cycle (not leached)
2. Leached flotation tailings – waste product of flotation cycle, after CIL processing
3. Leached flotation concentrate (LFC) tailings – primary product of flotation cycle, after CIL processing

Both physical (settling, consolidation and drying) and geochemical testing was undertaken. The physical testwork indicated:

- The tailings settles relatively quickly (within 1-2 days), releasing about 60-70% of total water as supernatant and achieving reasonably high densities.
- The tailings dries well, achieving densities of about 1.6 to 1.7 t/m^3 after evaporation losses of around 50 mm.
- The permeability of the tailings was around 10-6 to 10-7 m/s.

The geochemical testwork for the tailings indicated that:

- The flotation tailing is classified as non-acid-forming.
- The leached flotation concentrate is classified as potentially acid-forming.
- Both samples were enriched in known toxicants such as arsenic, molybdenum, antimony, chromium and selenium, and pose a significant environmental potential impact. Therefore lining of the tailings basin is required to reduce seepage.

18.2.4 Site Water Management

The water balance models for the TSF indicate that the water balance for the site is strongly negative under all conditions, due to the low rainfall and high evaporation rates that are characteristic of Egypt’s arid eastern coast region. A large volume of “make-up” water (typically 60-70% of total process water requirements) is needed to be extracted via the Red Sea intake.

The TSF has been designed to contain the 1-in-100-year extreme rainfall event at every stage. Any rainfall exceeding this event will be released in a controlled manner using an emergency spillway structure.

18.2.5 Tailings Storage Facility

The TSF consists of a “paddock” type impoundment, and is operated as a single cell receiving both Stage 1/2 leached tailings and Stage 4 flotation tailings.

The geochemical properties of the tailings together with the relatively high permeability of the in situ soil materials (wadi sands and gravels) mean that there is potential for contaminated seepage to occur. Given this potential and the proximity of the project area to the environmentally sensitive Red Sea, KP and SRK have incorporated an HDPE liner throughout the basin area and embankments. The design also includes an extensive underdrainage system, which will serve the dual functions of reducing the potential for seepage to occur through the bottom of the TSF basin and increasing the density that is achieved by the tailings.

Deposition of tailings into the facility uses the sub-aerial technique in order to increase the density that the tailings can achieve in storage.

The embankments of the TSF are designed to utilize upstream construction methods to achieve up to 33 m of embankment height.

18.3 Buildings

A mine camp facility and various mine buildings are required for the Sukari Gold Plant Project. The mine buildings are located within and adjacent to the processing plant compound. The camp facility is separate and is located approximately 1 km NNE of the processing plant.

The administration office complex and all other buildings located at the mine site are modular steel clad buildings custom built to suit the particular requirements. The camp facility consists of a mix of modular steel clad buildings and traditional brick domed-roof buildings.

Maintenance/workshop buildings are provided both outside the process plant for mining equipment maintenance and within the process plant area for fixed plant area. Workshops are provided with facilities for offices, ablutions, storage, tools and equipment, boiler-making/fabrication, machining and rubber works.

18.4 Communications

A communications network is established utilising satellite technology and fibre-optic communication for voice, fax, internet and PC network traffic. The communications and IT infrastructure comprises satellite link, PABX, ethernet LAN, IT servers, desktop computers, A4 & A3 printers, photo-copiers (b/w & colour), plotter, fax machine, UPS, copper and fibre cabling, and site two-way radio system.

VSAT satellite equipment comprises satellite antenna, transceiver, modem and bandwidth manager.

Fibre-optic connection is to the national communications fibre-optic network that runs along the Red Sea coast.

Ethernet LAN points are provided in all administration offices, mine office, stores and workshop.

A “trunked” repeater system provides the infrastructure to enable hand-held and mobile radio sets to communicate around the site.

All the satellite equipment and communications racks are protected by lightning protection and earthing systems.

IT system and desktop computers software programming is performed by the Operations IT personnel.

18.5 Security

To maintain controlled access to the plant areas, all persons entering the site are required to pass the continuously manned security gatehouse/boom gate.

A high security area fence is installed around the process plant site, offices/buildings, fuel depot, power stations and water ponds.

Added security in the form of a high security reinforced concrete building is provided for the gold room housing the electrowinning cells.

The security area fences are 2.4 m high wire mesh type fencing.

18.6 Power Supply

Power supply for the Stage 1/2/3 process plant and site infrastructure is supplied by a power station containing five MAK (Caterpillar) generator sets each capable of supplying 6.5 MW (de-rated) and six Cummins generator sets each capable of supplying 1.2 MW (de-rated).

The MAK and Cummins generator sets supply a common feed bus and are controlled from a central control room.

An additional Wartsila power station was installed in 2013 to supply power to process plant equipment installed on the Stage 4 expansion. This power station consists of five Wartsila generator sets each capable of supplying 7.8 MW (de-rated). The Wartsila station is a complete stand-alone installation with a control room, workshops, fuel storage and all auxiliaries.

The Wartsila and MAK generators are operated using diesel fuel but provision has been made in the design for them to be operated with heavy fuel oil (HFO).

A 20 MW bus-tie has been installed between the MAK and Wartsila power stations to be used if and when required to transfer 18 MW (de-rated) loads between stations.

18.7 Water Supply

Sukari is located in an arid desert terrain with little or no vegetation, where only a few Bedouin people live a marginal existence. There is no surface water and no regular rainfall. Rainfall is limited on average to one torrential downpour once every six or seven years.

Hydrological assessments commissioned by Centamin have determined that it will not be practical to develop ground water sources to consistently supply process plant requirements. Water is drawn directly from the Red Sea and if necessary from coastal bore holes located very close to the coast line where seawater freely infiltrates into the groundwater, using submersible pumps.

Three pipelines are installed with two sets of sea water intake pumps and booster stations to transport the water from the Red Sea to the Sukari mine site to, a distance of approximately 25 km. The combined capacity of the three pipelines is 1,700 m^3/h, which is sufficient to satisfy all process plant and mining requirements.

The seawater pipelines report to the raw water ponds located within the processing plant area.

Reverse osmosis water treatment plants draw a portion of the seawater for potable and fresh water supplies.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

No formal marketing studies have been completed.

Gold can be readily sold on numerous markets throughout the world and it is not difficult to ascertain its market price at any particular time. Since there are a large number of available gold purchasers, Sukari is not dependent upon the sale of gold to any one customer. Gold can be sold to various gold bullion dealers or smelters on a competitive basis at spot prices.

Gold doré bars are transported from the project site to an accredited gold refiner for smelting and refining into an LME grade gold bar on a regular basis, and the refined product credited to the company's revenue account.

For the economic analysis, a gold price of US$1,300/oz was used.

19.2 Contracts

SGM has numerous contracts with local and international companies pertaining to the operation of the mining and processing operations.

The estimated bullion transport costs, liability charges and refining costs used for the financial analysis were based on contract prices agreed with third party contracts. Foreign investments in the petroleum and mining sectors in Egypt are governed by individual revenue sharing agreements (Concession Agreements) between foreign companies and the Ministry for Petroleum and Mineral Resources or EMRA (as the case may be) and are structured as individual Acts of Parliament. Through its wholly owned subsidiary, PGM, Centamin entered into the Concession Agreement with EGSMA (now EMRA) and the Arab Republic of Egypt granting PGM and EMRA the right to explore, develop, mine and sell gold and associated minerals in specific concession areas located in the Eastern Desert of Egypt. The Concession Agreement came into effect under Egyptian law on 13 June 1995.

In 2009 SGM selected Barminco Egypt Underground Mining Services SAE (Barminco Egypt) as the successful tenderer to provide works and services for underground mining services at the Sukari Project.

The agreement had an initial term until December 2012 or the date of termination, whichever was the earlier. The agreement has now been extended to 30 November 2016. The contract will automatically continue after 30 November 2016 on a rolling basis unless either party terminates the agreement by giving the other party three months' prior written notice of termination at any time but not earlier than 3 months prior to the expiry of the term."

Under a Blasthole and Grade Control Drilling Agreement dated 9 April 2009 between PGM and Capital Drilling (Egypt) LLC (Capital), PGM engaged Capital to undertake blast and grade control drilling. The agreement commenced on 2 April 2009 for three years. A subsequent agreement with SGM was entered into commencing 1 December 2013 for a further one-year term, which has since been subsequently further extended.

Under the current Refining Agreement dated 13 December 2013 between SGM and Johnson Matthey Limited (assigned to Asahi Holdings Ltd) (''Asahi''), SGM must deliver all of the production of gold/silver doré from the Sukari mine to Asahi's appointed secure carrier at the Sukari mine for refining at its refinery in Ontario, Canada. The risk of loss and damage to the gold/silver doré passes from SGM to Asahi upon stowage of the material into the carrier's vehicle at the gold room at the Sukari mine. The agreement commenced on 1 March 2014 for an initial two-year term.

All the above contracts contain terms, rates and charges that are within industry norms.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

A core value at Centamin is to minimize the risks of an environmental impact resulting from its operations to a reasonably practical level. Centamin believe that environmental responsibility is an inherent part of to our business. In that respect, we have designated programs and systems to ensure that the environment is streamlined in all phases of our exploration, development, mining and processing activities.

Centamin's efforts in this area are guided by Centamin's corporate environmental policy that aims to ensure environmental protection and sustainable development. The policy outlines our commitment to pollution, safeguarding the environment, educating our employees and communities and applying sound environmental management practices to minimize and prevent environmental impacts.

20.2 Environmental Studies

In 2007, an Environmental and Social Impact Assessment (ESIA) was completed by Environics, an Egyptian company that specializes in Environmental Management Systems, and was approved by the Egyptian Environmental Affairs Agency.

Various addenda have subsequently been prepared, submitted to the authorities and approved. The ESIA and its addenda were prepared according to the Equator Principles.

The ESIA and its addenda set the framework for the environmental management scheme that is currently adopted at Sukari and delineate measures that should be taken to prevent or minimize potential impacts. As an integral part of the ESIA process, a comprehensive public consultation and stakeholders engagement component was undertaken. Public Consultation meetings were held in Marsa Alam during the course of preparation of the EIA. Individual and focus group meetings were also held. All concerns raised were satisfactorily considered or integrated in project design and plans. No significant objections are outstanding.

Baseline information is investigated and assessed in the ESIA. Baseline information included:

- Climate
- Geotechnical stability (global assessment), geomorphology and regional geology
- Hydrology (desktop level and field work with assessment of existing wells)
- Surface water assessment (including flooding)
- Biological environment including flora, fauna and habitats
- Heritage assessments
- Social, land use issues and project neighbourhood

Potential impacts are identified for construction, operation and closure phases. The site and project specifics as well as the project design details are analyzed in conjunction with identified impacts. Impacts were analyzed to indicate:

- Cases of no impacts where no further analysis is required
- Insignificant impacts that will be considered in the project management plan
- Significant impacts for which mitigation measures are implemented to render them either nil or insignificant.
- Residual impact are investigated and mitigated through management procedures

The environmental management plan includes measures taken to minimize and/or eliminate environmental impacts during construction, operation and closure. These preventive and/or mitigation measures are:

- Measures integrated into the design of project components (example selections and or decisions of locations or technologies applied in the project as well as addition of pollution abatement components)
- Management plans, procedures and instructions (such as preventive maintenance, or operational control procedures)
- Protective measures (such as personal protective equipment and workers awareness and training)
- Follow-up measures (including audits and monitoring plans)

20.3 Legislative Background

As a minimum, all Egyptian legislations and standards pertaining to the environment are considered. On site monitoring of various aspects such as air and water quality are undertaken on a monthly basis to ensure compliance with the relevant standards. External certified laboratories, recognized by the Egyptian Government, conduct quarterly or bi-annually monitoring to ensure compliance

The ESIA outlines the relevant legislation that will have to be complied with. Although the relevant articles of law are disjointed and do not fall under a single comprehensive act, the essence of such appears to be similar to that which would be applied to comparable operation in, for example, Australia.

20.4 Environmental Management Scheme of Sukari

SGM strives to maintain high standards of environmental performance and meet, and when practical exceed, relevant legal requirements. The system is supported by a robust multilayered training scheme and a comprehensive documentation system that ensures the maintenance of required registers, documents and renewal of required permits. The environmental management system addresses, inter-alia, waste management, material, water and energy management, management of hazardous substances and chemicals and biodiversity management.

A core element of SGM's environmental management system is to assess performance against objectives and obligations. The results of the assessment provide feedback and assurance regarding the level and effectiveness of controls and feed in needed performance enhancement initiatives and actions. An environmental monitoring system is in place to track performance on a periodic basis with different frequencies and approaches. Progress is evaluated through several tools including visual inspection, auditing, data collection and inventories, measurements and as well as systematic observations. Each of these tools has its own program and iterations. Monitoring is undertaken through SGM's in-house capabilities as well as third parties on a periodic basis.

The Sukari monitoring plan addresses different facets of the environmental aspects:

- Water quality, tailing storage facility water quality, groundwater quality and sewage;
- Air quality, air emissions and dust;
- Work environment parameters including dust, noise, illumination;
- Waste management practices;
- Potential impacts on biodiversity; and
- Potential impacts on cultural heritage.

SGM has focused on strengthening the in-house monitoring capabilities. A self-sustaining monitoring station was built with different equipment such as ambient and point source gas analysers, dust meters, noise meters, water kits and meteorological conditions. The station is managed by experienced competent personnel.

20.5 Project Permitting

Sukari is subject to a large number of permits. These permits are a function of:

- Project components - activities and material (source of impact)
- Site and its attributes (recipient of impact)

Each permit has its own conditions that are to be integrated in the design, construction or operation of the project component it addresses.

The permit application and processing requires the submittal of documents, data, maps and drawings, holding meetings with technical officers and gibing presentations, inspection of the site by officers as well as paying processing fees.

Sukari maintains a permit tracking system compiling all permits while indicating needs and timings for renewal and/or extension.

The permits include, inter alia:

- Environmental permit (approval of the ESIA)
- Permits for establishing the direct water intake
- Permits to establish beach wells
- Army permits to establish project components
- Permits related to use, import and purchase of explosives
- Clearance for the utilization of the concession area from the archaeological point of view

20.6 Social and Community Relations and Support

The Company is committed to making a long lasting positive impact on the communities where we do business. Centamin recognizes that it has a responsibility in supporting and enhancing the community in which it operates as a good corporate citizen. Enhancing the community and its livelihood will also reflect positively on our long-term blending in the community.

Prior to the commencement of development activities, stakeholder involvement was initiated as part of the process of the company's environmental and social impact assessment. For the Sukari operations, the Company has undertaken a comprehensive stakeholders' analysis to identify different groups and their interests. Our stakeholder engagement system maintains continuous communication and involvement with local community as well as local, regional and national entities and authorities.

The Company nurtures dialogue and builds relations with the local community in areas in which we operate. A public consultation has been maintained since the project design phase, during construction phase and through operation. The Company maintains open channels of communication with all our stakeholders for the purpose of information disclosure, raising concerns and grievances.

The Company has established a number of projects and initiatives to support provision of infrastructure for deprived areas and settlements. These activities are in response to needs assessments periodically updated and coordinated with governmental plans and projects. Such projects are implemented in different approaches from direct interaction with the community, though local NGOs and community support associations or through full cooperation with the governmental authorities. The Company has also supported projects to enhance health and livelihood conditions as well as education.

20.7 Waste Management

The Company generates waste as a result of our operations and activities. The Company strives to reduce and minimize the waste volume generated, and disposed of, at Sukari. Two types of waste are generated; mineral waste and non-mineral waste.

Mineral waste includes overburden waste rock and tailings. These are managed through placement in designed engineered structures that satisfy the following requirements:

- Being physically stable under all conditions
- Being chemically stable so as not to represent any harm to environment
- Being able to be closed in a manner compatible with surrounding landscape without impacts on the environment

Non-mineral waste includes a variety of waste. A waste management system is in place at Sukari. The system outlines the proper handling, storage and disposal of waste. The system is based on a full understanding of the value of waste we generate. Upon a detailed characterization of all types of waste and classifications to the waste as per nature and value, he system indicated the procedure needed to deal with the different wastes. The nature and value of the waste and the possibility for reuse, recycling or benefiting from it determines the method of managing the waste, within the hierarchy of waste management. The system is focused on:

- Waste minimization through maintaining an average stock for chemicals to avoid the cases of expired chemicals;
- Maximizing on-site recycling and reuse of different types of wastes;
- Recovery of valuable material;
- Reuse of treated wastewater streams; and
- Off-site recycling and treatment of waste.

Waste is recycled and reused to the maximum practical level. Material that can’t be recycled is disposed on site in a manner that is environmentally acceptable.

20.8 Remediation Obligations

Centamin is committed to leave a positive legacy for coming generations and development initiatives, and this includes early planning for the closure of the mine. The planning aims to ensure that mining activities are soundly phased out, the mine is closed in an environmentally sound manner, a physically and chemically stable landform is maintained, with minimal erosion and minimal potential for dust generation and that the hazards are reduced to levels equal to or below those naturally existing within the surrounding environment.

A draft restoration and rehabilitation plan was prepared during construction and is updated and refined annually to account for changes in mine development and operation as well as environmental and social conditions and requirements. The plan will be finalized prior to mine closure and after due consultation with stakeholders. The main activities of the rehabilitation process will range from dismantling infrastructure, winning-hauling-dumping-spreading of waste rock, ripping compacted surfaces and grading the area to topographically blend with the surroundings. Sukari is in an arid desert with no topsoil, as such there will be no attempt to establish vegetation as this would alter the nature of the area.

A provision for restoration and rehabilitation is included in the annual budget. The provision for future restoration costs is the best estimate of the present value of the expenditure required to settle the restoration obligation at the reporting date. Future restoration costs are reviewed annually and any changes in the estimate are reflected in the present value of the restoration provision at each reporting date.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

Centamin has, in 18 May 2015, published a five-year cost forecast from 2015 to 2019. The cost forecast includes sustaining capital of circa US$80 million per annum. No material expansion capital expenditure is planned.

21.2 Operating Costs

The operating cost estimate is in United States dollars with a base date of July 2015. There is no allowance for escalation or contingency in the operating costs.

21.2.1 Exchange Rates

The exchange rates used are as shown in Table 21.2-1.

Table 21.2-1 Currency Exchange Rates

Country	Unit of Currency	Exchange Rate (to US$)
USA	Dollar (US$)	1.00
Australia	Dollar (A$)	0.7319
Europe	Euro (EUR)	1.1046
Egypt	Pound (EGP)	0.1277

21.2.2 Taxes

Valued added tax on imported goods and all government duties and other taxes have also been excluded based on the concession agreement with the Egyptian Government.

21.2.3 Mining Costs

Mining costs were derived from

- Actual production and cost performance data incurred to date.
- Costs detailed in completed and active contracts (e.g. drilling contract, explosive and accessories supply and tyre management contracts)
- Consumption and productivity rates are based upon actual performance to date.
- Detailed labour organizational charts and salaries for current and projected requirements.
- The diesel fuel price used was US$0.70/litre.

A summary of the open-pit mining costs is detailed Table 16.1-1. The underground mining cost assumptions are detailed in Table 16.2-1.

21.2.4 Processing Costs

Processing costs were derived from actual production and cost performance data incurred to date, with the following considerations:

- Reagent consumption rates per tonne processed based on actual performance to date.
- Power generation costs and consumption rates per tonne based on diesel fuel price of US$0.70/litre and actual performance to date.
- Major wear item costs based on the wear rates derived from actual performance to date.

- Maintenance costs estimated based on historical breakdown frequency rates from plant performance to date.
- Detailed labour organizational chart and associated salary data.

For summary costs in tabular form, please refer to Table 16.1-2 and Table 16.2-1.

21.2.5 Labour

SGM currently has approximately 1,300 employees for the operation of the Sukari Gold Mine.

Expatriate levels are expected to reduce as training and familiarization with the operation increase with Egyptian employees.

SGM provided the base monthly remuneration rate in EGP for all local personnel according to skill level and responsibility. SGM also provided the applicable loadings to account for an annual salary increase, food allowance, public holiday allowance, annual leave, special allowances, site allowances and travel allowances. Indirect costs include elements such as social insurance and payroll taxes.

SGM provided expatriate remuneration costs, inclusive of all on costs.

The total unit labour costs included in Table 21.2-2 includes all allowances, including international flights for expatriates.

Table 21.2-2 Total Unit Labour Costs

Title	Grade	Total Annual Cost (US$)
Expatriate		
General Manager	EP1	367,000
Department Manager	EP2	278,000
Supervisor	EP3	209,000
Senior Operator	EP4	186,000
Operator	EP5	162,000
Title	**Grade**	**Total Monthly cost in EGP**
Egyptian Senior Staff		
Manager	M	25,000
Superintendent	S1	16,500
Senior Professional/Supervisor	S1	16,500
Professional	S2	9,700
Technical Officer	S3	7,400
Technician/Administration	S4	6,100
Graduate	S5	5,000
Egyptian Junior Staff		
Leading Hand	J1	4,700
Tradesman –Experienced	J1	4,700
Tradesman	J2	3,800
Operator - Experienced	J3	3,500
Operator	J4	2,700
Trades Assistant	J5	2,200
Trainee	J6	1,900

22 ECONOMIC ANALYSIS

As Centamin is a producing issuer, it has excluded information required by Item 22 of Form 43-101F1 as any forecast increase in plant throughput, in this technical report, is not considered to be a material expansion of current production.

23 ADJACENT PROPERTIES

No significant mining or exploration activity surrounds the Sukari Gold Project.

24 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

Sukari is viewed as a long-term strategic asset for Centamin. For the indicative schedule constructed by AMC using the open-pit and underground Mineral Reserves, the operation has a life of over 19 years. If the underground mine is extended beyond the life of the current underground Mineral Reserve, that contribution will allow the operation to extend beyond 19 years.

The recent Stage 4 expansion of the processing plant has been successfully implemented and commissioned.

The underground operations are progressing and extracting high-grade gold from the structurally-controlled zones of mineralization at depth. Interaction of the open-pit and underground is expected to occur when mining nears the base of Stage 3A of the open-pit, with the upper benches of Stages 3A and 3B currently being mined.

Development of the open-pit continues as planned, with further productivity improvements of the mining equipment expected to achieve total material movement of at least 66 Mtpa from 2015 onwards. Current supply of explosives is sufficient to meet the current production rates, and additional supplies of explosives can be secured to meet increased production rates, as required.

The strip ratio in the first five years of operation (2009 to 2014) was less than the LOM average. Strip ratio is forecast to increase to the LOM average (5.9:1), and in some future periods is likely to peak higher than the LOM average when waste pre-stripping is undertaken for the final three pit stages.

The Mineral Reserve was estimated at a gold price of US$1,300/oz and used a 10% discount rate. The financial analysis demonstrates that the project is economic.

26 RECOMMENDATIONS

26.1 Reconciliation

The QP, Patrick Smith, recommends a site-wide reconciliation process be developed and implemented to analyse all aspects of the production chain. With the major ramp-up in mining and processing rates at Sukari in recent years this process is needed to allow robust analysis of resource, reserve, and grade control models.

The reconciliation study would be undertaken as part of operational activities and budget.

26.2 Underground Mining Recovery

The QP, Declan Franzmann, recommends that SGM investigates stope backfilling opportunities in high-grade mineralization areas. Backfilling reduces the use of pillars for local stope stability, and extraction of the mineralization in these pillars can improve stope recovery.

The backfill study would be undertaken as part of operational activities and budget.

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28 DATE AND SIGNATURES

The effective date of this report is 30 June 2015.

The report is issued on 23 October 2015.

The data on which the contained Mineral Resource and Reserve estimate for the Sukari Gold Project is based were current as of the Effective Date, 30 June 2015.

The undersigned are all qualified persons and were responsible for preparing or supervising the preparation of parts of this Technical Report, as described in Section 2.

(Signed by) Declan Franzmann	23 October 2015
(Signed by) Patrick Smith	23 October 2015
(Signed by) Nicolas Johnson	23 October 2015
(Signed by) Mark Zammit	23 October 2015