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ASANKO GOLD MINE

DEFINITIVE FEASIBILITY STUDY

(AMENDED AND RESTATED)

National Instrument 43-101 Technical Report

Prepared by DRA Projects (Pty) Limited on behalf of

ASANKO GOLD INC.

Effective Date: 5 June 2017

Amended and Restated: 20 December, 2017 (Information as of 5 June 2017)

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This report titled "Asanko Gold Mine Definitive Feasibility Study (Amended and Restated)", is a National Instrument 43-101 Technical Report with an effective date of 5 June 2017 and was amended and restated December 20, 2017 (Information remaining as of 5 June 2017). It was prepared on behalf of Asanko Gold Inc. by Charles Muller, Malcolm Titley, Phil Bentley, Thomas Obiri-Yeboah, Glenn Bezuidenhout, Dave Morgan, Douglas Heher, Godknows Njowa and signed:

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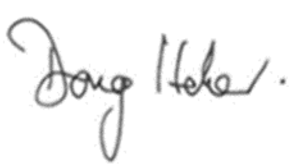

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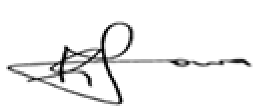

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1 SUMMARY

1.1 Introduction

This DFS report has been prepared on behalf of Asanko Gold Inc. (Asanko Gold"), a gold mining company listed on the TSX and NYSE, with headquarters at 680-1066 West Hastings Street, Vancouver, British Columbia.

In June 2015 Asanko Gold commissioned DRA, and various specialist consultants, to complete a DFS on the expansion of the Asanko Gold Mine ("AGM" or the "Project"), located in Ghana, West Africa, from the current open pit mining and processing operation to include an expanded processing facility and to bring the Esaase deposit into production, with construction expected to start in Q2 2017.

- Current operation (previously referred to as Phase 1) - as commissioned in Q1 2016:
  - CIL processing facility, located at the Obotan project site, operating at 3.6 Mtpa (design was originally 3 Mtpa)
  - Tailings Storage Facility ("TSF")
  - Life of Mine ("LoM") approximately 10 years to 2026
  - Ore sources: Nkran and Satellite pits

Note: Phase 1 was originally intended to process 3 Mtpa, but it was found that the nameplate capacity of the plant could be increased as the milling circuit had excess capacity.

This AGM 2017 DFS follows on from the Phase 2 Pre-feasibility Study ("Phase 2 PFS") published by Asanko Gold in June 2015. Phase 2 has been renamed and now consists of two discreet expansion projects, Project 5 Million ("P5M") and Project 10 Million ("P10M"), together the "AGM expansion projects".

The scope of the AGM 2017 DFS is as follows:

P5M (Q1 2017 to Q3 2018):

- Existing CIL processing facility at Obotan upgraded from the current 3.6 Mtpa to 5 Mtpa (Brownfields expansion)
- Overland conveyor constructed from Esaase to Obotan
- Power line from Obotan to Esaase constructed
- Esaase deposit brought into production at 2 Mtpa ROM
- Ore sources: Nkran, Satellite pits and Esaase, upon commissioning of the conveyor

  - On a standalone basis, LoM approximately 20 years to 2037

- P10M (Q3 2017 to Q3 2019):
  - o Second CIL plant constructed, adjacent to the current plant at Obotan, with a capacity of 5 Mtpa, thereby doubling processing capacity to 10 Mtpa
  - o Production from Esaase pit ramped up to 7 Mtpa ROM
  - o Resettlement of the village of Tetrem, comprising 250 structures
  - o Expansion of the footprint of the existing TSF
  - o Ore sources: Esaase, Nkran, Satellite pits
  - o LoM approximately 10 years to 2027

The purpose of the DFS is to demonstrate the economic and technical viability of the expansion which will allow the Asanko Gold Board to make a decision on the expansion and to bring the new Esaase pit into production. See Figure 1-1 below for a summary schedule which uses specific dates as it assumes current funding availability but must be adjusted to reflect any delays in securing necessary funding.

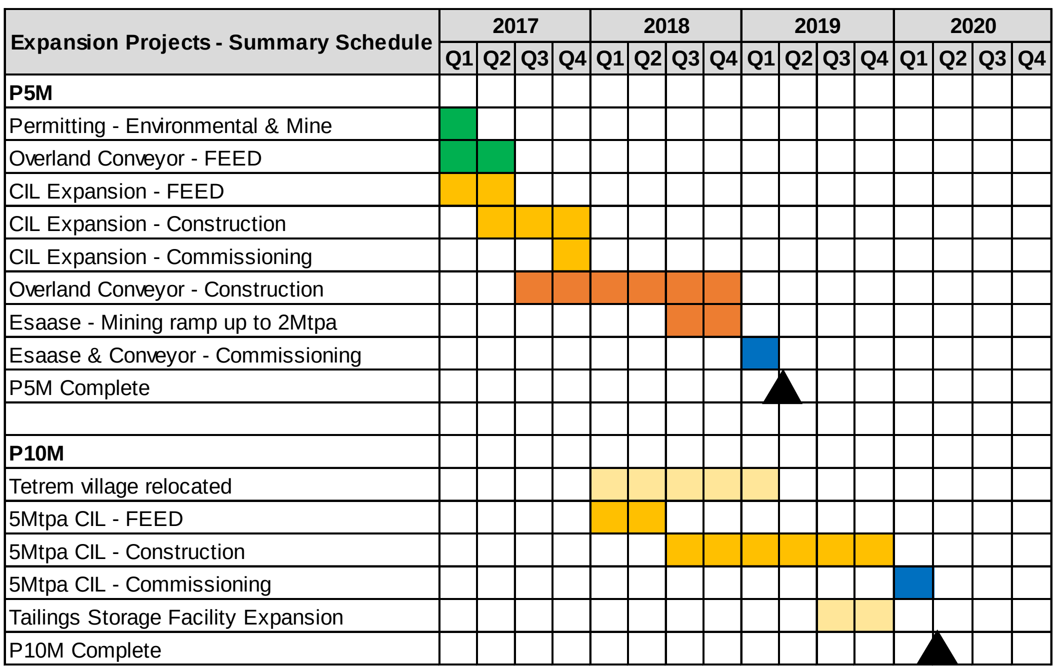


Figure 1-1: Summary DFS schedule (source: Asanko Gold 2017)

1.2 Project Description and Location

The AGM concessions, the Obotan and Esaase project areas, are located in the Amansie West District of the Ashanti Region of Ghana (Figure 1-2 and Figure 1-3). The Project concessions are owned 100% by Asanko Gold Ghana Limited ("Asanko Gold Ghana"), a 100% owned Ghanaian subsidiary of Asanko Gold.

The Government of Ghana retains the right to take a 10% free carried interest in the Project under Section 8 of the Ghanaian Mining Act.

Asanko Gold holds six mining leases (Table 1-1), as well as prospecting and reconnaissance licences, which collectively make up the AGM and span 30 km strike length of the Asankrangwa Gold Belt. The mining lease concessions cover an area of approximately 213.2 km^2, between latitudes 6° 11' 54.985" N and 6° 35' 33.074" N, and longitudes 2° 4' 59.195" W and 1° 51' 25.040" W.

The Esaase, Abore, Abirem, Datano, Jenni River and Adubea Mining Leases [NTD Obotan?]contain all of the mineral resources defined to date. All other concessions [delete references to “concessions”?] held by Asanko Gold in the area contain exploration potential defined to date and in some instances locations for infrastructure. The Ghana Environmental Protection Agency (“EPA”) grants permits on a perennial basis to conduct exploration. On advice from Asanko Gold, with respect to the AGM concession areas, all permitting within the afore-mentioned governmental permitting structure is up to date and accounted for.

Figure 1-2: The Asanko Gold Mine Location (source: Asanko Gold 2017)

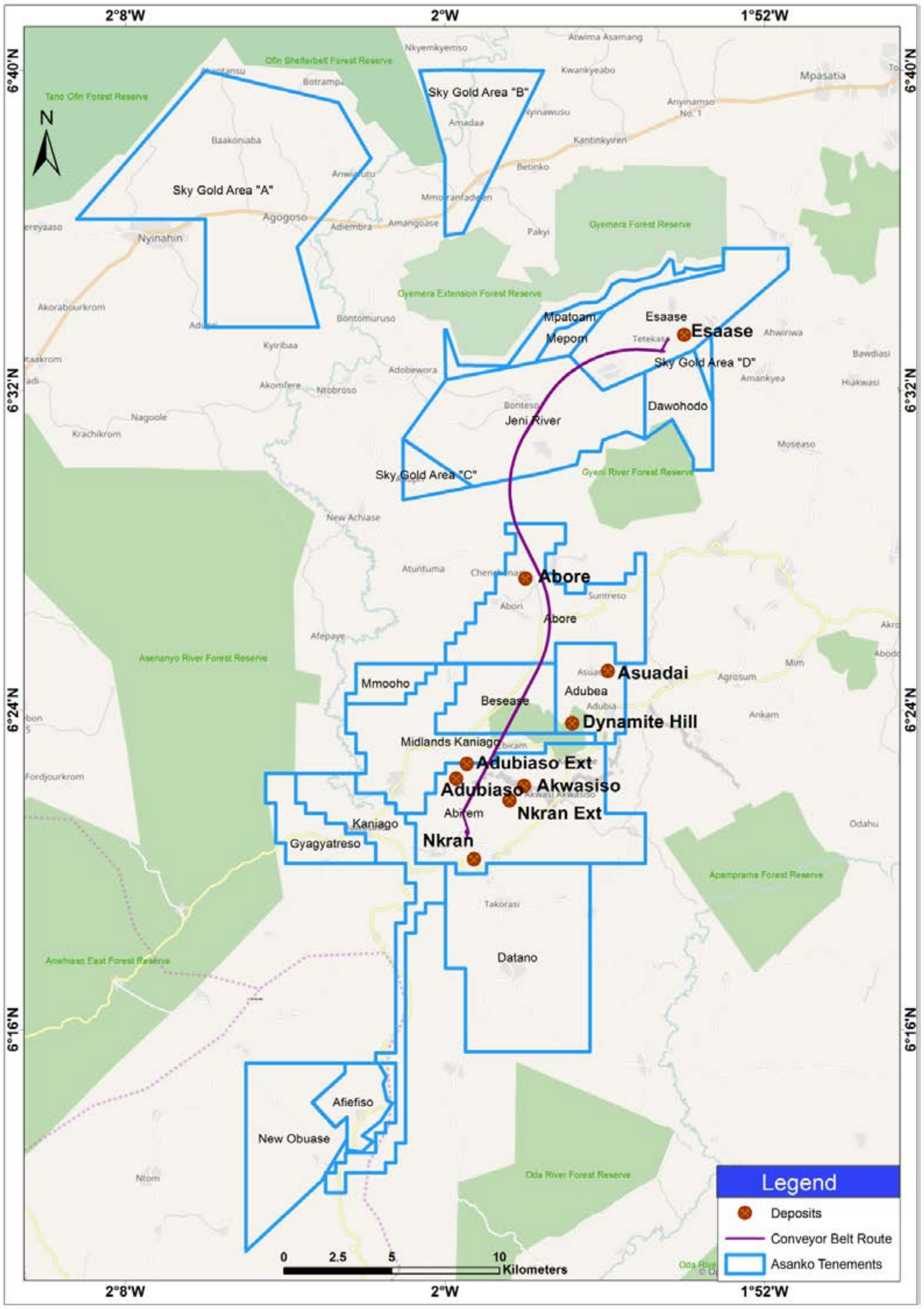


Figure 1-3: The Location of the Various Tenements making up the Asanko Gold Mine (source: Asanko Gold 2017)

Table 1-1: The Asanko Gold Mine Mining Licences (source: Asanko Gold 2017)

Tenement name	Licence Category	100% owned title holder	Minerals Commission file	Status of licence	Area Km²
Datano	Mining Lease	Asanko Gold Ghana – 100%	PL 6/32/Vol 3	Valid-ML renewal	53.78
Abore	Mining Lease	Asanko Gold Ghana – 100%	PL 6/303	Valid-ML received	28.47
Abirem	Mining Lease	Asanko Gold Ghana – 100%	PL 6/303	Valid-ML received	47.13
Adubea	Mining Lease	Asanko Gold Ghana – 100%	PL 6/310	Valid-ML received	13.38
Esaase	Mining Lease	Asanko Gold Ghana – 100%	PL 6/8/Vol.8	Valid-ML received	27.03
Jeni River	Mining Lease	Asanko Gold Ghana – 100%	RL 6/21	Valid	43.41

All concessions carry a 10% free carried interest in favour of the Ghanaian government. The government interest is reflected in a 10% ownership of the operating company, and the government has a right to 10% of any dividends paid by the subsidiary, Asanko Gold Ghana. The leases are also subject to a 5% royalty payable to the Government of Ghana. In addition, the Adubea concession is also subject to an additional 0.5% royalty to the original concession owner. The Esaase mining lease is also subject to an additional 0.5% royalty to the Bonte Liquidation Committee ("BLC").

On advice from Asanko Gold, under the current ownership arrangement and status of holdings, there is no environmental liability held over Asanko Gold for any of the AGM concessions relating to the Obotan project area, with the exception of project works to date.

There is a potential environmental liability on the Company's Jeni River concession which was inherited with the acquisition of the concessions and is not material to the Company, but is reported in its recent financial statements as an Asset Retirement Obligation.

1.3 Accessibility, Climate, Infrastructure and Physiography

The AGM concessions are located in the Amansie West District of the Ashanti Region of Ghana, approximately 250 km northwest of the capital Accra, and about 50 km to 80 km southwest of the regional capital of Kumasi. There are several local villages near the AGM project areas, with the villages of Tetrem and Esaase being in close proximity to the Esaase deposit.

Mining personnel are readily available in Ghana, due to its long tradition of gold mining, with a highly skilled workforce and numerous mining operations in the country.

There are daily flights from Accra to Kumasi flown by several different airlines. In addition, there is an airstrip located adjacent to the Obotan project site infrastructure west of the Nkran village, which is used by the AGM to transport staff and service providers to and from Accra. Existing road access to the site is available from the west, south and east, but the main access used will be from the ports of Tema and Takoradi to the south via Kumasi, or Obuasi. Total distance from Tema to the project site, via Kumasi is approximately 400 km.

The AGM is located in hilly terrain dissected by broad, flat drainages that typically form swamps in the wet season between May and late October. Hill tops are generally at very similar elevations, reflecting the elevation of a previous erosional peneplane that is now extensively eroded. Maximum elevations are around 80m above sea level, but the areas impacted by the AGM deposits generally lie at less than 50m elevation. Despite the subdued topography, hill slopes are typically steep. Ecologically the AGM area is situated in the wet evergreen forest zone.

1.4 History

1.4.1 Nkran Area

Nkran is important from the view point of historical artisanal gold mining that dates back many generations and remains quite extensive to the present day.

In the late 1980’s, this prospect attracted the attention of consultant Dr Alex Barko who recommended the area to one of his local Client groups and Obotan Minerals subsequently applied and received a prospecting concession covering about 106 km^2 over the general area. A minor amount of prospecting was carried out in the early stages. Some attention was paid to the alluvial gold potential because of the extensive gold in the nearby Offin River, (held then by Dunkwa Goldfields), as well as the alluvial gold project being developed at the time, a little further north in the Bonte area. In the early 1990’s, the Obotan concession was examined by American consultant Al Perry who was working on behalf of two related Australian juniors, Associated Gold Fields NL (“AGF”) and Kiwi International Resources Limited (“KIR”).

By early 1995, resource estimates (Measured, Indicated and Inferred) were reported as 4.8 Mt grading 3.7 g/t Au for an in-situ content of approximately 600,000 oz Au. A feasibility study was completed and a mining lease was granted in late 1995. In May 1996 the combined interests of KIR and AGF were bought out by Resolute Limited (“Resolute”) who immediately reviewed and expanded the scope of the project. This was achieved mainly by conducting further RC diamond drilling to increase resources to a depth of 150m at Nkran and to further assess the known mineralisation at nearby Adubiaso.

A revised mine development plan was completed by the end of July 1996 and a decision was made to proceed into production at a rate of 1.4 Mtpa. Initial mining was started early in 1997 and by May 1997, the first gold was poured. Mining operations ceased in 2002 due to low gold prices and the concessions were reclaimed and returned to the Government of Ghana.

1.4.2 Abore Area

The Abore area was covered in a prospecting concession granted to the Oda River Gold small scale mining licence (Asuadai prospect) at Adubea in 1991.

In the mid-1990's, Mutual Resources of Vancouver, Canada, in partnership with Leo Shield Exploration of Perth, Australia, completed a joint venture with the Oda River Group and commenced a regional exploration programme on the concession (covering approximately 73 km^2). Prospecting in the area north of Abore revealed extensive old and very recent artisanal mining in alluvial areas, as well as many old pits in the saprolite along a low hill immediately adjacent to the alluvial workings.

Soil geochemistry revealed a strong north-north-east trending gold anomaly over the area of artisanal mining (bedrock areas); the anomaly is several hundred metres wide and traceable along strike for about 3 km, well beyond the area of old workings. Extensive trenching in the area confirmed continuous bedrock mineralisation over a distance of at least 1,000m with widths in the range 50m to 100m. The mineralisation consists of a broad quartz stock work system hosted mainly by a north-north-east trending intermediate granitoid intrusion. The early artisanal pitting was focused mainly on narrow quartz veins associated with the stock work system. Extensive drilling in the area (mainly RC, but considerable diamond drilling as well) has outlined a sizeable resource (now known as the Abore north prospect). In the late 1990's, Mutual's interest in the project was bought out by Leo Shield, (now Shield Resources). In early 2001, an agreement was reached with Resolute whereby ore was trucked from Abore north to the Nkran plant for treatment.

1.4.3 Adubiaso Area

During the late 1990's, the Nkran plant started to process oxide ores from the Adubiaso gold deposit, located about 7.5 km north-north-west of Nkran. There were no known historical workings on this area.

1.4.4 Asuadai Area

The Asuadai prospect has predominantly been worked by local artisanal miners who have undertaken minor pitting in the region down to 5m to 10m through the oxide material to expose these stock work vein sets. There were no known formal historical mining workings on this area.

1.4.5 Dynamite Hill Area

There was no historical exploration or mining activity known at Dynamite Hill.

1.4.6 Esaase Area

Artisanal mining has a long history in the Bonte Area, associated with the Ashanti Kingdom. Evidence exists of adits driven by European settlers, between the periods 1900 to 1939. However, no documented records remain of their activity. Drilling was conducted on the Bonte River valley alluvial sediments during 1966 and 1967 to determine alluvial gold potential.

In 1990, the Bonte mining lease was granted to Akrokerri-Ashanti Gold Mines (“AAGM”) and was later transferred to Bonte Gold Mining (“BGM”), a local subsidiary of AAGM. BGM had reportedly recovered an estimated 200,000 oz of alluvial gold on the Esaase concession and another 300,000 oz downstream on the Jeni River concession, prior to entering into receivership in 2002. It should be noted that previous placer gold production is of no relevance to Asanko Gold’s development programme, which is entirely focused on the development of hard rock resources.

The Esaase mining concession, including the camp facilities at Tetrem, was bought from the BLC by Sametro Company Limited, a private Ghanaian company. In May 2006, Asanko Gold, then called Keegan Resources (“Keegan”), signed a letter of agreement with Sametro to earn 100% of the Esaase mining concession over a three year period of work commitments and option payments.

1.5 Geology & Resource Estimate

1.5.1 Regional Geological Setting

West Africa is underlain by the West African craton, an Archaean aged (>2,5Ga) stable crustal block which forms the geological basement in Sierra Leone, Liberia and parts of Guinea, Cote d'Ivoire and Ghana (Figure 1-2). The core of Archaean sequences is surrounded by younger Palaeo-Proterozoic (2,2Ga-2,0Ga) sedimentary and volcanic units, collectively known as the ”Birimian”, that form narrow (20 km to 60 km wide) alternating belts of mafic volcanic greenstone units, separated by wider basins of mainly marine clastic sediments. The greenstone belt-sedimentary basin trend is northeast-southwest across central and southern Ghana and can may be traced for hundreds of kilometres along strike. At least five greenstone belts are identified, namely the Ashanti, Asankrangwa, Sefwi, Kibi and Bui belts (Figure 1-3) The intervening sedimentary basins include from east to west; the Cape Coast Basin, Kumasi Basin and the Sunyani Basin.

The geology of Ghana is dominated by predominantly metavolcanic paleoproterozoic Birimian Supergroup (2.25 – 2.06 billion years ago) sequences inclusive of the clastic Tarkwaian Group sediments (2.12-2.14 billion years), after WAXI 2015) in the central-west and northern parts of the country. Clastic shallow water sediments of the Neoproterozoic Volta Basin cover the northeast of the country (Figure 1-3). A small strip of Paleozoic and Cretaceous to Tertiary sediments occur along the coast and in the extreme southeast of the country.

The Birimian rocks formed during two major orogenic phases, the Eoeburnian from ca. 2250-2150 Ma, and the Eburnian between ca. 2116-2060 Ma. These two orogenic stages were separated by a major extensional event during which flysch type basins developed throughout the southern and western parts of the Birimian. A marked break in the timing of events is apparent between eastern (Cote d’Ivoire/Ghana) and western parts of the Birimian, near the centre of the West African Craton and the western margin of the Comoé Basin in Cote d’Ivoire.

Key geotectonic events are:

- In Ghana Eoeburnian plutonism and contractional deformation occurred between ca. 2190-2140 Ma
- Basin formation between the Eoeburnian and Eburnian cycles occurred between ca. 2130-2116 Ma in Ghana
- Eburnian plutonism had essentially ceased by ca. 2095 Ma in Ghana
- Gold mineralisation in Ghana occurred during wrench deformation at ca. 2100-2090 Ma, after severe compression and probable basin inversion. At Nkran two distinct gold deposition periods are associated with early ductile-brittle deformation and a later cross cutting brittle quartz veining event

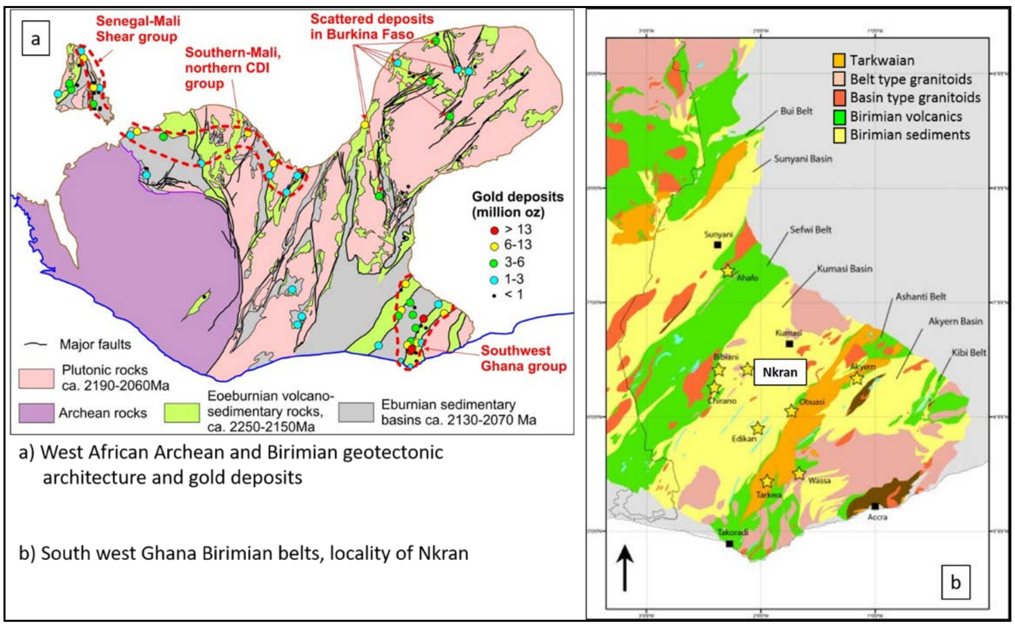


Figure 1-4: West Africa and southwest Ghana geological framework and location of the Nkran deposit (Asanko Gold Mine) (source: Asanko Gold)

The Birimian rocks consist of narrow greenstone (volcanic) belts, which may be traced for hundreds of kilometres along strike, but are usually only 20 km to 60 km wide. These belts are separated by wider basins (such as the Kumasi Basin) of mainly marine clastic sediments. Along the margins of the basins and belts, there appears to be considerable inter-bedding of basin sediments and volcanoclastic and pyroclastic units derived from the volcanic belts. Thin, but laterally extensive chemical sediments (exhalites), consisting of chert and fine-grained manganese-rich and graphitic sediments often mark the transitional zones. The margins of the belts commonly exhibit faulting on local and regional scales. These structures are fundamentally important in the development of gold deposits for which the region is well known.

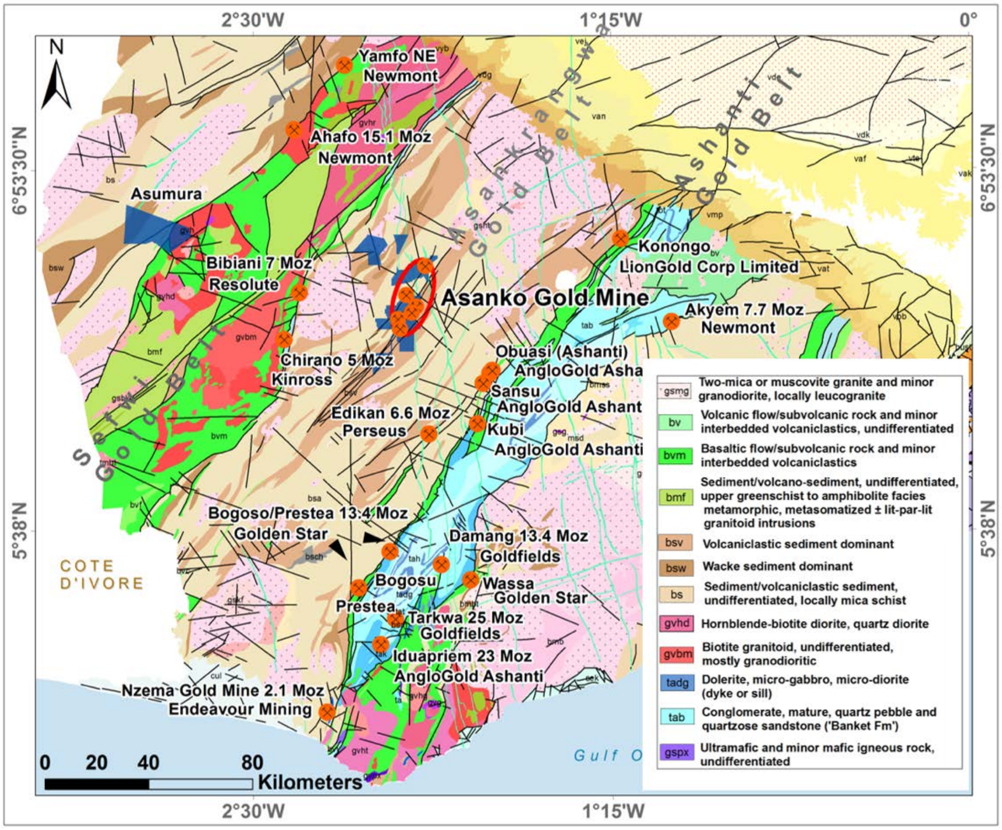


Figure 1-5:Geology of Southwest Ghana highlighting the Regional Geology around the AGM (source: Asanko Gold 2017)

The Tarkwaian rocks consist of a distinct sequence of metasediments (quartzite, conglomerate and phyllite) occurring within a broad band along the interior of the Ashanti Belt. Conglomerates host important palaeoplacer gold deposits in the Tarkwa district. Equivalent rock types occur in other belts of the region, but in relatively restricted areas. In the type locality at Tarkwa, the sequence is in the order of 2.5 km thick, whereas in the Bui belt, comparable units are approximately 9 km thick. These sediments mark a rapid period of erosion and proximal deposition during the late-stage of the orogenic cycle. They unconformably overlie the Birimian metavolcanics at the Damang mine near Tarkwa. (Figure 1-7) shows the generalised stratigraphy of southwest Ghana.

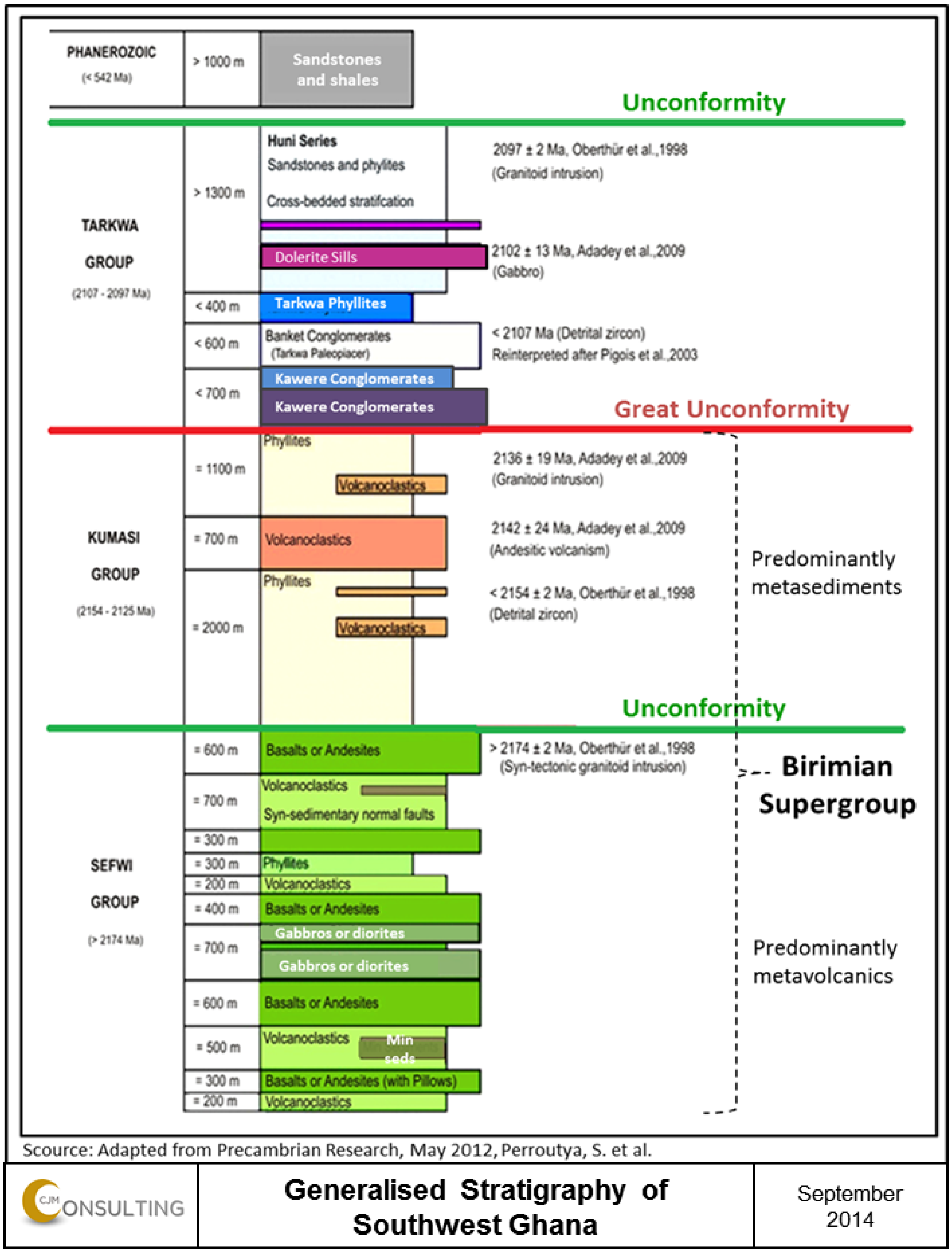


Figure 1-6: Generalised Stratigraphy of Southwest Ghana (source: CJM 2014)

The Birimian sediments and volcanics have been extensively metamorphosed to greenschist facies, although in many areas, higher temperatures and pressures are indicated by amphibolite facies. Multiple tectonic events have affected virtually all Birimian rocks with the most substantive being a fold thrust compressional event (Eburnean Orogeny) that affected both volcanic and sedimentary belts throughout the region and to a lesser extent, Tarkwaian rocks. For this reason, relative age relations suggest that final deposition of Tarkwaian rocks took place as the underlying and adjacent volcanic and sedimentary rocks were undergoing the initial stages of compressional deformation. Studies in the western part of the region (Milesi et al., 1992) have proposed several separate phases of folding and faulting suggesting a change in stress direction from northeast to southwest to north to south. However, a regional synthesis by Eisenlohr (1989) has concluded that although there is considerable heterogeneity in the extent and styles of deformation in many areas, most of the structural elements have common features, which (in his opinion) are compatible with a single, extended and progressive phase of regional deformation involving substantial northwest-southeast compression.

Ongoing studies of the geotectonic evolution of the Birimian have involved more extensive precision U/Pb age dating (eg WAXI 2016), the results of which have significantly improved the constraints on timing of deformation as well as the timing of gold mineralisation events. This work is supporting at least two periods of basin sedimentation (2135-2116 Ma and 2105-2070 Ma) and accretionary (oblique compression) tectonics (2116 – 2105 Ma and 2070-1980 Ma), as well as two ages of gold deposition in the Asankrangwa Belt.

1.5.2 AGM Gold Deposit and Geological Overview

The AGM is a collective term for the significant Nkran and Esaase gold deposits plus nine other satellite deposits. These are located on the Asankrangwa Belt within the Kumasi Basin sediments. Nkran was previously exploited by Resolute (1997-2001) and produced approximately 730,000oz Au. The Nkran pit was dewatered and reopened by Asanko Gold in 2015-2016.

Although each gold occurrence within the AGM has its own idiosyncrasies, geological and geophysical studies have advanced a similar mine scale setting for all the deposits discovered to date (Table 1-2).

There is an underlying structural relationship between reactivated WNW basement structures and the dominant NE-SW shears that have juxtaposed the sandstone, siltstone and lesser shale metasedimentary packages, coupled by N-S structures that may control flexures in the steeply dipping sediments.

All deposits have intrusive tonalitic – porphyritic granite dykes and small apophyses that have been emplaced after the main shortening deformation and syn-late with the transpressive shearing of the metasedimentary rock package.

Gold mineralisation has occurred at least twice at 2, or more distinct deformational events.

Gold occurs largely as free particles. It is deposited in economic concentrations predominantly around zones of rheological contrast between sandstone (porous) and siltstone facies (non-porous) that are commonly subvertical shear zones, as well as in late, shallow dipping conjugate quartz vein arrays that transgress rheologically contrasting metasedimentary units and the later granite intrusives.

Table 1-2: Summary of AGM Mineral Deposits (source: Asanko Gold 2017)

Deposit	Mineral Style	Main host rock	Measured & Indicated Mineral Resources	Proven & Probable Mineral Reserves
Nkran	D2 shear + granite + Late conjugate QV's	Qtz sandstone + granite + QV's	30.07Mt @ 1.78 g/t Au for 1.72 Moz	22.77Mt @ 1.91 g/t Au for 1.40 Moz
Esaase	D2 shear + tensional QV's	Highly deformed sandstone-siltstone-shale + QV's	84.02Mt @ 1.38 g/t Au for 3.72 Moz	62.56Mt @ 1.39 g/t Au for 2.94 Moz
Akwasiso	D2 shear + granite + Late conjugate QV's	Qtz sandstone + granite + QV's	6.33Mt @ 1.50 g/t Au for 0.31 Moz	4.95Mt @ 1.51 g/t Au for 0.24 Moz
Dynamite Hill	Granite + Late conjugate QV's	Qtz sandstone + granite	3.41Mt @ 1.48 g/t Au for 0.16 Moz	2.84Mt @1.49 g/t Au for 0.14 Moz
Adubiaso	D2 shear + granite dyke + Late conjugate QV's	Qtz sandstone + granite	2.73Mt @ 1.80 g/t Au for 0.16 Moz	2.28Mt @ 1.90 g/t Au for 0.14 Moz
Abore	D2 shear + granite dyke + Late conjugate QV's	Granite + QV's	5.33Mt @ 1.45 g/t Au for 0.25 Moz	3.18Mt @ 1.48 g/t Au for 0.15 Moz
Nkran Extension	D2 shear + Late conjugate QV's	Qtz sandstone	0.19Mt @ 2.70 g/t Au for 0.02 Moz	0.19Mt @ 2.24 g/t Au for 0.01 Moz
Adubiaso Ext	D2 shear + late conjugate QV's	Qtz sandstone	0.42Mt @ 1.61 g/t Au for 0.02 Moz	0.21Mt @ 1.53 g/t Au for 0.01 Moz
Asuadai	D2 + Granite + late	Granite + QV's	1.88Mt @ 1.22 g/t Au for 0.07 Moz	1.30Mt @ 1.09 g/t Au for 0.05 Moz

	conjugate QV's			

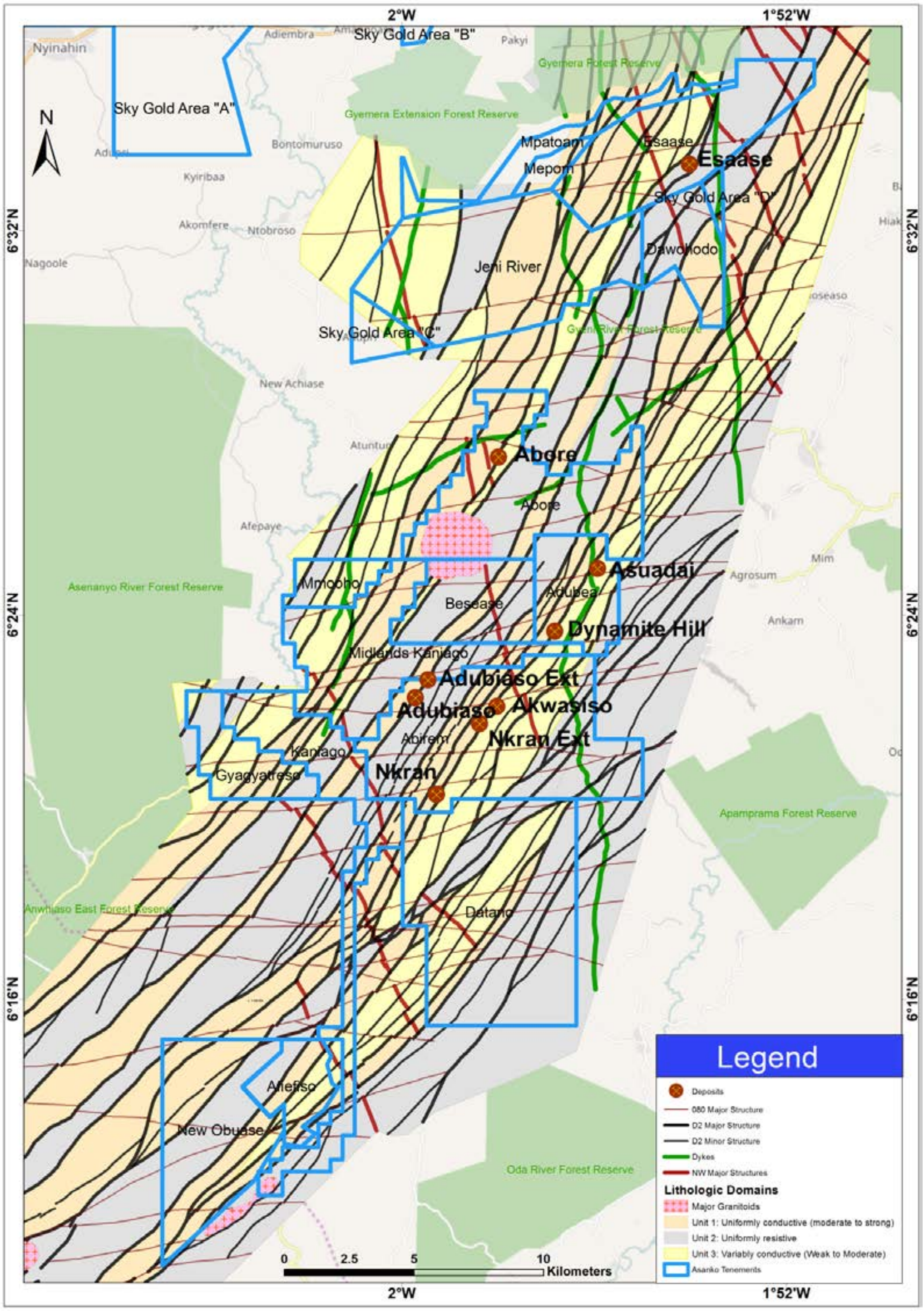


Figure 1-7: AGM leases and regional geological interpretation with gold deposits (source: Asanko Gold 2017)

In 2013 an exercise was initiated to generate 3D litho-structural models for all of the AGM deposits. This was intended to increase the geological and structural understanding of the AGM deposits, as well as introduce proper geological and structural controls into the updated Mineral Resource Estimate ("MRE"). The process involved geological and structural re-logging of drill core, interpretation of historic flitch diagrams, and use of Leapfrog® software to produce the 3D litho-structural models.

These models are currently being updated with recent drilling data and integrated into Micromine and Datamine 3D modelling software.

In addition to the creation of the 3D litho-structural models, Asanko Gold initiated a prospectivity mapping analysis of the Asankrangwa Belt. This exercise provided a basis to collate available regional geophysical and geological data, as well as drilling and geochemical survey information. An important outcome of the regional and local structural interpretation was the completion of a revised property geological map (Figure 1-8).

During 2016 Asanko Gold completed a full (72 borehole) relog of the Nkran deposit, and subsequently updated the 3D litho-domaining used for the Nkran MRE.

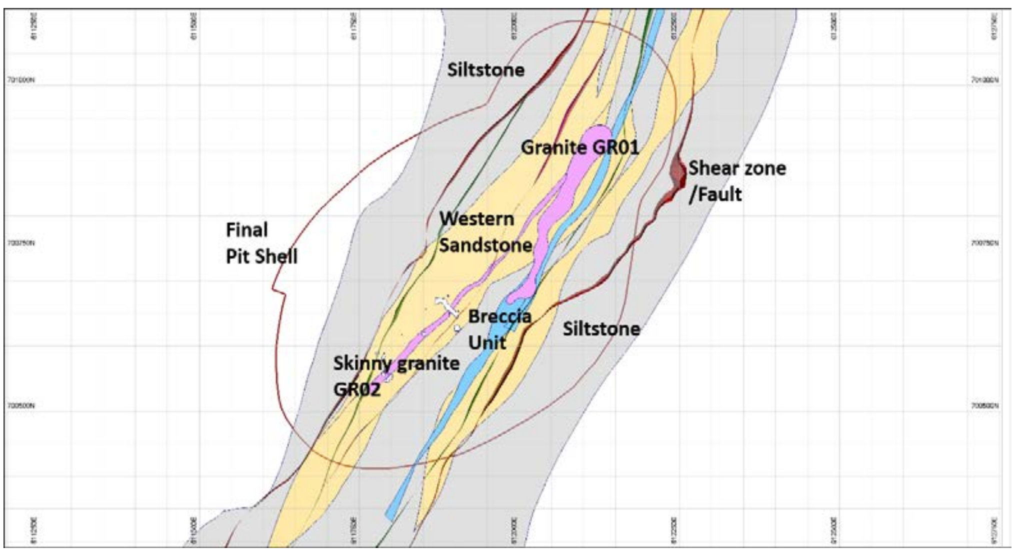


Figure 1-8: The Nkran Pit Geology (source: Asanko Gold 2016)

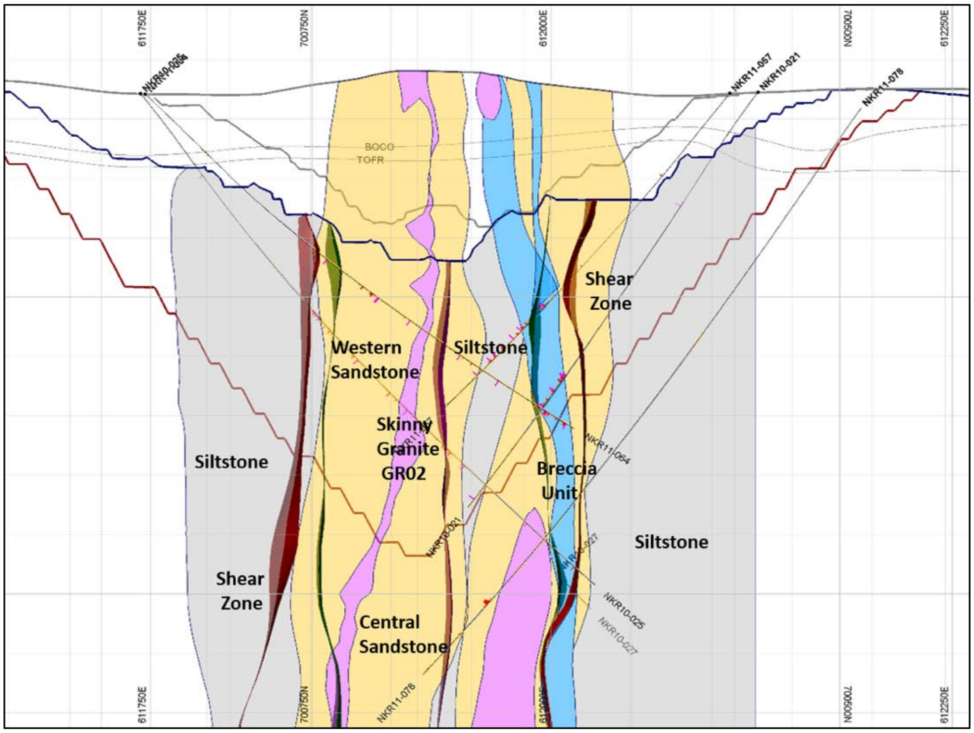


Figure 1-9: Nkran cross section at 10400N, looking north (source: Asanko Gold 2017)

1.5.3 Mineral Resources

Qualified Persons from CJM Consulting ("CJM") and CSA Global ("CSA") compiled the components of the MREs, in compliance with the definitions and guidelines for the reporting of Exploration Information, Mineral Resources and Mineral Reserves in Canada, "the CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines" (2014). These MRE also adhere to the Rules and Policies of the National Instrument 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1 and Companion Policy 43-101CP.

Furthermore, the Mineral Resource classifications are consistent with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of 2004 as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and the Mineral Council of Australia ("JORC").

On a global basis, both CJM and CSA are satisfied that the MRE globally reflects the ore bodies based on the available data.

Suitably experienced and qualified geologists, surveyors and other Mineral Resource practitioners employed by Asanko Gold were responsible for the capture of the drill hole information and geological information.

For the purposes of this disclosure CJM has estimated the MRE for the AGM projects Esaase, Abore, Asuadai, Adubiaso, Adubiaso Extension and Nkran Extension, and CSA estimated the MRE for Nkran and Dynamite Hill, and has overseen the Akwasiso MRE (April 2017). All other estimates are as at 31 December 2016. All gold grade estimation was completed using Ordinary Kriging ("OK"). This estimation approach was considered appropriate based on a review of various factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style and geometry of mineralisation.

1.6 Assumptions, Parameters and Methods used for MRE

The Asanko Gold projects are all located along major shear zones with cross-cutting faults. The mineralisation is directly related to the structural setting and specific lithological units. Delineated geological models based on lithological-structural interpretations form the basis for the MRE. Lower grade background lithologies host higher grade veins, and the individual mineralisation boundaries of these high-grade veins can be difficult to define.

The main lithological units, within specific fault blocks, form the basis for delineating geological domains. Within each of the domains, the continuous mineralised and waste proportions were delineated using an Indicator Kriging ("IK") method. In all CJM estimations a 0.3 g/t cut-off was used to flag data as mineralised, or not. Above 0.3 g/t is assigned a value of 1 and below a value of zero. Internal waste (values below 0.3 g/t) of 2m and less were also considered a part of the mineralised zone and flagged 1. The 1 and 0 values are then estimated into a block model using mineralised orientations and relationships observed and modelled from data. The Indicator Estimates produce a value between 0 and 1 as a probability for establishing if a cell is mineralised, or not. Each domain has specific probabilities that are related to expected mineralisation relationships. The unconstrained areas outside main lithological domains (where lithological boundaries do not constrain the mineralisation) were estimated using the same probabilistic approach. In most cases, these were part of the secondary mineralisation which was distinctly different from the main mineralisation.

All geological models and structural constraints for were provided by Asanko Gold after re-interpretations, as well as an internal re-logging programme of Nkran during 2016.

A considerable amount of exploration and evaluation drilling (in excess of 1,200 km of drilling) has been effected on the various deposits on the AGM property. Table 1-3 summarizes this work.

Table 1-3: Asanko Gold Summary Drilling to 31 December 2016 (source: Asanko Gold 2017)

Deposit	DDH		RCD		RC		GC		RAB	
	No	Meters	No	Meters	No	Meters	No	Meters	No	Meters
Nkran	279	83,567	29	8,532	556	26,810	97,939	620,028	-	-
Esaase	111	24,341	289	92,093	716	111,584	-	-	-	-
Abore	54	9,437	-	-	-	-	3,006	42,003	31	716
Abore N	15	1,985	-	-	409	31,594	-	-	-	-
Adubiaso	53	10,327	4	590	289	26,495	-	-	-	-
Adubiaso Ext	-	-	-	-	35	3,460	4,986	65,258	-	-
Asuadai	68	8,820	-	-	84	5,551	-	-	209	5,931
Dynamite Hill	12	2,502	1	249	158	18,303	-	-	-	-
Akwasiso	59	10,888	8	2,168	231	16,991	-	-	2	87
Nkran Ext	-	-	3	698	89	7,781	-	-	-	-
Total	651	151,868	334	104,330	2,567	248,570	105,931	727,289	242	6,734

A summary of key criteria for drilling, sampling and geology are tabulated below in Table 1-4.

Table 1-4: Confidence Levels of Key Criteria for Drilling, Sampling and Geology (source: Asanko Gold 2017)

Items	Discussion	Confidence
Drilling Techniques	RC/Diamond - Industry standard approach.	High
Logging	Standard nomenclature and apparent high quality.	High
Drill Sample Recovery	Diamond core and RC recovery adequate.	High
Sub-sampling Techniques and Sample Preparation	Industry standard for both RC and Diamond core.	High
Quality of Assay Data	Quality control conclusions outlined in Section 14. Some issues were identified. Recent improvements were noted.	Moderate
Verification of Sampling and Assaying	Dedicated drill hole twinning to reproduce original drill intercepts.	High
Location of Sampling Points	Survey of all collars with adequate down hole survey. Investigation of available down hole survey indicates expected deviation.	High
Data Density and Distribution	Core mineralisation defined on a notional 40 mE by 30 mN drill spacing with a small area drilled at 20 mE by 20 mN. Other areas more broadly spaced to approximately 80 mN.	Moderate to High
Database Integrity	Minor errors identified and rectified	High

Items	Discussion	Confidence
Geological Interpretation	The broad mineralisation constraints are subject to a large amount of uncertainty concerning localised mineralisation trends as a reflection of geological complexity. Closer spaced drilling is required to resolve this issue.	Moderate
Rock Dry Bulk Density	DBD measurements taken from drill core, DBD applied is considered robust when compared with 3D data.	High below top of transition, moderate in oxide material

The following aspects, or parameters are considered from a geostatistical aspect:

- Number of samples used to estimate a specific block:
  - Measured: at least 9 drill holes within variogram range and a minimum of 27 one meter composited samples
  - Indicated: at least 4 drill holes within variogram range and a minimum of 12 one meter composite samples
  - Inferred: 1 drill hole within search range
- Distance to sample (variogram range):
  - Measured: at least within 60% of variogram range
  - Indicated: within variogram range
  - Inferred: further than variogram range and within geological expected limits
- Lower confidence limit (blocks):
  - Measured: less than 20% from mean (80% confidence)
  - Indicated: 20 to 40% from mean (80 to 60% confidence)
  - Inferred: more than 40% (less than 60% confidence)
- Kriging efficiency:
  - Measured: more than 40%
  - Indicated: 10 to 40%
  - Inferred: less than 10%
- Deviation from lower 90% confidence limit, (data distribution within resource area considered for classification):
  - Measured: less than 10% deviation from mean
  - Indicated: 10 to 20%
  - Inferred: more than 20%

- Regression Slope:
    - Measured – 90%
    - Indicated - 60% to 90%
    - Inferred – less than 60%

The definition of a Measured Mineral Resource requires that the nature, quality, amount and distribution of data are such as to leave the CP with no reasonable doubt that the tonnages and grade of the mineralisation can be estimated to within close limits and any variation within these limits would not materially affect the economics of extraction.

CJM and CSA reviewed the spatial distribution of the relevant drill hole data, the robustness of the kriged model and the parameters for the area classified into Measured Resource. Both CJM and CSA concur that the geological models are well understood and defined in the Measured areas, and further drilling would not affect any material changes to the tonnages and grades.

The definition of Indicated Mineral Resource states that there should be sufficient confidence for mine design, mine planning or economic studies.

CJM and CSA are of the opinion that there is sufficient confidence in the estimate of the Indicated Resource areas to allow the appropriate application of technical and economic parameters and enable an evaluation of economic viability.

The following numeric resource category codes were assigned into the block models, based on the categorisation criteria listed above:

Measured Resource: RESCAT = 1

Indicated Resource: RESCAT = 2

Inferred Resource: RESCAT = 3, and

Unclassified Resource: RESCAT = 0

Mineral Resource classification areas, as well as the grades associated with those classifications, plus grade versus tonnage curves are presented in each deposit MRE.

1.7 Density

All density measurements for the AGM Obotan deposits were taken by means of the water immersion method. A large volume of data was collected by both Resolute and PMI Gold Corporation (“PMI”) over a range of rock types from half-core samples (667 total - Nkran 269, Adubiaso 129, Abore 184 and Asuadai 85; over a period of about 15 years). Subsequently Asanko Gold, with the commissioning of the Nkran pit in Jan 2016, has increased the density database.

The earlier historic data was viewed as being of sufficient quality and reliability for use in the resource estimation conducted by SRK in May of 2012 and is currently viewed by CJM as being of industry standard practice. CSA are of the opinion that Asanko Gold continues to build the database, especially with respect to Oxide densities (saprolite vs saprock), but that current data is adequate for the MRE.

PMI sampling data included 664 density measurements performed by SGS Bibiani and SGS Tarkwa on samples of DC. For solid core, densities were determined by water immersion with samples wrapped in cling film and coated in hairspray, or beeswax to prevent water absorption. For broken, crumbly, or clay core intervals, densities were determined by a volumetric method. The samples are weighed in air and weighed in water, while suspended beneath a balance. The density was then calculated from these measurements.

The density results used in the SRK May 2012 and the relevant weathered oxidation state, (as used in that resource estimate) are summarised in Table 1-5 below.

These density results were also used by CJM in the October 2014 and December 2016 MRE. The Fresh rock densities used for the Nkran pit have been adjusted to reflect ongoing monthly tests conducted during current mining operations.

The density measurements include Oxide / Saprolite, Transitional and Fresh (un-weathered) samples. On average across all the AGM project areas, measurements tend to show approximately 20% lower densities in the saprolite and saprock when compared to transitional samples, and 30% lower densities when compared to Fresh measurements.

It is the view of CJM and CSA that additional work should continue to be conducted to validate the historical data and to assist in increasing the tonnage confidence in ongoing MRE exercises.

Table 1-5: Bulk Densities Summarised by Deposit and Oxidation State (source: Asanko Gold 2017)

Project Area	Oxidation State	No Samples	Density (t/m³)		
			Minimum	Average	Maximum
Esaase	Saprolite - Oxide	579	1.66	2.29	3.01
	Transition	2,394	1.56	2.42	3.81
	Fresh	9,765	1.12	2.78	4.34
Nkran DD Core	Saprolite - Oxide	5	1.47	1.68	1.82
	Transition	10	1.94	2.10	2.29
	Fresh	253	2.17	2.66	2.79
Nkran In Pit	Saprolite - Oxide	0	0	0	0
	Transition	0	0	0	0
	Fresh	1,289	2.11	2.65	2.96
Abore	Saprock - Oxide	6	1.72	1.85	1.98
	Transition	6	2.14	2.42	2.57
	Fresh	21	2.08	2.67	2.87
Adubiaso & Adubiaso Extension	Saprolite - Oxide	16	1.52	1.97	2.20
	Transition	17	2.19	2.40	2.57
	Fresh	19	2.47	2.68	3.49
Asuadai	Saprolite - Oxide	0	0	0	0
	Transition	0	0	0	0
	Fresh	4	2.71	2.75	2.83
Akwasiso	Saprolite - Oxide	8	1.13	1.70	2.55
	Transition	3	1.48	2.18	2.76
	Fresh	33	2.50	2.73	2.87
Dynamite Hill	Saprolite - Oxide	26	1.48	1.76	2.21
	Transition	7	2.01	2.33	2.61
	Fresh	54	2.44	2.73	2.82
Nkran Extension	Saprolite - Oxide	0	0	0	0
	Transition	0	0	0	0
	Fresh	3	2.70	2.75	2.78

1.8 Mineral Resource Estimates

The global AGM MRE include the Nkran, Adubiaso, Adubiaso Extension, Nkran Extension, Abore, Dynamite Hill, Asuadai, Akwasiso, Esaase Main, Esaase B and Esaase D deposits.

Asanko Gold's Mineral Resources disclosures use a >0.5 g/t Au cut-off and a constraining pit shell equivalent to a US$1,500/oz Au price (May 2017). The previous Asanko Gold MRE (2014), undertaken by CJM, were stated at a 0.8 g/t Au cut-off, with the exception of the Esaase deposit, which was stated at 0.6 g/t Au cut-off. No other constraints were applied.

The updated AGM global MRE, as at December 31, 2016, has been adjusted to be in line with accepted resource disclosure practice and applied a 0.5 g/t Au cut-off and a constraining US$1,500/oz Au pit shell. The AGM mineral reserves, which are the economically viable portion within the US$1,500/oz pit shell resources, are estimated at a US$1,300/oz Au price. Based on the DFS, the AGM mineral reserves support a variable LoM between 21 years, if the Company only processes 5Mtpa (P5M) or 12 years if the Company decides to double its processing capacity to 10Mtpa (P10M).

With respect to the AGM Mineral Resource and Reserve disclosures, it should be noted that:

- Asanko Gold has adjusted the mineral resource constraining Au price shell from US$2,000/oz Au (refer to press release dated 24 February 2017) to US$1,500/oz Au
- All Mineral Resources are stated inclusive of Mineral Reserves
- The adoption of a US$1,500/oz Au resource shell does not impact stated Mineral Reserves
- Economic evaluation of the AGM is based on Mineral Reserves, not Mineral Resources
- Given that the AGM has between a 12 and 21 year LoM based on current mineral reserves, US$1,500/oz was deemed by CSA as an appropriate future gold price for the potential of "eventual economic extraction" and best practice Mineral Resource disclosure

1.8.1 Cut-off Grade Estimate

The AGM Mineral Resource cut-off grade (0.5 g/t Au) is based on the actual operating expenditure for the Nkran pit. The operating costs for the Nkran pit opencast mining from April 2016 (commencement of commercial production) to the end of May 2017 have been used as a basis of establishing a pay limit or break-even grade (Table 1-6) The analysis indicates a pay limit of 0.45 g/t Au. This supports the application of a cut-off grade of 0.50 g/t Au for Mineral Resource disclosure. This cut-off has been deemed appropriately conservative for this purpose and validated by Malcolm Titley (CSA).

Table 1-6: Nkran Pay Limit Calculation based on Actual Operating Costs (source: Asanko Gold 2017)

	Opex	Gold	Pay Limit	Ave. Selling Price US$/oz	Opex Costs US$/t		
	US$/t	US$/g	Au g/t		Mining	Processing	G&A
April 2016 – May 2017	17.50	39.15	0.45	1,247	3.66	13.00	0.84

1.8.2 Mineral Resource Summary

The >0.5 g/t Au resources are constrained within US$1,500/oz Au pit shells.

Summary Measured and Indicated resources are shown in Table 1-7 and Inferred resources in Table 1-8.

Table 1-7: AGM Measured and Indicated Resources (source: Asanko Gold 2017)

AGM Measured and Indicated Resources 31st December, 2016 (Akwasiso 25th April 2017) at a 0.5 g/t Au cut-off within a US$1,500/oz Au shell				
Deposit	**Total Measured and Indicated Resources**			
	Mt	**Au g/t**	**Au oz**	**Au Moz**
Esaase Main	84.02	1.38	3,722,211	3.72
Nkran Main	30.07	1.78	1,723,824	1.72
Abore	5.33	1.45	247,917	0.25
Dynamite Hill	3.41	1.48	161,818	0.16
Akwasiso	6.33	1.50	305,181	0.31
Adubaiso	2.73	1.80	158,108	0.16
Esaase D	2.00	1.29	82,741	0.08
Esaase B	2,65	0.84	71,660	0.07
Asuadai	1.88	1.22	73,600	0.07
Adubaiso Ext	0.42	1.61	21,627	0.02
Nkran Ext	0.19	2.70	16,316	0.02

Total	**139.01**	**1.47**	**6,585,001**	**6.59**

Table 1-8: AGM Inferred Mineral Resources 31 December 2016 (Akwasiso 25 April 2017) (source: Asanko Gold 2017)

AGM Inferred Resources 31 December 2016 (Akwasiso 25 April 2017): 0.5 g/t Au cut-off within a US$1,500/oz Au Shell				
Deposit	**Inferred**			
	Mt	**Au g/t**	**Au oz**	**Au Moz**
Esaase Main	0.09	1.08	2,812	0.003
Nkran Main	0.31	1.86	18,319	0.018
Abore	1.28	1.61	66,084	0.066
Dynamite Hill	0.21	1.58	10,726	0.011
Akwasiso	0.18	0.81	4,611	0.005
Adubiaso	0.01	1.92	364	0.000
Esaase D	1.01	1.26	40,916	0.041
Esaase B	2.12	0.86	58,278	0.058
Asuadai	0.63	1.75	35,115	0.035
Adubiaso Ext	0.14	3.10	13,741	0.014
Nkran Ext	0.01	1.02	254	0.000
Total	**5.96**	**1.31**	**250,966**	**0.251**

Notes for Tables 1-7 and 1-8

1. CSA re-estimated Nkran and Dynamite Hill in January 2017 and Akwasiso in April 2017.
2. CJM estimated Esaase in October 2012 and Abore, Adubiaso, Asuadai in April 2014.
3. The resource cut-off grade used for all deposits was 0.5 g/t Au within a Whittle pit shell at US$1,500/oz Au.
4. Columns may not add up due to rounding.
5. All figures are in metric tonnes.
6. The MREs are stated as in situ tonnes.
7. Individual densities were used per mineral zone.
8. The tonnages and contents are stated as 100%, which means no attributable portions are stated in the table.
9. Conversion from grams to troy ounces – 31.1035 grams per troy ounce

1.9 Mining & Reserves

1.9.1 Mining Strategy

Asanko Gold is intending to expand the AGM in two stages, namely P5M and P10M.

P5M is the upgrade of the existing CIL processing facility, located at Obotan, from a design of 3 Mtpa to a processing capacity of 5 Mtpa. Ore tonnages will be supplied from the Obotan deposits until the

Esaase overland conveyor is completed. The processing facility will be fed with 3 Mtpa from Obotan and 2 Mtpa from Esaase prior to January 2019.

P10M is an increase in processing capacity to 10 Mtpa by integrating an additional 5 Mtpa CIL processing facility into the upgraded 5 Mtpa facility at the Obotan site. It is planned that Esaase will supply 7 Mtpa and Obotan 3 Mtpa to the expanded 10 Mtpa processing facility.

The expanded 10 Mtpa processing facility will produce approximately 450,000 oz Au per year.

1.9.2 Open Pit Mineral Reserves

The Mineral Reserve Estimates ("MRev") for the Nkran, Dynamite Hill and Akwasiso deposits have been estimated using the updated MRE that was prepared by CSA. The MRev for Esaase, Abore, Adubiaso, Asuadai, Adubiaso Extension and Nkran Extension deposits were estimated using the resource model that was prepared by CJM (CSA validated the CJM Esaase MRE). The Mineral Reserves are the portion of the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining loses and the addition of waste dilution.

The MRev for each of the deposits were developed through the open pit optimisation analysis utilising industry standard and accepted 3D Lerchs-Grossmann algorithm to determine the economic pit limits based on technical, financial and cost inputs. These inputs include unit mining costs, processing costs, general and administrative costs, and unit revenue estimates. Pit optimisation technical parameters include pit footprint constraint, estimates of mining dilution, mining loss, process recovery, and pit overall slope angles. Pit overall slope angles are derived from geotechnical criteria adjusted for the expected haulage ramp layout.

In accordance with the guidelines of National Instrument 43-101 Standards of Disclosure for Mineral Projects and the CIM's Definition Standards for Mineral Resources and Mineral Reserves, only those ore blocks classified in the Measured and Indicated categories are allowed to drive the pit optimisation for a feasibility level study. Inferred resource blocks, regardless of grade and recovery, bear no economic value and are treated as waste.

The pit optimisation analysis process identified the pit shell for each of the deposits that should be used as the basis for the open pit detailed design. The additional Measured and Indicated Mineral Resources that are outside the limits of these optimised pit shells were not considered for an underground mining operation. The cut-off grade for the open pit mines was calculated to be 0.5 g/t and 0.7 g/t for oxide and fresh material at the Obotan deposits respectively. The economic cut-off grade of 0.6 g/t Au was determined at the Esaase deposit.

The identified pit-shells resultant from the open pit optimisation analysis were used in conjunction with the pit slope recommendations and technical parameters to execute a detailed mine design and associated mine schedule. Table 1-9 details the pit designs for the Obotan and Esaase deposits.

Table 1-9: AGM Obotan and Esaase Pit Design Details (source: Asanko Gold 2017)

Pit	Total Tonnes (Mt)	Ore Tonnes (Mt)	Waste Tonnes (Mt)	Stripping Ratio	Ounces (Moz)	Feed Grade (g/t)
Nkran	162.31	23.13	139.17	6.02	1.42	1.91
Akwasiso	36.07	5.04	31.04	6.16	0.24	1.50
Dynamite Hill	19.47	2.84	16.63	5.85	0.14	1.49
Adubiaso	18.21	2.28	15.93	6.98	0.14	1.90
Adubiaso Extension	2.62	0.21	2.41	11.37	0.01	1.53
Nkran Extension	2.38	0.20	2.18	10.95	0.01	2.20
Abore	20.55	3.21	17.33	5.40	0.15	1.49
Asuadai	6.46	1.32	5.15	3.91	0.05	1.09
Total Obotan[Note 1]	**268.07**	**38.23**	**229.84**	**6.01**	**2.16**	**1.76**
Esaase Main	411.18	62.57	348.61	5.57	2.94	1.46
D Zone	5.44	0.60	4.84	8.07	0.03	1.56
B Zone	0.33	0.10	0.23	2.28	0.00	0.83
Total AGM	685.02	101.50	583.52	5.75	5.14	1.57

Note [1] Includes 0.5Mt of Economic Inferred Material incidental to Mine Design (excluded from reserves)

The detailed pit designs formed the basis of reporting the MRev. Only Measured and Indicated material (modified for dilution and ore-loss) above the economic cut-off grades are reported as Mineral Reserves. Table 1-10 the Mineral Reserves for the Obotan and Esaase deposits.

Table 1-10: Mineral Reserve Estimate for the AGM (source: Asanko Gold 2017)

Deposit	Classification	Tonnage (Mt)	Grade (g/t Au)	Content (Moz Au)
Nkran	Proven	4.40	1.85	0.26
	Probable	18.37	1.93	1.14
	Total	22.77	1.91	1.40
Nkran Ext	Proven	0.11	2.47	0.01
	Probable	0.08	1.91	0.00
	Total	0.19	2.24	0.01
Abore	Proven	1.59	1.44	0.07
	Probable	1.60	1.53	0.08
	Total	3.18	1.48	0.15
Adubiaso	Proven	1.04	2.00	0.07
	Probable	1.04	1.82	0.07
	Total	2.09	2.08	0.14
Adubiaso Ext	Proven	0.12	1.66	0.01
	Probable	0.09	1.34	0.00
	Total	0.21	1.53	0.01
Dynamite Hill	Proven	-	-	-
	Probable	2.84	1.49	0.14
	Total	2.84	1.49	0.14
Akwasiso	Proven	-	-	-
	Probable	4.95	1.51	0.24
	Total	4.95	1.51	0.24
Asuadai	Proven	-	-	-
	Probable	1.30	1.09	0.05
	Total	1.30	1.09	0.05
Total Obotan Reserve[Note 9]	Proven	7.26	1.79	0.42
	Probable	30.47	1.76	1.72
	Total	37.74	1.76	2.14

Deposit	Classification	Tonnage (Mt)	Grade (g/t Au)	Content (Moz Au)
Esaase (Main Pit)Note 10	Proven	21.51	1.44	1.00
	Probable	41.05	1.47	1.94
	Total Main Pit	62.57	1.46	2.94
Esaase (B Zone)	Proven	0.10	0.83	-
	Probable	0.00	0.92	-
	Total B Zone	0.10	0.83	-
Esaase (D Zone)	Proven	0.20	1.05	0.01
	Probable	0.40	1.70	0.02
	Total D Zone	0.60	1.56	0.03
Total AGM Reserve	Proven	29.08	1.52	1.42
	Probable	71.92	1.59	3.68
	Total	101.00	1.57	5.11

Notes:

1. Mineral Reserves are defined within a mine design guided by Lerchs-Grossman ("LG") pit shells.
2. The LG shell generation was performed on Measured and Indicated materials only.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal.
4. Tonnage and grade measurements are in metric units.
5. Reserves for each pit are based on detailed pit designed informed by $1,300/oz pit shells.
5. Minimum economic cut-off grade for Esaase deposits is 0.6g/t Au and Nkran fresh 0.7g/t Au. All other pits use an economic cut-off grade of 0.5g/t Au and 0.7g/t Au for oxide and fresh material respectively.
6. No inferred, deposit, or mineralised waste contributes value to the pit optimisation.
7. No inferred, deposit, or mineralised waste is included in the Mineral Reserve.
8. Proven and Probable Mineral Reserves are modified to include ore-loss and dilution.
9. Reserve excludes Obotan surface stockpiles (as at 1st April 2011) of 1.95 Mt @ 1.22g/t Au.
10. Mineral Reserve excludes ~10Mt at 0.55g/t Au of very low grade material in the measured and indicated categories contained within the Esaase main pit design.

The MRev for the AGM ore sources are not, at this stage, materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issue. Furthermore, the estimate of Mineral Reserves is not materially affected by any known mining, metallurgical, infrastructure, or other relevant factor.

1.9.3 Mining Methods

The mining method selected for the AGM will be a conventional open pit, truck and shovel, drill and blast operation. Vegetation, topsoil and overburden will be stripped and stockpiled for future reclamation use. The ore and waste rock will be mined with 6m high benches, drilled, blasted and loaded into rigid frame haul trucks (94t) with hydraulic excavators (17m^3). The primary mining fleet of trucks and excavators will be supported by standard open-cut drilling and auxiliary equipment. Pit support equipment will consist of dozers, graders, fuel bowsers, water bowsers, hydraulic hammer, TLB and wheel loaders.

Several waste rock piles have been designed and located on in close proximity to the open pits in order to reduce waste hauling costs. Waste rock backfilling of the open pit will be performed where possible with attention given to timing and potential sterilisation.

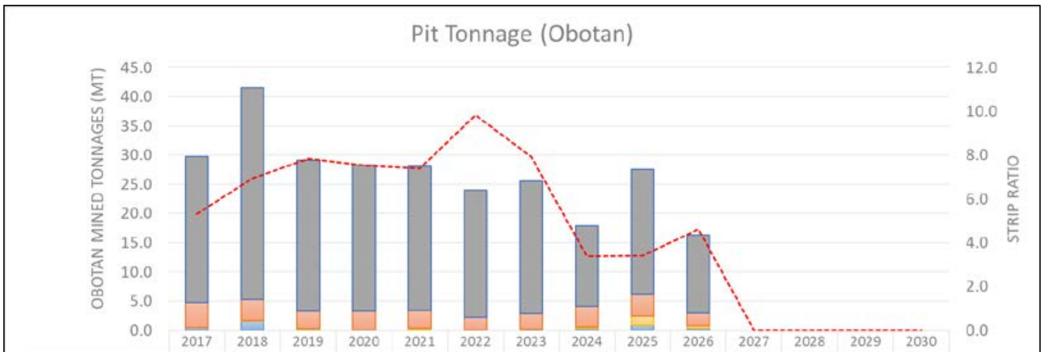
Mining operations at the AGM will be 50 weeks per year, operating around the clock on two 12 hour shifts. During this period, the CIL plant will be either fed from the run of mine ore stockpile and / or going through scheduled maintenance.

Esaase is planned for a peak total material movement of 52 Mtpa. The estimated increase in mining fleet requirements for Esaase is 96 pieces of mine equipment fleet, including 36 94-tonne haul trucks, four hydraulic excavators with 17 m^3 buckets, 12 diesel powered track production drills as well as various support and service equipment (graders, dozers, bowsers, front end loaders etc.).

The increase in the AGM workforce has been estimated to be 43 employees assigned to the processing plant. The estimated mining contractor workforce for 52 Mtpa peak mining rate has been estimated to be 479 mining contractor employees.

1.9.4 Mine Production Schedules

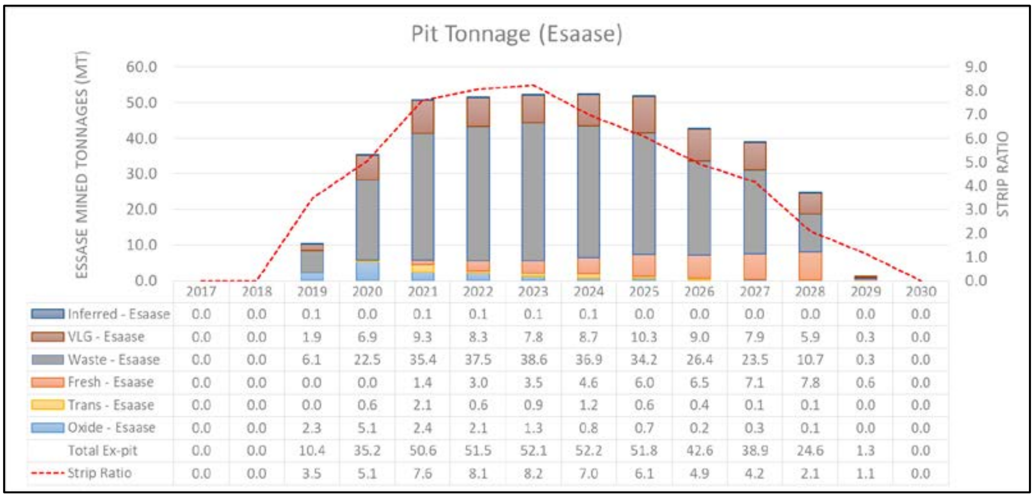
The Obotan mine production schedule as per updated mine planning (1st April 2017). Figure 1-10 below shows the updated Obotan mine schedule.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Waste - Obotan	25.1	36.3	25.8	24.9	24.7	21.8	22.7	13.8	21.3	13.4	0.0	0.0	0.0	0.0
Fresh - Obotan	4.3	3.5	3.1	3.3	3.0	2.2	2.8	3.6	3.9	2.2	0.0	0.0	0.0	0.0
Trans - Obotan	0.0	0.1	0.1	0.0	0.1	0.0	0.1	0.2	1.5	0.4	0.0	0.0	0.0	0.0
Oxide - Obotan	0.4	1.6	0.1	0.0	0.2	0.0	0.0	0.4	0.9	0.3	0.0	0.0	0.0	0.0
Total Ex-pit	29.8	41.6	29.1	28.3	28.0	24.0	25.5	17.9	27.6	16.3	0.0	0.0	0.0	0.0
Strip Ratio	5.3	6.9	7.8	7.5	7.4	9.8	7.9	3.4	3.4	4.6	0.0	0.0	0.0	0.0

Figure 1-10: April 2017 Obotan Mine Schedule (source: Asanko Gold 2017)

The Esaase mine production schedule is shown in Figure 1-11 below.

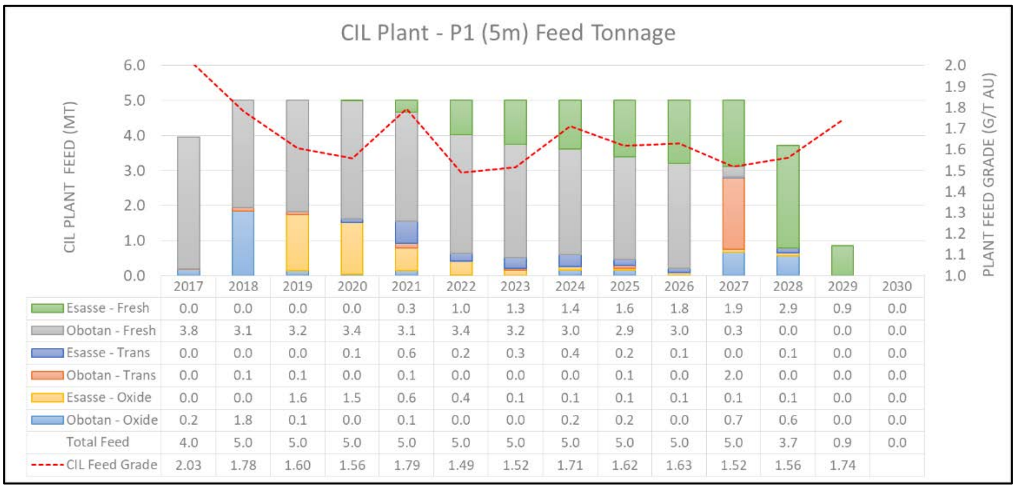


	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Inferred - Esaase	0.0	0.0	0.1	0.0	0.1	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0
VLG - Esaase	0.0	0.0	1.9	6.9	9.3	8.3	7.8	8.7	10.3	9.0	7.9	5.9	0.3	0.0
Waste - Esaase	0.0	0.0	6.1	22.5	35.4	37.5	38.6	36.9	34.2	26.4	23.5	10.7	0.3	0.0
Fresh - Esaase	0.0	0.0	0.0	0.0	1.4	3.0	3.5	4.6	6.0	6.5	7.1	7.8	0.6	0.0
Trans - Esaase	0.0	0.0	0.0	0.6	2.1	0.6	0.9	1.2	0.6	0.4	0.1	0.1	0.0	0.0
Oxide - Esaase	0.0	0.0	2.3	5.1	2.4	2.1	1.3	0.8	0.7	0.2	0.3	0.1	0.0	0.0
Total Ex-pit	0.0	0.0	10.4	35.2	50.6	51.5	52.1	52.2	51.8	42.6	38.9	24.6	1.3	0.0
Strip Ratio	0.0	0.0	3.5	5.1	7.6	8.1	8.2	7.0	6.1	4.9	4.2	2.1	1.1	0.0

Figure 1-11: Esaase Production Schedule (source: Asanko Gold 2017)

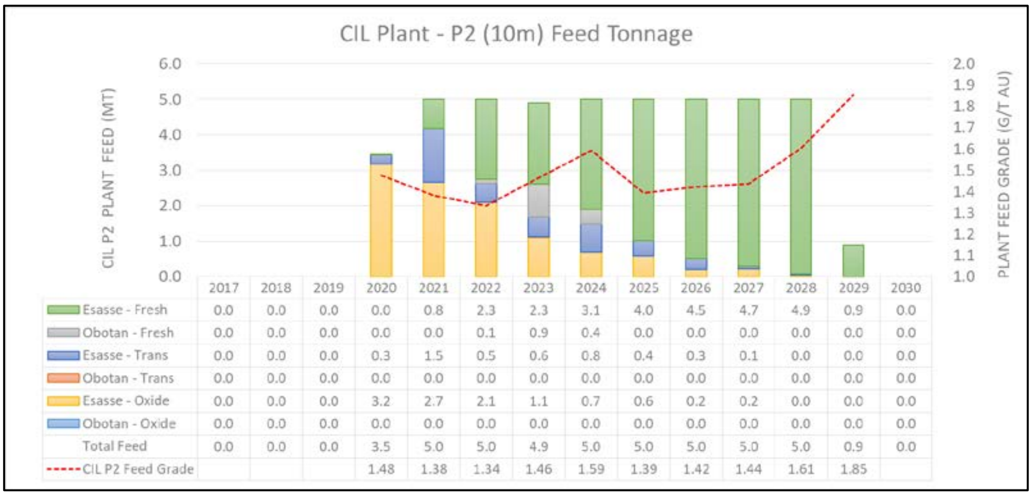
1.9.5 Plant Feed Schedules

The plant feed schedule is developed from the expected commissioning dates for the two plants and the aligned mine production schedules. Cognisance of stockpiling and re-handling is taken into account Figure 1-11 and Figure 1-12 below shows the P5M CIL and P10M CIL processing plant feed tonnage schedule respectively.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	0.0	0.0	0.0	0.0	0.3	1.0	1.3	1.4	1.6	1.8	1.9	2.9	0.9	0.0
Obotan - Fresh	3.8	3.1	3.2	3.4	3.1	3.4	3.2	3.0	2.9	3.0	0.3	0.0	0.0	0.0
Esasse - Trans	0.0	0.0	0.0	0.1	0.6	0.2	0.3	0.4	0.2	0.1	0.0	0.1	0.0	0.0
Obotan - Trans	0.0	0.1	0.1	0.0	0.1	0.0	0.0	0.0	0.1	0.0	2.0	0.0	0.0	0.0
Esasse - Oxide	0.0	0.0	1.6	1.5	0.6	0.4	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0
Obotan - Oxide	0.2	1.8	0.1	0.0	0.1	0.0	0.0	0.2	0.2	0.0	0.7	0.6	0.0	0.0
Total Feed	4.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	3.7	0.9	0.0
CIL Feed Grade	2.03	1.78	1.60	1.56	1.79	1.49	1.52	1.71	1.62	1.63	1.52	1.56	1.74	

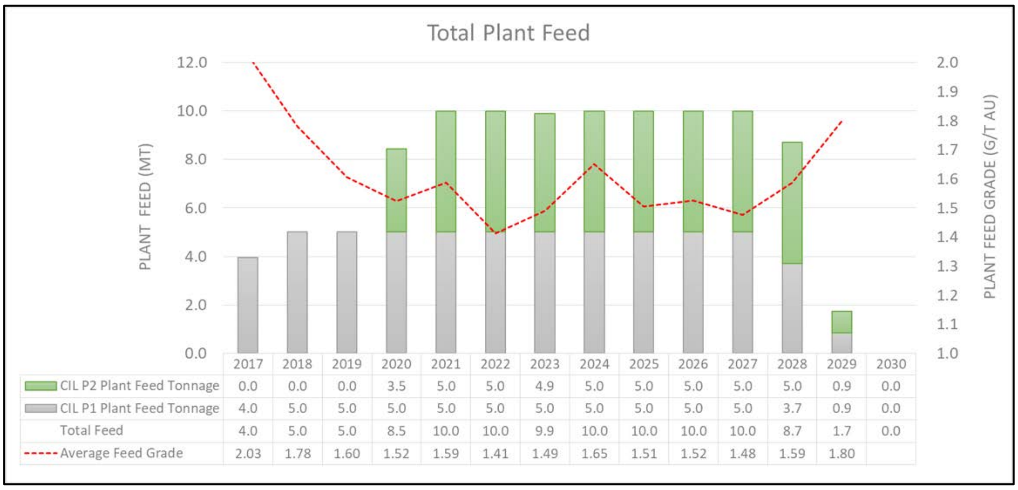
Figure 1-12: CIL P5M - Plant Feed Schedule (source: Asanko Gold 2017)



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	0.0	0.0	0.0	0.0	0.8	2.3	2.3	3.1	4.0	4.5	4.7	4.9	0.9	0.0
Obotan - Fresh	0.0	0.0	0.0	0.0	0.0	0.1	0.9	0.4	0.0	0.0	0.0	0.0	0.0	0.0
Esasse - Trans	0.0	0.0	0.0	0.3	1.5	0.5	0.6	0.8	0.4	0.3	0.1	0.0	0.0	0.0
Obotan - Trans	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Esasse - Oxide	0.0	0.0	0.0	3.2	2.7	2.1	1.1	0.7	0.6	0.2	0.2	0.0	0.0	0.0
Obotan - Oxide	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Feed	0.0	0.0	0.0	3.5	5.0	5.0	4.9	5.0	5.0	5.0	5.0	5.0	0.9	0.0
CIL P2 Feed Grade				1.48	1.38	1.34	1.46	1.59	1.39	1.42	1.44	1.61	1.85	

Figure 1-13: CIL P10M - Plant Feed Schedule (source: Asanko Gold 2017)

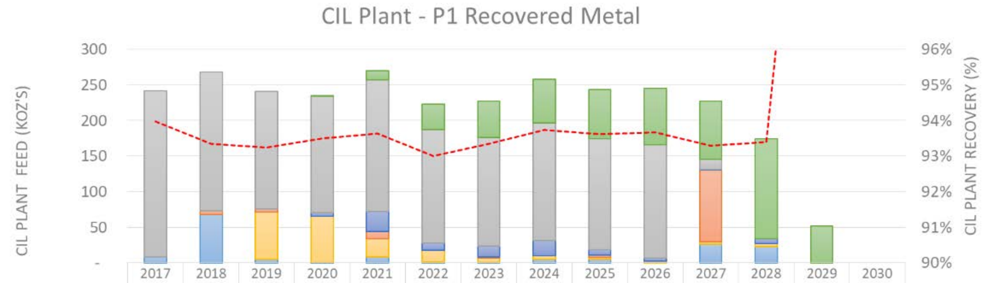
Figure 1-14 below shows the combined P10M CIL processing plant feed schedule.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
CIL P2 Plant Feed Tonnage	0.0	0.0	0.0	3.5	5.0	5.0	4.9	5.0	5.0	5.0	5.0	5.0	0.9	0.0
CIL P1 Plant Feed Tonnage	4.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	3.7	0.9	0.0
Total Feed	4.0	5.0	5.0	8.5	10.0	10.0	9.9	10.0	10.0	10.0	10.0	8.7	1.7	0.0
Average Feed Grade	2.03	1.78	1.60	1.52	1.59	1.41	1.49	1.65	1.51	1.52	1.48	1.59	1.80	

Figure 1-14: Combined Plant Feed Schedule (source: Asanko Gold 2017)

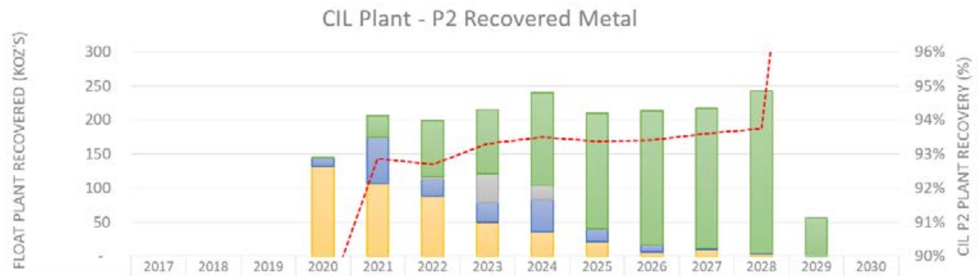
Figure 1-15 and Figure 1-16 below shows the P5M CIL and P10M CIL processing plant recovered metal respectively.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	-	-	-	0	13	36	52	61	69	79	82	141	52	-
Obotan - Fresh	234	195	165	163	185	159	152	166	156	160	15	-	-	-
Esasse - Trans	-	-	-	6	28	10	15	22	8	4	1	7	-	-
Obotan - Trans	0	5	4	0	11	-	2	-	3	-	100	-	-	-
Esasse - Oxide	-	-	66	64	26	16	6	5	2	2	4	4	-	-
Obotan - Oxide	8	68	4	1	8	1	0	5	5	-	26	22	-	-
Total Rec Au	242	268	241	234	270	223	227	258	243	245	228	174	52	-
Recovery	94.0%	93.3%	93.2%	93.5%	93.6%	93.0%	93.3%	93.7%	93.6%	93.7%	93.3%	93.4%	108.4%	

Figure 1-15: CIL P1 Plant Recoverable Gold (source: Asanko Gold 2017)

Note : Recovery in final year accounts for release of final gold lockup in the processing circuit.

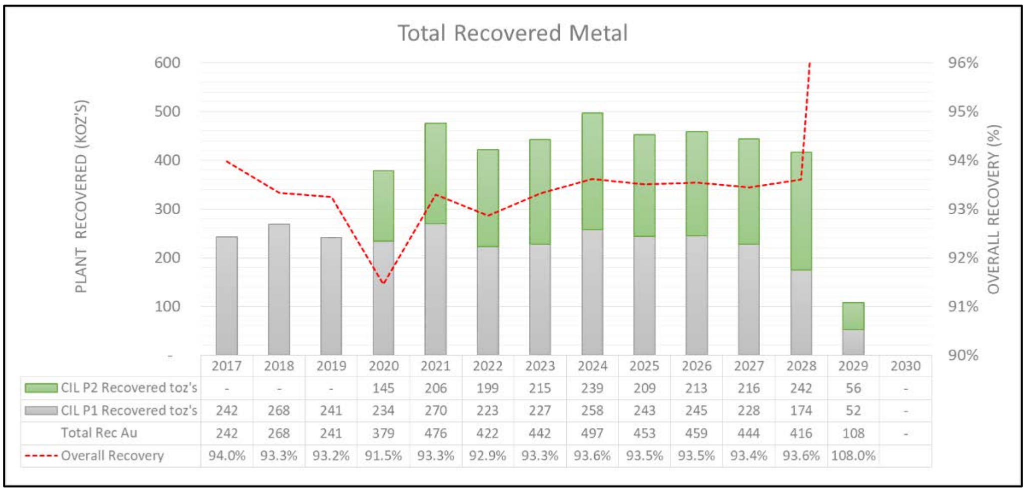


	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	-	-	-	1	32	82	94	135	170	198	205	237	56	-
Obotan - Fresh	-	-	-	-	-	5	43	21	-	-	-	-	-	-
Esasse - Trans	-	-	-	13	68	24	28	48	19	10	2	2	-	-
Obotan - Trans	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Esasse - Oxide	-	-	-	131	107	88	49	35	21	6	9	3	-	-
Obotan - Oxide	-	-	-	-	-	-	-	0	-	-	-	-	-	-
Total Rec Au	-	-	-	145	206	199	215	239	209	213	216	242	56	-
Recovery				88.3%	92.9%	92.7%	93.3%	93.5%	93.4%	93.4%	93.6%	93.7%	107.6%	

Figure 1-16: CIL P2 Plant Recoverable Gold (source: Asanko Gold 2017)

Note : Recovery in final year accounts for release of final gold lockup in the processing circuit.

Figure 1-17 below shows the combined P10M CIL processing recovered metal.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
CIL P2 Recovered toz's	-	-	-	145	206	199	215	239	209	213	216	242	56	-
CIL P1 Recovered toz's	242	268	241	234	270	223	227	258	243	245	228	174	52	-
Total Rec Au	242	268	241	379	476	422	442	497	453	459	444	416	108	-
Overall Recovery	94.0%	93.3%	93.2%	91.5%	93.3%	92.9%	93.3%	93.6%	93.5%	93.5%	93.4%	93.6%	108.0%	

Figure 1-17: Total Recoverable Gold (source: Asanko Gold 2017)

Note : Recovery in final year accounts for release of final gold lockup in the processing circuit.

1.10 Tailings Storage Facility

The TSF will consist of a multi-zoned downstream perimeter embankment, comprising a total footprint area of 378 ha (basin area 278 ha for the final TSF). The TSF is designed to store a total of 95 Mt over the LoM. Expansion of the TSF is feasible without significant change to the design parameters, and up to 120 Mt capacity has been considered in the design process.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the embankment. The design incorporates an upstream toe drain and basin under-drainage system in low lying basin areas to improve performance of the TSF. The under-drainage system comprises a network of collector and finger drains. The toe drain and under-drainage system drain by gravity to a collection sump located at the lowest point in the TSF.

Supernatant water will be removed from the TSF via submersible pumps located on a floating barge located within the supernatant pond throughout operation. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuits.

A downstream seepage collection system will be installed within and downstream of the TSF embankment, to allow monitoring and collection of seepage from the TSF in the collection sump located downstream of the final TSF.

Monitoring bores are being installed around the TSF, to constantly monitor water quality of the samples withdrawn from them. This will allow any seepage, or contamination to be detected, and will trigger the mitigation measures to be outlined in the TSF Management Plan.

The TSF embankment will be constructed in stages to suit storage requirements and the availability of suitable mine waste. It is envisaged that the upstream portions of the embankment will be raised annually by an earthworks contractor with the bulk embankment fill being placed as part of the mining operations on an ongoing basis.

The TSF design utilises a beach angle that has been calculated using published methods and the overall design is regarded as conservative, with no unique, or unusual design parameters, or methodologies utilised in the design. The use of downstream raise construction methods promote embankment stability, which has been demonstrated by the high factors of safety obtained for the stability assessment. The stability factors of safety comply with the latest Ghanaian mining regulations for up to 120 Mt TSF capacity. It should be noted that these regulations factors of safety are higher than accepted worldwide standards.

1.11 Environmental and Social

Asanko Gold's Environmental and Social Impact Assessment ("ESIA") for Nkran was approved in 2015 by the EPA, resulting in the issuing of an environmental permit. An ESIA for Esaase pit and the overland conveyor between Esaase and Obotan was submitted to the EPA in November 2016 and the environmental permit was granted in January 2017.

Detailed baseline studies were completed and this provided the required level of information for development of the ESIA.

Air quality, noise, surface water hydrology, groundwater hydrogeology, water quality, soil, fauna and flora baseline studies were completed and reports generated. Traffic, socio-economic and medical surveys were likewise completed.

Asanko Gold held a number of public forums as part of the environmental permitting for P5M and P10M, in 2016. This included a visit by senior dignitaries from the Esaase community and the EPA to the Sasol operation at Secunda in South Africa to see one of the world's longest single flight overland conveyors.

The Company engages regularly with a number of stakeholder groups and committees as platforms through which to provide project updates; address concerns and discuss matters of mutual interest. The Company also engages with local government and village leaders, including:

- Amansie West District Assembly
- Ministry of Food and Agriculture
- Ghana Health Service
- Land Valuation Board
- Environmental Protection Agency
- Forestry Commission
- Minerals Commission
- Inspectorate Division of Minerals Commission
- Water Resources Commission

1.12 Process

1.12.1 Metallurgical Test Work

The Obotan Gold Project was developed by Resolute during 1996-1997. In 1999 Resolute completed an upgrade study to expand the Obotan Gold Project to treat Oxide ore from the satellite pit Adubiaso and primary ore from the Nkran ore body. After the closure of the Resolute mining operations at Obotan in November 2002, further metallurgical test work campaigns were carried out for the treatment of Obotan ore. The AGM Phase 1 flow sheet was developed on historical operating data and testing conducted on Obotan samples in 2015. The AGM Phase 1 circuit was based on a typical single stage crushing, SABC milling circuit with gravity concentration followed by a CIL plant, and was commissioned in 2016.

The Esaase deposit has been subjected to six phases of metallurgical test work aimed at determining comminution, gravity, and leaching and flotation parameters. As a result of the findings from the Phase IV test work the final process for the Esaase 2012 PFS included ball milling, gravity gold recovery from the milling circuit and flotation of the milled product, at a grind of 80% passing 75 µm. The flotation concentrate would then be subjected to ultra fine grinding, treated in a gravity concentration circuit and leached with cyanide in a CIL circuit for gold recovery. In late 2014 and early 2015 further metallurgical test work was undertaken to support the AGM Phase 2 PFS which considered the combined treatment of Esaase and Obotan ore at a central processing facility consisting of a 3 Mtpa gravity-CIL circuit and a 5 Mtpa gravity-Flotation-Regrind-Concentrate CIL circuit. The PFS targeted the treatment of the majority of the Obotan and Esaase sulphide material in a new 5 Mtpa gravity-Flotation-Regrind-Concentrate CIL circuit, while the remaining material reported to existing 3 Mtpa gravity-CIL processing circuit.

Further test work was conducted as part of the 2016 AGM DFS, during which the opportunity was investigated to process Obotan gravity tailings and Esaase flotation concentrate as a combined feed stream to the existing Phase 1 CIL circuit. Laboratory recoveries were unaffected by the co-processing of the Obotan gravity tailings and Esaase flotation concentrate, however, carbon adsorption test work indicated slow carbon adsorption kinetics for the co-processed slurry due to carbon poisoning of the Esaase flotation reagents, which resulted in additional capital cost required to allow for extra adsorption stages to limit gold solution losses.

Based on the above, a decision was taken to re-evaluate the opportunity of a whole ore leach circuit for the Esaase material. Additional gravity-CIL test work on Esaase Sulphide material was then conducted as an addendum to the Phase 2 test work campaign, to investigate and support the decision to change the P10M processing plant from a 5 Mtpa gravity-Flotation-Regrind-Concentrate CIL circuit to a gravity-CIL circuit. During the addendum test campaign, the Esaase Sulphide material achieved similar recoveries to the Obotan Sulphide material when processed in the same gravity-CIL circuit configuration at a grind of 80% passing 106 µm. Final residue grades ranging from 0.100 g/t Au to 0.125 g/t Au where achieved resulting in overall recoveries of 93.5% to 95.6%.

1.12.2 Recovery Methods

The AGM expansion projects are phased in two stages. The first stage of the project, P5M, includes the upgrading of the existing Phase 1 CIL plant from the current throughput of 3.6 Mtpa to 5 Mtpa (Plant 1). The increased throughput will be achieved by supplementing the current 3.6 Mtpa of Nkran Sulphides material with oxide material, initially from a number of satellite pits and later from Esaase. A number of equipment upgrades are required to the existing circuit in order to process 5 Mtpa.

The second stage of the project, P10M, includes the addition of a new 5 Mtpa CIL processing facility (Plant 2), similar to the upgraded Phase 1 CIL plant design, to take the total processing throughput of the AGM DFS to 10 Mtpa. The additional feed will all come from the Esaase mine which will ramp up to 7 Mtpa.

1.12.2.1 AGM Phase 1 CIL Plant

The AGM Phase 1 processing plant was commissioned during Q4 2015 and is currently operating at a throughput of 3.6 Mtpa and achieving recoveries in excess of 94%. The AGM Phase 1 processing plant design is based on a typical single stage crushing, SAG and ball milling circuit followed by a CIL plant. The flow sheet includes a single stage jaw crusher that can either feed onto a live stockpile, or directly into an open circuit SAG, (complete with pebble crusher) and ball milling unit in closed circuit with classification cyclones. A gravity recovery circuit is utilised to treat a portion of the cyclone underflow stream to recover coarse free gold from the recirculating load.

The milled product (cyclone overflow) gravitates to a pre-leach thickener, via a trash removal screen. Thickener underflow is pumped directly to a pre-oxidation stage followed by a seven stage CIL circuit. Leached gold absorbs onto activated carbon, which flows counter-currently to the gold-bearing slurry. Loaded carbon is directed to the 5t elution circuit while tailings gravitates to the cyanide destruction circuit.

Provision was made in the design for the detoxification of cyanide in the CIL tailings by means of the SO2/Air process, during which the WAD cyanide concentration is reduced in a single tank by means of SMBS and air. The detoxified tailings product gravitates to the CIL tailings disposal tank via a sampling system from where it is pumped to the tailings storage facility.

Absorbed gold is eluted from the activated carbon by means of a heated solution of sodium cyanide and caustic soda via the split AARL procedure. Barren carbon from the batch elution process is directed to the carbon regeneration circuit, while the pregnant leach solution is routed to the electro winning circuit. After washing the gold sludge from the electro winning cathodes, the sludge is decanted and treated in a drying oven after which it is mixed with fluxes and loaded into an induction smelting furnace. After smelting the gold bullion bars are cleaned, labelled, assayed and prepared for shipping.

The Phase 1 CIL plant further incorporates water treatment, reagent preparation, oxygen generation and supply, compressed air and water services.

This process flow sheet is well known in industry, and is relatively low risk as it has historically been proven a successful processing route for Obotan region ores during Resolute's operations of 1998 to 2002.

Refer to Figure 1-18 for a simplified process flow diagram of the Phase 1 circuit.

A number of equipment upgrades are required to the existing circuit in order to process 5 Mtpa, as indicated in Figure 1-19

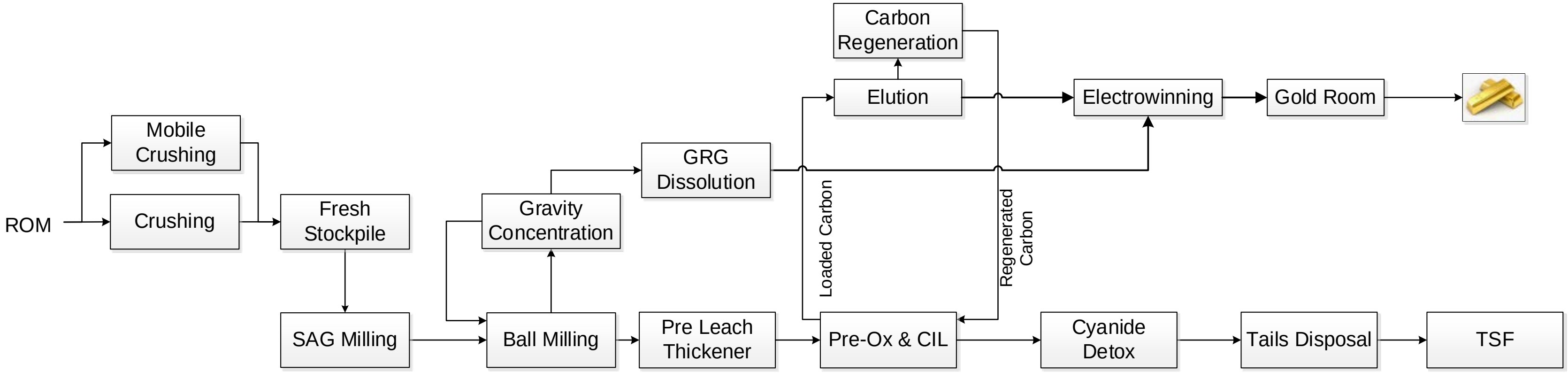


Figure 1-18: AGM Phase 1 CIL Plant Block Flow Diagram (source: DRA 2017)

1.12.2.2 Project 10M CIL Plant

The process flow sheet for the P10M CIL circuit will be based on the final P5M circuit, where feasible.

P10M circuit will consist of a ROM handling and a closed circuit, two-stage crushing circuit located at the Esaase mine pit, followed by stockpiling and loading of the stockpiled material onto the 35 kilometre overland conveying circuit to transport the crushed material to the Phase 1 processing site at Obotan where the milling, gravity recovery, and CIL circuit would be located. Provision is made for intermediate stockpiling of the Esaase crushed material and interlinking conveyors between the Project 5M and P10M CIL circuits to allow the processing of Esaase material in either of the CIL circuits.

The crushed Esaase material will feed onto a live, intermediate stockpile from where it can either be fed to the P5M (upgraded Phase 1) milling circuit, or to the P10M milling circuit once constructed.

The P10M ball milling circuit will operate in closed-circuit with a classification cyclone cluster. A gravity recovery circuit will be provided to treat the full cyclone underflow stream to maximize the recovery of coarse free gold from the recirculating load.

The milled product (cyclone overflow) will gravitate to the P10M pre-leach thickener, via a trash removal screen. Thickener underflow will be pumped directly to a pre-oxidation stage followed by a seven stage CIL circuit, as per the existing plant.

Leached gold will adsorb onto activated carbon, which flows counter-currently to the gold-bearing slurry. Loaded carbon will be directed to the elution circuit while tailings will gravitate to a dedicated P10M cyanide destruction circuit. As per the existing circuit, provision will be made in the design to cater for the destruction of cyanide in the P10M CIL tailings using the SO_2/Air process. WAD concentration will be reduced in a single tank by means of SMBS and air, after which the detoxified tailings will gravitate to a dedicated P10M CIL tails disposal system via a sampling system, from where it will be pumped to the expanded, common TSF.

As per the Phase 1 circuit, the absorbed gold will be eluted from the activated carbon by means of a heated solution of sodium cyanide and caustic soda via the split AARL procedure. Barren carbon from the batch elution process will be directed to a dedicated P10M carbon regeneration circuit, while the Pregnant Leach Solution ("PLS") will be routed to an upgraded electro winning circuit. A dedicated 5t elution facility will be provided for P10M.

The P5M electro winning circuit and gold room will further be expanded as part of P10M to cater for the additional gold loading due to the inclusion of the second 5 Mtpa CIL circuit. After washing the gold sludge from the electro winning cathodes, the sludge will be decanted and treated in either one of two drying ovens after which it will be mixed with fluxes and loaded into an induction smelting furnace. After smelting the gold bullion bars will be cleaned, labelled, assayed and prepared for shipping.

The P10M CIL plant will make use of the existing water treatment, reagent preparation, and reagent storage facilities where possible. Dedicated oxygen generation and supply, compressed air and water services will be provided for the P10M CIL plant.

A simplified process flow diagram for AGM DFS is provided in Figure 1-19.

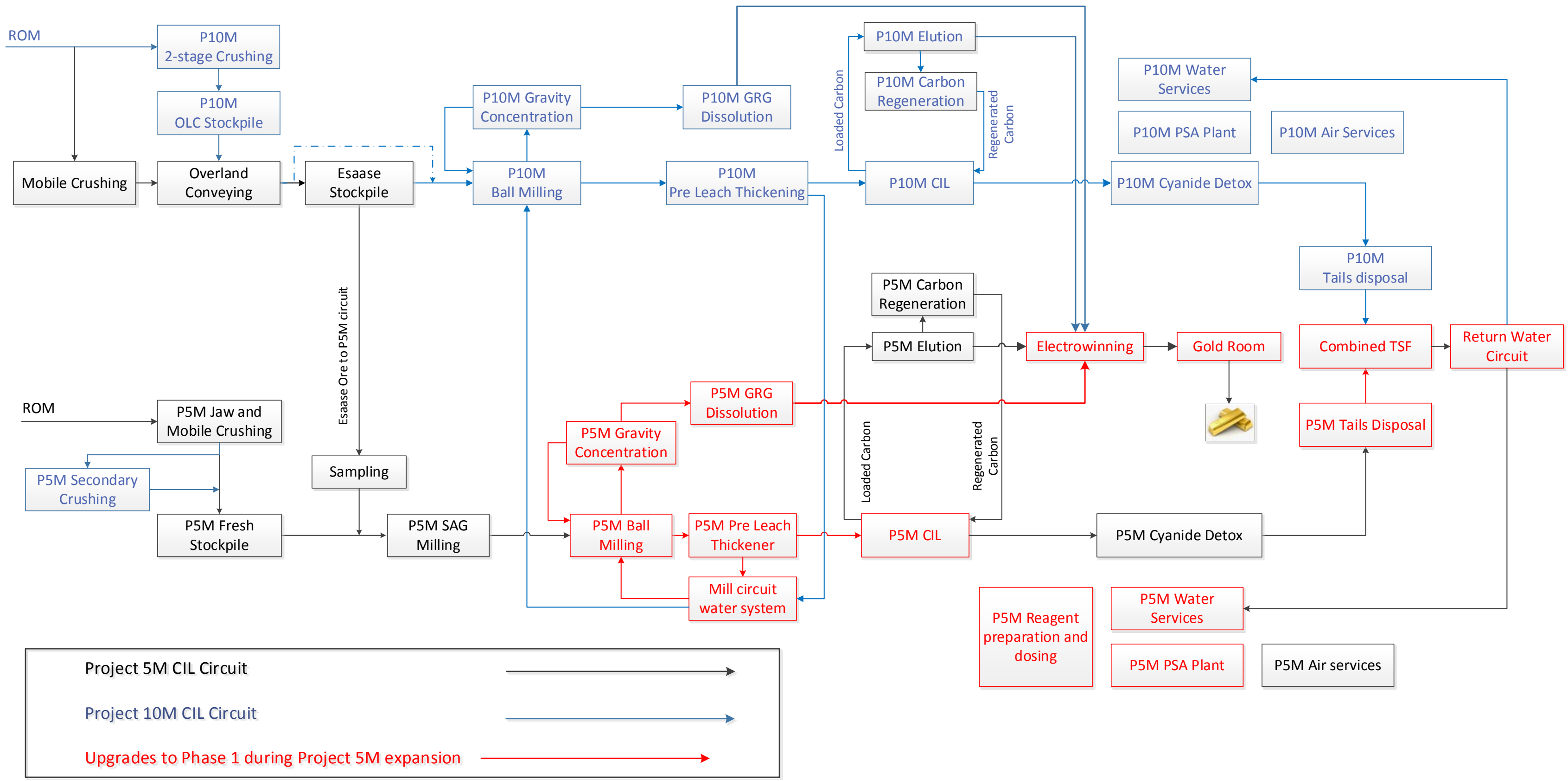


Figure 1-19: AGM CIL Plant Block Flow Diagram (source: DRA 2017)

1.12.3 Gold Recovery

Based on various metallurgical test work, as well as operational experience from previously mined deposits and the current Phase 1 Obotan plant, the overall expected gold recoveries for each of the AGM DFS CIL circuits is shown in Table 1-11 below.

Table 1-11: Predicted AGM DFS P5M and P10M CIL Plant Recoveries (source: DRA 2017)

		P5M CIL Plant	P10M CIL Plant
Mill Feed Tonnage	kt	58,525	44,219
Mill Feed Tonnage Split	%		
Obotan Oxide	%	*6.6%*	*0.0%*
Obotan Transitional	%	*4.3%*	*0.0%*
Obotan Fresh	%	*55.3%*	*3.2%*
Esaase Oxide	%	*8.0%*	*24.4%*
Esaase Transitional	%	*3.6%*	*10.2%*
Esaase Fresh	%	*22.3%*	*62.2%*
Mill Feed Grade	g/t Au	1.65	1.46
Gravity Recovery	%	52.5%	52.2%
CIL Feed Grade	g/t Au	0.78	0.70
CIL Residue Grade	g/t Au	0.10	0.09
Undiscounted Recovery	%	94.2%	94.1%
Recovery Discount	%	0.5%	0.8%
Discounted Recovery	%	93.7%	93.3%

1.13 Capital Costs

Capital and operating cost estimates for the scenarios have been prepared by Asanko Gold and their appointed independent consultants, with the basis of the estimates detailed in their respective sections of the NI 43-101.

1.13.1 Capital Costs

The capital costs provided for in the DCF model (Base Case) are summarised Table 1-12 and Figure 1-20.

Table 1-12: Base Case - Total Capital Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)*
Total Installation Capital	**-**
Total Stay-in-business Capital ("SIB")	**66**
On-Going Rehabilitation	4
Closure Cost	26
Tailings Dam	36

*Rounding applied

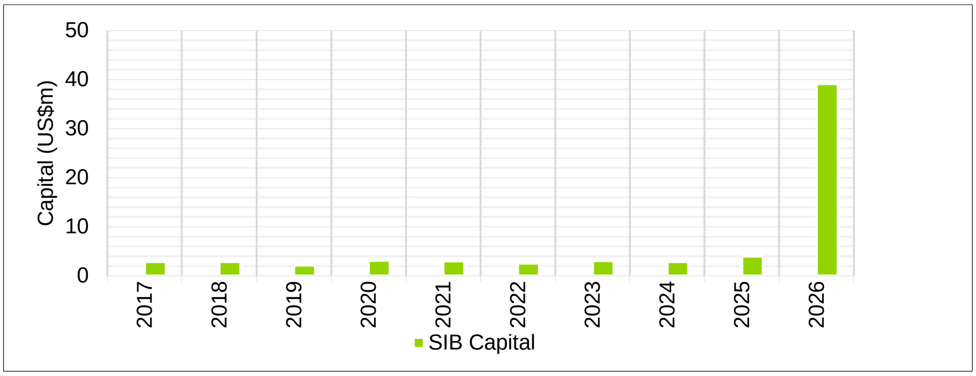


Figure 1-20: Base Case - Capital Scheduling (source: Venmyn Deloitte 2017)

The capital costs provided for in the DCF model (Base Case + P5M + P10M) are summarised in Table 1-13 and Figure 1-21.

Table 1-13: Base Case + P5M + P10M-Total Capital Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)*
Total Installation Capital	**349**
Process Plant	100
Overland Conveyor	78
Process Plant Infrastructure	55
RAP Project	24
Mining	8
Owners Cost	32
Project Indirect	29
Design Development	14
Contingency	9
Total SIB	**125**
On Going Rehabilitation at Obotan	8
Closure Cost (Obotan)	19
Tailings Dam	56
On Going Rehabilitation at Esaase	14
Closure Cost (Esaase)	12
Non-Mining Infrastructure	5
RAP Project	11

* Rounding Applied.

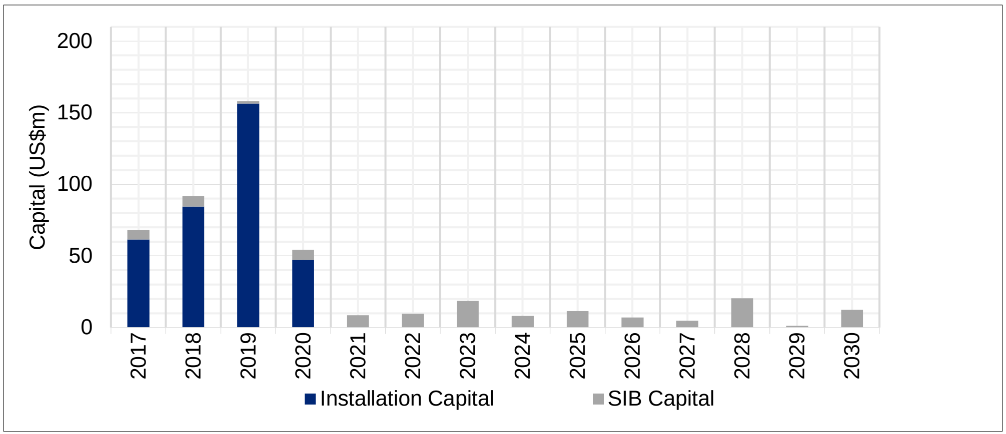


Figure 1-21: Base Case + P5M + P10M - Capital Scheduling (source: Venmyn Deloitte 2017)

The capital costs provided for in the DCF model (Base Case + P5M) are summarised in Table 1-14 and Figure 1-22.

Table 1-14: Base Case + P5M - Total Capital Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)*
Total Installation Capital	**147**
Process Plant	6
Overland Conveyor	90
Process Plant Infrastructure	13
RAP Project	-
Mining	1
Owners Cost	14
Project Indirect	13
Design Development	6
Contingency	4
Total SIB	**175**
On Going Rehabilitation at Obotan	8
Closure Cost (Obotan)	19
Tailings Dam	67
On Going Rehabilitation at Esaase	23
Closure Cost (Esaase)	12
Non-Mining Infrastructure	10
RAP Project	36

*Rounding applied.

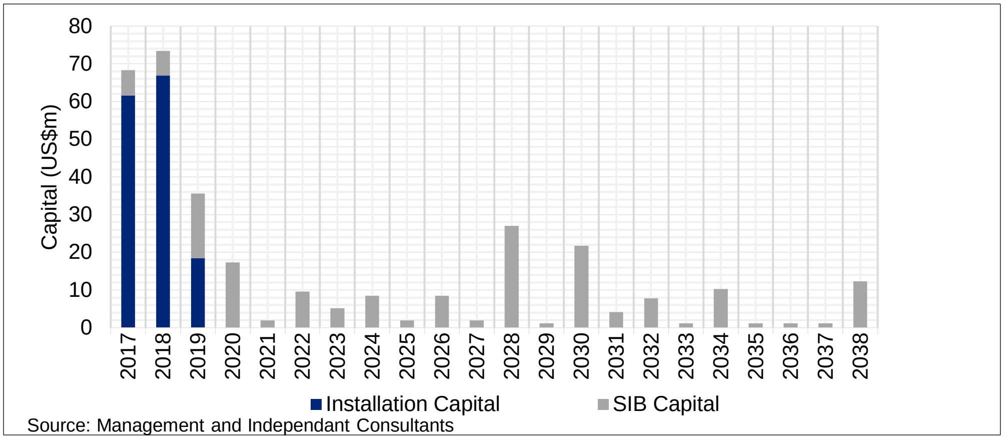


Figure 1-22: Base Case + P5M - Capital Scheduling (source: Venmyn Deloitte 2017)

1.14 Operating Costs

The operating costs accounted for in the financial model are shown in Table 1-15, Figure 1-23 and Figure 1-24.

Table 1-15: Base Case - Total Operating Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)	Amount (US$/oz)
Mining (Obotan)	832	443
Mining (Esaase)	-	-
Processing	424	226
Other (Refining and G&A)	241	128
Total OPEX	**1,497**	**797**

*Rounding applied

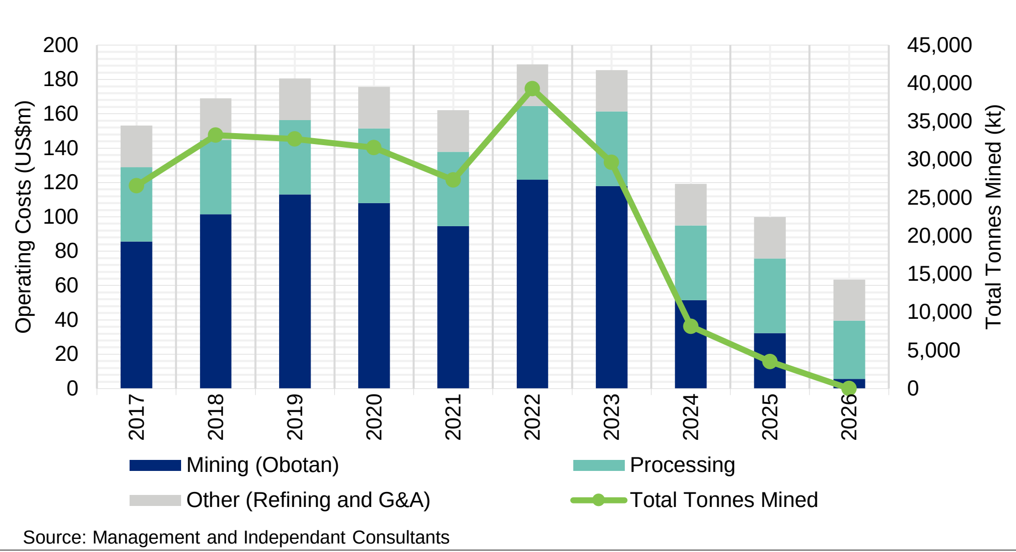


Figure 1-23: Base Case - Operating Cost Scheduling (source: Asanko Gold 2017)

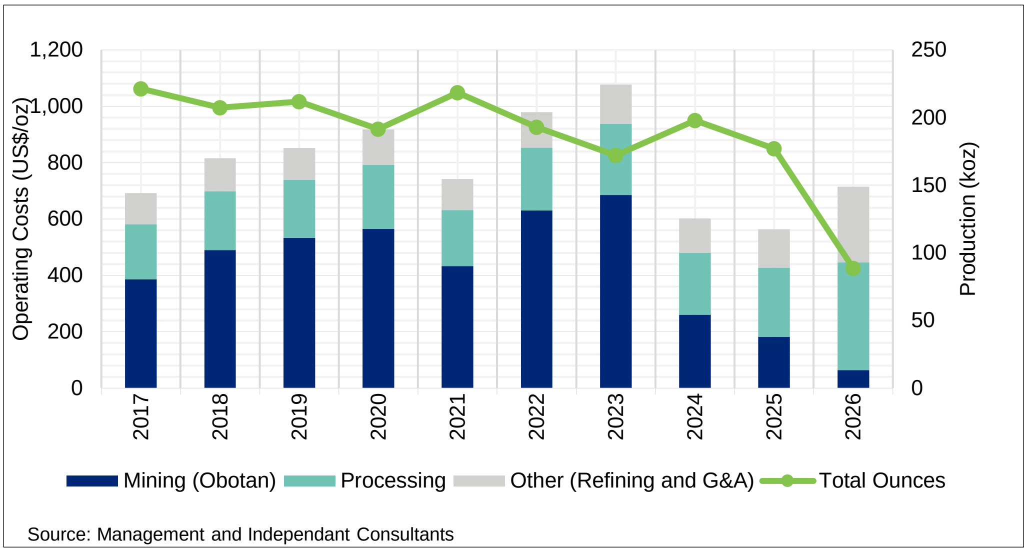


Figure 1-24: Base Case - Operating Cost US$/oz (source: Asanko Gold 2017)

The operating costs accounted for in the financial model for Base Case + P5M + P10M over the scheduled LoM are summarised in Table 1-16, Figure 1-25 and Figure 1-26.

Table 1-16: Base Case + P5M + P10M - Total Operating Costs (source: Asanko Gold 2017)

Aspect	Amount (US$m)	Amount (US$/oz)
Mining (Obotan)	903	186
Mining (Esaase)	1,351	279
Processing	1,109	229
Other (Refining and G&A)	323	67
Total OPEX	**3,686**	**761**

*Rounding applied

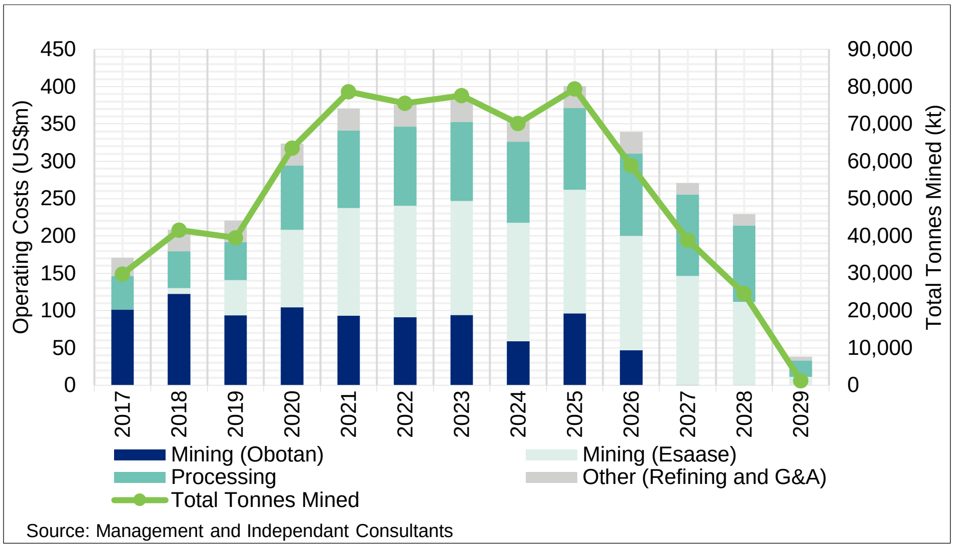


Figure 1-25: Base Case + P5M + P10M - Operating Cost Scheduling (source: Asanko Gold 2017)

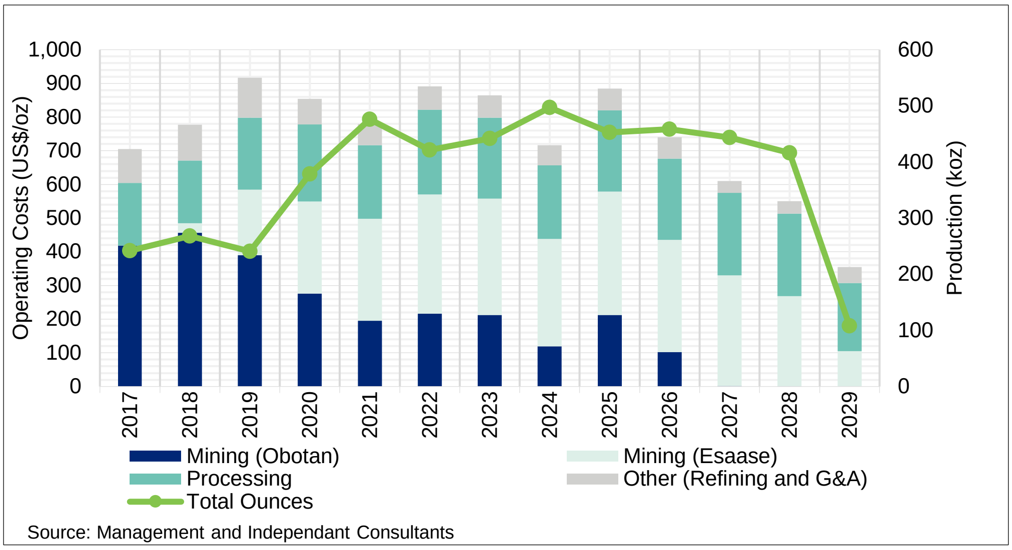


Figure 1-26: Base Case + P5M + P10M - Operating Cost US$/oz (source: Asanko Gold 2017)

The operating costs accounted for in the financial model for Base Case + P5M over the scheduled LoM are summarised in Table 1-17. Figure 1-27 and Figure 1-28.

Table 1-17: Base Case + P5M - Total Operating Costs (source: Asanko Gold 2017)

Aspect	Amount (US$m)	Amount (US$/oz)
Mining (Obotan)	903	186
Mining (Esaase)	1,460	301
Processing	1,192	246
Other (Refining and G&A)	505	104
Total OPEX	**4,060**	**837**

*Rounding applied

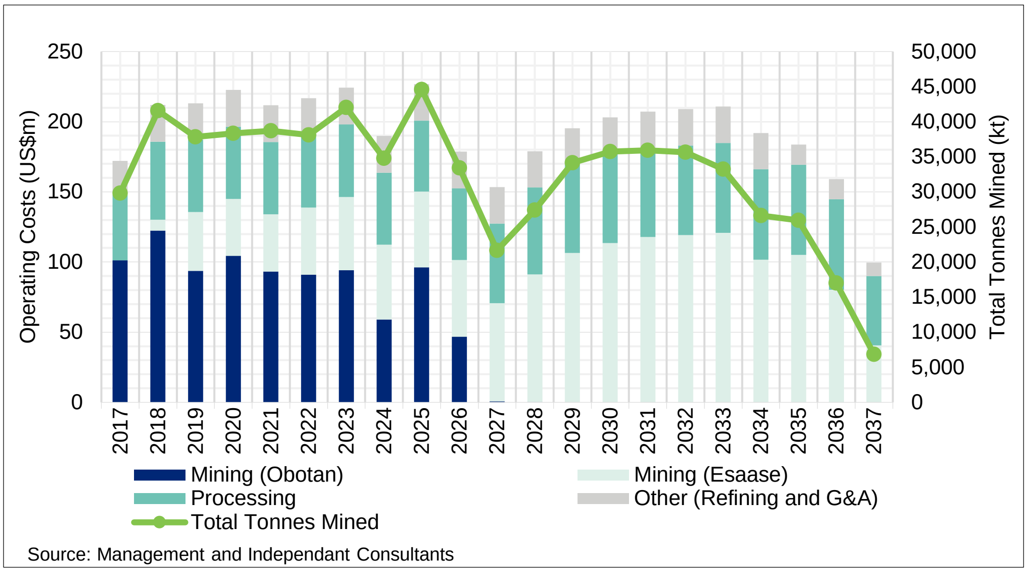


Figure 1-27: Base Case + P5M - Operating Cost Scheduling (source: Asanko Gold 2017)

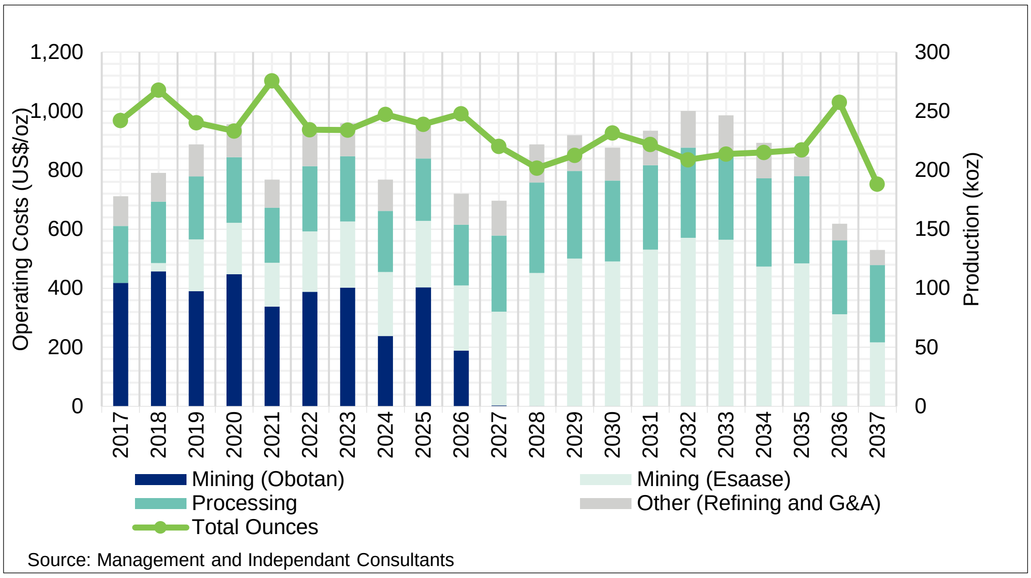


Figure 1-28: Base Case + P5M - Operating Cost US$/oz (source: Asanko Gold 2017)

1.15 Economic Analysis

1.15.1 The AGM – Economic Analysis

Asanko Gold has conducted an Economic Analysis for the Project based on the current LoM plan and cash flow model inputs supplied by Asanko Gold and its appointed contractors, which are summarised in Section 22.

1.15.2 Principle Assumptions

The principle assumptions employed in the Economic Analysis of the Project are presented Table 1-18.

Table 1-18: Principle Assumptions (source: Venmym Deloitte 2017)

Aspect	Unit Of Measure	Assumption
Discount Rate	(%)	5.00 (Industry common practice for economic analysis)
Obotan - Royalty Rate	(%)	5.00
Esaase - Royalty Rate	(%)	5.50
Corporate Tax	(%)	35.00
Capital Costs	(comment)	Section 4.1
Operations Costs	(comment)	Section 4.2
Model Calendar	(comment)	The year 2017 is Year 0 in the financial model. For the purposes of the NPV calculations, all values are discounted to the beginning of Year 0 (2017).
Commodity Price	(US$/oz)	US$1,250/oz (Adopted due to behavior of spot and historic trailing averages at time of analysis)
Financing	(comment)	100% Equity Financing

1.15.3 Cash Flow Approach

The annual cash flows for the Project Scenarios are summarised in Section 22.1.2.

1.15.4 NPV, IRR and Capital Payback Period

1.15.4.1 NPV and IRR

In consideration of the above assumptions, the following net present values (NPVs) and internal rates of return (IRR) were reached:

- Base Case:
  - NPV: US$481.82m
  - IRR: N/A as net cash flow of Base Case starts positively
- Base Case + P5M + P10M:
  - NPV: US$811.39m
  - IRR: N/A as net cash flow of Base Case + Project starts positively
- Base Case + P5M:
  - NPV: US$657.72m
  - IRR: N/A as net cash flow of Base Case + Project starts positively
- Project (P5M + P10M Incremental):
  - NPV: US$329.57m
  - IRR: 20.37%
- Project (P5M Incremental):
  - NPV: US$175.89m
  - IRR: 13.22%

1.15.4.2 Payback

In consideration of the above assumptions, the payback period for the:

- Project (Base Case + P5M + P10M) is 4 years
- Project (Base Case + P5M) is 5 years

1.16 Project Development

Phase 1 was approved by the Asanko Gold Board of Directors in July 2014 and DRA was awarded an EPCM contract immediately following approval. Contractor mobilization to site occurred in August 2014 and the Phase 1 development was successfully commissioned in Q1 2016 and commercial production was declared in Q2 2016, ahead of schedule and within budget.

P5M was approved by the Asanko Gold Board in November 2016 and a FEED programme commenced through to Q2 2017. Included in this phase was final designs on earthworks relating to the overland conveyor, detailed design of all structural and mechanical components of the overland conveyor and all detailed engineering and process design to allow the existing Phase 1 CIL plant to increase its design throughput capacity from 3 Mtpa to 5 Mtpa. The modifications to the existing plant include:

- An additional tailings pump chain and pipeline to the TSF
- One additional Knelson gravity gold concentrator
- An additional intensive leach reactor
- Installation of a larger diameter cyclone overflow pipeline to the preleach thickener
- Installation of a larger diameter thickener underflow pipeline to the CIL
- Increased capacity of the oxygen plant to deliver an additional 5 tpd oxygen
- An additional electo winning cell in the gold room
- Installation of larger aperture inter tank screens

These modifications will allow the existing CIL plant to process 5 Mtpa and will be completed in Q4 2017.

As part of P5M, a work stream will be executed to construct the terrace for the overland conveyor and installation of all related mechanical, electrical and instrument control systems. This will take 18 months to complete, which will allow ore from Esaase to fed at 2 Mtpa to feed the expanded plant at Obotan.

The infrastructure requirements at Esaase include:

- The refurbishment of the Esaase camp
- The resettlement of the Tetrem village
- The installation of a 33 kV power line to supply power to the conveyor and the Esaase site
- The establishment of the mine service area (“MSA”)
- The installation of a crushing system and stockpile to feed the overland conveyor
- Water management systems including seepage and holding dams

P10M (when approved by the Asanko Gold Board) will include the following:

- Construction of an additional 5 Mtpa CIL plant at Obotan to double processing capacity to 10 Mtpa
- An increase in the mining rate at Esaase from 2 Mtpa to 7 Mtpa
- A permanent primary and secondary crusher installation
- An expansion of the existing TSF

2 TERMS OF REFERENCE

2.1 The Issuer

This NI 43-101 technical report has been prepared on behalf of Asanko Gold, a gold mining company listed on the TSX and NYSE, with headquarters at 680-1066 West Hastings Street, Vancouver, British Columbia.

2.2 Terms of Reference

In June 2015 Asanko Gold commissioned DRA, and various specialist consultants, to complete a DFS on the expansion of the AGM, located in Ghana, West Africa, from the current open pit mining and processing operation to include an expanded processing facility and to bring the Esaase deposit into production, with construction expected to start in Q2 2017.

- Current operation (previously referred to as Phase 1) - as commissioned in Q1 2016
  - CIL processing facility, located at the Obotan project site, operating at 3.6 Mtpa (design was originally 3 Mtpa)
  - A TSF
  - LoM of approximately 10 years to 2026
  - Ore sources: Nkran and Satellite pits
  - Note: Phase 1 was originally intended to process 3 Mtpa, but it was found that the nameplate capacity of the plant could be increased as the milling circuit had excess capacity.

This AGM 2017 DFS follows on from the Phase 2 PFS published by Asanko Gold in June 2015. Phase 2 has been renamed and now consists of two discreet expansion projects, P5M and P10M.

The scope of the AGM 2017 DFS is as follows:

- P5M (Q1 2017 to Q3 2018):
  - Existing CIL processing facility at Obotan upgraded from the current 3.6 Mtpa to 5 Mtpa (Brownfields expansion)
  - Overland conveyor constructed from Esaase to Obotan
  - Power line from Obotan to Esaase constructed
  - Esaase deposit brought into production at 2 Mtpa ROM
  - Ore sources: Nkran, Satellite pits and Esaase, upon commissioning of the conveyor
  - On a standalone basis, LoM approximately 20 years to 2037

- P10M (Q3 2017 to Q3 2019):
  - Second CIL plant constructed, adjacent to the current plant at Obotan, with a capacity of 5 Mtpa, thereby doubling processing capacity to 10 Mtpa
  - Production from Esaase pit ramped up to 7 Mtpa ROM
  - Resettlement of the village of Tetrem, comprising 250 structures
  - Expansion of the footprint of the existing TSF
  - Ore sources: Esaase, Nkran, Satellite pits
  - LoM approximately 10 years to 2027

The purpose of the DFS is to demonstrate the economic and technical viability of the expansion which will allow the Asanko Gold Board to make a decision on the expansion and to bring the new Esaase pit into production.

2.3 Information Sources

Table 2-1 below summarises the key sources of information for the DFS. The Qualified Person who assumes responsibity for the section(s) of this Technical Report wherein such information is used remains rssponsible for it.

Table 2-1: Sources of Information

Area	Source
Geology and Mineral Resources	CJM / Asanko Gold / CSA
Mining	DRA
Mine Geotech	Mining One
Process and Engineering	DRA
Metallurgical Test Work	ALS Perth
Infrastructure	DRA
LIDAR Surveys	Southern Mapping
Overland Conveyor	DRA (Earthworks), ELB (Mechanical & Electrical)
Conveyor Simulation	Alex Lebedev
Geotechnical Data	Jones & Wagner
Tailings Storage Facility	Knight Piésold

Area	Source
Water Balance	Knight Piesold
Operating Costs	DRA (except G&A costs by Asanko Gold)
Capital Cost Estimate	DRA
Economic Assessment	Venmyn Deloitte
Project Development	DRA
Ownership and Permitting	Asanko Gold
Environment, Community	Asanko Gold & African Environmental Research and Consulting (AERC)
Closure Costing	Epoch

2.4 Site Visits by Qualified Persons

QP	Site Visit	Purpose
C. Muller	Yes	Core logging and resource definition
T. Obiri-Yeboah	Yes	Core and Pit inspection
G. Bezuidenhout	No	All laboratories used for testwork inspected
D.Morgan	Yes	Tailings dam real estate and construction
D. Heher	Yes	Overall site for the DFS
G. Njowa	No	Only did financial model
M. Titley	Yes	Core and pit inspection
P. Bentley	Yes	Core logging and resource definition

Note

Six out of the eight QPs have been to site. It was not deemed necessary for the other two to attend as G. Njowa was responsible for the economic analysis. In the case of G. Bezuidenhout on process, DRA designed and commissioned the Phase 1 plant and he was able to rely on the feedback of experienced colleagues.

2.5 See the QP certificates in Section 28 for the details of the site inspections. Site inspections focused mainly on Geology, Infrastructure and Permitting .

2.6 Acronyms, Abbreviations, Definitions, and Units of Measure

Unless otherwise indicated, all references to currency in this report refer to United States Dollars (US$). Frequently used acronyms and abbreviations are listed below.

Table 2-2: Acronyms and Abbreviations

Abbreviation	Meaning
AAGM	Akrokerri-Ashanti Gold Mines
AARL	Anglo American Research Laboratories
AAS	Atomic Absorption Spectrometry
ACSR	Aluminium Conductor Steel Reinforced
AERC	African Environmental Research and Consulting
Ag	Silver
AGF	Associated Gold Fields
AGM	Asanka Gold Mine
AHP	Analytical Hierarchy Process
AIG	Australian Institute of Geoscientists
ALS	Australian Laboratory Services Pty Ltd
Adansi	Adansi Gold Company Ghana
As	Arsenic
ASIC	All-In-Sustaining Costs
ASZ	Adubiaso Shear Zone
Au	Gold
AusIMM	Australasian Institute of Mining and Metallurgy
BFS	Bankable Feasibility Study

Abbreviation	Meaning
BGM	Bonte Gold Mining
BLC	Bonte Liquidation Committee
BLEG	Bulk Leach Extractable Gold
BNL	Bull Nose Lode
BoQ	Bill of Quantities
BSTP	Biological Sewerage Treatment Plant
CAPEX	Capital expenditure
CIL	Carbon in Leach
CJM	CJM Consulting
CRESCO	Cresco Project Finance (Pty) Ltd
Cu	Copper
CV	Central Vein
DC	Diamond Core
DDH	Diamond Drill Hole
DF	Design Feasibility
DFS	Definitive Feasibility Study
DPP	Definitive Project Plan for Phase 1 of the AGM, as filed on SEDAR in January 2015
DRA	DRA Projects (Pty) Ltd
EBL	Eastern Breccia Lode
EBSZ	Eastern Bounding Shear Zone
EGi	Environmental Chemistry International
EIS	Environmental Impact Statement
EL	East Lode
EL-N	East Lode North

Abbreviation	Meaning
EPA	Environmental Protection Agency
ESIA	Environmental and Social Impact Assessment
EV	Eastern Vein
FEED	Front End Engineering Design
FEL	Front End Loader
FS	Feasibility Study
FW	Foot Wall
G	Grams
G&A	General & Administration
g/t	Grams per metric tonne
GRES	GR Engineering Services
GV	Galamsey Vein
H&S	Hellman and Schofield
Ha	Hectare
Hg	Mercury
HMM	HMM Consultancy
HW	Hanging Wall
ICMC	International Cyanide Management Code
IK	Indicator Kriging
IMR	In-situ Mineral Resource
IP	Induced Potential
JORC	Australian Institute of Geoscientists and the Mineral Council of Australia
KE	Kriging efficiency
Keegan	Keegan Resources Inc, Asanko Gold's former name until the name change in February 2013

Abbreviation	Meaning
Kg	Kilogram
KIR	Kiwi International Resources
KNA	Kriging neighbourhood analysis
Km	Kilometres
Koz	Kilo ounce (troy)
KP	Knight Piésold (Pty) Limited
KRGL	Keegan Resources Ghana Limited
l	Litres
LEI	Layered Earth Inversions
LG	Lerchs-Grossman
Li's	Legislative Instruments
LoM	Life of Mine
LVD	Land Valuation Division
m	Meters
Ma	Million years
mAMSL	Meters above mean sea level
MCC's	Motor Control Centres
MICA	Mineral Industry Consultants Association
MOP	Mining Optimisation Process
MRE	Mineral Resource Estimate
MRev	Mineral Reserve Estimate
MSA	Mining Services Area
Mtpa	Million tonnes per annum
NAFHAS	Local specialist facilities management company

Abbreviation	Meaning
NPV	Net Present Value
NS	Nitro Shear
OLC	Overland Conveyor
OK	Ordinary Kriging
OPD	Out Patient Department
OPEX	Operating cost
Oz	Ounce
Pb	Lead
PFC	Power Factor Correction
PFS	Pre-feasibility Study
PMI	PMI Gold Corporation
PSA	Pressure Swing Absorption
QA/QC	Quality Assurance / Quality Control
QP	Qualified Persons
QSP	Quartz Sericite Pyrite
RAP	Resettlement Action Plan
RC	Reverse Circulation
Resolute	Resolute Mining Ltd
RF	Revenue Factor
RFQ	Request for Quote
RK	Red Kite Mine Finance
RoM	Run of Mine
RQD	Rock Quality Designation
SABC	SAG and Ball Milling Circuit
SAG	Semi Autogenous Grinding

Abbreviation	Meaning
Sb	Antimony
SIB	Stay-in-business Capital
SMBS	Sodium Meta Bi Sulphite
SMD	Stirred Media Detritor
SMU	Selected Mining Unit
SRK	Steffen, Robertson and Kirsten
STP	Sewage Treatment Plants
T	Degrees relative to true north
t	Tonnes
Tpa	Tonnes per annum
TSF	Tailings Storage Facility
TSS	Total Suspended Solids
UCS	Unconfined Compressive Strength
US$	United States Dollars
VLF	Very Low Frequency
VRA	Volta River Authority
VTEM	Versatile Time-Domain Electromagnetic Surveying
WAD	Weak Acid Dissociable
WAXI	West Africa Exploration Initative
WBSZ	Western Bounding Shear Zone
WRD	Waste Rock Dump
WRDF	Waste Rock Dump Facility
Zn	Zinc

3 RELIANCE ON OTHER EXPERTS

This report has relied on the report entitled; "Root to Title Asanko Gold Mine" dated July 2014 and prepared by Kimathi and Partners Legal and Advisory Services, Accra Ghana (the "Tenement Report") in connection with mineral title matters.

For the Ghanaian corporate income tax rate of 35% used in the economic analysis the authors have relied on publicly accessible Ghanaian legal and tax information compiled by International Comparative Legal Guides. ICGL compiles its information form named local legal practitioners and makes this information available for purchase.

https://iclg.com/practice-areas/mining-laws-and-regulations/ghana#chaptercontent3

4 PROPERTY DESCRIPTION AND LOCATION

Ghana is located in West Africa, sharing boundaries with Togo to the east, Cote d'Ivoire to the west, Burkina Faso to the north and the Gulf of Guinea to the south.

Agriculture accounts for nearly 25% of GDP and employs more than half of the workforce, mainly small landholders. Ghana is one of the leading exporters of cocoa in the world, with lesser cash crops including palm oil, rubber, coffee and coconuts. Cattle are farmed in northern Ghana. The services sector accounts for about 50% of GDP. Gold and cocoa exports and individual remittances are the major sources of foreign exchange. The expansion of Ghana’s recently established oil industry has boosted economic growth, although the recent oil price crash reduced by half Ghana’s 2015 oil revenue. Production at Jubilee, Ghana's offshore oilfield, began in mid-December 2010 and currently produces roughly 110,000 barrels per day. The country’s first gas processing plant at Atubao is also producing natural gas from the Jubilee field, providing power to several of Ghana’s thermal power plants.

Gold represents Ghana’s major export commodity, followed by cocoa and timber products. Ghana is the world’s tenth and Africa’s second largest producer of gold. Manganese, bauxite and diamonds are also mined. Tourism is growing rapidly.

Ghana has an estimated population of 26.9 million (2015 est.) and covers an area of approximately 238,500 km^2. Ghana has a large variety of African tribal, or sub-ethnic units. The main groups include the Akan (45%), Moshi-Dagomba (15%), Ewe (12%) and Ga (7%) people. English is the official language, a legacy of British colonial rule. Twi is the most widely spoken local African language. Birth rates are high compared with world averages, and the annual rate of population growth (2.2%) is one of the highest in the world, although about average for sub-Saharan Africa. The majority of the population are Christian (69%) whilst the northern ethnic groups are largely Muslim (16%) and indigenous beliefs (21%) are also practiced throughout the country.

In 1957 Ghana, (formerly known as the Gold Coast), became the first country in sub-Saharan Africa to gain independence. After leading the country for nine years, the nation's founding president, Kwame Nkrumah was overthrown in a coup d'etat in 1966. After Kwame Nkrumah, Ghana was ruled by a series of military despots with intermittent experiments with democratic rule, most of which were curtailed by military takeovers. Ghana has been a stable democracy since 1992, which marked the drafting of a new constitution, and it is this that has gained Ghana recognition as a leading democracy in Africa. Ghana is governed under a multi-party democratic system, with elected presidents allowed to hold power for a maximum of two terms of four years.

Following the death of President John Atta Mills in July 2012, Vice President John Dramani Mahama became interim head of state. He was elected President in December 2012. The next Presidential election was on 7th December 2016 and Nana Akufo-Addo of the New Patriotic Party was elected President.

Most of the major international airlines fly into and from the international airport in Ghana’s capital city, Accra. Domestic air travel is thriving and the country has a vibrant telecommunications sector, with six cellular phone operators and several internet service providers. Ghana predominantly has a tropical climate and consists mostly of low savannah regions with a central, hilled forest belt. Ghana's one dominant geographic feature is the Volta River, upon which the Akosombo Dam was built in 1964. The damming of the Volta created the enormous Lake Volta, which occupies a sizeable portion of Ghana's south-eastern territory.

Ghana's economy was strengthened by a quarter century of relatively sound management, a competitive business environment, and sustained reductions in poverty levels, but in recent years has suffered the consequences of a loose fiscal policy, high budget and current account deficits and a depreciating currency. Ghana has a market based economy with relatively few policy barriers to trade and investment in comparison with other countries in the region. Ghana has substantial natural resources and a much higher per capita output than many other countries in West Africa. Nevertheless, it remains dependent on international financial and technical assistance.

Currently one of the biggest single economic issue facing Ghana is the lack of consistent electricity. While the Mahama administration took steps to improve the situation, little progress has been made. Ghana signed a US$920 million extended credit facility with the IMF in April 2015 to help it address its growing economic crisis. The IMF fiscal targets will require Ghana to reduce the fiscal deficit by cutting subsidies, decreasing the bloated public sector wage bill, strengthening revenue administration and increasing revenues. The December 2016 elections have yielded a change in Government, which will hopefully positively impact on the general power facilitation and administration in Ghana.

4.1 Project Location

The AGM tenements are located in the Amansie West District, of the Ashanti Region of Ghana, approximately 250 km north west of the capital Accra and some 50 km to 80 km south west of the regional capital Kumasi (Figure 4-1).

The AGM areas are accessed from the town of Obuasi, northward towards Kumasi on the Kumasi-Dunkwa highway to the Anwian-Kwanta junction. The concessions cover an area of approximately 370 km^2 between latitudes 6º 19’40” N and 6º 28’ 40” N; and longitudes 2º 00’ 55” W and 1º 55’ 00” W.

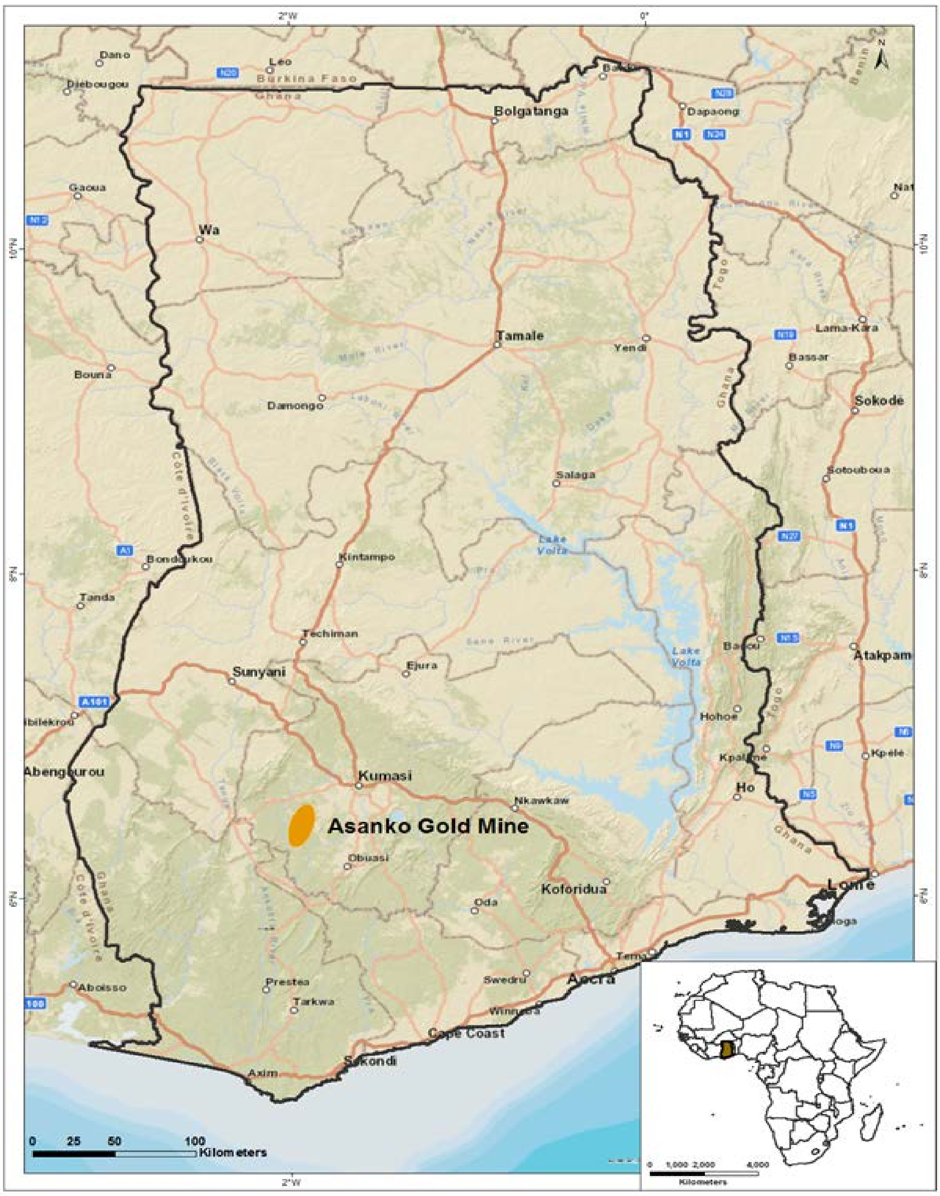


Figure 4-1: The Location of the Asanko Gold Mine in Ghana, West Africa (source: CJM 2014 Technical Report)

The AGM is being developed in two phases. Phase 1 was the construction of a 3 Mtpa CIL processing facility, associated mine infrastructure and the development of the Nkran pit as the primary ore source.

The second phase of expansion comprises P5M, which is the upgrade of the existing Phase 1 processing facility to 5 Mtpa and the development of the Esaase deposit, which is located 28 km north of the project (Figure 4-2) with an overland conveyor, as well as the optimal integration of other satellite Oxide and Fresh ore sources, and P10M, which consists of an additional CIL processing facility to double processing capacity to 10 Mtpa.

The AGM is located 70 to 80 km from Kumasi and is accessed by travelling 35 km south to Anwiankwanta Junction, and then west into the project area on surfaced and un-surfaced all weather roads.

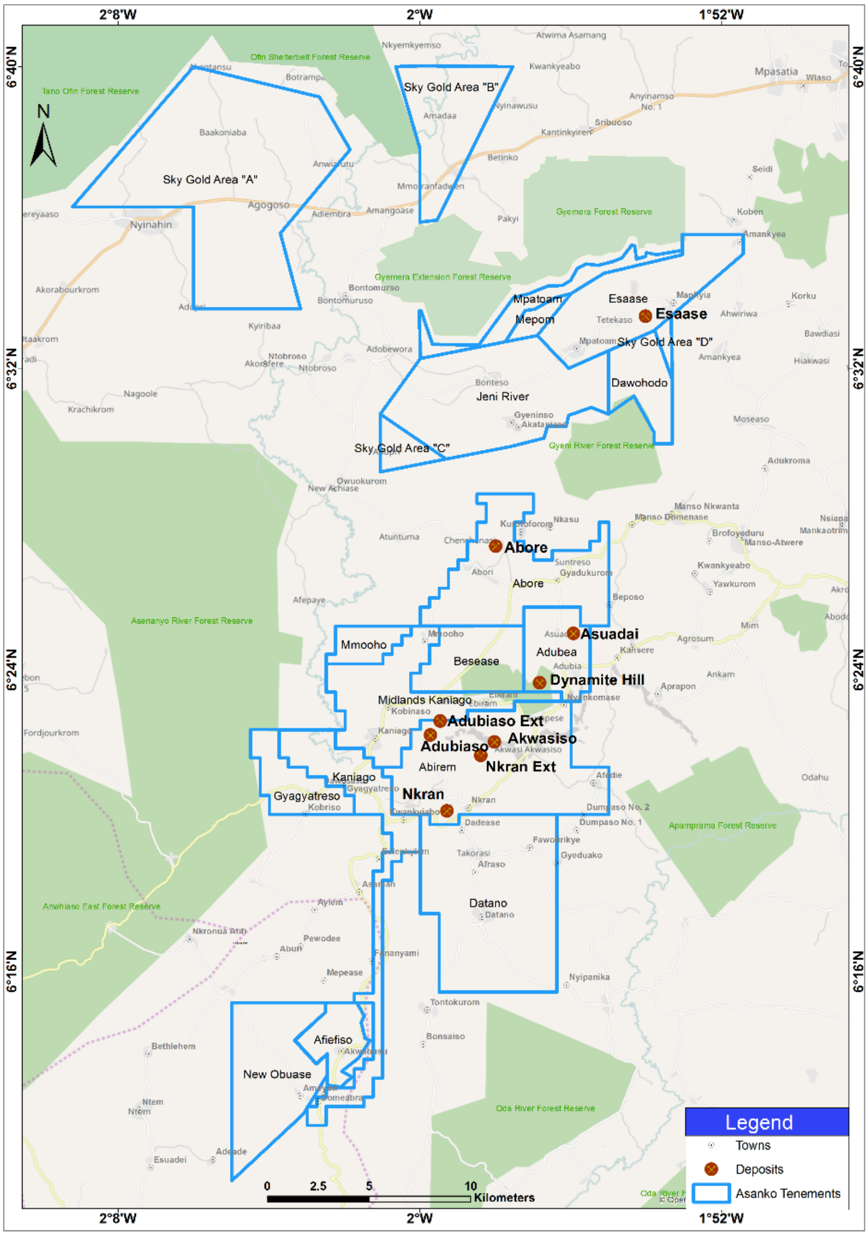


Figure 4-2: The Location of the Various Tenements making up the Asanko Gold Mine (source: Asanko Gold)

4.2 Status of Surface and Mineral Title

The legal status of the mineral properties in Ghana in which Asanko Gold has an interest have been verified by Asanko Gold staff, but not by an independent legal entity. As at 31st December 2016 all mineral tenements were in good standing with the Government of Ghana. Furthermore, it has been confirmed that the properties are lawfully accessible for evaluation and also mineral production. During 2015 and 2016 Asanko Gold restructured its Ghanaian subsidiaries, which resulted in a number of transfers of mineral rights. The areas of the respective mining leases and prospecting licences with respective company owners are tabulated in (Table 4-1).

Table 4-1: Asanko Gold Mine Mining Lease and Prospecting Concession Areas

Licence				Licence Owner
Name	Mincom Reference No	Km^2	Type	
Abore	PL 6/303	28.47	Mining Lease	Asanko Gold Ghana 100%
Abirem	PL 6/303	47.13	Mining Lease	Asanko Gold Ghana 100%
Adubea	PL 6/310	13.38	Mining Lease	Asanko Gold Ghana 100%
Datano	PL 6/32-Vol 3	53.78	Mining Lease	Asanko Gold Ghana 100%
Esaase	PL 6/8 Vol 8	27.03	Mining Lease	Asanko Gold Ghana 100%
Jeni River	RL 6/21	43.41	Mining Lease	Asanko Gold Ghana 100%
Kaniago	PL 6/307	25.27	Prospecting	Asanko Gold Ghana 100%
New Obuase-Afiefso	PL 3/84	122.20	Prospecting	Asanko Gold Ghana 100%
Switchback/Adansi	PL 6/32	10.83	Prospecting	Asanko Gold Ghana 100%
Mepom	Pl 6/245	2.37	Prospecting	Asanko Gold Ghana 100%
Dawohoda	PL 6/43	10.36	Prospecting	Asanko Gold Ghana 100%
Asumura	PL 7/107 Vol 2	82.11	Prospecting	Asanko Gold Ghana 100%
Fosukrom	PL 2/413R/Vol 2	62.16	Prospecting	Asanko Gold Ghana 100%
Sky Gold	RL 6/86	91.50	Reconnaissance	Asanko Gold Ghana 100%
Pomakrom	Under application	102.69	Application	Asanko Gold Ghana 100%
Kaniago	PL 6/289	25.50	Prospecting	Asanko Gold Ghana 100%
Besease	Under application	15.55	Prospecting	Asanko Gold Ghana 100%
Mimooha	PL 6/352	5.76	Prospecting	Asanko Gold Ghana 100%

The lease / concession boundaries have surveyed by GPS and are correlated with the latitude and longitude via degree co-ordinates as per the Ghanian Mining Cadastre (July 2016).

4.3 Location of the Property

Asanko Gold holds 6 mining leases, 11 prospecting licences, 1 reconnaissance licence and 1 prospecting licence in application, which collectively make up the AGM and span over 40 km strike length of the Asankrangwa Gold Belt. The AGM is made up of the Obotan Project area (Figure 4-2) which consists of a series of contiguous concessions, and the Esaase Project area, (Figure 4-2). These concessions cover an area in total of approximately 670 km^2, between latitudes 6° 11' 54.985" N and 6° 35' 33.074" N, and longitudes 2° 4' 59.195" W and 1° 51' 25.040" W.

The mining law divides the various licences that can be granted for a mineral right into three sequential categories, reconnaissance licence, prospecting licence and a mining lease, defined under the Minerals and Mining Act 2006 (Act 703), as follows:

4.4 Reconnaissance Licence (Sections 31-33)

A reconnaissance licence entitles the holder to search for specified minerals by geochemical, geophysical and geological means. It does not generally permit drilling, excavation, or other physical activities on the land, except where such activity is specifically permitted by the licence. It is normally granted for 12 months and may be renewed for a period not exceeding 12 months, if it is in the public interest. The area extent is negotiable, related to the proposed reconnaissance programme.

4.5 Prospecting Licence (Sections 34-38)

A prospecting licence entitles the holder to search for the stipulated minerals and to determine their extent and economic value. This licence is granted initially for a period of up to three years covering a maximum area of 150 km^2. This may be renewed for an additional period of two years, but with a 50% reduction, or "shedding off" in the size of the licence area if requested. A prospecting licence will only be granted if the applicant shows adequate financial resources, technical competence and experience and shows an adequate prospecting programme. It enables the holder to carry out drilling, excavation and other physical activities on the ground.

Mining Lease (Sections 39-46) When the holder of a prospecting licence establishes that the mineral to which the licence relates is present in commercial quantities, notice of this must be given to the Minister of Lands, Forestry and Mines and if the holder wishes to proceed towards mining, an application for a mining lease must be made to the Minister within three months of the date of the notice.

4.6 Financial Agreements

All concessions carry a 10% free carried interest in favour of the Ghanaian government. The government interest only comes into effect at the production stage. The mining leases are also subject to a 5% royalty payable to the Government of Ghana. In addition, the Adubea mining concession is subject to an additional 0.5% royalty to the original concession owner. The Esaase mining lease is also subject to an additional 0.5% royalty to the Bonte Liquidation Committee ("BLC").

4.7 Environmental Liabilities

On advice from Asanko Gold, under the current ownership arrangement and status of holdings, there is no environmental liability held over Asanko Gold for any of the AGM concessions, with the exception of project works to date and a portion of the Jeni Concession over which the overland conveyor passes from Esaase to the processing facility.

4.8 Permitting Status

The Esaase, Abore, Abirem, and Adubea Mining Leases contain all of the resources defined to date. All other concessions held by Asanko Gold in the area contain exploration potential defined to date and in some instances locations for infrastructure. The EPA grants permits on a perennial basis to conduct exploration. On advice from Asanko, with respect to the project areas, all permitting within the afore-mentioned governmental permitting structure is up to date and accounted for.

Prior to approval to commence operations being granted by the Ghanaian government, the Company completed an ESIA and EIS for Esaase which have been submitted to the EPA for final approvals, for which record of payment has been received and approvals imminent.

The environmental and social studies have been expanded to include the integration of Phase 1, P5M and P10M as part of the AGM 2017 Expansion DFS. In particular, the servitude for the overland ore conveyor has been designed and permitted as part of this DFS.

In November 2012, the Company formally received mining leases on the Abore-Abirem and Adubea prospecting licences. The mining leases have been granted for different periods, with the Abore lease expiring on 1st November 2017, the Abirem lease expiring on 27th March 2026 and the Adubea lease due to expire on 1st November 2018. All leases are renewable under the terms of the Minerals and Mining Act, 2006. In conjunction with the formal issue of the mining leases, the Company also received a key water discharge permit which has allowed dewatering of the Nkran and Adubiaso pits. Phase 1 is now fully permitted and operational, and all the P5M and P10M metallurgical plant expansions and related infrastructural permiting (TSF expansion and overland conveyor) have been permitted.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, Vegetation and Climate

The AGM lies in the Amansie West District of the western region of Ghana. The AGM is located in hilly terrain dissected by broad, flat drainages that typically form swamps in the wet season between May and late October. Hill tops are generally at very similar elevations, reflecting the elevation of a previous erosional peneplane that is now extensively eroded. Maximum elevations are around 80 metres above sea level, but the areas impacted by the project generally lie at less than 50 metres elevation. Despite the subdued topography, hill slopes are typically steep. Ecologically the AGM is situated in the wet evergreen forest zone.

The concession areas are covered by a series of low, gently undulating hills, which rarely exceed 680 masl in elevation. The soils of the area fall within the Bekwai and Nzema Oda classification. The soils of the Bekwai series are found on the summits and some upper slope sites of the hills of the area. They are generally deep to very deep (over 20 cm), humus, well drained, red in colour, loam to clay loam, gravelly and concretionary, with well developed sub angular blocky structure and clay cutans within sub-soils. The soils are acidic throughout the profile. A typical profile of the Bekwai series consists of a thick topsoil of up to 19 cm dark brown to dusky red, humus stained, loam to clay loam, with weak fine granular structure with moderate acidic reaction. The soils of the Nzema Oda series are heavy textured soils developed on alluvial deposits along streams of the area. The soils are poorly drained and are subjected to flooding during the wet seasons and are greyish in colour with prominent yellowish orange mottles. The soils are deep, acidic with clay loam to clay textures, but are structureless in the sub-soils. Limited quartz gravels and stones may be encountered at the base of the profile. Evaluation of the soils for agricultural production revealed that the upland soils are suitable for the production of a number of climatically suited tree and food crops, as well as cereals, legumes and vegetables. Tree crops such as cocoa, coffee, citrus, oil palm, avocado pear and mangoes do well on these soils. Cassava, yams, plantain, banana and maize are successfully grown on these soils.

In general, the concession areas have been largely transformed, having experienced extensive degradation in recent years. The main land uses include secondary forest, subsistence and cash crop farming, and artisanal gold mining.

There are several local villages near the AGM site. The closest to the plant site is the Manso Nkran village, while the villages of Tetrem and Esaase are in close proximity to the Esaase deposit.

The annual rainfall is in the range of 1,500 mm to 2,000 mm and temperatures range from 22°C to 36°C. The major rainy season takes place from April to July followed by a minor rainy season from September to October. The AGM has operated without cessation, or delay throughout both of the rainy seasons.

5.2 Access

The Obotan project area is accessed by road from the city of Kumasi, south towards Obuasi on the Kumasi–Dunkwa highway to the Anwian–Kwanta junction then approximately 20 km west from this junction through Poano and Antoakurom on a tarred road onto a laterite road for approximately 30 km through Manso Akropon, Manso Atwere, Manso Nkwanta, Suntreso, Gyadukurom to Abore. At Abore, the road branches northwest to Akuntam, then northeast to Nkasu. At Gyadukurum, the road branches off south to Asuadai, Dynamite Hill, and Adubea. At Adubea, the road continues south to Kumpese and westward to Abirem, to Besease, then north to Mmooho. At Kumpese, the road branches south to Akwasiso, south west to Koninase and to Nkran and Adubiaso. Areas of interest within the concession are reached via a combination of secondary roads, four wheel drive tracks, logging roads, and farming / hunting footpaths.

The Esaase property is accessed by road from the city of Kumasi by taking the tarred Sunyani-Kumasi road west for 10 km to the Bibiani Junction at Abuakwa and then southwest for 10 km along the tarred Bibiani-Kumasi highway to the village of Wiaso. A secondary tarred road is taken 8 km south from Wiaso to the village of Amankyea. Secondary gravel roads can be taken for a further 11 km via the villages of Ahewerwa and Tetrem.

The Esaase deposit itself is accessed by a series of secondary roads constructed either by the former Bonte Gold Mines or by Asanko Gold.

5.3 Existing Infrastructure and Services

Current site infrastructure with respect to the Obotan concessions consists of an office complex, metallurgical facility, TSF, senior and junior accommodation and mess facilities, workshops, power distribution facility, a new core storage facility, potable and operational water supplies, a waste rock dump facility, an upgraded dry weather air strip, a haul road from Abore pit to Nkran pit, all west of the village of Nkran:

- Local facilities of importance to exploration and mining include towns, villages, roads, trails, power lines, rivers and rail roads
- The principal towns within the area are Abore and Adubea
- Akwasiso and Nkran are the principal towns within the Phase 1 mining area
- Surrounding villages are connected to the national electrical grid
- There is grid power to the Nkran area, the former processing plant and town site
- Most areas are adequately serviced by several cellular telephone suppliers
- The principal towns have potable water and health posts which cover local needs
- Ghana has a good base of skilled mining and exploration personnel
- The following resources are available for the Esaase concession area:

- The Esaase exploration camp and surrounding villages are connected to the national electrical grid
- The project is in an area well serviced by the Ghana national power grid with at least two alternate points of supply within a 50 km radius of the open pit mining site
- Mobile phone communication is accessible in most parts of the concession
- Hospitals and most government offices are available in Kumasi
- Food and general supplies are also purchased in Kumasi
- Ghana has a mature mining industry that has resulted in the local availability of both skilled and unskilled personnel

5.4 Sufficiency of Surface Rights

The AGM concessions are owned 100% by Asanko Gold Ghana (the merged entity of KRGL and Adansi), a fully owned operating subsidiary of Asanko Gold. The Government of Ghana retains the right to take a 10% free carried interest in the AGM under Section 8 of the Ghanaian Mining Act.

When Asanko Gold acquired the Esaase concessions, there was a mining lease in place from the historical alluvial mining operations. The Minister of Lands and Natural Resources granted the other Mining Leases for the Obotan Project to PMI in November 2012, prior to the acquisition of PMI by Asanko Gold in early 2014.

In order to develop the Esaase concession area a requirement was that Asanko obtained the community's consent for the overall project, as well as for the proposed resettlement as preconditions for granting of the Environmental Permit by the EPA. This, among others, is what necessitated the Public Hearing prior to issuance of the Environmental Permit. The Minerals Commission, in approving the Esaase Mining Area (or Moratorium), also approved the resettlement activities as per section A (ii) of their Approval Letter. The receipt of the Environmental Permit, along with the approval of the Esaase Moratorium by the EPA and Minerals Commission, respectively, signified government approval of the resettlement associated with Phase 2.

As part of the development process of the Esaase concessions in April 13, 2017 the Ghana Minerals Commission approved the Esaase active mining area including the conveyor route and the site for the resettlement which initiated active engagements on the resettlement process. In April, the Minerals Commission declared moratorium at Esaase area with all the key stakeholders including the District Assembly and the Traditional Authorities present. In May 17, 2017, through a due process, Tetrem and Esaase/Manhyia communities elected representatives from various stakeholder groups in the communities, in addition to nominated traditional authorities to form the resettlement negotiation committee (RNC) to deliberate on the modalities for the resettlement. To ensure properly constituted committee and outcomes, a sub chief of Tetrem community, who is also an Appeal Court Judge in Ghana, also assisted the RNC to nominate

Meetings are held on monthly basis to deliberate on various issues i.e.; room size, selection of site, infrastructure facilities, roofing types, modalities for relocation in lieu of resettlement, water systems, other social support for the vulnerable etc. and these will culminate to a signed agreement by 2018. The deliberation is ongoing and is expected to complete by the third quarter of 2018. Similarly, a duly constituted crop rate review committee reconvened to negotiate for new crop rates to pay for compensation for farms and properties that would be affected during construction. A signed crop rates agreement is expected before the end of December 2017. The estimated cost associated with the resettlement action plan (RAP) has been included in both the initial capital cost estimate, as well as the estimate of ongoing capital cost. These estimates were based on Asanko Gold's recent experience with the resettlement of villages associated with Phase 1. In light of the collective nature of the negotiations with representative of Tetrem, the mature legal framework within Ghana, the advanced stage of the resettlement process under Ghanaian law, the receipt of the Environmental Permit, approval of the Esaase Moratorium and the Company's recent experiences with resettlement efforts and associated costs during Phase 1, the resettlement of the Tetrem community is not considered to pose a material risk to the potential development of mineral resources and reserves.

The formal grant of these three Mining Leases, renewable under the terms of the Minerals and Mining Act, 2006, followed the favorable recommendation by the Minerals Commission of Ghana in September 2012.

The Mining Leases cover a total area of 167 sq. km, encompassing the two main deposits, Nkran and Esaase and the smaller satellite deposits, Abore, Adubiaso, Dynamite Hill, Akwasiso, Asuadai, Adubiaso Extension and Nkran Extension.

In conjunction with the formal issue of the Mining Leases, PMI also received a key water discharge permit for the AGM Phase 1 dewatering of the Nkran and Adubiaso pits.

6 HISTORY

6.1 Introduction

The AGM Project Area encompasses over 70 km of strike from Esaase in the north to the Fromenda targets ~20 km south of Nkran pit (Figure 6-1). Formerly the Project area was owned by Resolute's Amansie Mining operations, commonly known as the Obotan Mine, but now referred to as the AGM.

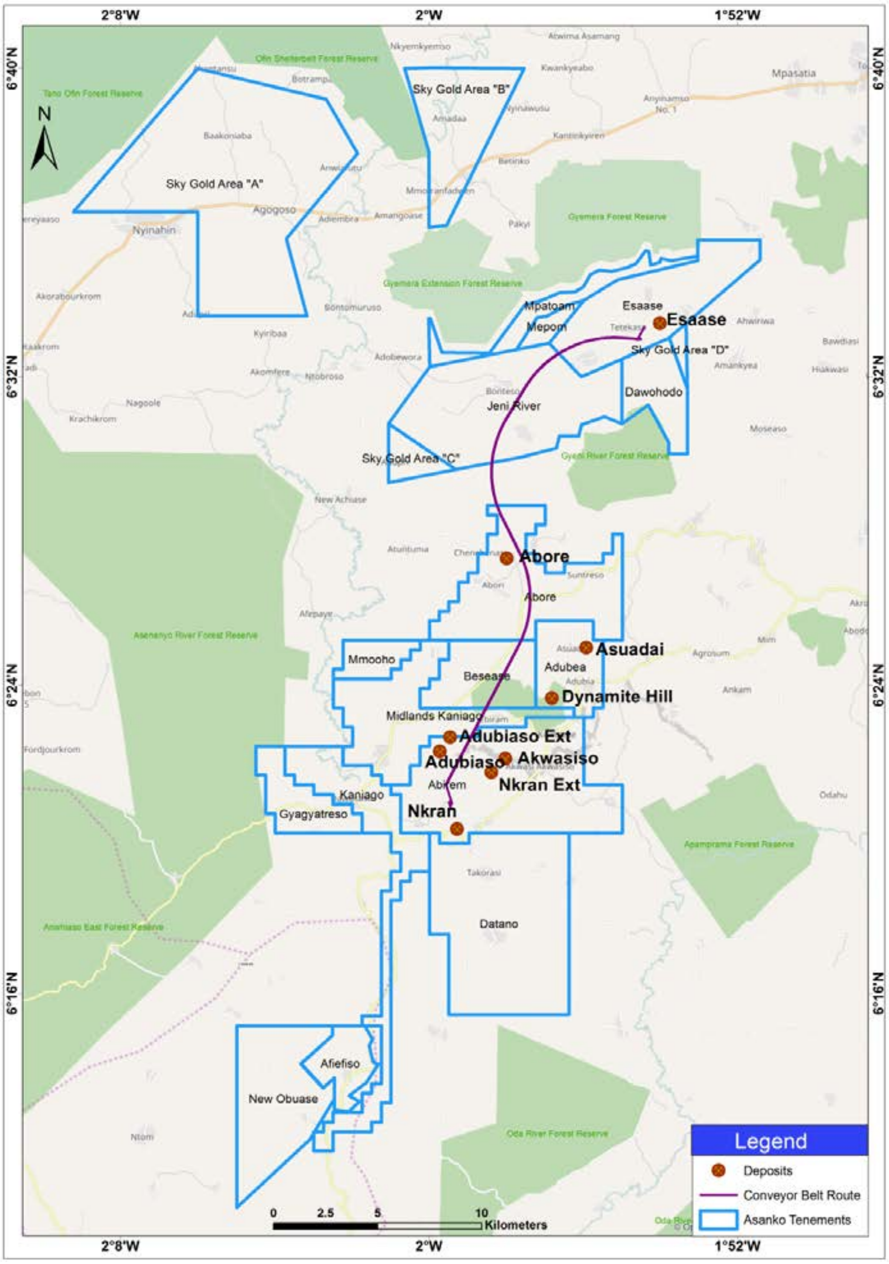


Figure 6-1: Extent of AGM permit area & locality of gold deposits (source: Asanko Gold)

This summary of previous exploration and mining activities is largely reliant on those recorded in the Minerals Commission book "Gold deposits of Ghana" (Griffis, R.J., Gold Deposits of Ghana, 2002, pp 154 to 156). Historically the main producing mine in the area was the Obotan Mine, now referred to in this document as the Nkran pit, which closed in 2001 having produced 590,743 oz Au at an average grade of 2.35 g/t Au. Asanko Gold re-commenced mining operations at Nkran in February 2015.

The Esaase area was previously mined by an alluvial mining company called Bonte Resources which went into liquidation, returning the concessions to the Government of Ghana. The concessions were obtained by Keegan and were subject to a systematic exploration and drill evaluation programme which discovered the Esaase deposit. Keegan was subsequently renamed Asanko Gold and then acquired PMI to establish the current Asanko Gold asset base, including the AGM.

6.2 Previous Exploration

A number of Companies have effected exploration and mining activities in the AGM area, mostly from the mid 1990's to present. There are no details of work programmes, or expenditures incurred during these exploration and mining projects, and the QP absolves from the any responsibilities for the following, largely sourced from Griffis 2002. Figure 6-2 below shows the extent of the various exploration surveys up to 2014.

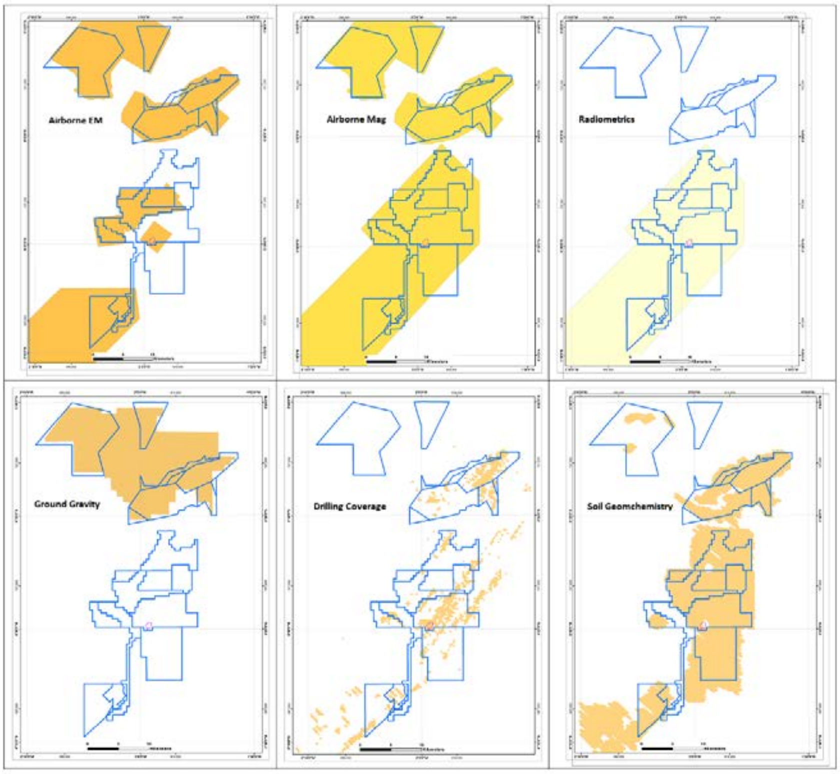


Figure 6-2: Extent of historic geophysical, drilling, and ground Exploration surveys on the AGM properties up to 2014. (source: Asanko Gold 2015)

6.2.1 Obotan Minerals

In the late 1980’s, the Nkran prospect attracted the attention of Dr Alex Barko, a consultant, who recommended the area to one of his local client groups. Obotan Minerals subsequently applied for and received a prospecting concession covering about 106 km² over the general area.

Only minor prospecting was carried out in the early stages. Some attention was paid to the alluvial gold potential due to the extensive gold in the nearby Offin River, which was previously mined by the State Gold Mining Company in the 1960’s and was later held by Dunkwa Continental Goldfields, as well as the alluvial gold project being developed at the time, located further north within the Bonte area.

6.2.2 AGF/ KIR

In the early 1990’s, the Project concessions were examined by American consultant, Al Perry who worked on behalf of two related Australian junior companies: Associated Gold Fields (“AGF”) and Kiwi International Resources (“KIR”). Perry negotiated an option on the concession and proceeded to focus on the known prospects at Nkran (formerly known as Jabokassie) and to carry out a regional soil geochemical survey that identified numerous anomalies around Nkran (Figure 6-3).

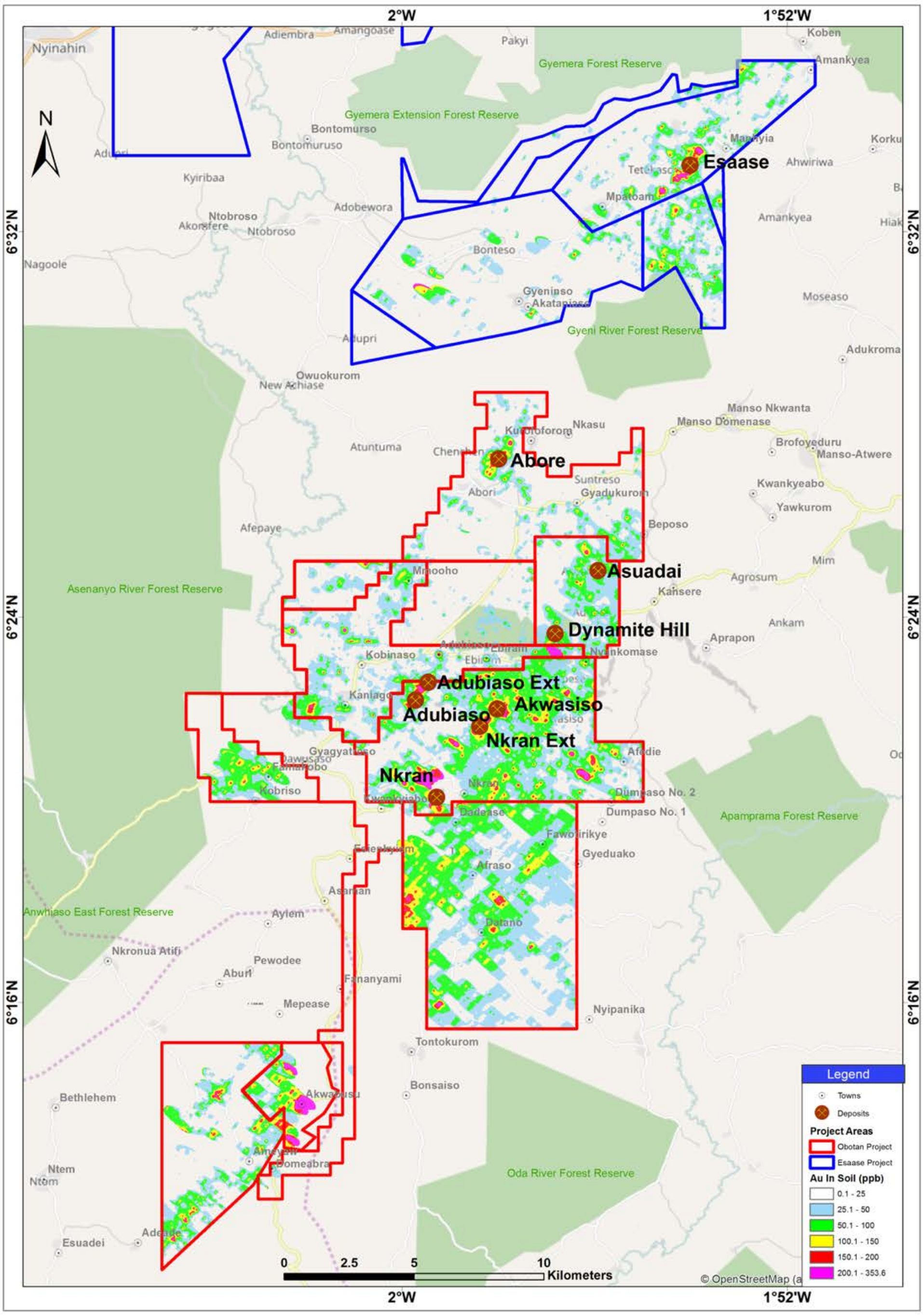


Figure 6-3: Distribution of Soil Sampling Grids and Gold Anomalies across the AGM properties. (source: Asanko Gold 2016)

AGF and KIR quickly assessed the open pit potential of the Nkran prospect and carried out an early stage RC drilling programme that returned very encouraging results over a wide zone of bedrock mineralisation, which extended along strike for about 600m.

By early 1995, resource estimates for Nkran, (measured, indicated and inferred) were reported as 4.8 Mt at approximately 3.7 g/t Au for an *in situ* gold content of nearly 600,000 oz Au. A feasibility study was completed and a mining lease was granted in late 1995.

This successful exploration work was noted by the emerging Australian gold producer, Resolute Samantha (now Resolute).

6.2.3 Resolute

A deal was completed by May 1996 whereby the combined interests of KIR and AGF were bought out by Resolute, who immediately reviewed and expanded the scope of the project with a ground magnetic survey (Figure 6-4). This was followed up by further RC diamond drilling to increase resources to a depth of 150m at Nkran and to further assess the known mineralisation at the nearby Adubiaso prospect.

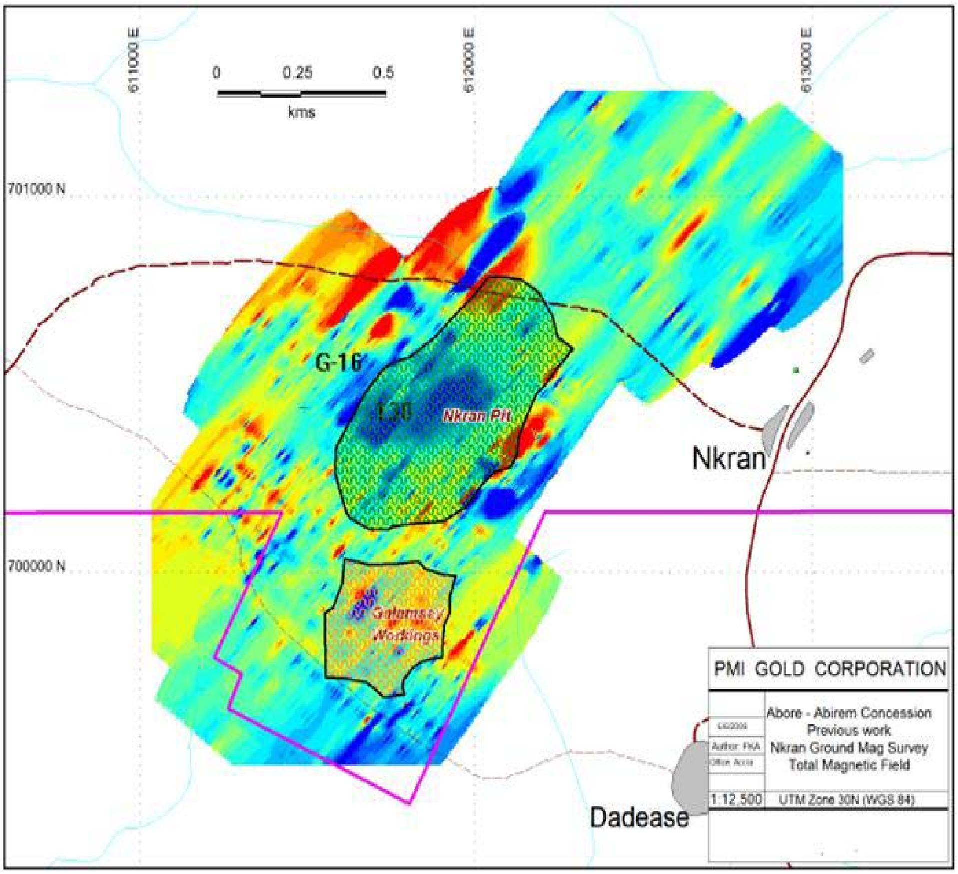


Figure 6-4: Abirem Nkran pit – An Example of a Ground Magnetic Survey by Resolute. (source: Spiers 2011)

A revised mine development plan was completed by the end of July 1996 and a decision to produce at a rate of 1.4 Mtpa was made. Initial mining was started early in 1997 and the first gold was poured by May 1997. During the late 1990's, the Nkran plant started to process oxide ores from the Adubiaso gold deposit, located about 5 km north northwest of Nkran.

Throughout the 1988-1998 period, significant drilling was undertaken by the various companies previously mentioned, over the Nkran project area. (SRK, 2012; see Section 10)

6.2.4 Leo Shield

The Abore area was covered in a prospecting concession granted to the Oda River Gold Mining Company, which was headed up by Nana Asibey-Mensah of Kumasi. Local villagers held a small scale mining licence (Asuadai prospect) at Edubia and Adubiaso.

In the mid-1990's, Mutual Resources of Vancouver, Canada, in partnership with Leo Shield Exploration of Perth, Australia, completed a joint venture with the Oda River group and commenced a regional exploration programme on the concession (covering approximately 73 km^2).

Prospecting north of Abore revealed extensive old, as well as very recent artisanal mining in alluvial areas, as well as many old Ashanti pits in the saprolite along a low hill immediately adjacent to the alluvial workings.

Soil geochemistry revealed a strong north-northeast trending gold anomaly over the area of artisanal mining (bedrock areas); the anomaly was several hundred metres wide and traceable along strike for about 3 km, well beyond the area of old workings (Figure 6-5).

Extensive trenching in the area confirmed continuous bedrock mineralisation over a distance of at least 1,000m with widths varying between 50m to 100m. The mineralisation consisted of a broad quartz stockwork system hosted mainly by a north northeast trending, intermediate granitoid intrusion. The early artisanal pitting focused mainly on narrow quartz veins associated with the stock work system.

Extensive drilling in the area (mainly RC, but with some diamond drilling as well) outlined sizeable resources, (now known as the Abore, Adubiaso, Asuadai and Akwasiso prospects).

In the late 1990's, Mutual's interest in the project was bought out by Leo Shield (later Shield Resources) (SRK, 2012).

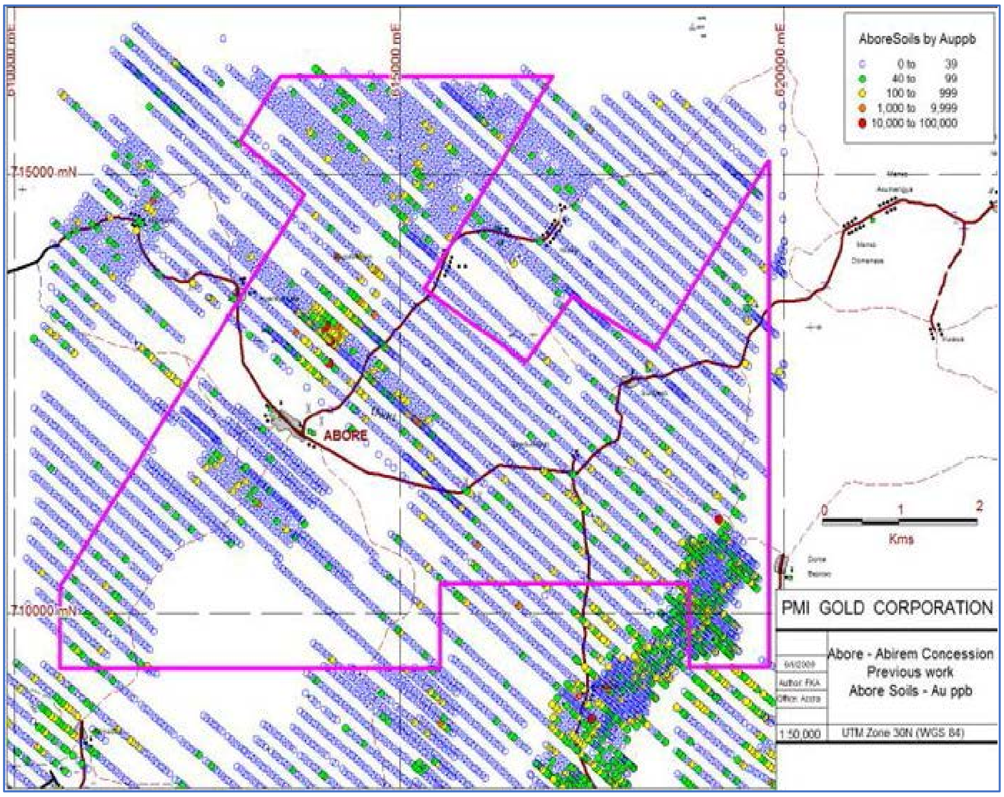


Figure 6-5: Abore – Regional Soil Sampling by Leo Shield. (source: PMI, Asanko Gold 2014 Technical report)

Throughout the 1996 to 1998 period, significant drilling was conducted by various companies, but most predominantly by Leo Shield over the Abore project area, in line with exploration and development undertaken over the Nkran project area. Drill testing was dominated by RC drilling, accounting for 85% of the total current drilling dataset.

6.3 Midlands Minerals Corporation Ltd

Asanko Gold acquired the Midlands Kaniago tenements in January 2015 from Midlands Minerals Corporation Ltd ("Midlands").

From 2006 through 2010 Midlands conducted stream sediment sampling (116 samples) followed by three phases of soil geochemistry (4,917 samples). In April 2010 they conducted an airborne VTEM survey totalling 560 line kilometres.

In 2011 and 2012 Midlands completed 162m of trenching on gold in soil anomalies and 13,509m of RC and DC drilling. No further work was completed by Midlands.

6.4 Work Conducted By PMI

6.4.1 Ground Geophysical Surveys – IP and VLF

A 5 km² Induced Potential (“IP”) ground geophysical survey was carried out in the Nkran pit area by Geotech Airborne to determine its usefulness in this environment as an exploration tool to help resolve and map the sub-surface. A total of 12,640m (12.64 km) of baseline were cut and picketed by local crews in Figure 6-6 top left, and 76,240m (76.24 km) of cross lines were cut, picketed, and GPS surveyed.

The contoured Apparent Resistivity results, interpreted at 119m depth slice after inverting the data with the UBC 2D inversion software, are presented in Figure 6-6 bottom left. The Nkran structural trends are clearly outlined.

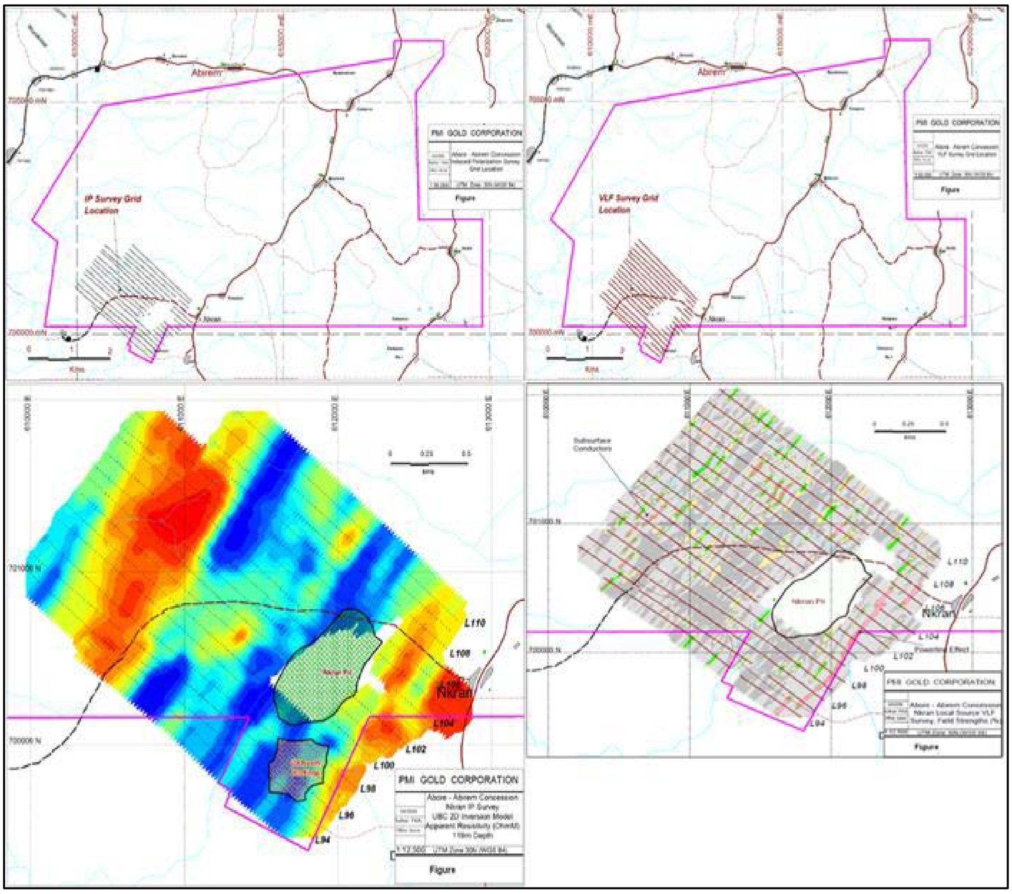


Figure 6-6: Ground Geophysical Surveys. (source: Spiers 2011, PMI and Asanko Gold 2014 Technical Report)

Note:

IP grid location (top left) and apparent resistivity grid (bottom left); VLF survey grid (top right) and VLF Field Strengths (bottom left) Source: Spiers, 2011 SRK Consulting, Page 57, GLEE/NAID/BINO/GUIB/wulr PMI003 MRE Update Report NI_43-101 March 2012 Rev1 25 May 2012

PMI purchased a local source Very Low Frequency ("VLF") transmitter and rented two analogue receivers in order to carry out conductivity surveys over target areas outlined by geochemical surveys (soil and stream sediment) and by structures interpreted from airborne resistivity surveys.

The 5 km² area surrounding the Nkran pit that was surveyed with PMI's in-house IP equipment and geophysical crew, was also surveyed with the in-house VLF equipment, see Figure 6-6 bottom right.

Dip and gain readings were taken every 12.5m, data processed and a Frazer Filtered map constructed. The survey was carried out by Fred Akosah, Chief Geophysicist, Eddie Norman, Technician, and a local field crew. Fred Akosah and Douglas MacQuarrie experimented with various field techniques and data rendition, manipulation and presentation.

It was determined that the technique was useful in picking out narrow conductivity features. The Nkran survey was carried out at 100m cross line spacing. In Figure 6-6 top right, illustrates the grid surveyed and its location. The test survey and data processing had determined that the gain is the most useful survey parameter in the environment encountered. A contoured map of field strengths percent was constructed and is presented in Figure 6-6 bottom right.

A 5.1 km antenna line was cut and 42 line km on 42 lines were surveyed.

6.4.2 Airborne Geophysical Survey – Heli-borne VTEM

Condor Consulting Inc. completed the processing and analysis of a VTEM EM and magnetic survey flown for PMI by Geotech Airborne Ltd, centred over the Nkran pit (Figure 6-7). The primary purpose of this programme of work was to test the VTEM response over the Nkran deposit to 400m depths, and to assist in the identification of gold targets in close proximity to the Nkran mine through an interpretation of conductive and magnetic features. The outcome of this work has been to identify a number of target zones based on the recognition of discrete conductive features and a structural analysis of the conductivity and magnetic outcomes.

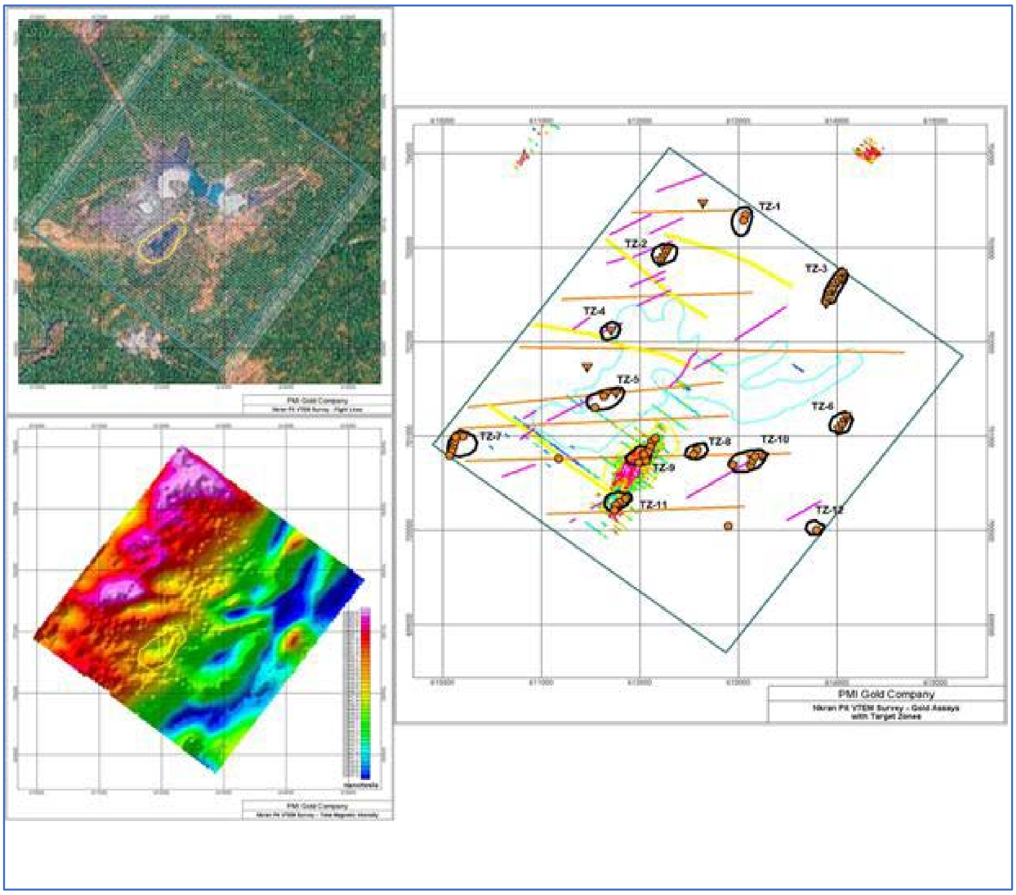


Figure 6-7: VTEM Survey Nkran Pit (yellow outline) (source: Spiers, 2011)

Notes: extent and flight lines (top left); TMI magnetic grid (bottom left) and interpreted target zones and structures (right).

6.5 Work Carried Out by Keegan at Esaase

Keegan discovered and evaluated the Esaase deposit over the period 2006 – 2013.

6.5.1 Geophysical Programmes

An IP programme was completed in 2006 which successfully identified significant faults that are interpreted as significant mineralisation boundaries. In order to identify other such structures, Keegan contracted Geotech Ltd to perform an airborne VTEM geophysical programme on the project area. The survey was carried out during the period October 11 to October 25, 2007. The principal geophysical sensors included Geotech's versatile time Domain Electromagnetic System ("VTEM"). Ancillary equipment included a GPS navigation system and a radar altimeter. A total of 2,266 line-km were flown. In-field data processing involved quality control and compilation of data collected during the acquisition stage, using the in-field processing centre established in Ghana.

The survey was flown at nominal traverse line spacing of 200m. Flight line directions were N130°E/N50°W. The helicopter maintained a mean terrain clearance of 122m. The data was processed and interpreted by Condor Consulting, Inc., who performed AdTau time constant analysis on line data in order to determine the best time delay channels to use. Condor performed Layered Earth Inversions ("LEI"), generated depth slices for the survey and characterized the 2D and 3D nature of the survey.

The 10 channel map shown in Figure 6-8 is relatively deep penetrating channel that avoids noise disturbance and provides an overall picture of the resistive characteristics of the rocks. The 92m Layered Earth Inversion is useful for a more detailed view of bedrock resistivity at the fresh bedrock surface.

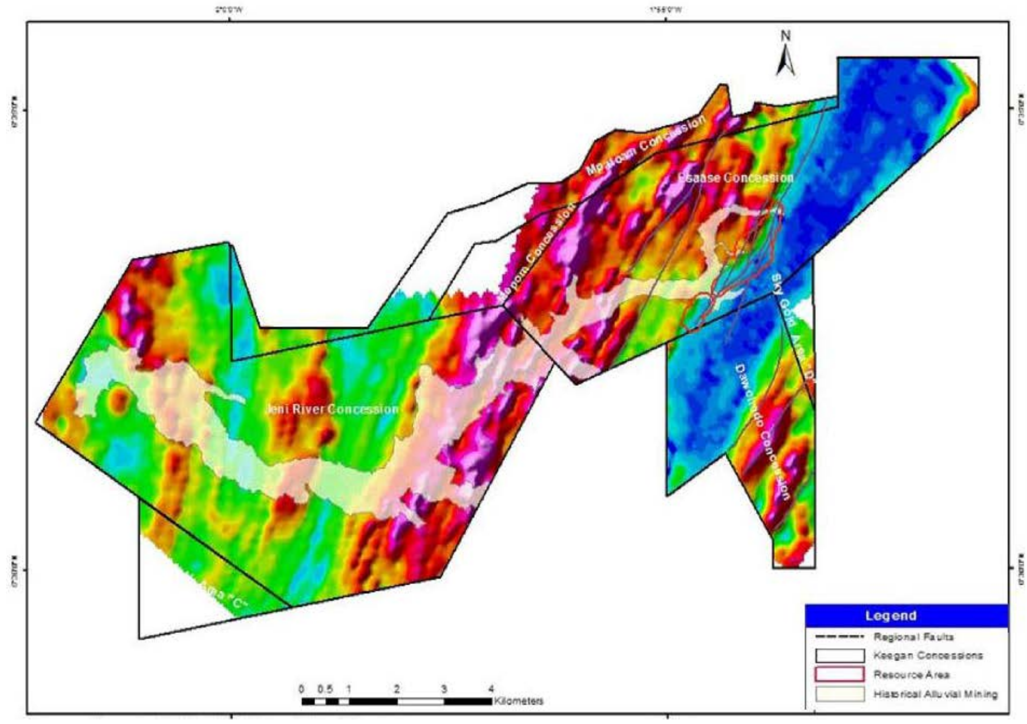


Figure 6-8: VTEM 92 m Layered Earth Inversion (LEI) for Esaase. (source: Esaase Technical report 2011, Coffey Mining)

The image indicates significant packages of higher and lower resistivity rock masses with changes in the resistivity values of the rocks across what are interpreted as northeast oriented structures. These breaks correlate with the position and orientation of gold anomalies, which are expressed both in the surface soils that overlie these breaks and in the subsurface as indicated by drilling.

6.5.2 Sampling Methods and Sample Quality

6.5.2.1 Soil Sampling Programme

Keegan commenced a soil sampling programme upon acquisition of the Esaase Concession in June 2006 and received assay results from over 4,000 soil samples. Sampling was undertaken on NE oriented lines spaced 100m to 400m apart with samples taken at 25m intervals along the lines Figure 6-8 This programme extended the initial soil sampling completed in March 2006 as part of initial due diligence on the concession. After the acquisition of the Jeni River Concession, Asanko Gold expanded its soil programme to the Jeni River Concession and has obtained over 2,100 samples from this concession using an identical sampling regime (Figure 6-9 and Figure 6-10.

Figure 6-11 shows the gold-in-soil contour map derived from these samples. Some 1,630 soil samples were collected over the Dawohodo concession in 2011. Soil samples were obtained wherever there were no obvious alluvial disturbance, or alluvial material and care was taken to sample below the organic horizon. The material below the organic horizon on ridge tops, or steep slopes from higher elevations is weathered bedrock, whereas that taken nearer to the alluvial creek bottoms is underlain by colluvium and/or saprolite. Drilling and trenching indicate that soil samples from weathered bedrock, on average, have gold levels within an order of magnitude of the underlying rock values. Soil samples from non-bedrock sources, (i.e. alluvial), tend to have much lower gold values than the underlying bedrock.

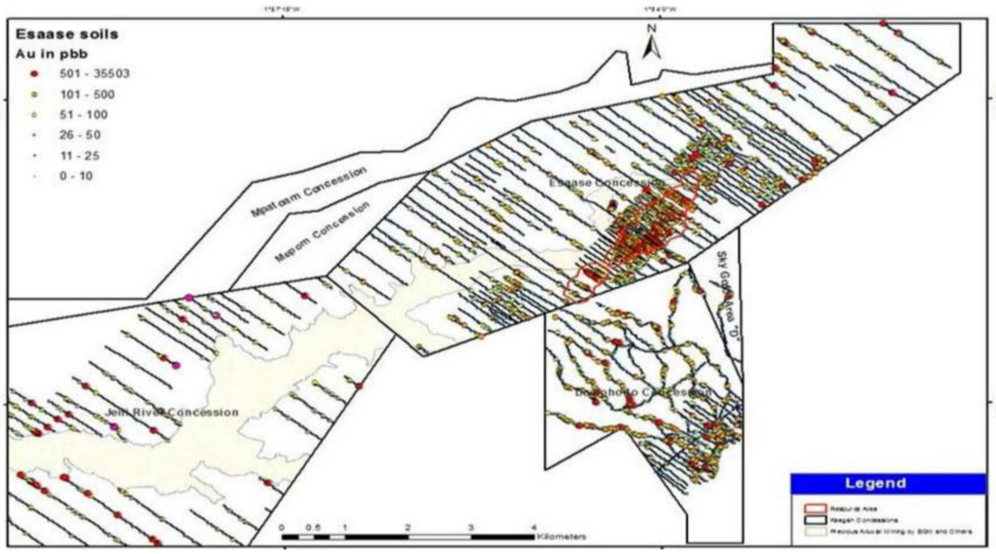


Figure 6-9: Gold in Soil Thematic Map for the Esaase Concession. (source: Esaase Technical Report 2011, Coffey Mining)

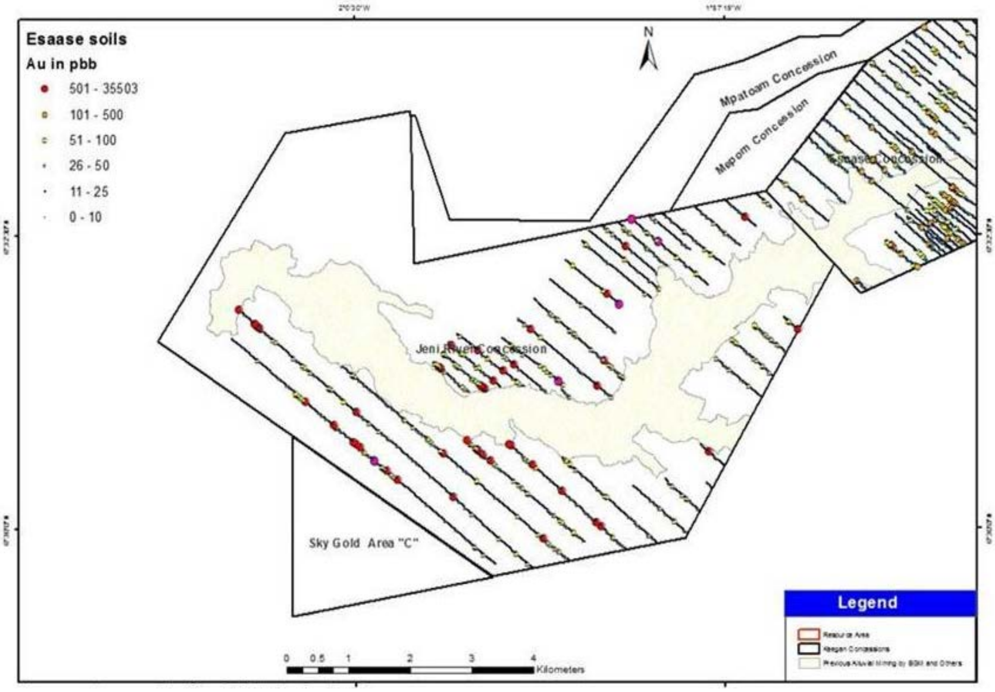


Figure 6-10: Gold in Soil Thematic Map for the Jeni River Concession. (source: Esaase Technical Report 2011, Coffey Mining)

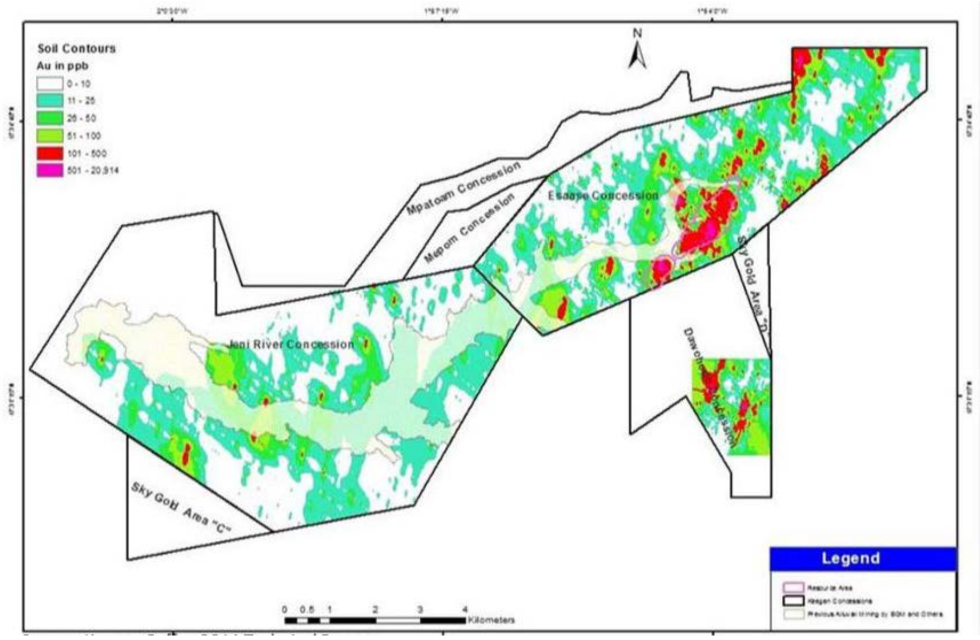


Figure 6-11: Gold in Soil Contour Map for the Esaase Prospect. (source: Esaase Technical Report 2011, Coffey Mining)

6.6 AGM Project Area Previous Mining and Satellite Pit Activities

A number of satellite pit mining and evaluation projects were effected prior to the Asanko Gold consolidation of the Keegan and PMI asset (Figure 6-12 and Figure 6-13).

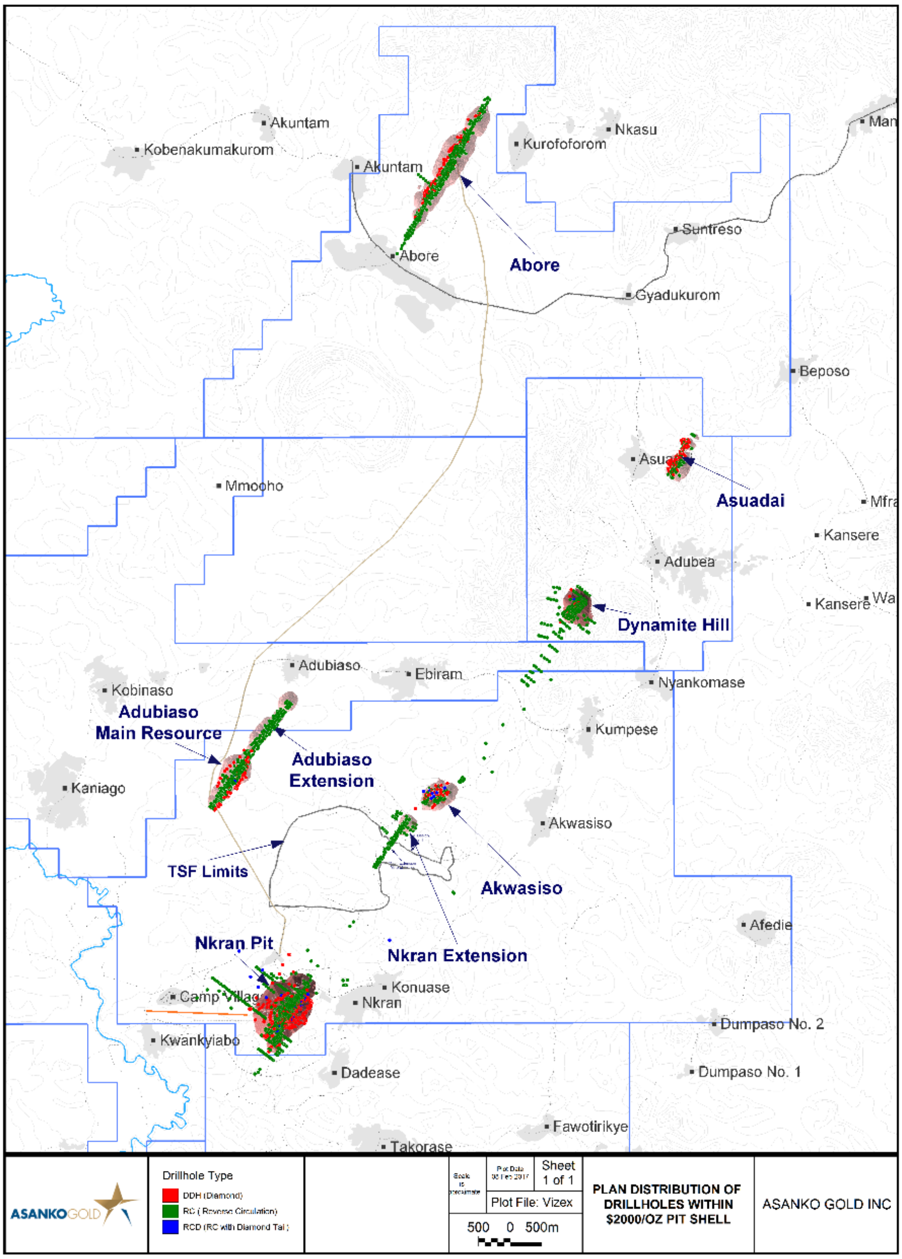


Figure 6-12: Asanko Gold Tenement Area and Locality of Satellite Gold Deposits proximal to Nkran Mine (source: Asanko Gold 2016)

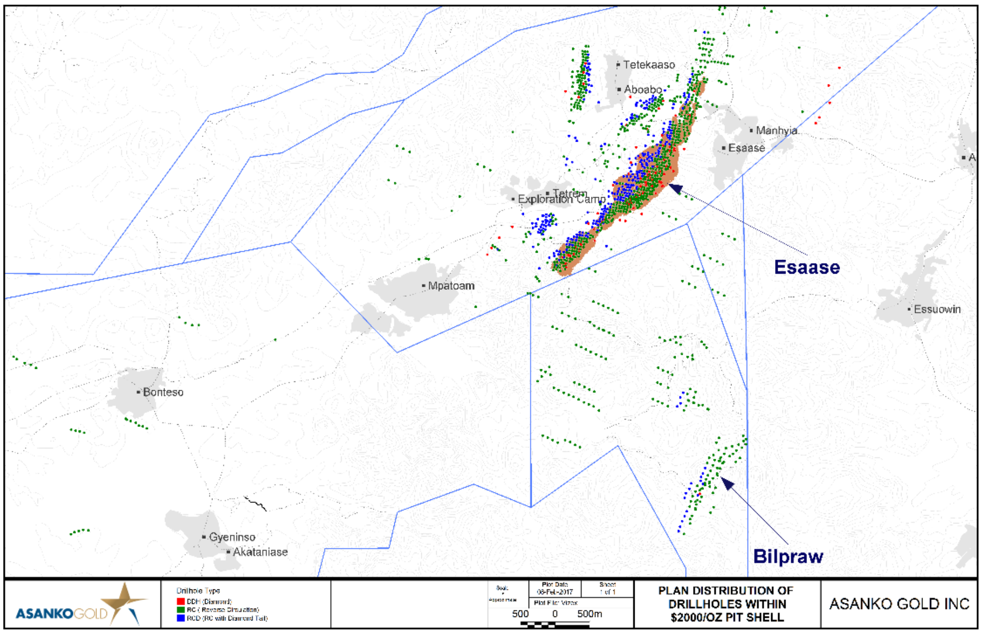


Figure 6-13: Asanko Gold Tenement Area and Locality of Esaase and Satellite Gold Deposits (source: Asanko Gold 2016)

6.6.1 Abore Area

The Abore prospect is located approximately 45 km south-west from Kumasi and about 9 km west of Manso Nkwanta, the district capital. The Nkran open pit operation is located about 12.5 km south of Abore and the Esaase gold project is about 10 km to the north. The area falls within a relatively high rainfall area (1,500 - 2,000 mm/yr) and the vegetation is mainly of the tropical forest deciduous type that dominates south-western Ghana.

The immediate area of Abore has only moderate relief. However, to the north are the Manso Nkwanta highlands, (Gyeni River Forest Reserve with maximum elevation of about 600m amsl) where the relief is about 300m. Immediately south of the town of Abore (approximate elevation of 180m amsl in town area) there is a very prominent, circular hill of bare granite with a relief of about 170m.

Early examinations of the district by Gold Coast Survey geologists (Cooper, GC Geological Survey Annual Report 1932 - 33, pp 37 - 38), noted the very widespread pitting in the district, (see section on the Nkran pit). The artisanal miners were mainly targeting alluvial and eluvial gold occurrences.

On a regional basis, it was observed that many of the bedrock occurrences are aligned along parallel northeast to north northeast structural zones. The western structure includes old pits near Aburi, (now Abore, or Abori). Junner (1935) summarised descriptions of the major known deposits in the district, but no specific mention was made of any occurrences north of Abore.

The general area certainly attracted some attention in the gold rushes of 1898 to 1901 and again in the 1930’s, as indicated on concession maps from those periods. However, there appears to be no record of the work done on most of the concessions, with the exception of the Bilpraw mine. In the

1930's, the well known Chief Joseph Biney of Ashanti Goldfields fame held a small concession (13 km^2) close to Abore, (the Adubea concession now held by Asanko Gold) and a group known as the Akan Syndicate held a 26 km^2 concession in the highlands north of Abore village.

It was not until the early 1990's that the area attracted further attention and this was mainly because of the successful exploration work carried out on the adjacent concession where the Nkran deposit is located. Alluvial mining had also started up on the nearby Jeni and Bonte rivers to the north. The Abore area was covered in a prospecting concession granted to the Oda River Gold small scale mining licence (Asuadai prospect) at Adubea.

In the mid-1990's, Mutual Resources of Vancouver, Canada, in partnership with Leo Shield Exploration of Perth, Australia, completed a joint venture with the Oda River group and commenced a regional exploration programme (soil geochemistry and follow up trenching) on the concession (covering 73 km^2). Full details of this work are not available, but Griffis (2002) notes that prospecting in the area north of Abore revealed artisanal mining in alluvial areas, as well as many old Ashanti pits in the saprolite along a low hill immediately adjacent to the alluvial workings.

Soil geochemistry revealed a strong north-north-east trending gold anomaly over the area of artisanal mining (bedrock areas). The anomaly is several hundred metres wide and traceable along strike for about 3 km, well beyond the area of old workings. Trenching in the area confirmed continuous bedrock mineralisation over a distance of at least 1,000m with widths in the range 50m to 100m.

In the late 1990's, Mutual's interest in the project was bought out by Leo Shield, (now Shield Resources). In early 2001, an agreement was reached with Resolute whereby ore was trucked from Abore north to the Nkran plant for treatment.

6.6.2 Nkran Area

The Nkran pit is located approximately 40 km west northwest of Obuasi and just east of the Offin River. Good road access to the area is available via a partially tarred road from the Bekwai junction (Awiankwanta) on the main Obuasi-Kumasi highway. The area is part of the general Manso Nkwanta, or Amansie West District and features rolling hills with topographic relief generally in the range 50m to 100m, but rising considerably in the rugged hills north of Manso Nkwanta where relief is usually more than 200m. The high rainfall (1,500 - 2,000 mm/yr) in the area produces thick vegetation, although the primary forests have largely been degregated and are mainly restricted to a few forest reserves. There are many small farming villages throughout the general area along a network of gravel roads. Most of the farming is intended for local needs and consists mainly of plantain, corn and cassava, but important cash crops include cocoa, oil palm and citrus fruit.

The area appears to be quite important from the view point of historical artisanal gold mining that dates back many generations and remains quite extensive to the present day. As noted by Junner (1935) "One of the outstanding features of this area (Manso Nkwanta District), which was examined by Dr Cooper in 1932, is the colossal number of old native pits sunk chiefly for alluvial and elluvial gold."

The Nkran deposit appears to correlate with Junner's description of the Jabokassie (Tewoakrom) prospect, which he described very briefly as being "one mile west of Nkrang and 19 km south-west of Mansu Nkwanta. The exposure of the ore body was a "stock work of sub-vertical and flat lying irregular narrow veins of quartz in a dyke of quartz porphyry intruding spotted phyllites". The European workings consist of adits and drives, which run 80m into the hill on the site of old native workings, which extend for nearly 610m in a north-east-south-west direction. The samples dollied by Dr Cooper gave variable results, mostly poor."

In the late 1980's, this prospect attracted the attention of consultant Dr Alex Barko who recommended the area to one of his local Client groups and Obotan Minerals subsequently applied and received a prospecting concession covering about 106 km^2 over the general area. Limited evaluation work was undertaken in the early stages (Griffis, 2002; p.171). Some attention was paid to the alluvial gold potential because of the extensive gold in the nearby Offin River, (held by Dunkwa Goldfields), as well as the alluvial gold project being developed at the time, a little further north in the Bonte area. In the early 1990's, the Obotan concession was examined by American consultant Al Perry who was working on behalf of two related Australian juniors, Associated Gold Fields NL and Kiwi International Resources Limited.

According to Griffis (2002) Perry negotiated an option on the concession and proceeded to focus on the known prospects at Nkran deposit, (formerly known as Jabokassie) and to carry out a regional soil geochemical survey that identified numerous anomalies around Nkran. The new group moved quickly to access the open pit potential of the Nkran prospect. An early phase of RC drilling (details not available) yielded encouraging results over a wide zone of bedrock mineralisation, which extended along strike for about 600m. The broad, low-lying Nkran had relief of only about 40m with oxidation extending to depths of 40m.

By early 1995, MRE (Measured, Indicated and Inferred classes) were reported as 4.8 Mt averaging 3.7 g/t Au with contained gold of approximately 600,000 oz Au. A feasibility study was completed and a mining lease was granted in late 1995 (Griffis, 2002; p 154).

This exploration work was noted by the emerging Australian gold producer, Resolute Samantha, (now Resolute), who were keen to gain an entry into the rapidly expanding gold mining sector of Ghana. A deal was completed by May 1996 whereby the combined interests of Kiwi and Associated Gold Fields were bought out by Resolute who immediately reviewed and expanded the scope of the project. This was achieved mainly by conducting further RC and DC drilling to increase mineral resources to a depth of 150m at Nkran and to further assess the known mineralisation at nearby Adubiaso.

A revised mine development plan was completed by the end of July 1996 and a decision was made to proceed into production at a rate of 1.4 Mtpa. Initial mining was started early in 1997 and by May 1997, the first gold was poured. The fast tracking of this project into production in about 8 months was made possible by the very experienced team assembled by Resolute and the prime contractor, Lycopodium, who were responsible for the design and construction of the treatment plant.

The Nkran Mine closed in 2001 having produced 590,743 oz Au at an average grade of 2.35 g/t Au (Resolute Mine closure document, 2001). The mine was dewatered and re-opened in 2015-2016 by

Asanko Gold as a deeper opencast operation with an estimated life of mine of approximately 12 years.

6.6.3 Adubiaso Area

During the late 1990's, the Nkran plant started to process Oxide ores from the Adubiaso gold deposit, located about 7.5 km north-north-west of Nkran. At Adubiaso, there is a steep dipping (approximately 65°east) quartz vein system cutting across Birimian metasediments, which dip steeply (approximately 75°) to the west. The vein system appears to be related to a north-east fracture system (distinct from the Nkran structure) along the contact zone between dominantly phyllitic units on the east and coarser greywackes on the west, which host most of the gold bearing veins. The central part of the vein system is 15m to 20m wide, but it tapers to about 10m at both ends; the vein system has a strike length of about 700m although the main area of economic significance is the central 300m of the zone.

As at Nkran, there are narrow granitoids running generally parallel to the Adubiaso orebody in the pit area, but these are un-mineralised. It is also noteworthy that, at Adubiaso, the gold mineralisation is restricted to the quartz veins and the metasedimentary host rocks are essentially barren, whereas at Nkran the gold values extend well into the host rocks.

6.6.4 Asuadai Project Area

There are numerous other prospects in the immediate vicinity that may eventually be mined. The Asuadai prospect is within a 10 km radius of the Nkran project area. The prospect features a massive intermediate (tonalite) granitoid hosting a quartz stock work system. Local artisanal miners have undertaken minor pitting in the region down to 5m to 10m through the Oxide material to expose these stock work vein sets.

6.6.5 Dynamite Hill – Akwasiso - Nkran Extension Project Area

Exploration activity since 2013 on the north-eastern extension of the Nkran structure delineated a number of zones of mineralisation vis Dynamite Hill, Akwasiso and Nkran Extension that have all been drilled (2016) to Indicated resource classification. Dynamite Hill mineralisation is hosted within veins and disseminations in and around a granitic intrusive locating on the Nkran structure. Akwasiso is a smaller version of Nkran, with mineralisation hosted in shears on siltstone / sandstone contacts, and around and within a granitic intrusive. Nkran Extension is more planar shear controlled mineralisation locating on splays off the Nkran structure.

6.6.6 Esaase Project

Artisanal mining has a long history in the Bonte Area, (where the project area is located), associated with the Ashanti Kingdom. Evidence exists of adits driven by European settlers between 1900 and 1939. However, no documented records remain of their activity. Drilling was conducted on the Bonte River valley alluvial sediments during 1966 and 1967 to determine alluvial gold potential.

In 1990, the Bonte mining lease was granted to Akrokerri-Ashanti Gold Mines ("AAGM") and was later transferred to Bonte Gold Mining ("BGM"), a local subsidiary of AAGM. BGM had reportedly recovered an estimated 200,000 oz Au of alluvial gold on the Esaase concession and another 300,000 oz Au downstream on the Jeni River concession (Griffis, 2002; p.178), prior to entering into receivership in 2002. It should be noted that previous placer gold production is of no relevance to Asanko Gold's development programme, which is entirely focused on the development of hard rock resources.

The Esaase mining concession, including the camp facilities at Tetrem, was bought from the BLC by Sametro Company Limited, a private Ghanaian company. In May, 2006, Asanko Gold, then called Keegan, signed a letter of agreement with Sametro to earn 100% of the Esaase mining concession over a three year period of work commitments and option payments. Since mid 2006, Asanko Gold has undertaken an aggressive exploration programme combining soil geochemistry and geophysical surveys followed by DC and RC exploration drilling.

6.7 AGM – Phase 1 - Historical Mineral Resource and Reserve Estimates

The historical Mineral Resources and Reserves quoted in this section have been superseded by the current Mineral Resources (Section 14) of this report. The historical estimates have not been verified by the Qualified Persons ("QPs") of this report and therefore should not be relied upon. The current resources and reserves were not based on the historical estimates in any way, and the historical estimates are only quoted to provide historical context. A summary of the Resource and Reserve estimation history is provided in Table 6-1 below.

Table 6-1: Obotan MRE History (source: Asanko Gold)

Company	Mineral Resource Reporting Code	Category	Year
Kiwi / Associated Gold Fields	JORC	Resource	1995
Resolute	JORC	Resource and Reserve	1996
Resolute	JORC	Resource and Reserve	1998
Leo Shield	JORC	Abore Resource	1998
Resolute	JORC	Resource and Reserve	1999
Hellman & Schofield	JORC	Resource	2011
SRK	CIM / NI 43-101	Resource	2011
SRK	CIM / NI 43-101	Resource	2012
GRES	CIM / NI 43-101	Reserve	2012
CJM	CIM / NI 43-101	Resource	2014
CSA / DRA	CIM / NI 43-101	Resource and Reserve	2017

The first comprehensive MRE for the Nkran ore body was prepared by the Kiwi / Associated Gold Fields joint venture in early 1995 (Table 6-2). This was based on 4,100m of RC drilling and an additional 1,750m of DC drilling.

Table 6-2: Kiwi / Associated Gold Fields 1995 – Nkran Historical Gold Resource Estimates

Category	MRE
Measured	3.2Mt @ 3.5g/t Au (360,000 oz Au in situ)
Indicated	1.4Mt @ 2.8g/t Au (120,000 oz Au in situ)
Inferred	1.5Mt @ 2.8g/t Au (130,000 oz Au in situ)
Total	6.1Mt @ 3.2g/t Au (610,000 oz Au in situ)

The above estimates did not distinguish between Oxide and primary (sulphide) resources since both are free milling. However, there does appear to have been a near surface enrichment and mushrooming of the ore body close to surface and it is likely that a majority of the measured resources were within the Oxide zone. The resource estimates were taken to depths of about 100m below surface.

When Resolute took over, it became an immediate priority to expand resources in order to justify a larger open-pit operation. This was achieved by defining resources at Nkran to a depth of 150m and by adding modest resources from the nearby Adubiaso prospect. In their 1996 Annual Report, Resolute reported the Mineral Resource and Reserve Estimates for the Obotan concession shown in Table 6-3, which were based on about 6,570m of RC and 4,860m of DC drilling.

Table 6-3: Resolute 1996 - Obotan Concessions Historical Mineral Resource and Reserve Estimates

Category	Mineral Resource and Reserve Estimates
Proven and Probable Reserves	10.15 Mt @ 2.11 g/t Au (689,000 oz Au)
Measured and Indicated Resources[1]	4.05 Mt @ 1.86 g/t Au (242,000 oz Au)
Inferred Resources	7.70 Mt @ 1.81 g/t Au (448,000 oz Au)

Note[1]: M&I Resources stated exclusive of mineral reserves.

This was the basis for a Bankable Feasibility Study ("BFS") by the Lycopodium group with the view to processing 1.4 Mtpa. Ongoing exploration drilling expanded the Mineral Resource and Reserve Estimates after the mine started production in the second quarter of 1997. The Mineral Resource and Reserve Estimates, as given in the 1998 Annual Report, are shown in Table 6-4.

Table 6-4: Resolute 1998 - Obotan Concessions Historical Mineral Resource and Reserve Estimates

Category	Mineral Resource and Reserve Estimates
Proven and Probable Reserves	9.97 Mt @ 2.7 g/t Au (849,259 oz Au)
Measured and Indicated Resources[1]	27.25 Mt @ 2.2 g/t Au (1,927,246 oz Au)
Inferred Resources	0.975 Mt @ 1.66 g/t Au (46,834 oz Au)

Note[1]: M&I Resources stated exclusive of mineral reserves.

Table 6-5 shows that, as of June 1999, Reserve and Resource Estimates had little changes.

Table 6-5: Resolute 1999 - Obotan Concessions Historical Mineral Resource and Reserve Estimates

Category	Mineral Resource and Reserve Estimates
Proven and Probable Reserves	6.1 Mt @ 2.46 g/t Au (485,190 oz Au)
All Resources²	28.4 Mt @ 2.17 g/t Au (1,982,245 oz Au)

Note ² All resources include Inferred resources.

Note ² Tables 6-2 to 6-5 do not comply with current NI 43-101 disclosure practice.

The main difference in the estimates historically is that whilst the overall resource remains quite high, the mineable reserves had dropped considerably and had not been replaced. Subsequent estimates indicated some addition in the resource estimates, but the impact of the higher and rising gold prices resulted in the company lowering the cut-off grade and increase in resources ounces at a lower grade.

The Nkran project came on stream in May 1997, under budget and at least one month ahead of schedule. The plant was designed and constructed by the Lycopodium group, whose extensive previous experience in Ghana was important in bringing the mine into production in about an eight month period at lower than projected costs.

The capital cost was US$32 million for an operation with an annual plant throughput of 1.4 Mtpa and gold production of about 120,000 oz Au per year. These costs include mainly plant, infrastructure and pre-mining costs, but exclude the mining equipment, which was provided by the mining contractor.

The design capacity of the plant was 1.4 Mtpa for primary ore and it was expected that considerably higher throughput 2 Mtpa would be achievable very early on because most of the ore would be from the softer Oxide zone. Very early on, plans were made to expand the primary ore production to about 2 Mtpa with very modest additional capital cost.

The plant design was a standard gravity - CIL design, incorporating a coarse crusher and SAG mill, a Knelson concentrator, CIL circuit and a desorption plant. As expected, the throughput of Oxide ores exceeded 2 Mtpa. A later design modification incorporated an additional crushing stage so that a finer and more homogenous feed would be supplied to the SAG mill and thereby increasing the treatment of the harder sulphide ores to at least 2 Mtpa. The plant design originally had only one Knelson concentrator, but with the increased throughput and relative abundance of easily liberated free gold, a second Knelson bowl was added. The severe power shortages from the national grid starting in 1997 resulted in the installation of a standby power plant, which allowed Resolute to maintain a steady throughput.

The official production figures published by the Ghana Chamber of Mines since the start-up of operations in 1997 are as given in Table 6-6 below.

Table 6-6: Obotan Historical Processed Ore (Source: Griffis 2002, Resolute Annual Reports 1997, 1998, 1999)

	1997	1998	1999
Processed Ore (Mt)	0.664	0.664	0.664
Gold Production (oz Au)	39,903	39,903	39,903
Recovered Grade (g/t Au)	1.87	1.87	1.87

The original cash operating costs were projected to be just below US$220/oz Au, but with the advent of a declining gold price environment, considerable efforts were made to reduce the operating costs. The high throughput and favourable recovered grades in 1998 resulted in a reported cash operating cost of US$162/oz Au. This would equate to about US$12.33/t of ore processed. The lower throughput and recovered grades in 1999 were expected to improve in the remaining years of production to about 150,000 oz Au per year at an anticipated cost in the range of US$180-200/oz Au.

Table 6-7 gives the Leo Shield (1998 annual report) estimate of Indicated and Inferred Abore resources (based on JORC reporting standards).

Table 6-7: Leo Shield 1998 - Abore Historical MRE

Category	Mineral Resource Estimates
Indicated Resources	2.057 Mt @ 2.4 g/t Au (156,000 oz Au in situ)
Inferred Resources	2.588 Mt @ 1.7 g/t Au (139,400 oz Au in situ)

The above MREs were based on a cut-off grade of 0.7 g/t Au and an average width of mineralisation of approximately 35m. The Fresh ore (sulphidic) resources were estimated to an average depth of approximately 100m. The above estimates include both Oxides and primary zone Sulphides, whilst the Transitional material is included with the Sulphides. The total Oxide resource is estimated to be approximately 1.68 Mt @ 2.2 g/t Au (approximately 119,000 oz Au in-situ). The above estimates are based on 355 RC (about 28,600m) and 13 DC drill holes (1,560m). As noted earlier, the limited resource base made a stand-alone project not viable with the then gold prices.

Hellman and Schofield ("H&S") were commissioned by PMI in 2010 to conduct a MRE for the project areas utilising analytical data made available to H&S up to the close of the data set on the 23rd of June 2010. The H&S February 2011 MRE was based on a 0.5 g/t Au cut-off for all deposits except for the Nkran underground which utilised a 1.5 g/t Au cut-off. Results of the 2011 MRE are shown in Table 6-8.

Table 6-8: Historical MREs for Obotan 2011 (source: H&S)

Project Area	Indicated			Inferred		
	Tonnes	Au g/t	Gold Content Oz	Tonnes	Au g/t	Gold Content Oz
Abore	1,020,000	1.51	49,000	2,235,000	1.40	98,000
Adubiaso	1,033,000	1.58	53,000	2,667,000	1.30	113,000
Asaudai	390,000	1.29	16,000	1,131,000	1.30	48,000
Nkran OC	539,000	1.58	27,000	5,946,000	2.00	385,000
Nkran UG	82,000	4.12	11,000	3,658,000	3.50	409,000
Total - All Areas	**3,064,000**	**1.59**	**156,000**	**15,637,000**	**2.10**	**1,053,000**

Subsequent to the H&S 2010/2011 estimation and during 2011, PMI drilled a further 85 DC drill holes totalling 26,605m, focussing mostly on the Nkran mineral resource. SRK was then commissioned to conduct an updated MRE in accordance with NI 43-101 guidelines for the Nkran pit and other satellite deposits, including the additional drilling data from June 2010 to August 2011. The SRK October 2011 MRE utilised a 0.5 g/t Au cut-off grade for all opencast deposits and is summarised in Table 6-9 below.

Table 6-9: SRK October 2011 MRE (source: SRK 2011)

2011	Measured			Indicated			Inferred		
	Tonnes (millions)	Grade (g/t Au)	Moz Au	Tonnes (millions)	Grade (g/t Au)	Moz Au	Tonnes (millions)	Grade (g/t Au)	Moz Au
Nkran	11.10	2.76	0.98	19.70	2.42	1.52	12.60	2.54	1.02
Adubiaso	1.07	2.78	0.09	2.60	2.30	0.19	0.87	2.06	0.05
Abore	2.50	1.88	0.15	3.99	1.80	0.23	3.40	1.72	0.18
Asuadai	N/A	N/A	N/A	1.21	1.71	0.06	0.67	1.95	0.04
Total	**14.67**	**2.66**	**1.22**	**27.5**	**2.32**	**2.00**	**17.54**	**2.35**	**1.29**

Subsequent to the SRK October 2011 MRE, from August 2011 to January 2012, PMI drilled a further 110 DC drill holes on all 4 deposits which totalled 28,835m. SRK then completed an updated MRE in accordance with NI 43-101 guidelines. The SRK March 2012 MRE utilised a 0.5 g/t Au cut-off grade for all deposits and is summarised in Table 6-10 below

Table 6-10: SRK March 2012 MRE (source: SRK 2012)

Deposit	Measured			Indicated			Inferred		
	Tonnes (millions)	Grade (g/t Au)	Moz Au	Tonnes (millions)	Grade (g/t Au)	Moz Au	Tonnes (millions)	Grade (g/t Au)	Moz Au
Nkran	11.74	2.55	0.96	20.41	2.12	1.39	14.74	2.21	1.05
Adubiaso	1.50	2.98	0.14	2.67	2.41	0.21	1.25	1.91	0.08
Abore	2.33	1.78	0.13	3.70	1.53	0.18	3.92	1.50	0.19
Asuadai	N/A	N/A	N/A	2.44	1.28	0.10	2.00	1.33	0.10
Total	**15.57**	**2.47**	**1.23**	**29.21**	**2.00**	**1.88**	**21.91**	**1.99**	**1.40**

Subsequent to the SRK March 2012 MRE, PMI commissioned GR Engineering Services ("GRES") to complete a DFS Study and Mineral Reserve Estimate ("MRev") for the project. The MRev stated in the 2012 GRES DFS was based on the SRK March 2012 MRE and a gold price of US$1,300/oz Au. A summary of the 2012 MRev is found in Table 6-11.

Table 6-11: GRES September 2012 MRev (source: GRES)

Deposit	Classification	Tonnage (Mt)	Au Grade (g/t)	Moz Au
Nkran	Proven	11.5	2.47	0.92
	Probable	14.6	2.17	1.02
Adubiaso	Proven	1.2	2.80	0.1
	Probable	1.3	2.62	0.11
Abore	Proven	2.1	1.70	0.11
	Probable	1.9	1.70	0.10
Asuadai	Proven	0	0	0
	Probable	1.6	1.22	0.06
Total	Proven	14.8	2.39	1.14
	Probable	19.4	2.08	1.30
	Total	34.2	2.21	2.43

Subsequent to the SRK March 2012 MRE, Asanko Gold commissioned CJM to complete an updated MRE for the Nkran and the satellite deposits. CJM completed an updated MRE in accordance with NI 43-101 guidelines.

The CJM October 2014 Obotan MRE included no additional drilling on the Nkran, Abore, Adubiaso, and Asuadai deposits, but included a maiden resource for the Dynamite Hill deposit. A total of 17,977m of additional drilling on Dynamite Hill were included in the updated 2014 MRE and a 0.8 g/t Au cut-off grade was utilised for all these deposits. A summary of the CJM October 2014 MRE is found in Table 6-12.

Subsequent to the mine development and initiation of production from Nkran in January 2016, CJM have updated the MRE for Nkran, inclusive of depletion to 31 December 2016.

CSA were employed between August 2016 - January 2017 to audit the Nkran reconciliation, up to 31st December 2016, and, based on estimated vs actual, to undertake reconciliations from September to December 31st 2016 as well as audit the Nkran, Esaase and Dynamite Hill MREs. The other satellite Resources and Reserves remain as per the CJM October 2014 estimate. Furthermore, all the AGM pits MREs are now reported as constrained to >0.5 g/t Au within a US$1,500/oz Au pit shell, cf previously unconstrained at >0.8 g/t Au cut-off. These results are reported in Section 14.

6.8 Historical Mineral Resource and Reserve Estimates for Esaase

MREs for the Esaase deposit were released by Coffey Mining on behalf of Keegan in 2007, 2009 and 2011. These resource tables are summarised in Table 6-12, Table 6-13 and Table 6-14.

Table 6-12: MRE for the Esaase Gold Project dated Dec 2007 (source: Coffey Mining)

Resource Category	Cut-off Grade	Tonnage	Gold Grade	Gold Content
	g/t Au	Mt	g/t Au	Moz Au
Indicated	0.4	6.94	1.20	0.26
	0.6	5.41	1.40	0.24
	0.8	3.97	1.60	0.21
	1.0	2.85	1.90	0.17
	1.2	2.10	2.20	0.15
Inferred	0.4	43.89	1.10	1.62
	0.6	31.94	1.40	1.43
	0.8	23.16	1.70	1.24
	1.0	17.07	1.90	1.06
	1.2	12.99	2.20	0.92

Table 6-13: MRE for the Esaase Gold Project dated April 2009 (source: Coffey Mining)

Resource Category	Cut-off Grade	Tonnage	Gold Grade	Gold Content
	g/t Au	Mt	g/t Au	Moz Au
Indicated	0.4	57.99	1.20	2.28
	0.5	49.25	1.40	2.15
	0.6	41.94	1.50	2.02
	0.7	35.75	1.70	1.89
	0.8	30.66	1.80	1.77
	0.9	26.32	2.00	1.66
	1.0	22.78	2.10	1.55
Inferred	0.4	41.66	1.20	1.65
	0.5	34.05	1.40	1.55
	0.6	28.57	1.60	1.45
	0.7	24.43	1.70	1.36
	0.8	20.65	1.90	1.27
	0.9	17.91	2.10	1.20
	1.0	15.85	2.20	1.14

Table 6-14: MREs for the Esaase Gold Project dated November 2011 (source: Coffey Mining)

Resource Category	Cut-off Grade	Tonnage	Gold Grade	Gold Metal
	g/t Au	Mt	g/t Au	Moz Au
Measured	0.3	5.34	1.20	0.200
	0.4	5.03	1.20	0.196
	0.5	4.66	1.30	0.191
	0.6	4.21	1.40	0.183
	0.7	3.73	1.40	0.173
	0.8	3.26	1.50	0.162
	0.9	2.83	1.70	0.150
	1.0	2.44	1.80	0.139
Indicated	0.3	102.90	1.10	3.543
	0.4	93.71	1.10	3.441
	0.5	83.03	1.20	3.288
	0.6	72.04	1.30	3.096
	0.7	61.67	1.50	2.882
	0.8	52.51	1.60	2.663
	0.9	44.72	1.70	2.451
	1.0	38.14	1.80	2.251
Inferred	0.3	50.04	1.00	1.598
	0.4	45.90	1.10	1.553
	0.5	40.54	1.10	1.476
	0.6	34.82	1.20	1.375
	0.7	29.39	1.30	1.262
	0.8	24.79	1.40	1.153
	0.9	20.61	1.60	1.039
	1.0	17.12	1.70	0.932

The 2011 MRE (Table 6-15) was carried out by Coffey Mining on behalf of Keegan in accordance with the Canadian NI 43-101. The MRE was based on a cut-off of 0.4 g/t Au.

Table 6-15: Historical MRE for the Esaase Gold Project 2011 (source: Coffey Mining)

Deposit	Mineral Reserves								
	Proven			Probable			Total		
	Tonnage	Grade	Content	Tonnage	Grade	Content	Tonnage	Grade	Content
	Mt	g/t Au	Moz Au	Mt	g/t Au	Moz Au	Mt	g/t Au	Moz Au
Esaase	5.1	1.20	0.199	74.3	1.10	2.685	79.4	1.10	2.884
Note: Based on a cut-off of 0.4 g/t Au									

In May 2013 DRA carried out a MRE (Table 6-16) based on a PFS for the Esaase deposit as a stand-alone project.

Table 6-16: Historical MRE for the Esaase Gold Project 2013 (source: DRA)

Deposit	Mineral Reserves								
	Proven			Probable			Total		
	Tonnage	Grade	Content	Tonnage	Grade	Content	Tonnage	Grade	Content
	Mt	g/t Au	Moz Au	Mt	g/t Au	Moz Au	Mt	g/t Au	Moz Au
Esaase	22.85	1.43	1.050	29.49	1.40	1.320	52.3	1.41	2.370
Note: Based on a cut-off of 0.6 g/t Au									

7 GEOLOGICAL SITUATION AND MINERALISATION

The descriptions of the regional and local geology are sourced from Siddorn and Lee (2005) and Asanko Gold staff observations and mapping, and the descriptions of the prospect geology and mineralisation are based on the work of McCuaig and Williams (2002) updated to recent Asanko Gold observations and interpretations. More recent geotectonic dating work has been undertaken by the West Africa Exploration Iniative (“WAXI”) (2012-2015).

7.1 Regional, Local and Property Geology

7.1.1 Regional Geology

The geology of Ghana is dominated by predominantly metavolcanic paleoproterozoic Birimian Supergroup (2.25 – 2.06 billion years ago) sequences inclusive of the clastic Tarkwaian Group sediments (2.12 - 2.14 billion years), after WAXI 2015) in the central-west and northern parts of the country. Clastic shallow water sediments of the Neoproterozoic Volta Basin cover the northeast of the country (Figure 7-1). A small strip of Paleozoic and Cretaceous to Tertiary sediments occur along the coast and in the extreme southeast of the country.

The Birimian rocks formed during two major orogenic phases, the Eoeburnian from ca. 2250-2150 Ma, and the Eburnian between ca. 2116-2060 Ma. These two orogenic stages were separated by a major extensional event during which flysch type basins developed throughout the southern and western parts of the Birimian. A marked break in the timing of events is apparent between eastern (Cote d’Ivoire / Ghana) and western parts of the Birimian, near the centre of the West African Craton and the western margin of the Comoé Basin in CDI.

Key geotectonic events are:

- In Ghana Eoeburnian plutonism and contractional deformation occurred between ca. 2190-2140 Ma
- Basin formation between the Eoeburnian and Eburnian cycles occurred between ca. 2130-2116 Ma in Ghana
- Eburnian plutonism had essentially ceased by ca. 2095 Ma in Ghana
- Gold mineralisation in Ghana occurred during wrench deformation at ca. 2100-2090 Ma, after basin inversion / severe compression. At Nkran two distinct gold deposition periods are associated with early ductile-brittle deformation and a later cross cutting brittle quartz veining event

The Birimian rocks consist of narrow greenstone (volcanic) belts, which may be traced for hundreds of kilometres along strike, but are usually only 20 km to 60 km wide. These belts are separated by wider basins, (such as the Kumasi Basin) of mainly marine clastic sediments. Along the margins of the basins and belts, there appears to be considerable inter-bedding of basin sediments and volcanoclastic and pyroclastic units derived from the volcanic belts. Thin, but laterally extensive chemical sediments (exhalites), consisting of chert and fine-grained manganese-rich and graphitic sediments often mark the transitional zones. The margins of the belts commonly exhibit faulting on local and regional scales. These structures are fundamentally important in the development of gold deposits for which the region is well known.

The Tarkwaian rocks consist of a distinct sequence of metasediments (quartzite, conglomerate and phyllite) occurring within a broad band along the interior of the Ashanti Belt. Conglomerates host important palaeoplacer gold deposits in the Tarkwa district. Equivalent rock types occur in other belts of the region, but in relatively restricted areas. In the type locality at Tarkwa, the sequence is in the order of 2.5 km thick, whereas in the Bui belt, comparable units are approximately 9 km thick. These sediments mark a rapid period of erosion and proximal deposition during the late-stage of the orogenic cycle. They unconformably overlie the Birimian metavolcanics at the Damang mine near Tarkwa. The unconformity separating the Birimian from the overlying Tarkwaian is colloquially known as the “Great Unconformity”. (Figure 7-2) shows the generalised stratigraphy of southwest Ghana.

The Birimian sediments and volcanics have been extensively metamorphosed. The most widespread metamorphic facies appears to be greenschist, although in many areas, higher temperatures and pressures are indicated by amphibolite facies. Multiple tectonic events have affected virtually all Birimian rocks with the most substantive being a fold thrust compressional event, (Eburnean Orogeny), that affected both volcanic and sedimentary belts throughout the region and to a lesser extent, Tarkwaian rocks. For this reason, relative age relations suggest that final deposition of Tarkwaian rocks took place as the underlying and adjacent volcanic and sedimentary rocks were undergoing the initial stages of compressional deformation. Studies in the western part of the region (Milesi et al., 1992) have proposed several separate phases of folding and faulting suggesting a change in stress direction from northeast to southwest to north to south. However, a regional synthesis by Eisenlohr (1989) has concluded that although there is considerable heterogeneity in the extent and styles of deformation in many areas, most of the structural elements have common features, which (in his opinion), are compatible with a single, extended and progressive phase of regional deformation involving substantial northwest-southeast compression.

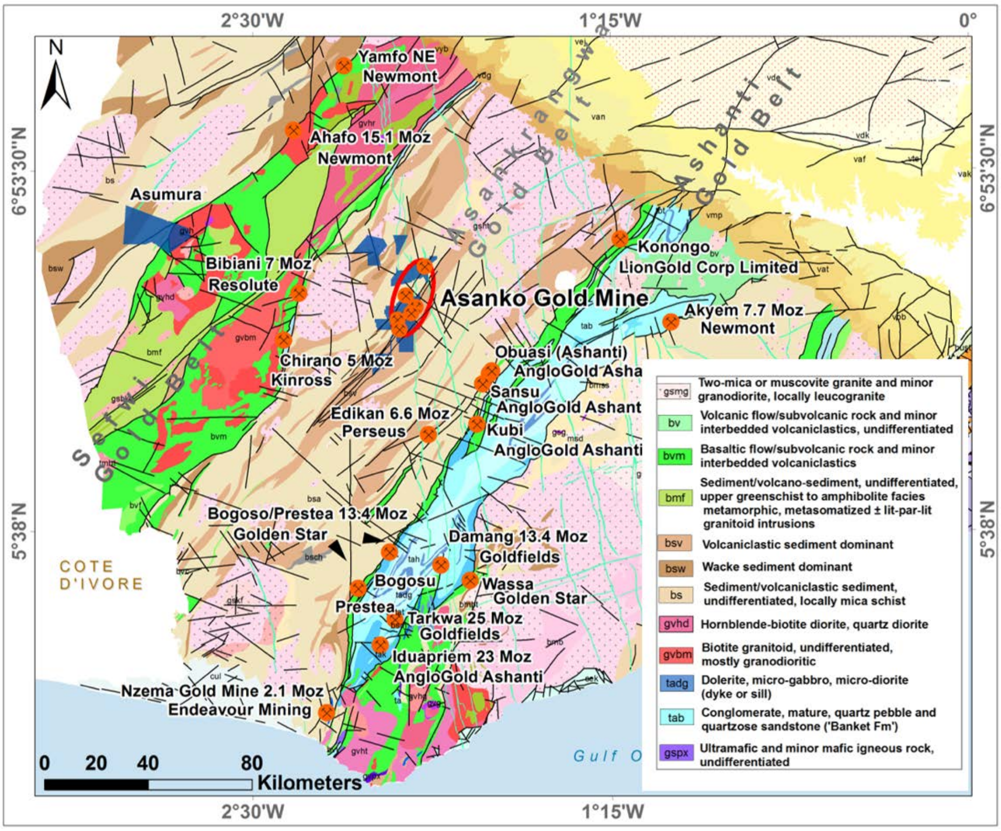


Figure 7-1: Geology of Southwest Ghana highlighting the Regional Geology around the AGM Gold Mine (Ghana Geological Survey / Asanko Gold Technical Report 2014)

Ongoing studies of the geotectonic evolution of the Birimian have involved more extensive precision U/Pb age dating (eg WAXI 2016), the results of which have significantly improved the constraints on timing of deformation as well as the timing of gold mineralisation events. This work is supporting at least two periods of basin sedimentation (2135-2116 Ma and 2105-2070 Ma) and accretionary (oblique compression) tectonics (2116 – 2105 Ma and 2070-1980 Ma), as well as two ages of gold deposition in the Asankrangwa Belt.

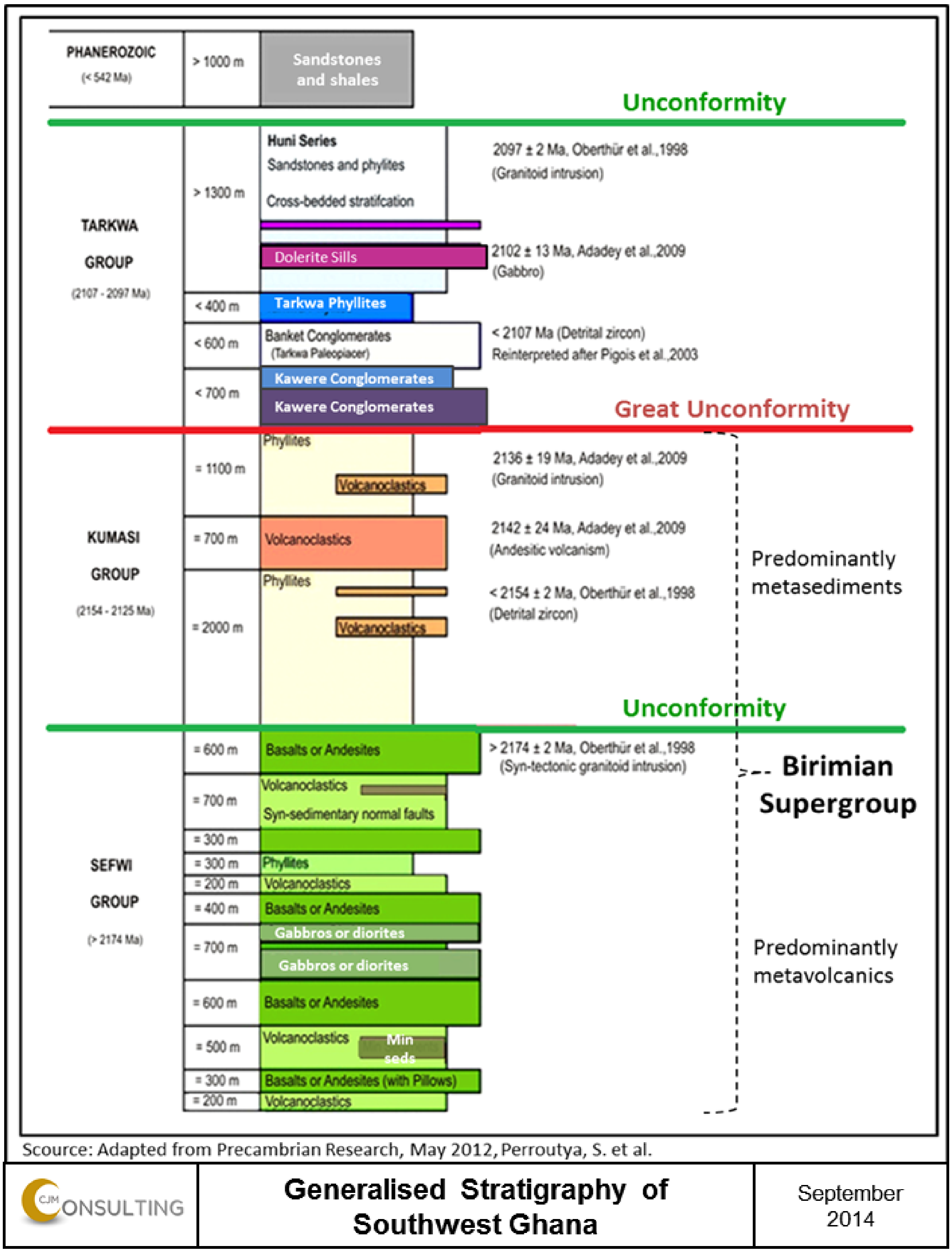


Figure 7-2: Generalised Stratigraphy of Southwest Ghana (CJM Asanko Gold Technical Report 2014, adapted from Perrouyta et al., 2012)

7.2 Local Geology - Asanko Gold Mine Environment

Asanko Gold has undertaken further airborne geophysical surveys during 2015/2016 to advance the understanding of the geological and structural settings of the Asankrangwa Belt. Although final interpretations are still being completed, the regional magnetic and VTEM data for the Ashanti Belt and adjacent Kumasi Basin provide a good indication of the distribution of the principal geological units occurring in the region (Figure 7-3). In the east, moderately magnetic mafic volcanic rocks results in a high magnetic zone corresponding to the Lower Birimian super group, and the infolded, strongly magnetic rocks of the Ashanti Belt volcano sedimentary and Tarkwaian sedimentary packages. This domain is in sharp contact with the weakly, to non-magnetic rocks of the upper Birimian metasediments, which dominate the Kumasi Basin in the west. This zone of contrast coincides with the prominent, shear zone which bounds the northwest margin of the Ashanti volcanic belt that plays host to most of the large gold deposits in the area.

Historically, it was thought that the Kumasi Basin consists of a broad, monotonous package of turbiditic sediments (Upper Birimian metasediments), extending as far west as the Sefwi-Bibiani Belt, with little intervening lithological, or structural variability. However, the presence of the Asankrangwa Gold Belt was recognised after decades of galamsey activity (artisanal mining) in gold-bearing, shear zone hosted quartz reefs. In the central portion of the Kumasi Basin newly acquired geophysical data has also provided a more in-depth view of the geological and structural setting of the area.

The Asankrangwa Gold Belt straddles two broad domains of distinct magnetic character. The western portion is characterised by the low magnetic relief that is typical of the Kumasi Basin as a whole. However, the eastern portion exhibits a strong magnetic relief, much alike to the Ashanti Belt further east. Indeed, the only portion of the known Ghanaian stratigraphy that produces such a characteristic magnetic fabric is the infolded package of Lower Birimian metavolcanics and Tarkwaian metasediments that defines the Ashanti Belt.

The magnetic intensity of the eastern Asankrangwa areas is not as pronounced as that of the Ashanti Belt, but it is interpreted to represent the same rock package on the basis of its distinct magnetic fabric.

A sharp north-east trending break separates the two distinct magnetic domains of the Asankrangwa Belt. It also truncates the dominant east northeast to west southwest trends typical of the eastern domain. Evident, dramatic changes in the structural grain in the area indicate the presence of a major shear zone separating the two domains. This interpretation results in the Upper Birimian meta-sediments of the western domain occurring in the hanging wall of the shear zone, and above Lower Birimian meta-volcanics of the eastern domain which occur in the shear zone footwall. This arrangement of ‘younger-over-older’ supports a longer and intense thrusting history on the shear zone.

Notably, the Obuasi deposit in the Ashanti Belt is hosted by an identical structural-stratigraphic configuration i.e. within the Upper Birimian metasediments, structurally above the Lower Birimian metavolcanics.

One of the structural setting interpretations of the Asankrangwa Gold Belt that explains these relationships is an inverted half-graben, in which growth faulting controlled accumulation of the upper Birimian metasediments, above the Lower Birimian metavolcanics in the foot wall.

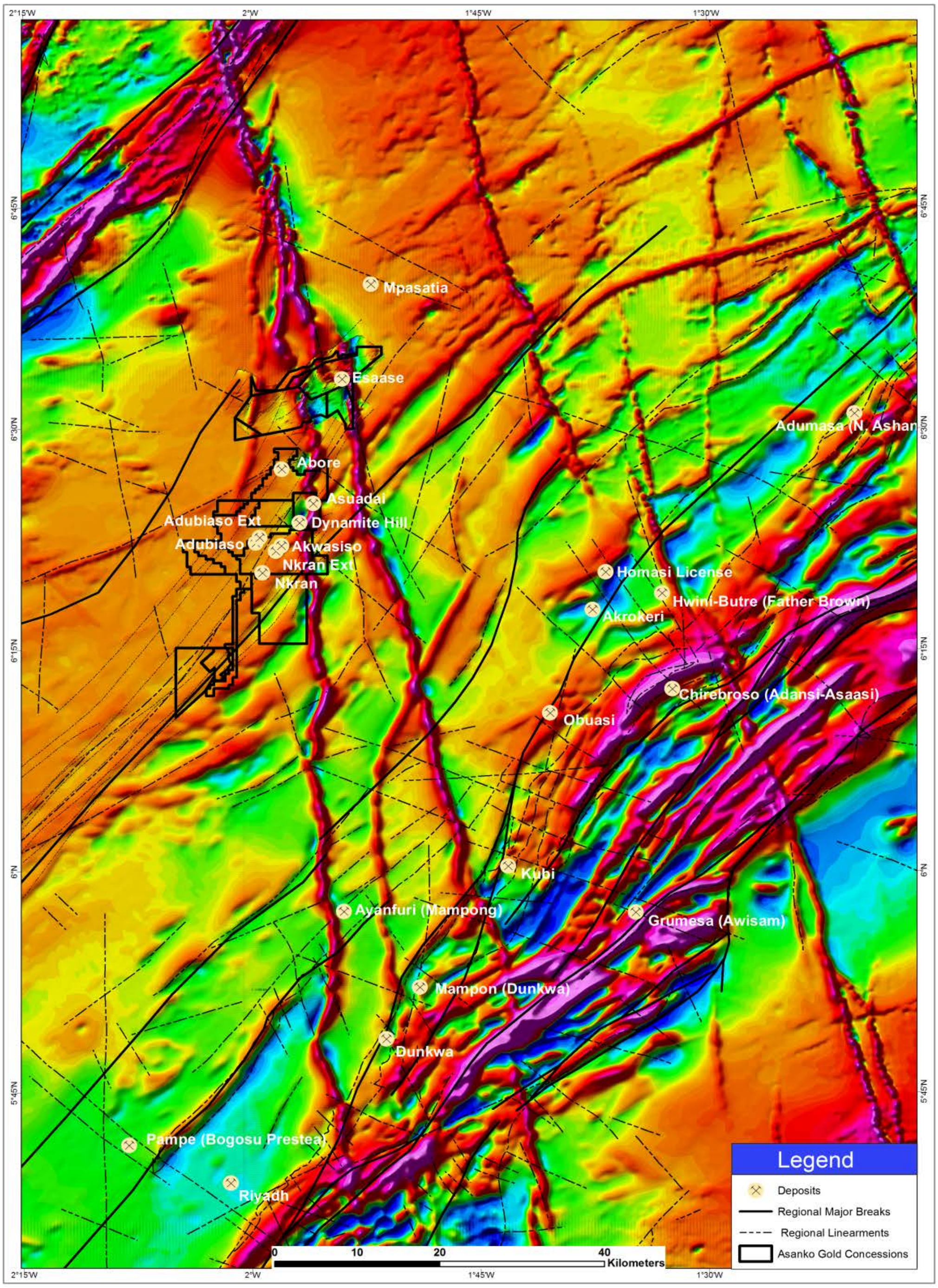


Figure 7-3: Regional Total Field Aeromagnetic Image of the Ashanti Belt (east) and adjacent Kumasi Basin (West) (Source of Image Ghana Geological Survey)

Basin inversion and/or exhumation of the Birimian metasediments during deposition of the Tarkwaian sediments is consistent with their occurrence above a regional angular unconformity, and predominantly within fault-bound basins. The "Asankrangwa Shear Zone" is a fundamental crustal structure that has repeatedly been active throughout the entire geological history of the region, from initial rifting, right through orogenesis (northwest-southeast compression, northeast-southwest left lateral shearing).

The regional magnetics also indicate the presence of a set of subtle eastwest trending structures. One of these structures forming the southern boundary of the Birimian metavolcanic / Tarkwaian package in the Asankrangwa Belt passes through the southern Ashanti concessions and is continuous with a regional east-northeast trending structure that may be traced to the Obuasi deposit, where it roughly coincides with a major right-stepping flexure in the Ashanti Belt (Figure 7-3).

This major east-north-east trending structure could be interpreted as a transfer fault, originally associated with the formation of the Kumasi Basin. The parallel, but less extensive, east-west structures that pervade the Asankrangwa Belt in the vicinity of the Ashanti II concessions, may be splays from this same regional-scale transfer fault, or independent transfer faults in their own right.

7.3 Property Geology

7.3.1 Introduction

An exercise was initiated in 2013 to produce 3D litho-structural models for all of the AGM deposits. This was intended to increase geological and structural understanding of the AGM deposits, as well as introduce proper geological and structural controls into the updated MRE. The process involved geological and structural re-logging of drill core, interpretation of historic flitch diagrams, and use of Leapfrog® software to produce the 3D litho-structural models.

Asanko Gold initially utilised HMM Consultancy (“HMM”) to create the 3D litho-structural models for all the Obotan deposits, with the exception of the Adubiaso deposit, where Optiro created the models. The sections below summarize and describe the findings of these reports. These models are currently being updated with recent drilling data and integrated into Micromine 3D modelling software.

In addition to the creation of the 3D litho-structural models, Asanko Gold initiated a prospectivity mapping analysis of the Asankrangwa Belt. This exercise provided a basis to collate available regional geophysical and geological data, as well as drilling and geochemical survey information. An important outcome of the regional and local structural interpretation was the completion of a revised property geological map (Figure 7-4).

During 2016 Asanko Gold completed a full (72 borehole) relog of the Nkran deposit, and subsequently updated the 3D litho-domaining used for the Nkran MRE (as at 31st December 2016).

7.3.2 Nkran

The Nkran deposit is located within the Kumasi Basin within the Asankrangwa Gold Belt. The basin is bound to the south by the Ashanti fault / shear and the Bibiani shear to the north. The Nkran deposit is located in the northern, central portion of the Obotan lease concessions. The Asankrangwa Gold

Belt expresses itself as a complex of northeast trending shear zones situated along the central axis of the Kumasi Basin.

The Nkran deposit is located on a jog within a regional shear corridor, which is a zone of about 15 km in width and may be traced on a northeast to southwest trend for a distance of some 150 km. This is one of several major northeast trending shears / structures that bisect the Kumasi Basin / Asankrangwa Gold Belt.

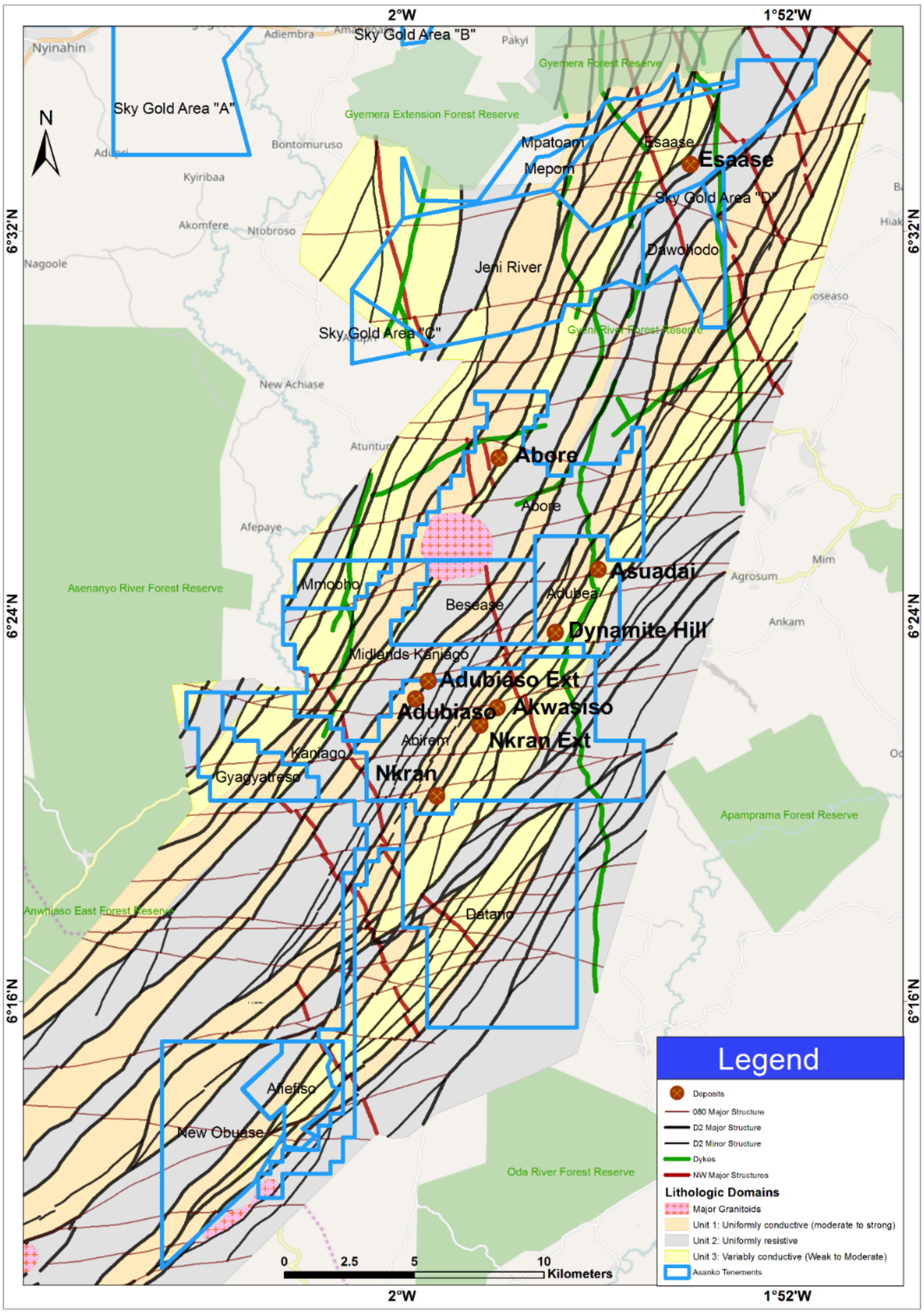


Figure 7-4: Asanko Gold Mine Property Geological Map (source: Asanko Gold)

The regional / local (Kumasi Basin) geological setting is heavily faulted and consists of an isoclinally folded sequence of metasediments, dominated by turbiditic sequences of greywackes and shales, intercalated with rare andesites and volcanoclastics, previously described as greywackes, phyllites, argillites and shales. 5 deformation events were recognised in the study conducted by Davis (March 2014), which correlate well to regional events described by De Kock et al., (2002), Allibone et al., (2002), Perrouty et al., (2012) and work conducted by Siddorn et al., (2005). Notwithstanding, very little geological work has been conducted regionally within the Kumasi Basin, with the local geology very poorly constrained or understood. Work carried out in 2005 (Siddorn et al.,) divided the stratigraphy into 6 basic units; non-magnetic turbidites, EM and Magnetically responsive turbidites with strong graphitic content, non-magnetic sediments, dominated by greywacke (minor argillite), metavolcanics (non-magnetic), metavolcanics (magnetic) and granite.

A more detailed interpretation is currently being compiled post the 2015/2016 regional geophysical surveys and interpretations.

7.4 Nkran Pit Geology

In plan, the Nkran pit covers approximately 850m in a northeast southwest direction along the strike length of the ore body, and at its widest point measures 450m across strike (Figure 7-5). The pit was mined to a final elevation of 4951 RL (Local RL) from a surface elevation of 5124 RL (Local RL; Brinckley, 2001a). The general geology of the Nkran deposit is shown in the map. The main rock types at Nkran pit consist of thinly bedded greywacke and thickly bedded to massive sandstone, phyllite and carbonaceous shale. The distribution of the geology is best shown on production maps and comprises of phyllite dominated domains on the western and eastern sides, separated by a core of wacke-sandstone dominated sediments (Brinckley 2001, Standing 1998 and McCuaig et al., 2002). The metasediments have been isoclinally folded and sheared, and generally dip steeply to the north-west at between 70º to 80º, with a steep 70º southerly plunge. Intruding the metasediments are two granitic (tonalite in Resolute data) intrusions. The granite is largely restricted to the northeast portion of the pit, with isolated pods of granite in the southern portion. Granite is present at depth in the south end of the pit. The recent re-opening of the Nkran deposit has provided extensive in pit exposure. The granites intrude structures marked by a stratigraphic discordance, and are variably sericite altered. A weak metamorphic aureole exists and is characterised by localised hornfelsing immediately at the intrusive contact with sediments, as well as nematoblastic andalusite within the carbonaceous shale horizons. Of note is that the granites are post D1-D2, and yet host a brittle vein style of gold mineralisation.

Figure 7-5 and Figure 7-6 are updated geological plans derived from a detailed re-log of all the PMI diamond core drilled under the Resolute pit.

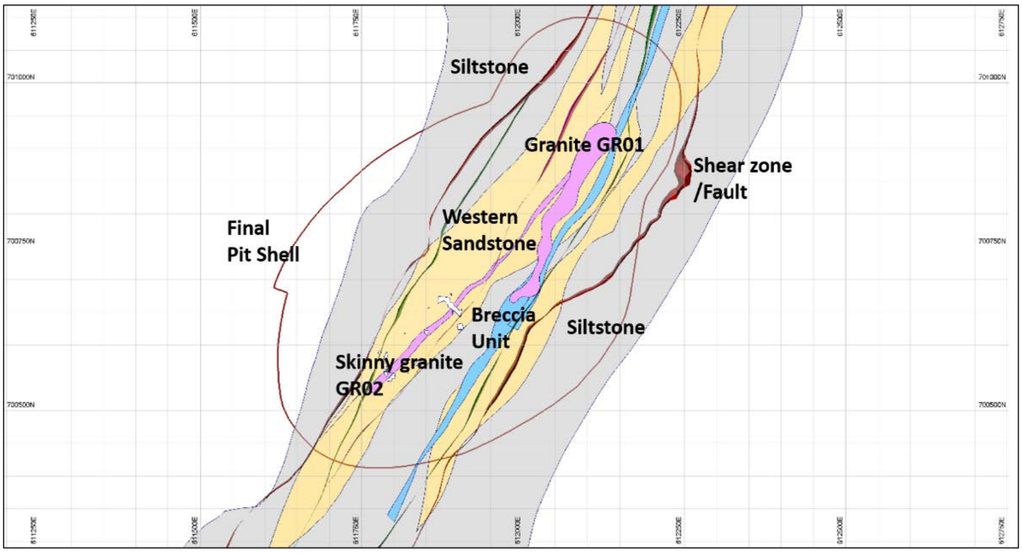


Figure 7-5: The Nkran Pit Geology, Asanko Gold Mine 2016 (source: Asanko Gold)

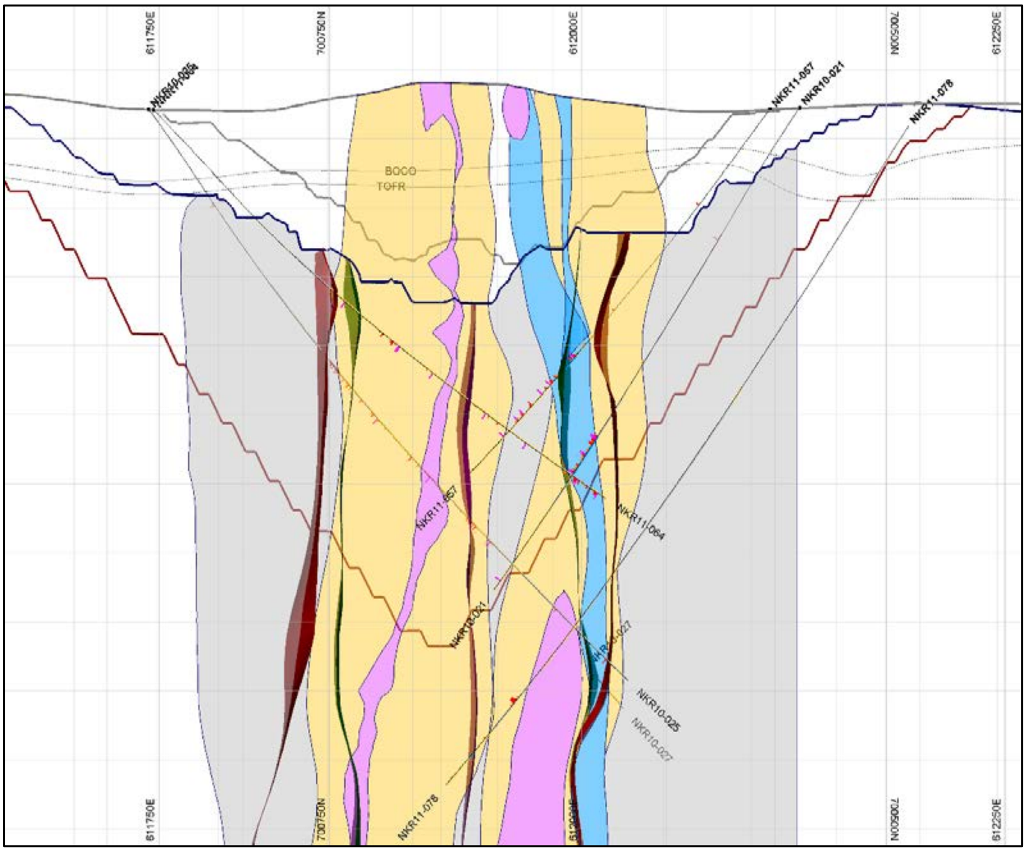


Figure 7-6: Nkran cross section at 10400N, looking north. Same rock types as labelled in Figure 7-5 (source: Asanko Gold)

The regional stratigraphy trends northeasterly, while in the middle of the Nkran pit the stratigraphy trends north. The stratigraphic discordance in the centre of the pit correlates with (1) the southern extent of the granite intrusion (GR01) at upper levels, and (2) the presence of sandstone-dominant stratigraphy. Phyllites locally are observed to splay and merge along strike, and are mapped as being sheared fine grained silts and sandstones. These phyllites mark zones of higher strain (shear zones) within a more competent sandstone dominant package and granite. Duplex structures present through the centre of pit, cut the GR01 granite and repeat the sandstone-dominant stratigraphy along sheared phyllites and granite contacts.

Generally, in the country rock, the base of full oxidation fluctuates between 5059 to 5071 mRL and the base of transition fluctuates between 5035 to 5047 mRL. The weathering gradient runs from freshest in the northeast to most weathered in the southwest at the specified reduced level.

7.5 Nkran Structural Interpretation

A number of structural studies have been undertaken on the Nkran deposit, initially whilst being exploited by Resolute, and subsequently post the PMI diamond drilling under the flooded Nkran Pit. The orientation of the PMI Nkran core has proved problematic, even though it is claimed to have been orientated. Asanko Gold has conducted relogs of 35 (in 2014) then all 72 boreholes (in 2016) informing the Nkran geological model. This exercise has enabled much needed parity in the rock description and lithocoding, as well as more structural interpretations. The 2016 Nkran model is now analysed in 3D in Micromine and Datamine. The 2016 relog geological model has been used to inform the litho domains for the current MRE for the Nkran Pit. The outcome of these studies currently suggests a simplification into 3 predominant deformation / geotectonic phases.

7.5.1 Closure of the Kumasi Basin through NW-SE Compression

7.5.1.1 D1 Northwest – Southeast Shortening

The earliest phase of deformation that can be recognised is that of northwest southeast shortening, resulting in a north-east trending, steep northwest dipping stratigraphic package. This early deformation fabric has been termed S0/S1 due to bedding being approximately parallel to foliation. Original bedding fabrics (notably younging directions) are rarely observed when not overprinted by D2 (S2) fabrics. Younging directions are observed to oscillate between west and east, indicating near isoclinal folding. This fabric correlates with the regional D1 event seen throughout Ghana.

7.5.1.2 D2 West Northwest – Southeast Shortening

The second observable deformation event, resulting from a rotation of the regional stress field to a west northwest orientation, and resulting in subsequent north northeast trending, steep north west dipping foliation that cuts the earlier D1 (S0/S1) fabric. The S2 fabric is well developed in the shale / silt rich horizons. The major controlling structures within the pit are also parallel to the S2 fabric. D2 deformation can be correlated with the earliest (first) gold event seen within the Nkran deposit

7.5.2 Change / Rotation in Stress Field resulting in SW-NE Compression

7.5.2.1 D3 Southwest – Northeast Shortening

The third deformation event that can be recognised is as a result of southwest northeast shortening that manifests itself as an oblique west northwest trending crenulation cleavage. This crenulation cleavage cuts and folds the S2 fabric. The orientation of this S4 fabric is similar to that of the late D5 mineralisation event. However, importantly, the two are separated by the intrusion of the granitic bodies within the central and southern portion of the pits, which are un-foliated, but does contain D5 related mineralised quartz carbonate veins that appear to be conjugate sets.

7.5.3 Change / Rotation in Stess Field resulting in NE-SW Compression

7.5.3.1 D4a / D4b – Northeast – Southwest Shortening

The field relationships within the Nkran deposit suggest two phases of gold related veins over printing a pre-existing steeply dipping package of sediments adjacent to a granite intrusion that appears to be a rheological control on focussing deformation. The first set of veins was linked to northeast - southwest shortening. The second set of phase of veins was linked to north northeast - south southwest extension. The time between the two events is not constrained (i.e. the second set of veins may represent an orogenic collapse event after the northeast southwest shortening linked to the first phase of veins, or it could be a completely separate event).

7.5.4 D5 Sinistral Movement

The final phase of deformation that is recognised in the Nkran deposit is interpreted to be a sinistral reactivation of the major shears, resulting in barren brecciated zones and barren laminated veins. The orientation of the shears and formation of the breccia and bucky / laminated quartz veins is consistent with northeast - southwest compression and the formation of breccia in dilational jogs in the central portion of the pit.

7.5.5 Adubiaso

McCuaig and Williams (2002) describe the Adubiaso geology in some detail, comprising a sub-vertical stratigraphy of inter-bedded greywacke and phyllite, with three sub-vertical granite (porphyry) dykes obliquely cross-cutting the stratigraphy.

Mineralisation at Adubiaso is split into two phases:

1) Ductile, shear hosted mineralisation, within the north-northeast striking, steeply west dipping Adubiaso Shear Zone ("ASZ"). This zone measures approximately 25m in width in the central area, thinning to approximately 6m at the northern and southern ends of the pit. It appears to be reasonably vertically and continuous in Resolute Grade Control data, however, it loses resolution within the Resolute resource and PMI exploration data.

2) Cross cutting, north-west to north northwest striking, moderately east dipping brittle quartz carbonate vein hosted mineralisation. This mineralisation cross cuts the ASZ and porphyry zones, and clearly post dates the early phase of mineralisation and can be found in the hanging wall and foot wall to the central mineralised zone. These structures appear to be spaced 35m to 60m vertically, and are readily apparent in Resolute Grade Control data."

The mineralisation in plan shows an overall north-south trend and a broadly anastomosing character (Figure 7-7). The undulations in the grade outlines are considered to correlate with interpreted northeast to southwest striking shears. Davis (2014) terms these as the "C-shears", and they appear to off-set (in a dextral sense) the lithology and mineralisation to differing degrees. The overall movement is on a metre to tens of metre scale, with small off-sets noted in the geological modelling.

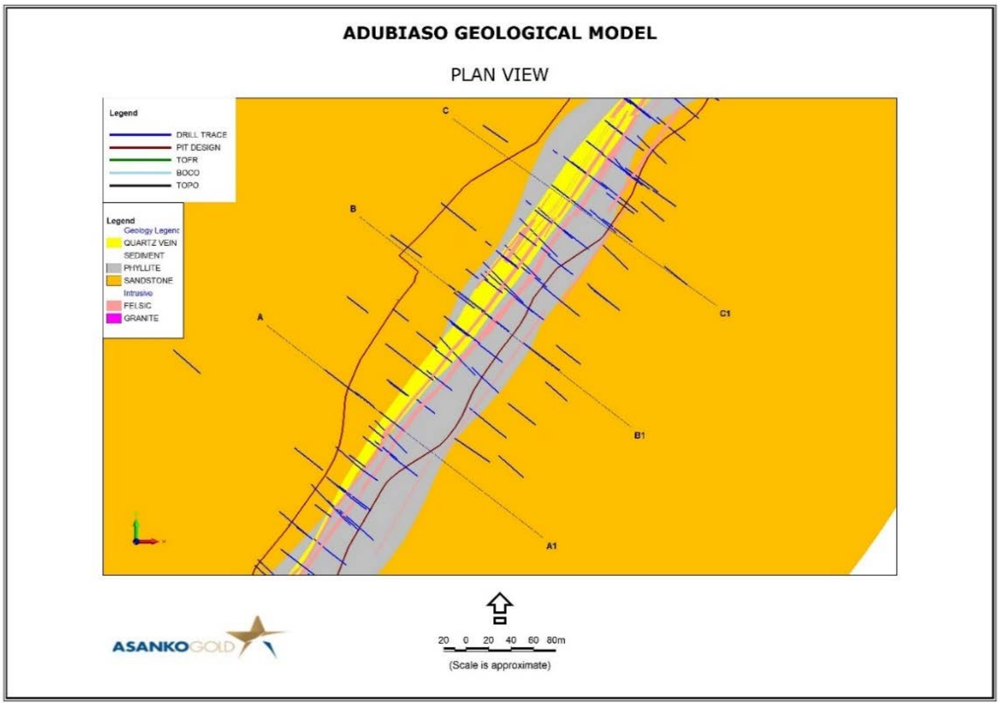


Figure 7-7: Plan view of Adubiaso Geology (source: Asanko Gold)

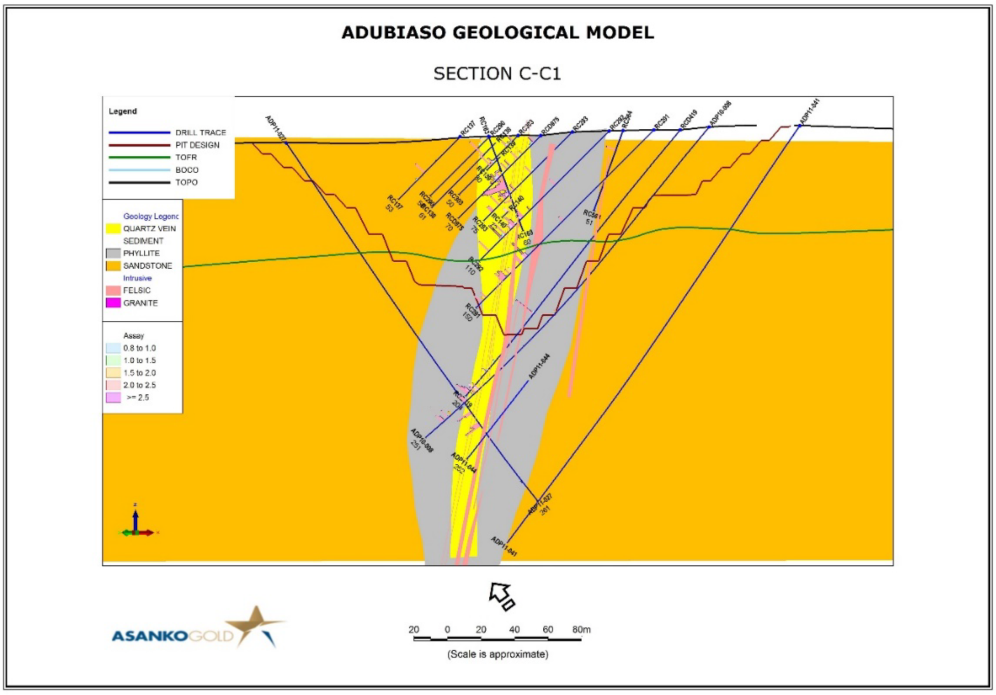


Figure 7-8: Cross Section C-C1 of the Adubiaso Orebody looking North (source: Asanko Gold)

7.5.6 Adubiaso Extension

PMI originally tested the northeastward extension of the Adubiaso Main mineralisation during 2011. Asanko Gold, post consolidating the Kaniago Midlands PL to the north, continued testing the extension during 2016 with a 3,413m RC programme. The exercise was partly successful in terms of tracking out over 500m of strike of multiple thin mineralised zones (Figure 7-9), which have subsequently been infill drilled. The geology is similar to that described above for the Adubiaso main pit, that Resolute exploited between October 1999 and July 2001.

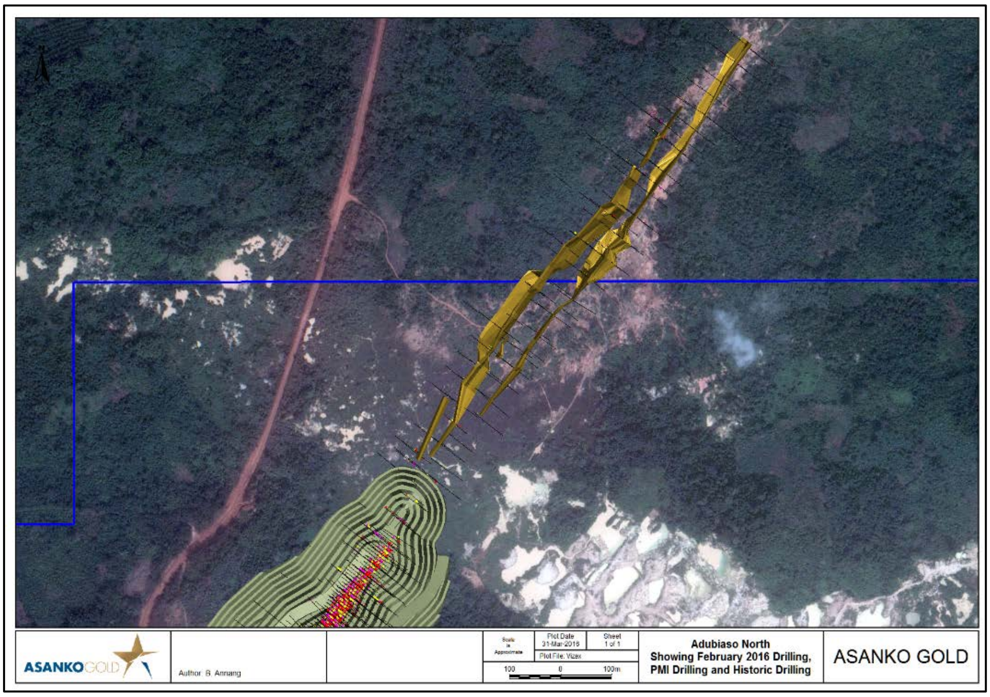


Figure 7-9: Plan view showing Adubiaso Extension Mineralised Zones (source: Asanko Gold)

7.5.7 Abore

The main rock types observed within the Abore Deposit consist of carbonaceous shale, siltstone (phyllite), thinly bedded wacke and thickly bedded sandstone. This sedimentary sequence has been intruded by a granitic (tonalitic) intrusion. For the purpose of the development of the geological model, the various lithologies have been grouped into the following:

- Inter-bedded Siltstone Dominant:
  - The thinly bedded siltstone and shale dominant (with a minor inter-bedded wacke component) is the principal geology domain on the western portion of the deposit. This forms the hanging wall host sequence to the granite intrusion. There are reported inter-beds of carbonaceous shale on both the hanging wall and foot wall to the

for the metallurgical recovery of gold

granite intrusion. It is unknown if this created any preg-robbing issues for the metallurgical recovery of gold

- Inter-bedded Sandstone Dominant:
  - o The foot wall sandstone and greywacke inter-bedded sedimentary sequence is the principal lithology on the eastern portion of the deposit. This forms the foot wall host sequence to the granite intrusion. The sediments dip steeply to the northwest between 70° to 85°
- Granite Intrusion:
  - o An elongate granite (tonalitic) intrusion has intruded parallel to the main lithological domain boundary. The granite, which is foliated, dips steeply to the northwest. The plagioclase-quartz granite has a medium to coarse grained texture. The granite has been boudinaged, creating discrete individual boudins of granite over the strike length

Dyke:

A late cross-cutting west-east striking dyke features in the northern section of the deposit. The dyke, interpreted from airborne magnetic data would post date mineralisation and would be barren of gold mineralisation. The dyke would have most likely intruded along a regional west east structural feature. The thickness of the dyke is unknown and for the purpose of the geological model has been inferred to be 15m to 25m.

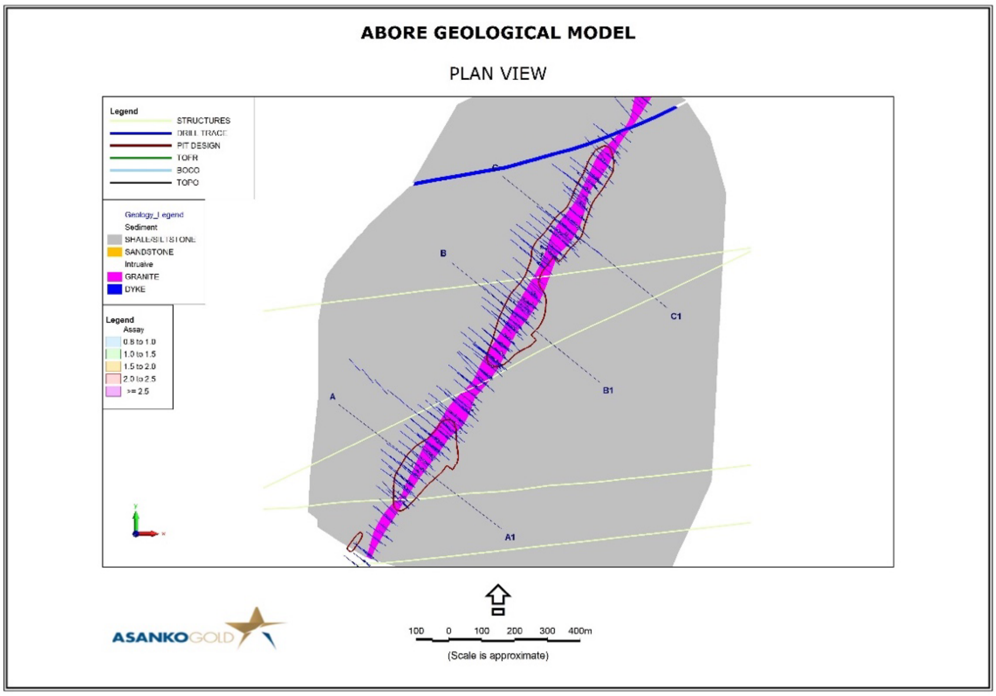


Figure 7-10: Plan View of the Abore Deposit (source: Asanko Gold)

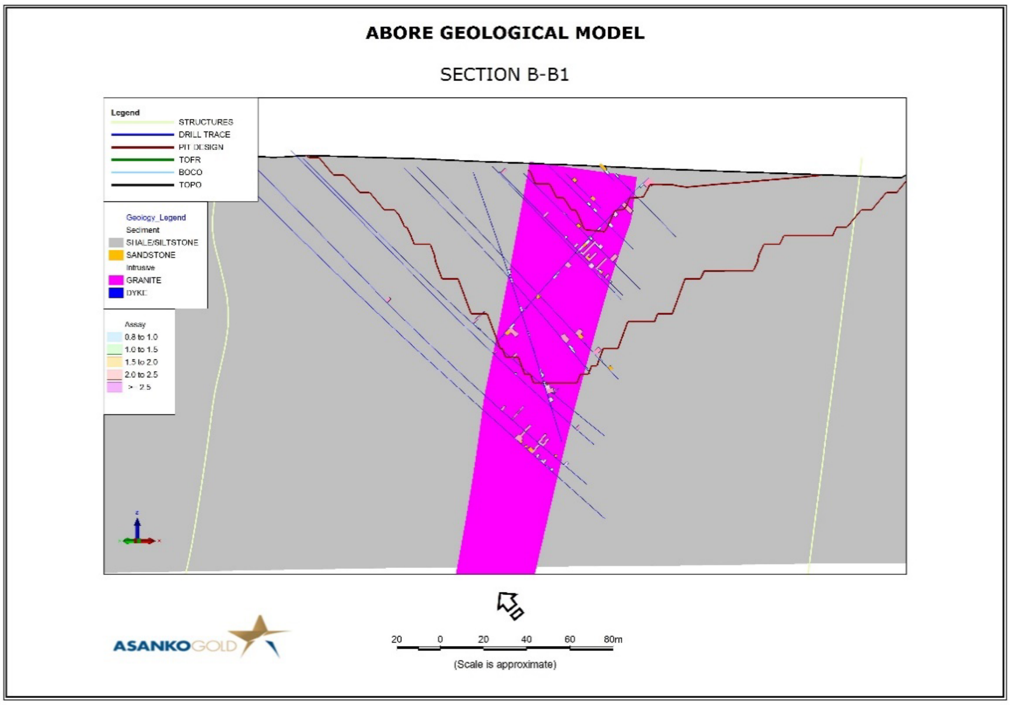


Figure 7-11: Cross Section looking North of the Abore Deposit (source: Asanko Gold)

Several foliation and bedding relationships may be observed at the Abore deposit. The most common feature is a pervasive foliation that is well developed in the fine grained siltstones and shales and to a lesser extent in the coarser inter-bedded sandstone and wacke sequences. A strong foliation is also present on both the hanging wall and footwall margins of the granite, indicating emplacement prior to deformation. The dominant foliation observed from diamond drill core was observed to strike at 040°, with a steep dip to the west (Figure 7-10, Figure 7-11 and Figure 7-12). The dip was seen to vary in places, either due to refraction of the foliation through the silt and coarser wacke units and/or subsequent folding which rotated the foliation.

The hanging wall sequence of inter-bedded siltstone, shale and wacke is significantly more deformed and foliated than the footwall sequence of sandstone with minor wacke, with several shears, trending on a bearing of 020°. These shears developed preferentially within the carbonaceous shale-rich units.

The presence of large-scale folding within the Abore pit is supported by the observation of opposing foliation / bedding relationships within drillcore. At least two, and potentially three, phases of compressional deformation may be recognised. Younging directions are seen to oscillate between west and east, which indicates near isoclinal folding, with bedding and the 040° trending foliation being seen to be close to parallel. This earliest phase of deformation (folding) results in the northeast trending steep northwest dipping stratigraphic package. This macro-scale fabric provides the dominant grain which controls the distribution of the lithological units.

The second notable folding event is a north-northeast trending, steep north-westerly dipping foliation which cuts the earlier 040° trending fabric. The north-northeast trending foliation is expressed as north-northeast trending structures which have developed within carbonaceous shale units, often transposing the unit into a north northeasterly orientation. Several of these structures are recognisable within the pit. These structures appear to be spatially associated with high grade gold trends.

Previous work conducted by McCuaig et al. in 2002, suggests that these structures are sinistral, and form a conjugate pair of structures with the east northeast trending structures that cut across the pit and offset the granite. However, lack of mineralisation associated with the second (east northeast trending structures) appears to suggest they did not develop syn-kinematically. Similar structures may be observed at Nkran, which show similar north northeast trending brittle structures which are now known to control high grade gold mineralisation within the pit.

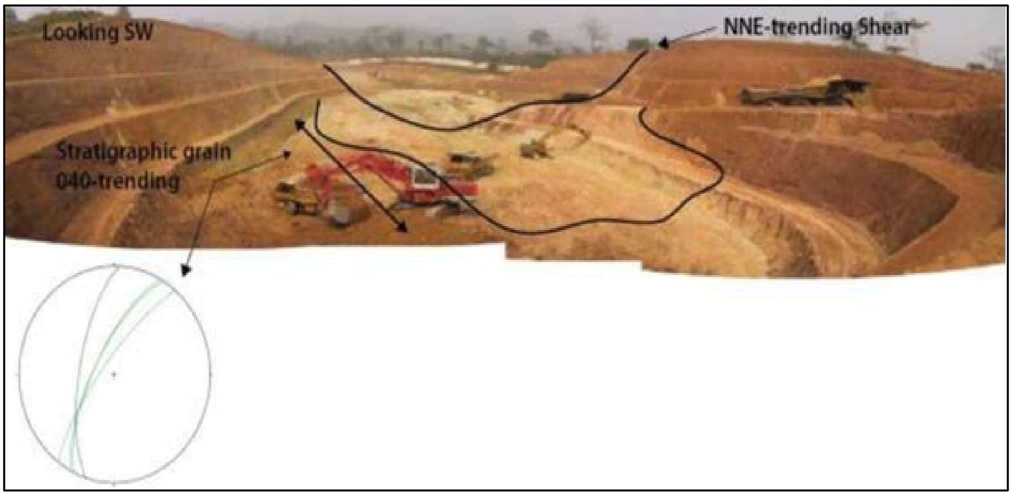


Figure 7-12: Abore Pit 2002 SW View. Pale Grey Granite in Centre of the Pit, NNE trending Siltstone-Shale Rich Units Cutting the Pit (source: CJM Asanko Gold Technical Report, 2014)

7.5.8 Dynamite Hill

The main rock types observed at Dynamite Hill consist of thinly bedded carbonaceous shale, siltstone (phyllite), and more thickly bedded wacke and sandstone. A granitic intrusion exists and is observable in core and outcrop. It intrudes the meta-sedimentary sequence along the Nkran Shear, near the centre of the prospect. Locally, porphyroblasts of possible andalusite are recognised within slaty horizons. These horizons are associated with thermal aureoles and with contact metamorphism against the granite intrusion. This rock has a distinct medium to coarse grained and crowded porphyritic texture. It is extensively altered and veined. Emplacement is clearly post the main compressive phases, and the quartz veining transgresses both granite and surrounding deformed metasediments.

The prospect may be broadly sub-divided into two sedimentary domains or sequences which are: a) the western zone, and; b) the eastern zone, (Figure 7-13). The western zone consists of inter-bedded wacke and siltstone, with a minor shale component, whilst the eastern domain consists of predominantly sandstones and wackes, with thick inter-beds of siltstones. The sedimentary sequences are bounded by the elongate granite and the associated 020° Nkran Shear Zone which locally consists of discrete zones of high-strain. The granite dips steeply to the west and ranges in thickness from 20m to 50m.

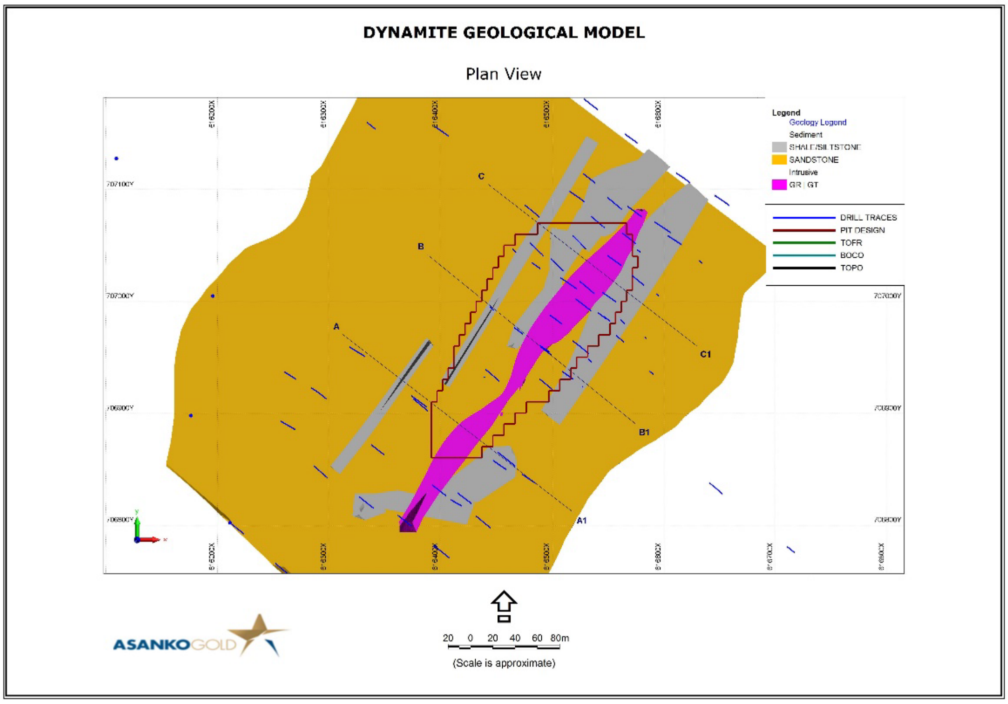


Figure 7-13: Plan View of Dynamite Hill, Narrow Granitic Intrusion (Purple) (source: Asanko Gold)

Discordant cleavage and bedding relationships of opposing vergence, are consistent throughout the Dynamite Prospect which indicates the presence of several fold structures. Two phases of cleavage development are observable at Dynamite Hill. Within the thickest packages of phyllitic rocks, this cleavage attains the intensity of slaty cleavage. The strike of the main cleavage in the exposures is at 045°. However, the dip is often influenced by slumping in outcrop. The second phase of cleavage development is developed close to the cross cutting shears (e.g. "Nitro Shear") and is orientated on a bearing of 020°. This cleavage decreases in intensity as one moves away from these zones. As one moves away from these shears, a degree of variation in cleavage orientation from 030° to 060° between the phyllite and sand-rich units is observable. This fabric is thought to be a product of refraction of the main cleavage through the incompetent phyllites, which is supported by drillcore analyses, which indicates bedding and foliation trending 040° and dipping approximately 70° to the west. The second phase of cleavage development has not been recorded in the drilling.

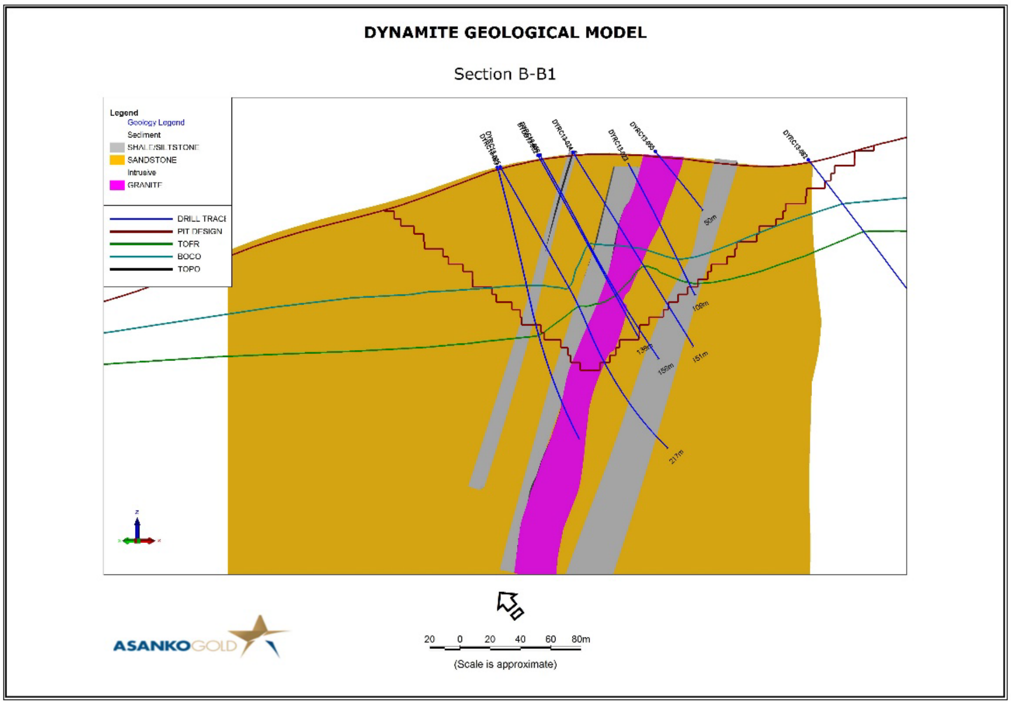


Figure 7-14: Cross Section of Dynamite Hill looking North (source: Asanko Gold)

7.5.9 Asuadai

The Asuadai deposit is located on the regional northeast trending "Nkran" shear zone, approximately 8 km along strike from Nkran and Dynamite Hill. The main rock types observed within the Asuadai pits consist of thinly bedded carbonaceous shale, siltstone (phyllite) and more thickly bedded wacke and sandstone. Two narrow granitic intrusions (diorite dykes) intrude the meta-sedimentary sequence on the boundary between the two main sedimentary domains. Extensive shearing in places associated with silica flooding, (alteration) makes it difficult to determine volcanic component of these rocks.

The general geology of the deposit may be broadly sub-divided into two main sedimentary domains which are:

- The northwest sedimentary sequence comprising interbedded wacke and siltstone (with a minor shale component)
- The southeast sedimentary sequence consisting of interbedded sandstones and wacke lithologies, with a minor shale component. The sequence is separated by a granitic dyke intruding parallel to this main lithological boundary (Figure 7-15)

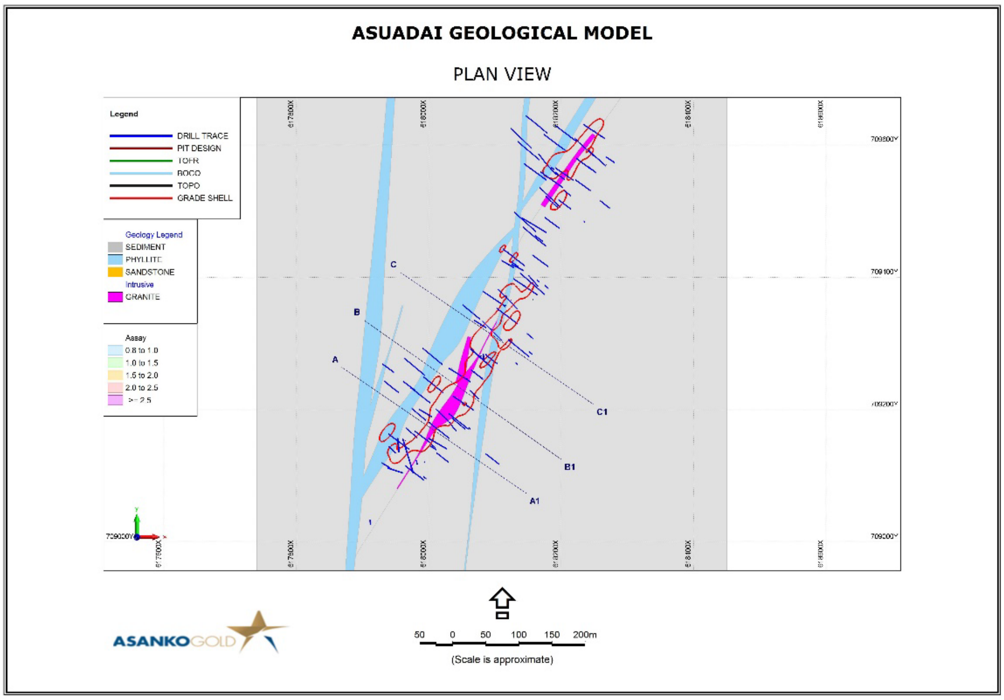


Figure 7-15: Plan View of the Asuadai Deposit (source: Asanko Gold)

Bedding trends on a bearing of 040°, with local variations, along major structures of up to 020°.The stratigraphy at Asuadai has, like Nkran, been isoclinally folded and dips steeply (approximately 70°) to the west. The stratigraphy at Asuadai is locally imbricated and transposed along major structures which trend on a bearing of 020°. It is deemed possible that these structures have potentially been mis-logged as phyllite versus a ductile shear fabric. The 020° bearing structures offset the granitic intrusions, the main sedimentary domain contact and the inter-bedded siltstones (phyllites).

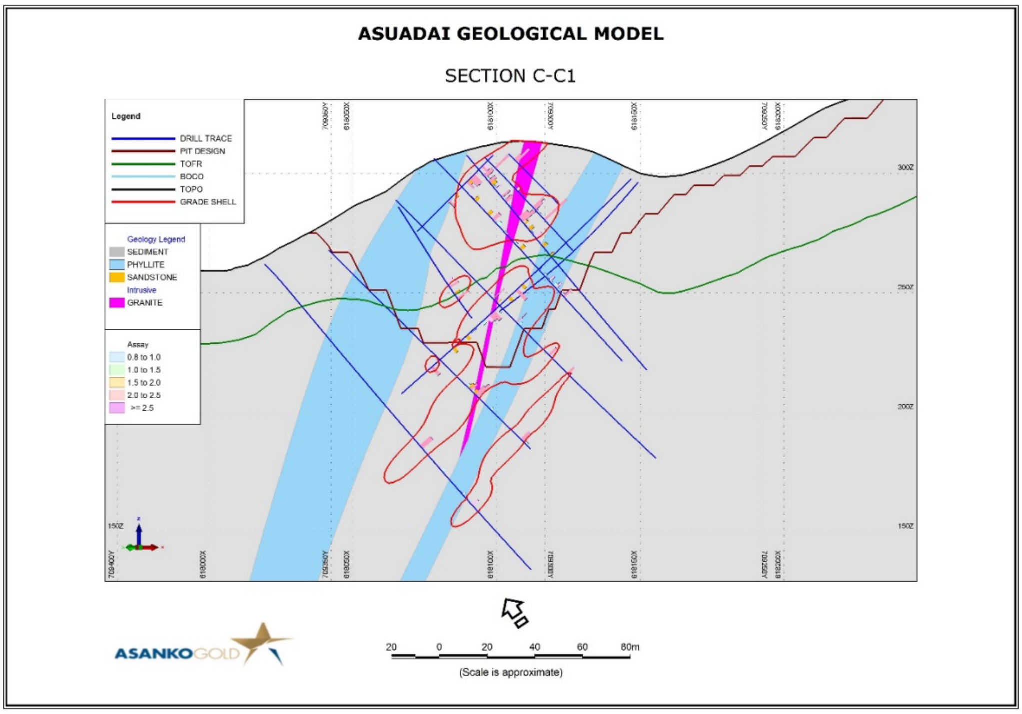


Figure 7-16: Cross Section C-C1 looking north of the Asuadai Deposit (source: Asanko Gold)

The granite forms the core of the deposit and is bounded by the two main sedimentary sequences (Figure 7-16).

It is notable that the geology of Asuadai is similar to that of both the Nkran and Dynamite Hill deposits. All of these deposits are located on a 020º trending jog on a regional 035-040º trending structure. The deposits are characterised by sheared siltstones (phyllites) dominant on the western margin and sandstone dominant on the eastern portion of the deposit. The core of the Nkran deposit also consists of a series of wacke and sandstone dominated stratigraphy that has been intruded by several granitic intrusions.

This is similar to the centre and eastern margin of Asuadai which is dominated by granite and wacke / sandstone stratigraphy. The stratigraphy at Nkran has been repeated by a series of thrusts, and subsequently transposed along these structures. It would appear a similar structural framework exists at Asuadai with several imbricating structures consisting of strongly sheared carbonaceous shale and transposed zones.

7.5.10 Akwasiso

The Akwasiso deposit is located on the Nkran shear between Nkran Extension and Dynamite Hill. Geologically it has many similarities to Nkran Main, with a granite intrusion hugging a 080o cross structure and mineralisation hosted in bounding 035°N subvertical shear structures transgressing a sandstone / siltstone sequence (Figure 7-17 and Figure 7-18). Resolute originally tested the Oxide material at Akwasiso in 2001 and then disposed of the area to Small Scale Miners. Asanko Gold managed to re-acquire the area during 2016, and have subsequently completed further RC and Diamond core drilling programmes.

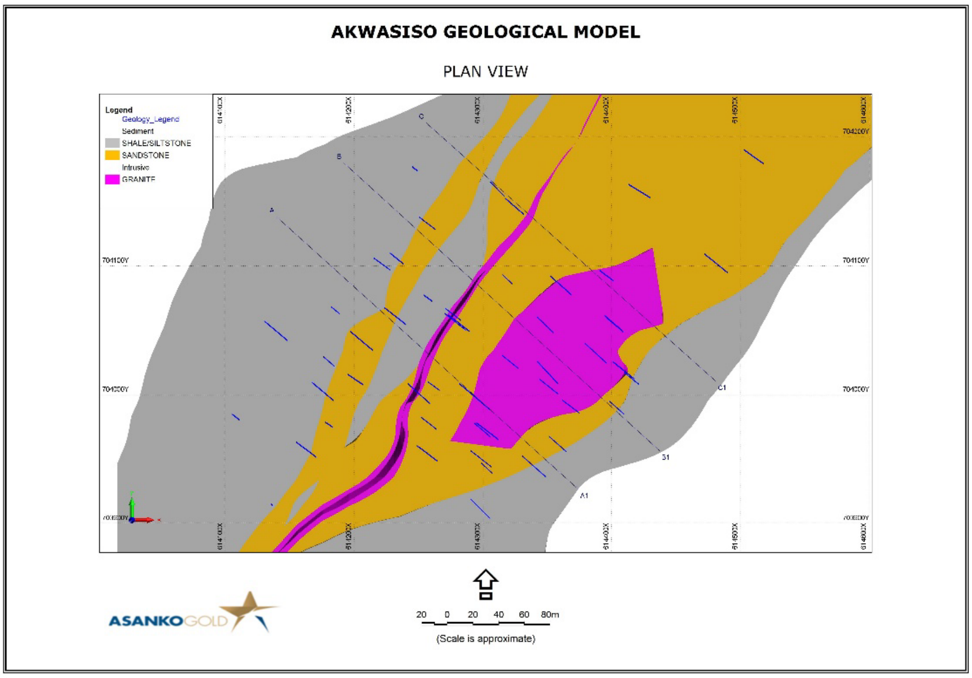


Figure 7-17: Plan View of the Akwasiso Deposit (source: Asanko Gold)

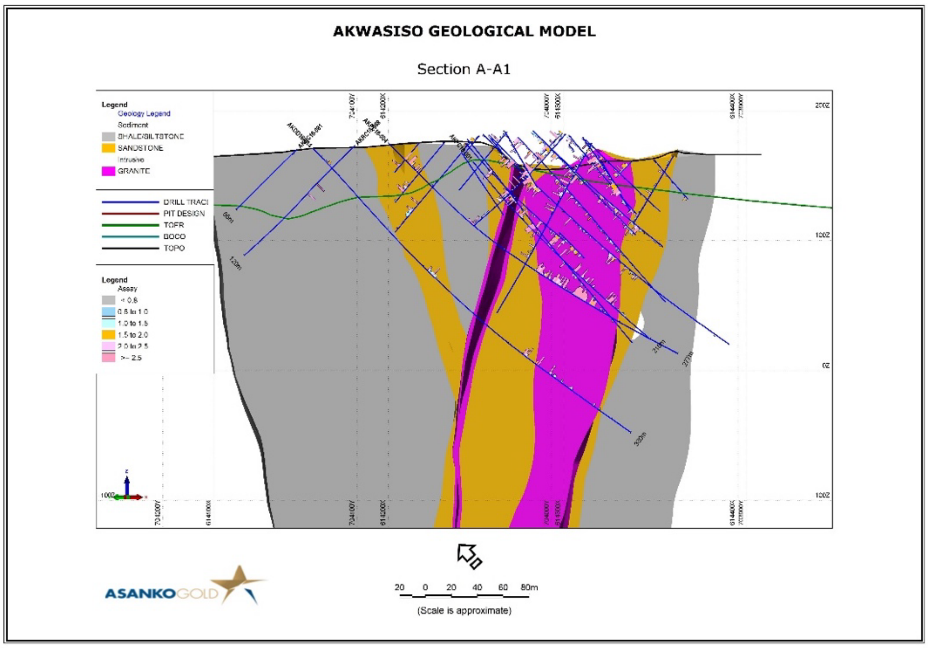


Figure 7-18: Cross Section looking North of the Akwasiso Deposit (source: Asanko Gold)

7.5.11 Nkran Extension

Nkran Extension is a north easterly trending zone on the Nkran shear, immediately NE of the Nkran pit. The zone was initially detected by sterilization drilling for the TSF. A follow-up phase of RC drilling showed generally narrow and erratic continuity of mineralisation for approximately 900 meters to Akwasiso to the NE (Figure 7-19).

The Nkran Extension zone of mineralisation is hosted within the Nkran shear, and truncates predominantly siltstone and sandy-siltstone meta-sedimentary units. Granitic intrusives are notably absent. There are commonly two mineralised structures, and at the northern end the mineralisation tenor increases where the shear splits into up to 4 discrete structures, on which a small Whittle pit has been designed.

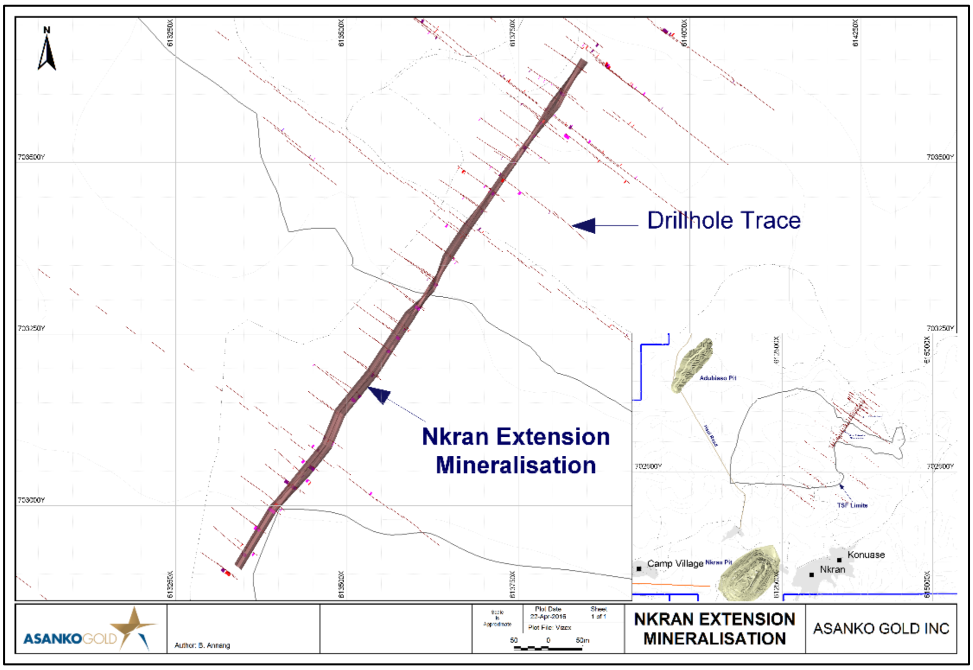


Figure 7-19: Plan View Showing the Extent of the Nkran Extension Mineralisation (source: Asanko Gold)

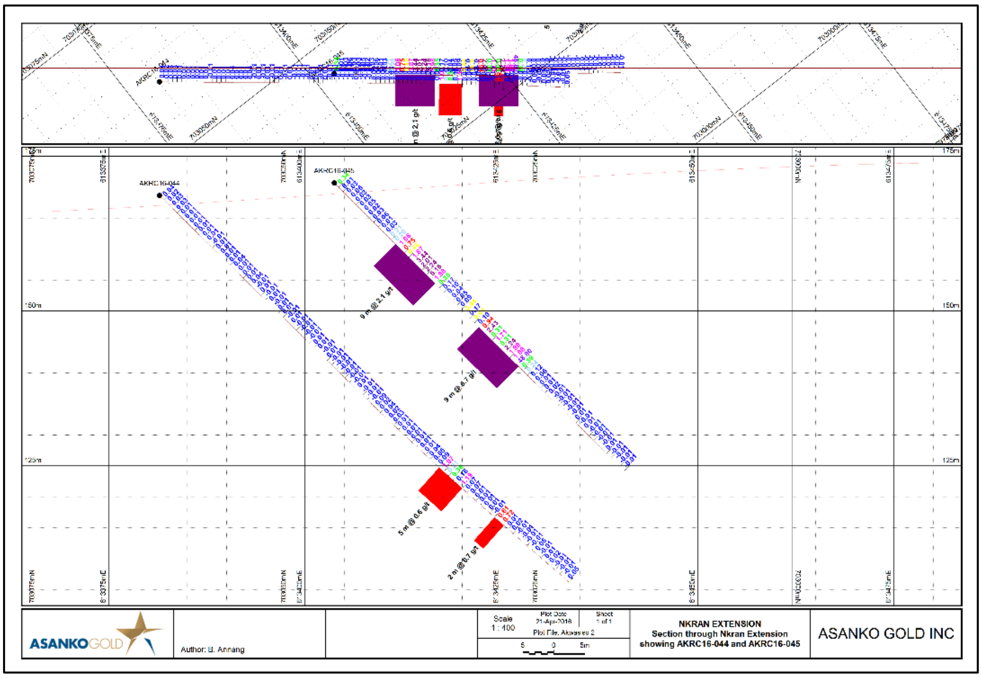


Figure 7-20: Borehole Cross Section looking North of the Nkran Extension Mineralisation (source: Asanko Gold)

7.5.12 Esaase

The Esaase project area was explored by Keegan from 2006 to 2012. Esaase contains a system of gold-bearing quartz veins hosted by tightly folded Birimian-age sedimentary rocks. Geological units on the Esaase property have been interpreted from a combination of airborne geophysical resistivity mapping (Versatile Time-Domain Electromagnetic Surveying, or “VTEM”), mineral resource definition drilling and associated outcrop mapping. The lithologies of the property may be divided into meta-sedimentary units with higher electrical and electromagnetic resistivity and units of relatively higher conductivity (Figure 7-21). Within the mineral resource zone, the host rocks may be divided between phyllite and siltstone (with substantial carbonate in matrix, with the ore zone predominant in the hanging wall of a resistivity break) and siltstone / greywacke, (predominant in footwall of the resistivity break). Host rocks to ore range from massive, thinly layered phyllite through interlayered phyllite and siltstone, to thick-bedded siltstone and wacke (Figure 7-22). Although recognisable stratigraphy appears to be present, the similarity of rock types, folding and faulting precludes correlation of individual stratigraphic units at the current stage of drilling and outcrop density.

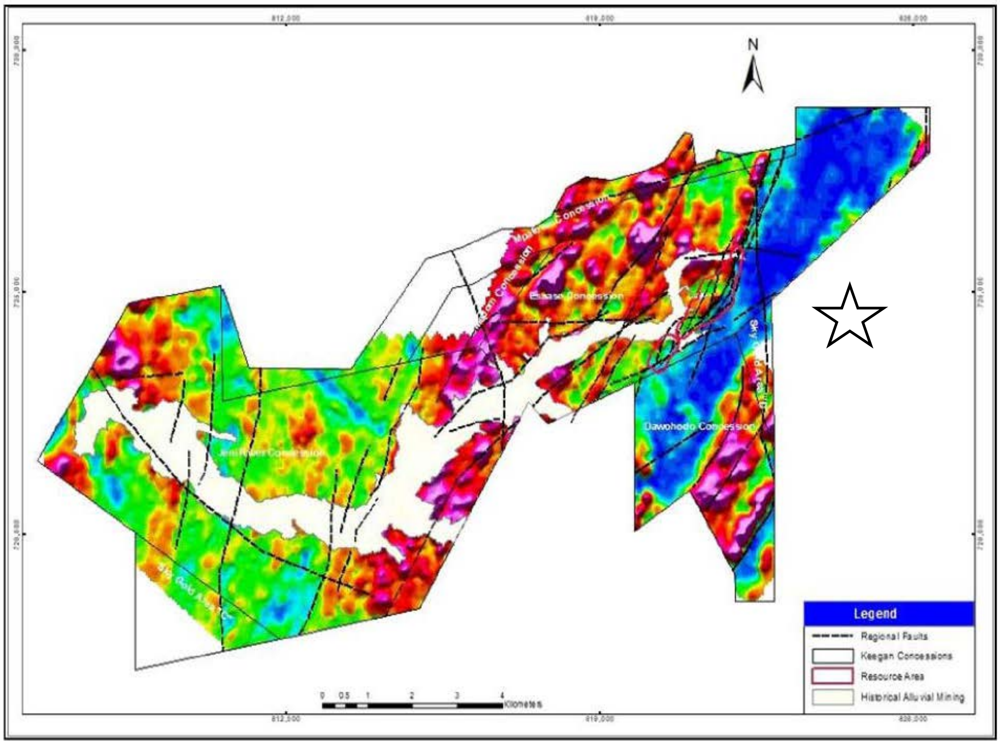


Figure 7-21: Esaase deposit with IP Resistivity, Northeast Structures and Alluvial Mining (White River Course) (source: Esaase NI-43101, 2012)

The structural architecture (Figure 7-23) is dominated by fold-thrust patterning followed by a late stage strike-slip deformation event. Open to tight, northwest-dipping folds (axial planes of the folds strike 020° to 035°, and plunge northeast 30° to 70°), are asymmetric and climb to the southeast as shear zones are approached. Folds tighten and deformation increases systematically to the southeast as shear zones are approached. This patterning repeats itself on the 10m to 100m scale. Folding in the deformed siltstone / shale package is open to tight, locally approaching isoclinal. Fold orientation ranges from upright to moderately inclined with their dip direction to the northwest. The pattern of deformation is consistent with regional interpretations of tectonic transport to the southwest. The fold limbs steepen as high strain zones (shears / thrust faults) are approached from the northwest. Within these shear zones shearing commonly shows lower, or lesser strain and repeats the pattern of low to high strain at the next shear. This pattern repeats itself at many scales (micro to macro), but for mapping purposes it is typically on the 10m to 50m scale. These northeast striking, northwest dipping syn-kinematic shears, which roughly parallel fold axial planes, appear to demarcate zones of mineralisation. In many instances, the basal shear / thrust, divides the more deformed, altered, mineralised and electrically conductive siltstone shale unit in the hanging wall from the more massively bedded and less deformed siltstone / greywacke in the footwall. It is common to see broken rock, often carbonaceous at/or near this basal contact indicating likely late brittle faulting. As fault planes cannot be measured on these surfaces, their orientation cannot be clearly determined; thus it cannot be conclusively determined whether this fault, or series of faults provide a conclusive footwall boundary. The resistivity contrast provides the best evidence for this contact on a property wide scale and consistent gold assays provide the best evidence on a sectional scale (Klipfel, 2009).

The metasediments are intruded by post, or late kinematic dykes and small stocks of intermediate to felsic composition, i.e. tonalite to granodiorite. In the southern portion of the deposit, these intrusions are intensely brecciated and mineralized, and occur at/or near the footwall of mineralisation (Klipfel, 2009). The existence of a weathering profile on the Esaase Gold Project is strongly influenced by topography. The weathering horizon is typical of tropical settings in West Africa, (Figure 7-25), consisting of ferricrete duracrust, saprolite, oxidised bedrock, and bedrock (there is often a gradational zone, “saprock” between the saprolite and oxidised bedrock). At higher elevations the laterite and saprolite, and much of the saprock, has been weathered away, leaving behind oxidised bedrock. At intermediate elevations the weathering profile is mostly intact and may be covered by transported colluvium. At the lowest elevations, the entire profile is covered by either alluvium, or residual tailings from previous alluvial operations (Klipfel, 2009).

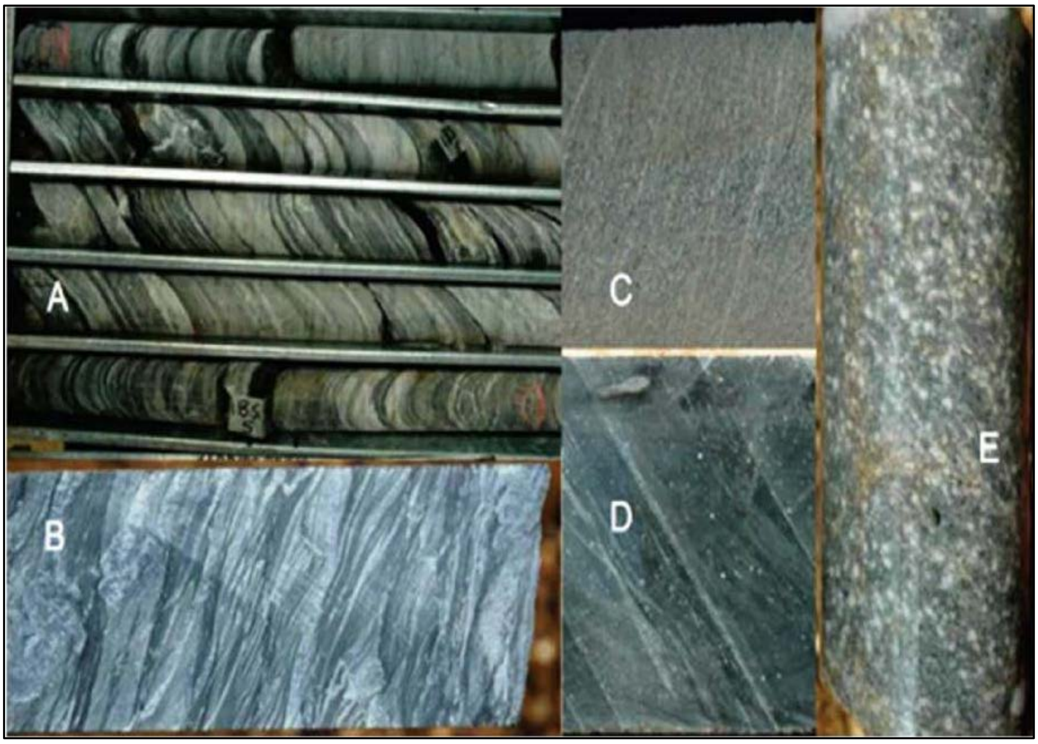
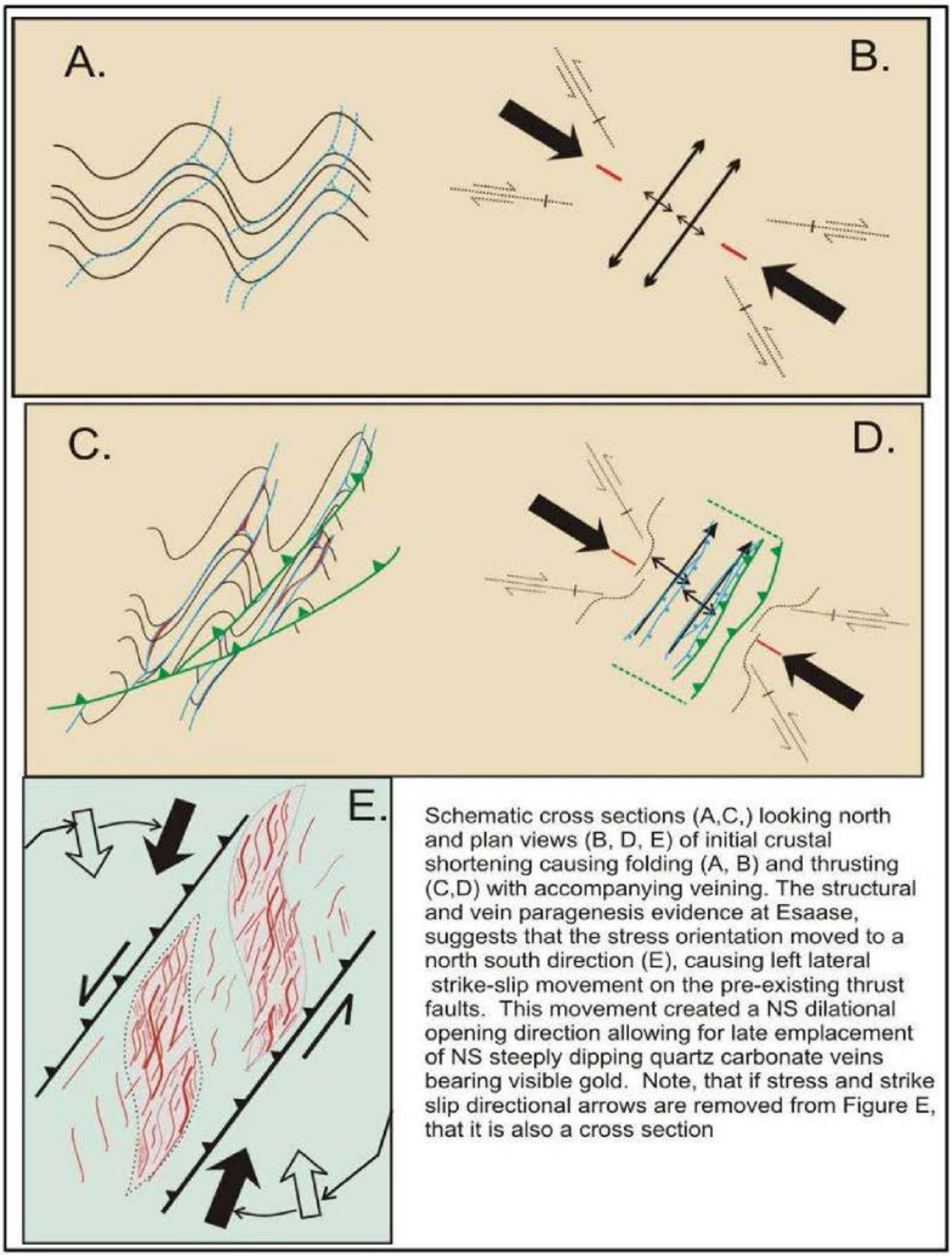


Figure 7-22: Core Photos of Lithology from the Esaase Deposit (source: Esaase NI 43-101 Technical Report, 2009)



Schematic cross sections (A,C,) looking north and plan views (B, D, E) of initial crustal shortening causing folding (A, B) and thrusting (C,D) with accompanying veining. The structural and vein paragenesis evidence at Esaase, suggests that the stress orientation moved to a north south direction (E), causing left lateral strike-slip movement on the pre-existing thrust faults. This movement created a NS dilational opening direction allowing for late emplacement of NS steeply dipping quartz carbonate veins bearing visible gold. Note, that if stress and strike slip directional arrows are removed from Figure E, that it is also a cross section

Figure 7-23: Schematic Structural Model for the Esaase Deposit and Vicinity (source: Esaase Technical Report 2009)

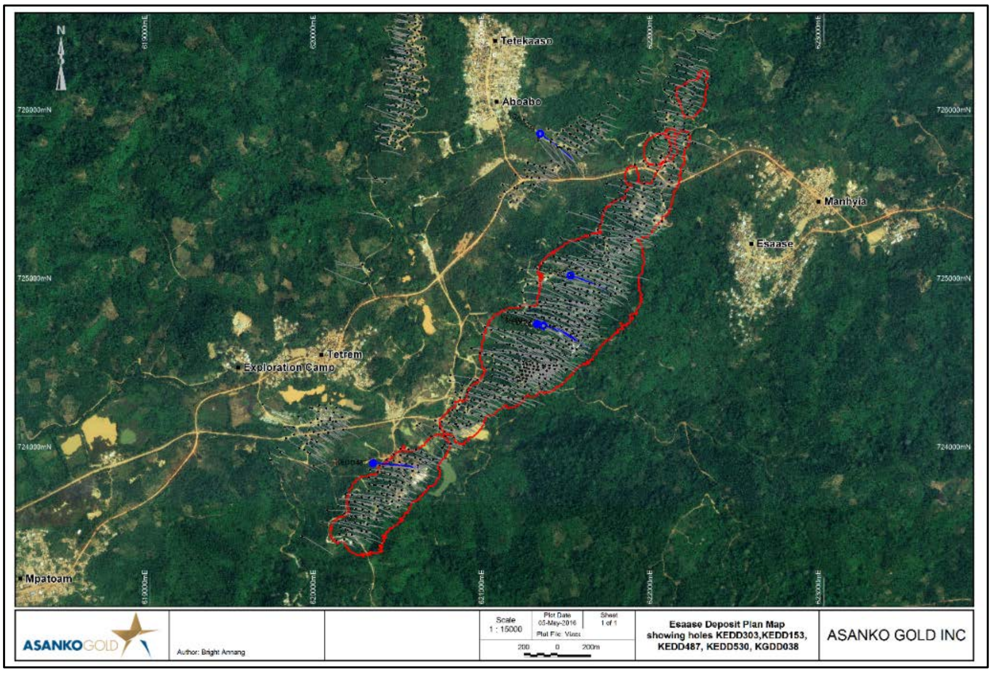


Figure 7-24: Image of the Esaase Project, Final Pit Outline and Drillhole Distribution (source: CJM Esaase Technical Report 2012)

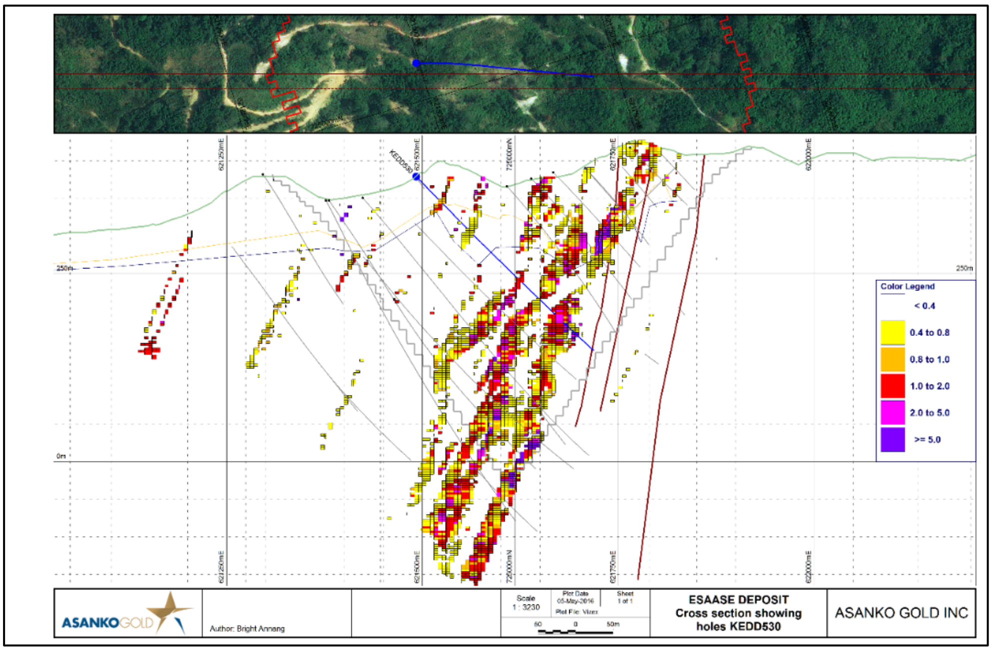


Figure 7-25: Esaase Cross Section looking North (source: CJM Esaase Technical Report 2012)

7.6 Mineralisation

This section draws from documentation by McCuaig and Williams (2002), Asanko Gold staff observations and the 2016 Nkran core relog, which involved independent structural geology consultant Brett Davis from Orefind (Pty) Ltd. There is a clear consensus that the primary controls on mineralisation at Nkran (and the Asankrangwa Belt in general) are structural in origin, and that certain sandstone units within the metasedimentary package provided favourable rheological (rock differnces) conditions that optimised gold deposition. There are at least two separate gold mineralising events that are linked to the structural evolution of the area. Mineralisation is linked to

- Early isoclinal folding, shearing and/or duplexing of stratigraphy controlling the location of deformation zones and fluid flow
- A late approximate east-west compressional event that generated shallow dipping to flat orientated conjugate vein sets that cross cut the earlier fabric and gold mineralisation
- This brittle style deformation postdates the emplacement of granitic intrusives into the core of the existing deformed and sheared sediments

7.6.1 Nkran

Economic gold mineralisation, mined in the Resolute pit and now by Asanko Gold, is associated with several lodes within the pit vis original terminology *Galamsey* Vein ("GV"), Central Vein ("CV"), Eastern Vein ("EV") and the Central Stockwork Zone (Figure 7-26)The EV system is a north northeasterly striking, sub-vertical to steeply easterly dipping zone of mineralisation. The EV marks the eastern margin of mineralisation at Nkran. The zone weakens in the north-eastern portion of the pit, where the main zone of mineralisation swings to the north and strikes across the pit. The EV strikes northeast following the margin of the granite. The GV comprises several zones in the western most corner of the pit. These zones strike north northeast, dip steeply to the east and are arrayed in an en-echelon pattern, stepping to the right in plan view and to the west down dip in section. The CV is a large quartz vein striking northeast and dipping steeply towards the east. The CV bounds zones of differing vein density in the central stock work zone. The central stock work zone is formed between the EV and GV. It is generally better developed adjacent to these structures. Veins comprise massive quartz, with strike extents ranging from centimeters to up to several meters, with widths ranging from centimeters to metres. These veins are relatively undeformed, with a two major orientations:

1) North northwest strikes and moderate east northeast dips.
2) North to south and shallow to sub-horizontal dips.

The GV and EV were recognised as major ductile shear zones overprinted by a brittle-ductile mineralising event (McCuaig et al., 2002).

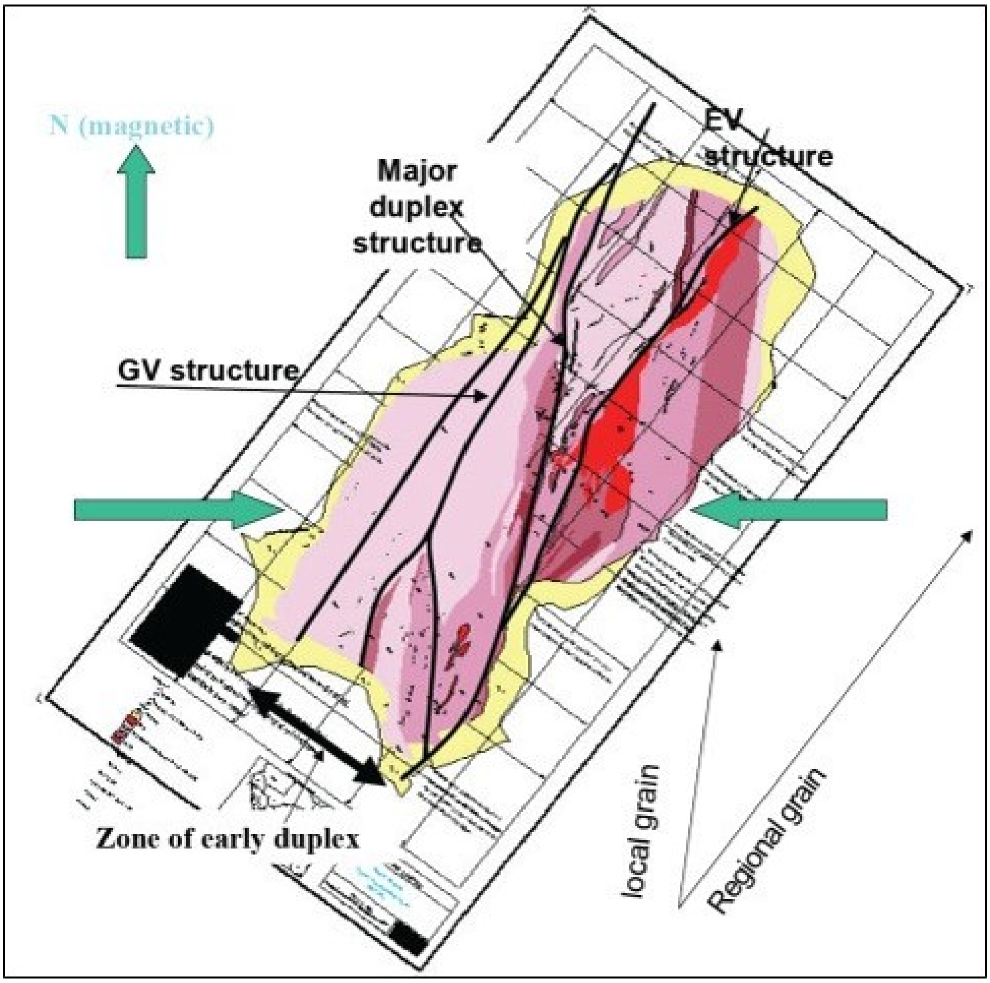


Figure 7-26: 2001 Geology Map of the Resolute Pit (Source: SRK Technical Report 2011)

During Q1 2016 Asanko Gold relogged all 72 PMI diamond boreholes that inform the Nkran MRE. This underpinned a revised geological model and lithodomaining for the updated mineral resource estimation in this document. Figure 7-27 shows the revised geological interpretation.

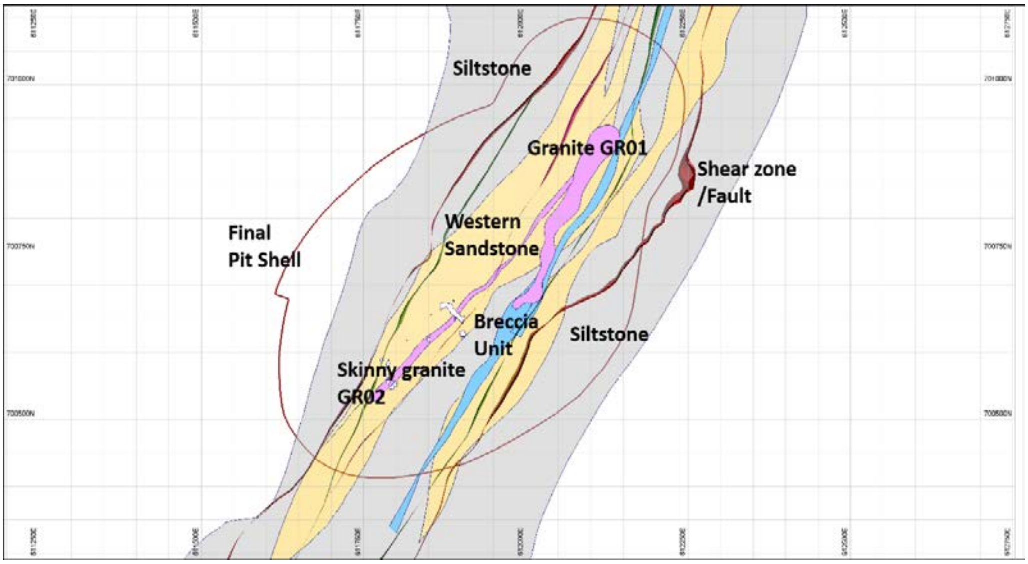


Figure 7-27: 2016 Geological re-interpretation of the Nkran Deposit (Source: Asanko Gold 2016)

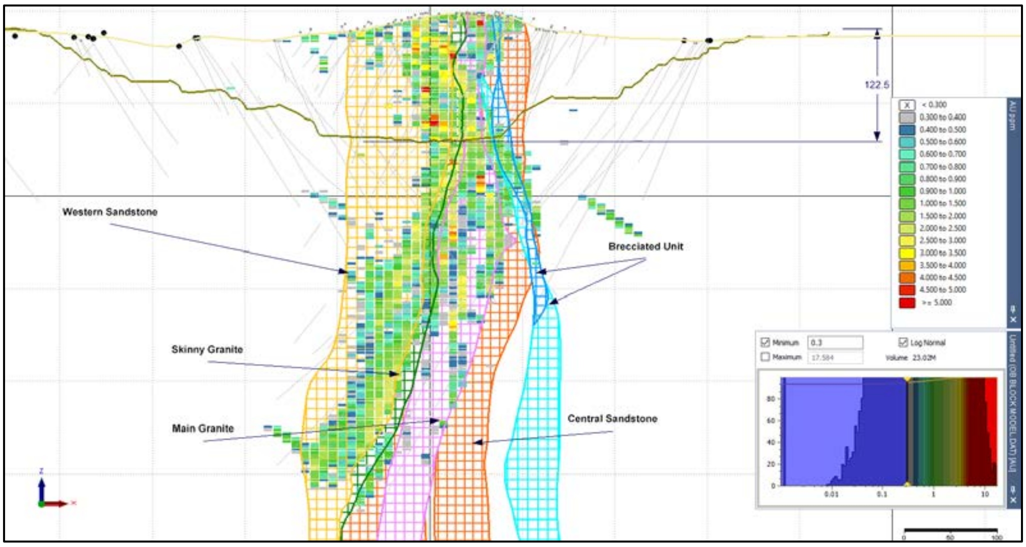


Figure 7-28: Cross section at 7800N showing Nkran Gold Distribution and Main Lithological Units (Source: Asanko Gold, 2017)

The new model included better definition of the granite intrusives and the delineation of a Central sandstone unit.

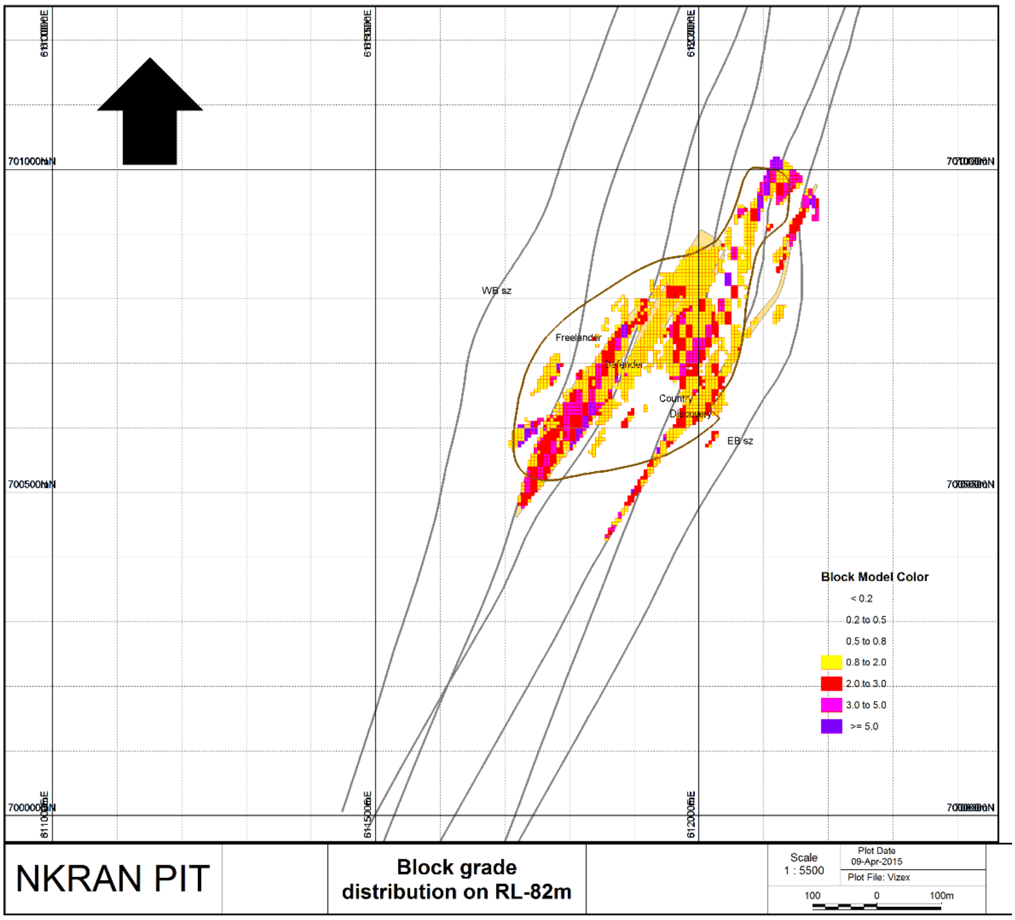


Figure 7-29: Nkran Pit plan view of RL 82 Grade Distribution (Source: Asanko Gold)

There is a very strong control on the gold mineralisation distribution by structures associated with the Western Sandstone and the Eastern Breccia

7.7 Adubiaso

At Adubiaso, the gold mineralisation occurs along the main northeast to southwest striking shear vein system in sub-vertically inter-bedded greywackes and phyllites intruded by later felsic intrusives. Subtle jogs in the felsic intrusives give rise to higher grade ore shoots. The ore body plunges shallowly to the northeast at 20^{o} parallel to the intersection of east northeast dipping veins with the main strike direction.

The deposit extends for some 1,000m along strike and is known to exist down to a depth of 180m below surface due to current drilling information. The mineralised zones are typically 1m to 4m in width, but may occasionally reach up to 20m in width. The gold mineralisation occurs as free gold and is associated with the northeast plunging quartz veins, along the intersection of the metasediments and sheared porphyries.

The mineralised vein set strikes north-northwest to north-northeast and dips towards the east, cross-cuts the regional northeast striking foliation, and is variably deformed near the shear zone.

A subtle jog in the strike of the porphyries and carbonaceous schist, correlates with ore zone terminations. The ore shoots plunge shallowly to the north, parallel to the intersection of the east northeast dipping veins with the sub-vertical north to south striking shear zone, and sub-parallel stretching lineation. The ore body occurs parallel to the strike inflection, which would be parallel to the north plunging stretching lineation.

7.8 Abore

There are at least two and potentially three, phases of mineralisation that may be recognised at Abore. Mineralisation is effectively constrained to the granite, with the overall trend of mineralisation being parallel to that of the stratigraphy. However, as noted by McCuaig et al., (2002) not all the granite is mineralised. The economic mineralisation is developed primarily along the eastern margin of the granite. Granite hosted mineralisation can be split into two phases:

- Quartz vein hosted
- North northeast trending shear hosted

A phase of steep quartz veining hosted within the granite is observable from drill core, but this is currently poorly constrained.

The dominant phase of mineralisation at Abore is hosted in shallow west dipping 1 cm to 10 cm thick quartz vein arrays which have developed primarily along the eastern margin of the granite contact and the sandstone-wacke dominated stratigraphy. Very little disseminated alteration was observed, despite the significant hydrothermal, (sericite and arsenopyrite) alteration associated with the mineralised zones. Vein density, rather than vein thickness, seems be indicative of higher grade zones. This is demonstrated well in drill hole ABP11-038 which was logged by James Davis. Analysis of vein orientations showed that two vein types shallow west dipping and steep west dipping occur. Both vein sets strike towards the north. The shallow westerly dipping vein arrays create shallow stacked westerly dipping ore shoots within the granite. The relationship between the two vein sets is not clear. Two scenarios are plausible either set are related to separate deformations events, or else the steep set is related to orogenic collapse after a compressional phase resulting in the shallow west dipping vein array.

Analysis of the grade control data shows discrete north-northeast trending zones of high grade mineralisation that have developed in the boudin necks of the granite bodies (Figure 7-30) relating to early north northeast trending structures. It is probable that this pre-dates the quartz vein hosted mineralisation, and is similar to that of Nkran, where shallow west dipping vein arrays overprint steep, high grade mineralisation.

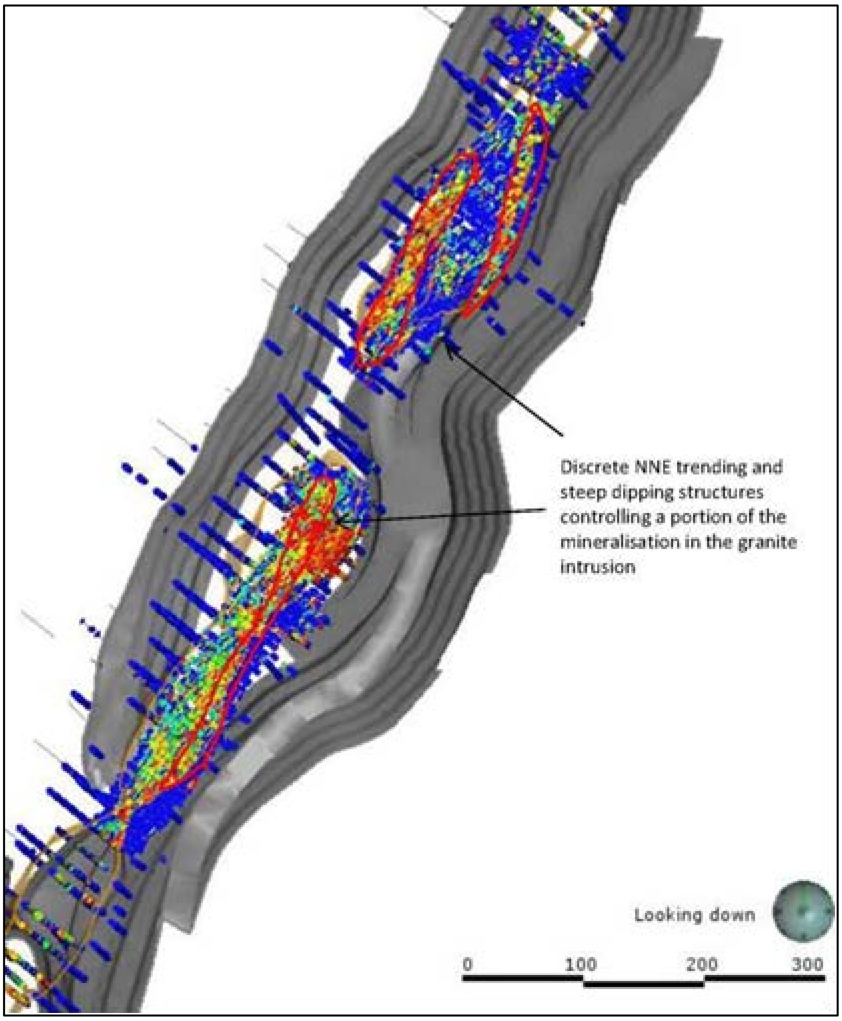


Figure 7-30: Abore Deposit Plan View 155mRL Mineralisation is hosted in the Granite Intrusion (Source: Asanko Gold, 2014)

The ore shoots have a shallow plunge to the northwest and southwest. The south westerly plunging shoots appear to be the most continuous and are likely related to the fault intersections with the granite. The northeast plunging shoots can be explained from vein orientation data. Two dominant vein trends are observed:

- Veins parallel to bedding
- Shallow dipping extension veins

The line of the intersection between the two vein sets is the same orientation as the grade defined in the ore shoots which have a shallow plunge to the northeast. The arrays of extension veins are consistent with the formation in a compressional stress regime. A model of dip variation for the foot wall contact of the intrusion also shows several low dip domains which plunge to the northeast. Such inflections in the strain margins of the granite are interpreted as dilational jogs. These sites likely control the distribution of well developed extension vein arrays.

7.9 Dynamite Hill

The mineralisation at the Dynamite Hill deposit can be classified into two distinct types:

- Steep west dipping shear zones with ductile, shear hosted mineralisation, within the north northeast striking, steeply west dipping "Nitro" and "TNT" shears. These zones typically measure approximately 2m in width. It is not known if these lodes are overprinted by a barren quartz breccia event, such as that at Nkran (Figure 7-31), which significantly disrupts the continuous nature of the mineralisation. At Dynamite Hill the alteration zone is marked by the presence of arsenopyrite.
- Shallow quartz vein hosted cross cutting, northwest to north northwest striking, shallow to moderately northeast dipping brittle quartz-carbonate vein hosted mineralisation (Figure 7-32) and associated sericite-albite-arsenopyrite-magnetite alteration. This gold mineralisation is predominantly hosted in the granite intrusion and on the margins within the sandstone. The stacked vein array sets plunge steeply to the north, controlled by the bounding eastwest structures.

Figure 7-31: Dynamite Hill presence of Steep Breccia Style Lode in Core (DYDD13-003) (Source: Asanko Gold, 2014)

Figure 7-32: Dynamite Hill drill core DYDD13-003 showing Steep and Shallow Lodes (Left) and Shallow Veins which cut the Granite (Right). (Source: Asanko Gold 2014).

7.10 Asuadai

The Asuadai deposit is characterised by preferential alteration of the sandstone and wacke (to a lesser extent) lithologies to a sericite-magnetite (+/- albite) assemblage. This alteration style appears to be distinctive to mineralisation associated with the Nkran regional structural trend. Various stages of arsenopyrite and pyrite are observed, either disseminated throughout the core, or as selvages to gold bearing quartz veins. Arsenopyrite appears to be dominantly associated with the shallow southwest dipping vein arrays, with significant disseminated alteration occurring within the granitic intrusion. Siltstone (and carbonaceous shale) lithologies are generally unaltered.

Early ductile mineralisation appears to be associated with silicification and minor pyrite. The extensive over printing and later reactivation of these structures makes it difficult to establish a distinct alteration package.

The Asuadai deposit is relatively complex with a number of controls of mineralisation which influences the geometry of the mineralisation. Two distinct styles of mineralisation may be recognised which include:

- Steep ductile type mineralisation, this was observed within drill hole ASP11-014 which was selectively logged and is associated with the metasedimentary lithologies. This style was selectively overprinted by a later brittle brecciating event. This style of mineralisation would be analogous to the CV, GV and Eastern Breccia Lode (“EBL”) at Nkran, or the Nitro Shear (NS) at Dynamite Hill. This mineralisation parallels bedding, or foliation. Stereographic projections of vein arrays show a 020º to 040º orientation dipping steeply towards the west. The steep ductile mineralisation is seen to bind the granitic intrusion. This mineralisation is also associated with parallel structures to the main granitic intrusion

- Shallow dipping quartz veins. This is the dominant phase of gold mineralisation at Asuadai and consists of veins that vary in thickness from 1 cm to 60 cm. The flat lying vein arrays are best developed in the granite. The veins have associated sericite-albite-arsenopyrite-magnetite alteration

7.11 Esaase

Gold mineralisation in the Esaase Project area occurs in quartz-carbonate veins hosted within parallel northeast trending, moderately to steeply west dipping bodies of extremely deformed siltstone-shale. One form of disseminated alteration most commonly noted in oxidised rocks is Quartz Sericite Pyrite ("QSP") alteration. This alteration type is not distinctly different in colouration in fresh core and is thus difficult to detect in that state. Surface weathering converts the sericite to white kaolinite creating a bright white colour alteration distinguishable even at great distance when exposed in trenches, road cuts, and drill pads. At closer scale, pyrite pseudomorphs can be distinguished. The second stage consists of pervasive carbonate alteration in the form of carbonate porphyroblasts, particularly after andalusite in phyllitic rocks. Carbonate flooding is more prevalent in siltstone where precursor andalusite porphyroblasts did not form (Klipfel, 2009).

Quartz veins formed within the mineralisation envelopes throughout the duration of the extensive fold and thrust and strike slip deformation events. Four stages of veins can be identified. These include an early un-mineralised quartz-only vein stage which has undergone deformation and brecciation. A second vein stage consists of a myriad of fine spider web like quartz carbonate veins. These veins are also early and are consistently deformed and offset. The third stage consists of quartz-carbonate-sulphide veins with visible free gold. The associated sulphide is generally pyrite, but up to 15% of it can be chalcopyrite, with minor arsenopyrite. Finally, late stage post-mineral calcite veins crosscut all previous features (Klipfel, 2009).

Veins that contain visible gold predominantly strike 350° to 020°, have sub-vertical dips and are either planar, or S-shaped. Thus they are oblique in orientation to the overall strike and dip of mineralisation and appear to be bounded by aforementioned thrust faults and can be described as en-echelon vein sets. They are likely to have been emplaced during a transition from fold thrust deformation to left lateral strike slip deformation (Figure 7-33 and Figure 7-34 Klipfel, 2009).

Figure 7-33: Example of Folded and Broken Early Veins (Source: Esaase Technical Report 2009)

Figure 7-34: Example of sheeted veining with visible gold (Source: Esaase Technical Report 2009)

8 DEPOSIT TYPES

The following description of deposit types is sourced predominantly from Siddorn and Lee (2005). The QP does not disclaim responsibility for the content contained in the following sub-sections.

8.1 Geological Characteristics of Structurally Hosted Gold Deposits in Southwest Ghana

Two broad styles of gold deposits are present in Southwest Ghana:

- Paleoplacer disseminated gold deposits in Tarkwaian conglomerates
- Structurally controlled lode or orogenic gold deposits

This section reviews only the characteristics of structurally controlled lode gold deposits. Most of these deposits are hosted in Birimian metasediments, often close to major lithological contacts with either Birimian metavolcanics, or Tarkwaian metasediments.

Two distinct gold events are recognised in southwest Ghana:

- D2 gold related to regional north eastsouth west compression and reverse faulting (ca. 2110 Ma.)
- D4/D5 gold related to regional sinistral strike slip faulting (ca. 2090 Ma)

Gold mineralisation is associated with major northeast striking, 5m to 40m wide graphite-chlorite-sericite fault zones. In particular, gold mineralisation is developed where the northeast fault zones intersect major east northeast striking fault zones, and especially where they are recognised to have influenced granite emplacement, alteration and gold geochemical trends.

Left stepping flexures (10 km to 30 km scale) in the north east striking fault zones, (which produce more northerly striking fault sections), are important for the localisation of gold mineralisation. Other local complexities in stratigraphy and fault geometry, associated with major northeast striking faults, are also important for example, folds in stratigraphy that may produce saddle reef style mineralisation, or fault duplexes.

The Ashanti and Bogoso deposits (Figure 8-1) are hosted in shear zones, close to, or at, the contact between Birimian metasediments and Birimian metavolcanics, or Tarkwaian metasediments. The shear zones that host the Bogoso deposit occur at the contact between Birimian metasediments and Tarkwaian metasediments. The shear zones have also imbricated a series of moderately magnetic mafic igneous rocks (doleritic sills), that strike northeast concordant with the shear zones. The Ashanti deposit is hosted in shear zones either within the Birimian metasediments, (Obuasi, Ashanti, and F fissures), or at the contact between Birimian metasediments and Birimian metavolcanics (Cote d’Or fissure).

The Ashanti and Bogoso deposits are both dominated by D5 regional strike slip gold, deposited in re-activated D2 reverse faults. They both contain quartz vein free-milling gold lodes and sulphide (arsenopyrite rich) disseminated refractory gold lodes. The sulphide lodes are interpreted to form alteration haloes around the quartz vein lodes. Alteration is typically graphite, quartz, ankerite, sericite, tourmaline, chlorite, arsenopyrite, and pyrite.

The Chirano and Nkran deposits both demonstrate a late (2nd) phase of gold mineralisation hosted in granitoids, (Chirano belt type granite, Nkran basin type granite), emplaced in regional shear corridors. The Chirano deposit is situated close to the contact between Birimian metavolcanics and Tarkwaian metasediments. The Obotan deposit is situated within the Birimian metasediments, but the granitoid and mineralisation, both occur at contacts between greywacke and carbonaceous phyllite units. The Chirano and Nkran deposits are both dominated by D2 regional reverse faulting gold, and only contain quartz vein-hosted free-milling gold lodes.

The deposit types discussed in this document are structurally controlled mesothermal quartz vein style mineralisation. This is the most important type of gold occurrence in West Africa and is commonly referred to as the Ashanti-type in recognition of the Obuasi area being the type locality and the largest gold deposit in the region. Milesi et al. (1992) recognised that mesothermal quartz vein style deposits are largely confined to tectonic corridors that are often over 50 km long and up to several kilometres wide and usually display complex, multi-phase structural features, which control the mineralisation.

The most common host rock is usually fine-grained metasediments, often in close proximity to graphitic, siliceous, or manganiferous chemical sediments. However, in some areas, mafic volcanics and belt intrusions are also known to host significant gold occurrences. Refractory type deposits feature early-stage disseminated sulphides in which pyrite and arsenopyrite host important amounts of gold overprinted by extensive late stage quartz veining in which visible gold is quite common and accessory polymetallic sulphides are frequently observed. This type includes important lode / vein deposits in Ghana such as at Obuasi, Prestea, Bogosu, Bibiani and Nkran. However, a second non-refractory style of gold mineralisation occurs in which gold is not hosted within sulphide minerals either in early, or late stage mineralisation. These type deposits have lower sulphide content in general and in particular, often lack the needle-like arsenopyrite that is common in the refractory type deposits. Such deposits include the Chirano, Ahafo and Nkran type deposits.

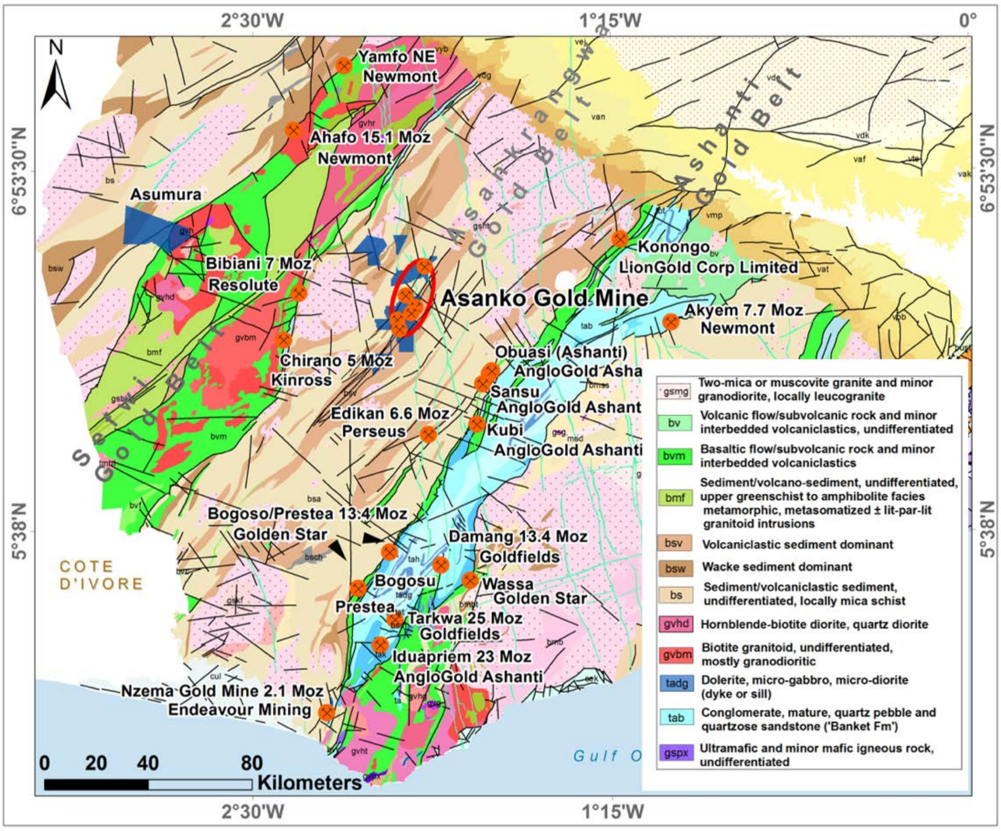


Figure 8-1: Distribution and geological setting of significant gold deposits, SW Ghana (Source: Geological Survey of Ghana and Asanko Gold)

9 EXPLORATION

Asanko Gold finalised the consolidation of the Keegan and PMI assets early in 2014. Exploration up to that juncture has been discussed in Section 6.

Asanko Gold have initiated an ongoing systematic exploration programme, that includes regional (generative) and near mine (advanced) programmes, targeting new gold deposits as well as incremental mineral resources (oxide and fresh). The regional prospectivity work was initiated during 2014, and advanced drilling programmes since Q1 2015. The key elements completed to date:

- Prospectivity analysis of the Kumasi Basin (Figure 9-1)
- June 2014 CJM Technical report on the Asanko Gold assets and revised mineral resource estimations
- Review of previous exploration and drilling programmes
- Exploration target prioritisation for generative and advanced exploration
- 3,000 line kilometre Heli-borne VTEM survey infilling previous gaps in coverage (Figure 9-2)
- Updated regional geological interpretation based on the interpretation of the VTEM survey (Figure 9-3)
- 3-D inversion study of the VTEM data
- Two relogs of Nkran DD core, detailed
- Updated Nkran geological model
- Evaluation drilling to Indicated resource classification at the Dynamite Hill, Akwasiso, Nkran Extension and Adubiaso Extension projects;
- Updated MRE for Nkran, Dynamite Hill, Akwasiso, Nkran Extension and Adubiaso Extension

The statistics of this exploration work are shown below in Table 9-1.

Table 9-1: Statistics of Exploration Work (source: Asanko Gold)

Surface Geochem Stats.	Sample Type			
Year	**Grab**	**Rock**	**Soil**	**Grand Total**
2016	24	7	190	221
2017	17		619	636
Grand Total	41	7	809	857

Drilling Statistics	Hole Type				
Year	**DDH**	**RC**	**RCD**	**TR**	**Grand Total**
2014	9	48			57
2015		65		17	82
2016	25	167	11	23	226
2017	10	28	26	7	71
Grand Total	44	308	37	47	436

Grade Control Sampling	Sampling Method				
Open Pit and Year	**RC**	**RCD**	**RL**	**TR**	**Grand Total**
2015	1,902		107	2	2,011
Nkran Pit	1,902		107	2	2,011
2016	3,322				3,322
Adubiaso Extension	323				323
Nkran Pit	2,999				2,999
2017	1,379	1	15		1,395
Akwasiso Pit	313		15		328
Nkran Pit	1,066	1			1,067
Grand Total	**6,603**	**1**	**122**	**2**	**6,728**

Notes

DDH	Diamond Drill Hole
RC	Reverse Circulation
RCD	Reverse Circulation with a Diamond Core Tail
TR	Trench
RL	Rip Line

The expenditure incurred over this period is shown in Table 9-2 below.

Table 9-2: Summary Asanko Exploration Expenditure 2015 – 2017 (source: Asanko Gold)

Exploration Expenditure	2015 US$	2016 US$	2017 US$ YTD	Total to June 2017 US$
Exploration Total	**2,785,520**	**6,387,446**	**3,650,000**	**12,822,966**

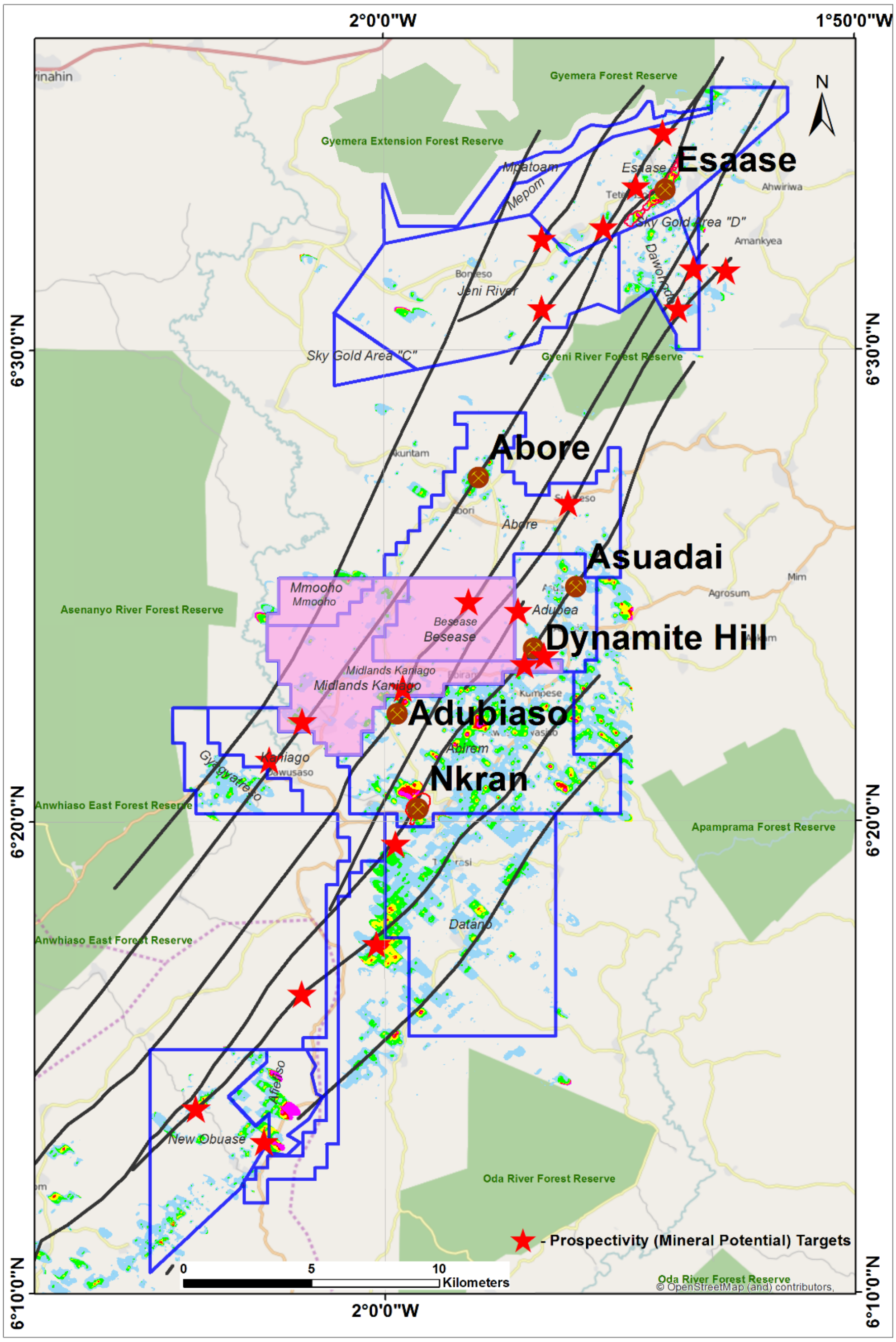


Figure 9-1: Propectivity analysis of the AGM tenements, Asankrangwa Belt (source: Asanko Gold)

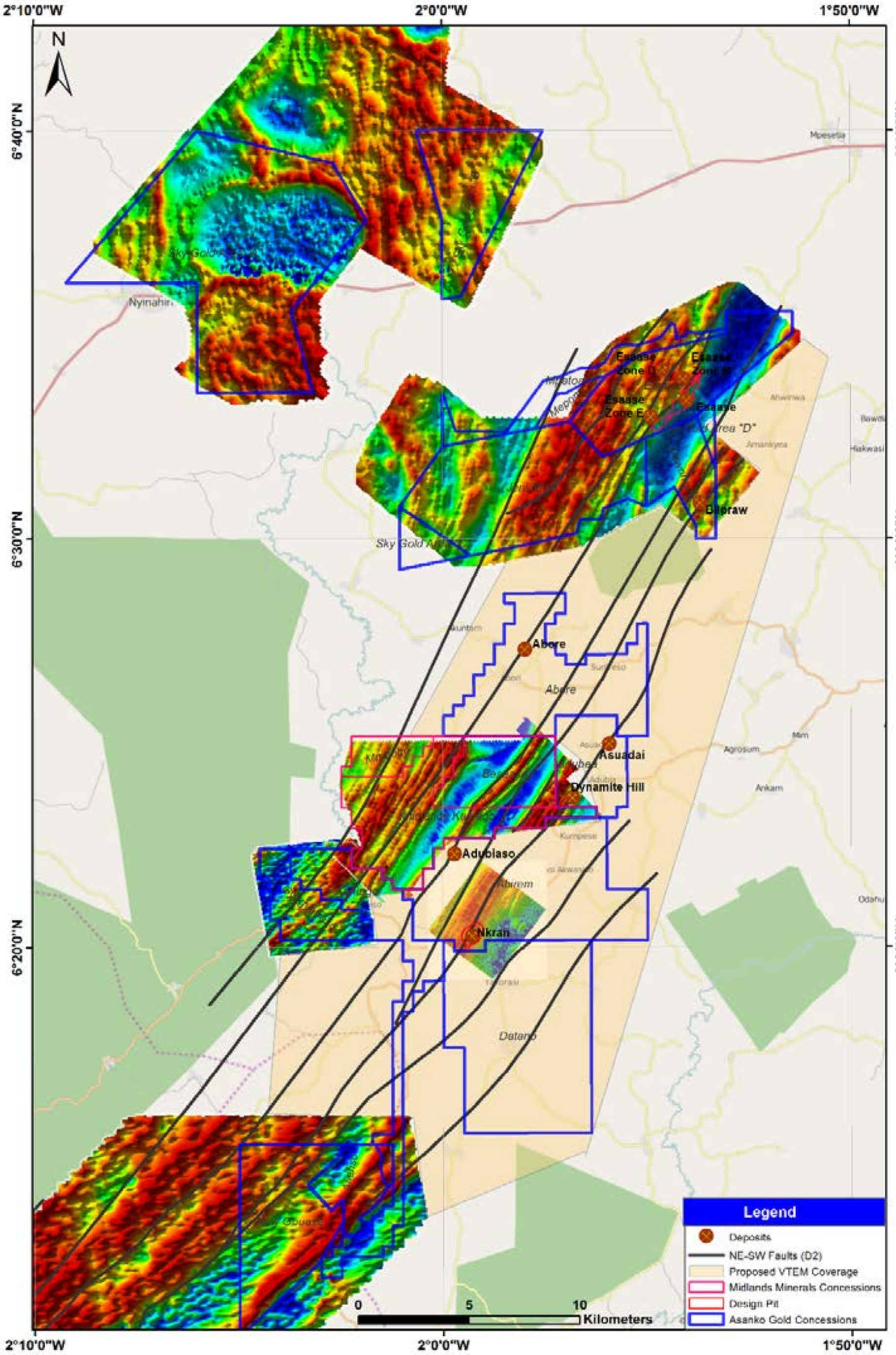


Figure 9-2: Extent of Asanko Gold 2015 VTEM Survey shown in pink (Source: Fathom Geophysics 2016)

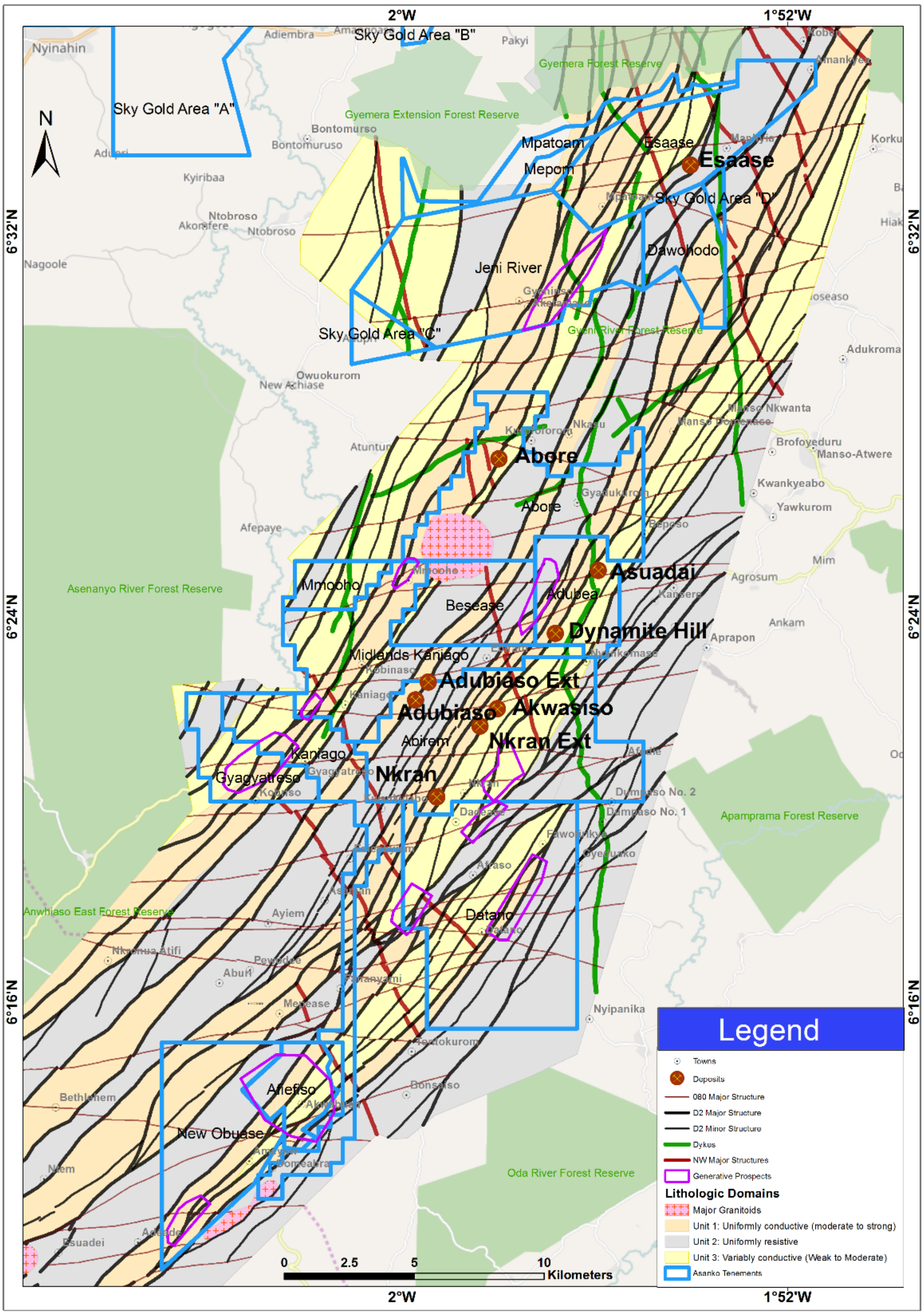


Figure 9-3: Regional geological interpretation from VTEM survey (Source: Asanko Gold 2016)

10 DRILLING

10.1 Introduction

Drill traverses for all project areas are generally aligned perpendicular to the local northeast southwest mineralised trends. A variety of drilling methods have been used for exploration and pit evaluation over the years. These are summarised in Table 10-1.

Table 10-1: Asanko Gold GM Summary Drilling Statistics by Type

Type of Drilling	# Holes	Meters
Diamond Core ("DDH")	841	181,706
RC Pilot, Diamond Core Tail	656	205,800
Reverse Circulation ("RC")	3,006	347,192
Grade Control RC	43,549	415,380
Air Core ("AC")	426	20,677
Reverse Air Blast ("RAB")	240	6,647
Various Geotech and Related	158	16,100

To date, a total of 3,552 evaluation DD, RC and RCD boreholes totalling 504,768m have been drilled in the AGM deposits, as well as nearly 103,000 Grade Control boreholes totalling over 685,000m. Table 10-2 summarises the drilling undertaken by each Company and Table 10-3 that completed to date on each deposit.

Table 10-2: Previous Operator Evaluation Drilling Statistics on the various Deposits (source: Asanko Gold)

Company	**Historic Pit Evaluation Drilling Statistics**											
	DDH		**RCD**		**RC**		**Grade Control RC**		**Air Core**		**RAB**	
	No	**Meters**	**No**	**Meters**	**No**	**Meters**	**No**	**Meters**	**No**	**Meters**	**No**	**Meters**
Keegan	230	50,478	611	193,563	1,544	241,201	0	0			-	-
PMI	284	76,463	7	3,170	163	19,220	0	0	426	20,677	-	
Resolute	214	43,165	28	6,481	1,096	68,988	62,382	311,909			240	6,647
Total	**755**	**175,905**	**656**	**205,800**	**3,006**	**347,192**	**102,925**	**685,286**	**426**	**20,677**	**240**	**6,647**

Company	**Asanko Gold Pit Evaluation Drilling**											
	DDH		**RCD**		**RC**		**Grade Control RC**		**Air Core**		**RAB**	
	No	**Meters**	**No**	**Meters**	**No**	**Meters**	**No**	**Meters**	**No**	**Meters**	**No**	**Meters**
Asanko Gold	27	5,800	10	2,586	203	17,783	40,543	373,377	-	-	-	-
Total	**755**	**175,905**	**656**	**205,800**	**3,006**	**347,192**	**102,925**	**685,286**	**426**	**20,677**	**240**	**6,647**

Table 10-3: Evaluation Drilling completed to Date on each Deposit (source: Asanko Gold)

Deposit	DDH		RCD		RC		GC		RAB	
	No	Meters	No	Meters	No	Meters	No	Meters	No	Meters
Nkran	279	83,567	29	8,532	556	26,810	97,939	620,028	-	-
Esaase	111	24,341	289	92,093	716	111,584	-	-	-	-
Abore	54	9,437	-	-	-	-	3,006	42,003	31	716
Abore N	15	1,985	-	-	409	31,594	-	-	-	-
Adubiaso	53	10,327	4	590	289	26,495	-	-	-	-
Adubiaso Ext	-	-	-	-	35	3,460	4,986	65,258	-	-
Asaudai	68	8,820	-	-	84	5,551	-	-	209	5,931
Dynamite Hill	12	2,502	1	249	158	18,303	-	-	-	-
Akwasiso	59	10,888	8	2,168	231	16,991	-	-	2	87
Nkran Ext	-	-	3	698	89	7,781	-	-	-	-
Total	**651**	**151,868**	**334**	**104,330**	**2,567**	**248,570**	**105,931**	**727,289**	**242**	**6,734**

The resource drill hole spacing varies between the projects, ranging from 10m to 20m across strike to 20m to 50m along strike (to define near mine surface projections of mineralisation). Drill coverage at depth is variable approaching the maximum drilled depth on the property of 590m from surface in drill hole RCD802A at the Nkran project.

The drilling density is considered appropriate to define the geometry and extent of the mineralisation for the purpose of estimating gold resources, given the understanding of the local project geology, structure and confining formations. Asanko Gold's strategy is to conduct drilling sufficient to assume geology and grade continuity to a level to support at least Indicated Mineral Resources and thus support the application of modifying factors in sufficient detail to support mine planning and evaluation of economic viability. Recent analysis suggests that intersection spacing of 25m to 40m fulfils this criteria.

10.2 Drilling Summaries for Each Deposit

10.2.1 Nkran

The Nkran project database utilised during the 31st December 2016 MRE included the historical and recent Grade Control drill hole data summarised in Table 10-1 and Figure 10-1.The following drill statistics are based on meterage. Most of the surface holes used in the resource estimation are from the Resolute–Amansie programmes (approximately 70%). All the PMI drilling is diamond core ("DC") which accounts for the remaining 30% of the drilling. DC drilling accounts for 68% of all holes and RC the remaining 32%. PMI diamond drilling undertaken during 2007 through to January 2012 accounts for 33% of the total dataset, with the remaining 67% being mixed RC and DC completed by Resolute during an unspecified period.

In the opinion of the QP, there are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the MRE. It is noted that on an ongoing basis the density database will be updated on a monthly basis. Asanko Gold is also in the process of upgrading the rotary splitters on the grade control drilling rigs, which will give parity to drill sampling methods as well as more homogenous samples, inclusive of field duplicates.

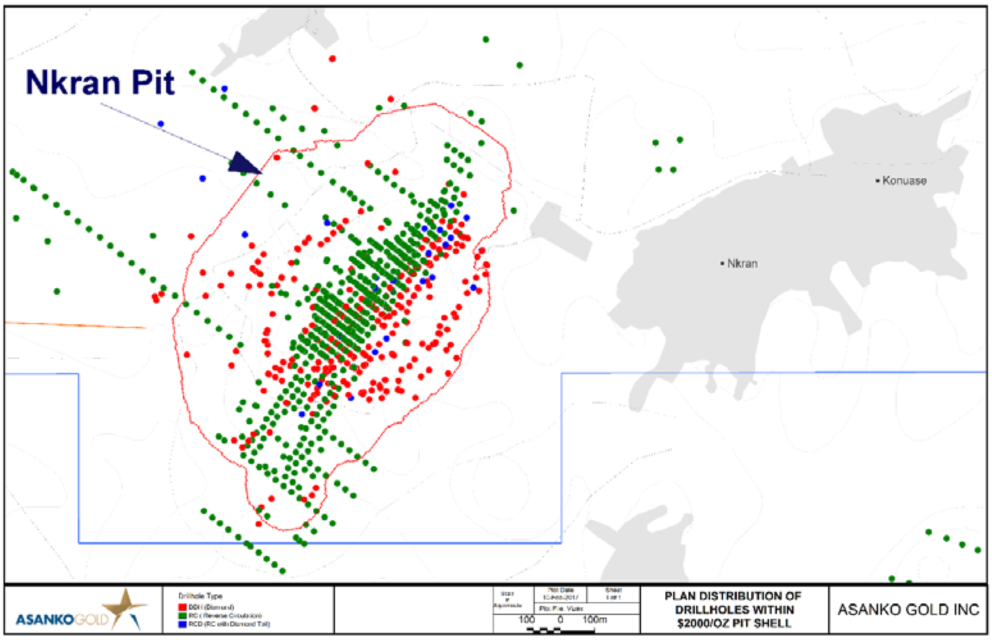


Figure 10-1: The Current Drill Collar Distribution for the Nkran Pit.(Source: Asanko Gold)

Since mid 2015, Asanko Gold has undertaken systematic 10m x 5m grade control drilling as the Nkran Pit push backs and the mining operation progressed. This drilling is down-the-hole ("DTH") hammer RC on a 10m x 5m -45° heel-toe design to 22.5 and 45m depths, which generates coverage of 6m x 6m mining benches. Figure 10-2 shows the distribution and a representative cross section of historic Resolute (purple) and Asanko Gold GC (red) drilling to 31st December 2016.

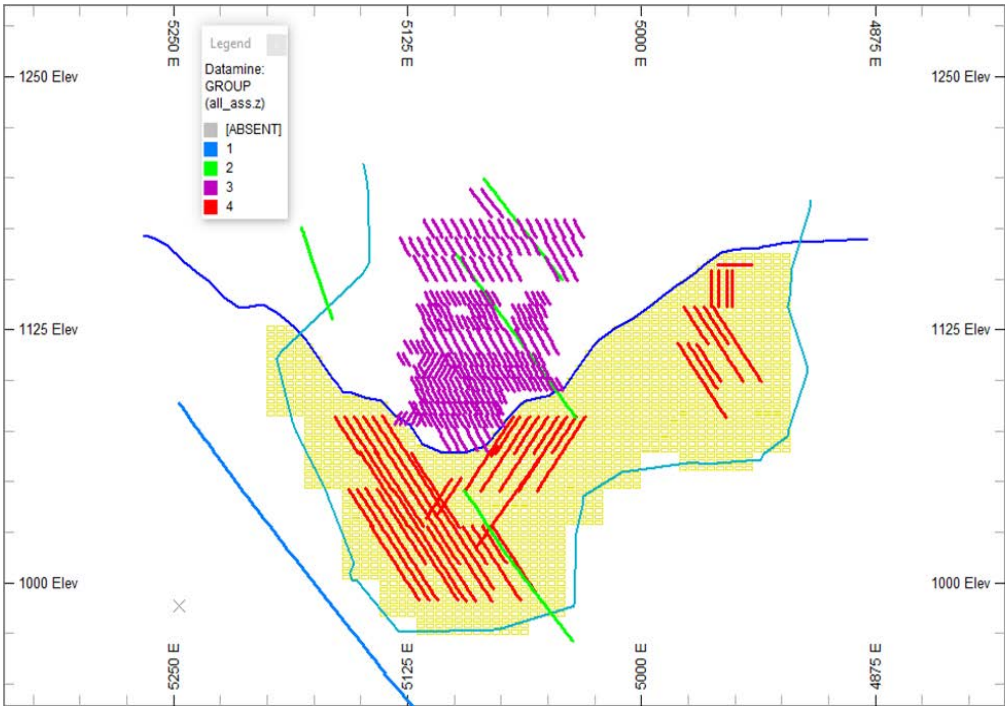


Figure 10-2: A cross Section showing the Relationship of Nkran Historic Resolute and Current Asanko Gold Grade Control Drilling Distribution (Source: Asanko Gold 2017)

10.2.2 Adubiaso

The Adubiaso project database utilised during the MRE included the historical data as shown in Table 10-4 and Figure 10-3.

Table 10-4: Adubiaso – Summary of the Historical Drilling Dataset (source: Asanko Gold)

RC			DDH	Total RC +DDH
Resolute	Number of holes	277	5	282
	Metres	25 547	635	26 182
PMI	Number of holes		45	45
	Metres		9 021	9 021
Total	Number of holes	277	50	327
	Metres	25 547	9 656	35 203

The following drill statistics for Adubiaso are based on meterage. The Adubiaso Main orebody drilling remains as per 2014 (a combination of Resolute 75% and PMI 25% DC drill holes).

PMI diamond drilling undertaken during 2007 through to January 2012 accounts for 25% of the total dataset with the remaining 75% being mixed RC and DC drilling completed by Resolute during an unspecified period.

During 2016 Asanko Gold conducted RC exploration and RC infill drilling on the NE extension of the Adubiaso pit mineralisation. These programmes resulted in further resources and open pittable reserves being estimated for what is known as Adubiaso Extension (Figure 10-3 and Table 10-5)

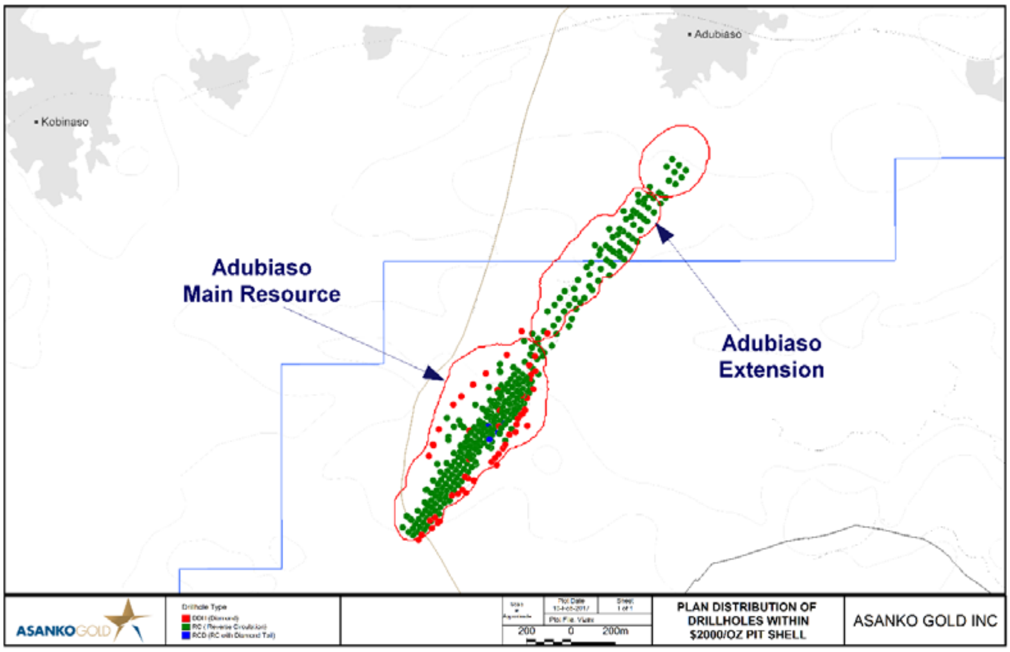


Figure 10-3: Current Drill Collar Locations for the Adubiaso Main and Adubiaso Extension Deposits (Source: Asanko Gold)

Table 10-5: Asanko Gold Grade Control meters Adubiaso Extension (source: Asanko Gold)

Locality / Pit	Asanko Gold Grade Control Hole Type Meterage			
	RC	RCD	RL	Grand Total
Adubiaso Ext 2016	11,877	0	0	11,877

10.2.3 Abore

The Abore project database comprises historical drill hole data from Resolute and PMI, as summarised in Table 10-6 and Figure 10-4.

Table 10-6: Abore - Summary of Historical & Recent Drilling Dataset (source: Asanko Gold)

RC			DDH	Total RC +DDH
Resolute	Number of holes	408	6	414
	Metres	31,639	928	32,567
PMI	Number of holes	0	51	51
	Metres	0	9 260	9 260
Total	Number of holes	408	57	465
	Metres	31,639	10 188	41,827

The following drill statistics for Abore are based on meterage. Most of the holes used in the MRE are from the Resolute programmes (some 78%). The combined PMI and Resolute DC drilling accounts for 25% of all holes and RC the remaining 75%.

PMI DC drilling undertaken during 2007 through to January 2012 accounts for 22% of the total dataset with the remaining 78% being mixed RC and DC drilling completed by Resolute during an unspecified period.

Figure 10-4 shows the drill collars based on the drill type for Abore.

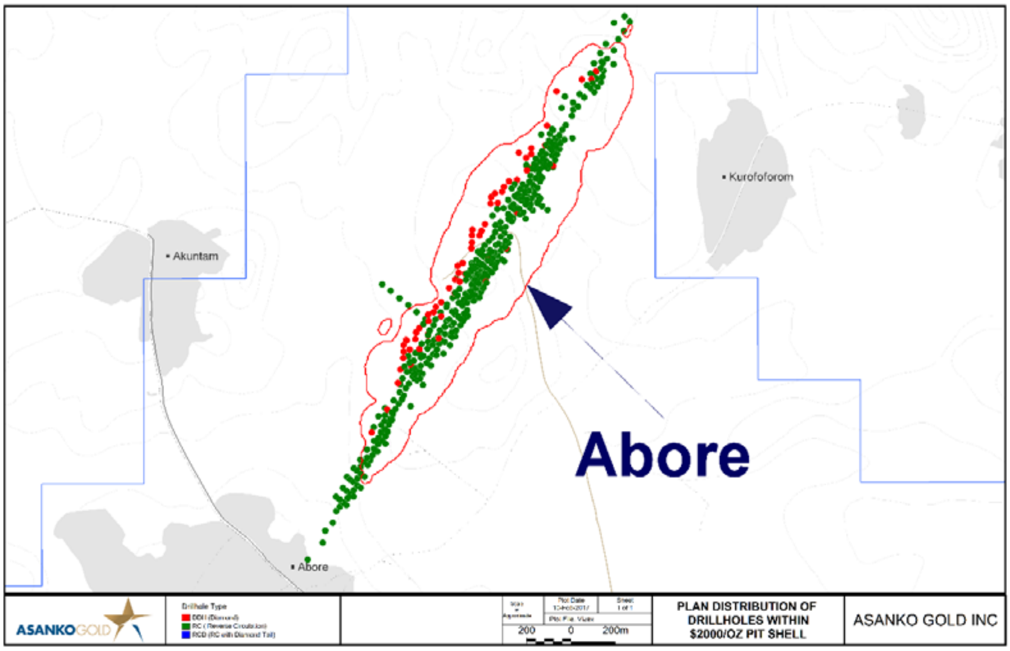


Figure 10-4: Abore – Plan of Drill Collar Locations showing Hole Type (Source: Asanko Gold)

10.2.4 Dynamite Hill

The Dynamite Hill project database includes historic PMI and recent Asanko Gold infill drill hole data as summarised in Table 10-7.

Table 10-7: Dynamite Hill – Summary of Historical and Recent Drilling Dataset (source: Asanko Gold)

Deposit		RC	DDH	Total
PMI	Number of holes	119	9	128
	Metres	15,057	2,094	17,151
Asanko Gold	Number of holes	39	0	39
	Metres	3,246	0	3 246
Total	Number of holes	158	9	167
	Metres	18,303	2,904	20,397

Most of the holes are from the PMI programmes (some 63%). The combined PMI and Asanko Gold DC and infill RC drilling accounts for 14% of all holes and RC the remaining 86%. Figure 10-5 shows the current Dynamite Hill drill collar plan.

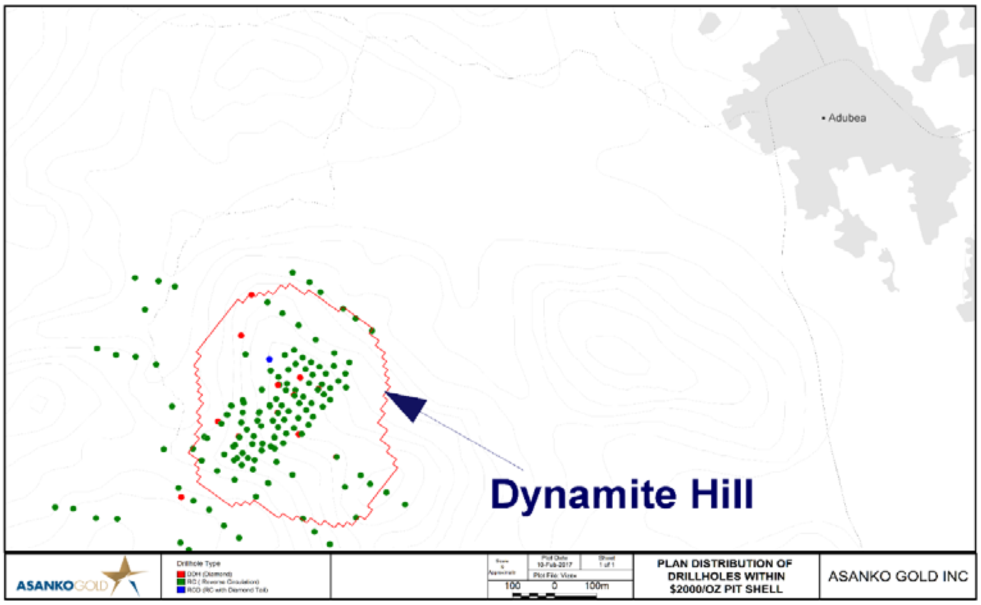


Figure 10-5: Dynamite Hill Drill Collar Plan (Source : Asanko Gold)

10.2.5 Asuadai

The Asuadai project database comprises historical drill hole data from Resolute and PMI, as summarised in Table 10-8.

Table 10-8: Asuadai – Summary of Historical and Recent Drilling Dataset (source: Asanko Gold)

RC			DDH	Total RC +DDH
Resolute	Number of holes	84	3	87
	Metres	5,606	294	5,900
PMI	Number of holes	0	56	56
	Metres	0	7 406	7 406
Total	Number of holes	84	59	143
	Metres	5,606	7 700	13,306

Most of the holes are from the Resolute programmes (some 44%). The combined PMI and Resolute DC drilling accounts for 58% of all holes and RC the remaining 42%. PMI DC drilling undertaken during 2007 through to January 2012 accounts for 56% of the total dataset with the remaining 44% being mixed RC and diamond drilling completed by Resolute. Figure 10-6 shows the Asuadai drill collar plan based on the drill type and company programme.

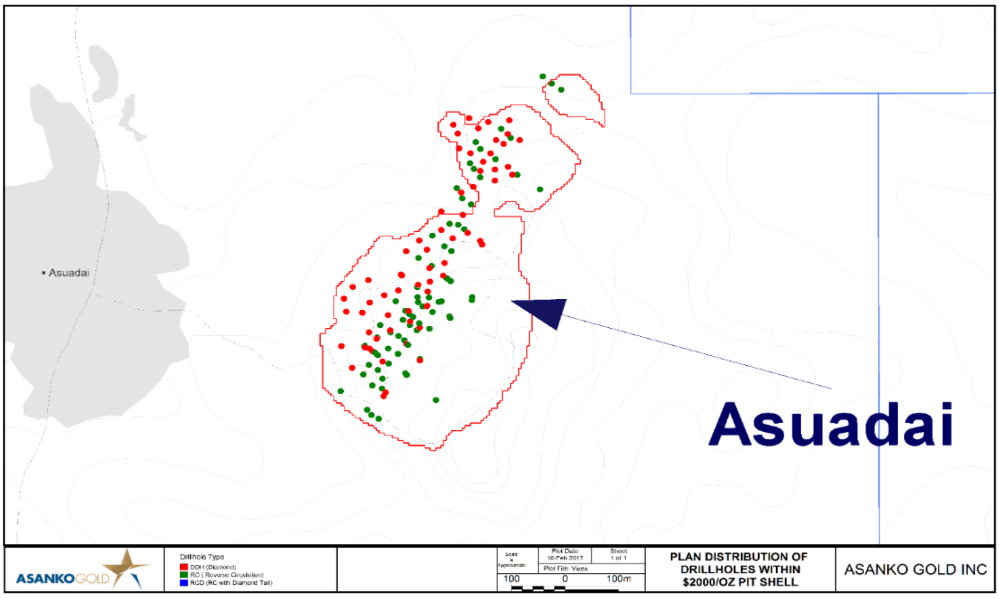


Figure 10-6: Asuadai – Plan of Drill Collar Locations Showing Hole Type (Source: Asanko Gold)

10.2.6 Esaase

Drilling at the Esaase Gold Project was undertaken by Keegan from 2006 – 2012, and focused mainly on the northwest striking main gold bearing structures in the Esaase orebody. Other targets attracting investigation include Jeni, Dawohodo, Mpatoam and Binappco concessions (Figure 10.7). Surface RC and DC drilling has been completed at the project. The project drill programmes were designed to test the mineralised corridor delineated from soil sampling, trenching, drilling and geophysical interpretations. The initial 14 diamond drill holes were completed by Eagle drilling contractors with the remainder completed by Geodrill contractors. Both of these drilling companies are reputable Ghana based companies providing RC and DC drilling services consistent with current industry standards. Table 10-9 summarises drilling statistics for all holes drilled at the Esaase Gold Project as at the beginning of February 2012. A total of 1,496 drill holes have been completed on the project area (Table 10-9 and Figure 10-7). Of these 1,187 drill holes in the currently defined resource area were used for the Esaase MRE.

Table 10-9: Summary Drilling Statistics for Esaase (source: Asanko Gold)

Type	Number	Type	Metres Drilled
RC holes	987	RC metres	149,906
RC pre-collars with Diamond tails	340	RC pre-collar with Diamond tail metres	100,360
Diamond holes	112	Diamond hole metres	24,811
Water wells	57	DTH metres	3 573
Total drill holes	**1,496**	**Total Metres drilled**	**268,249**

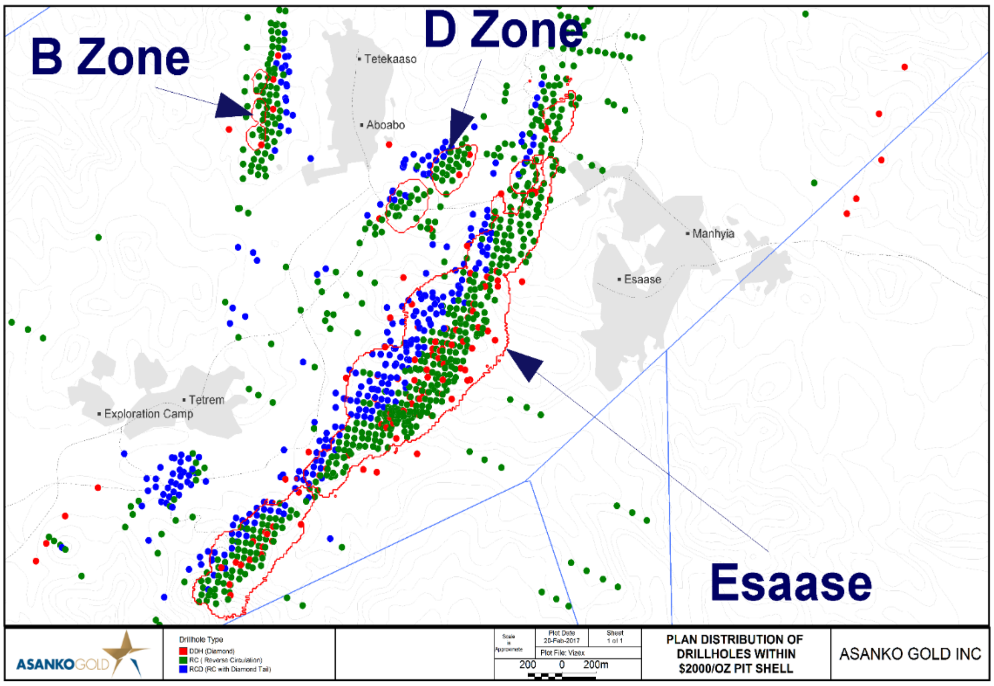


Figure 10-7: Drill Hole Locations for the Esaase Gold Project (Source: Esaase NI-43101, 2012)

10.2.7 Akwasiso

The Akwasiso project database includes historic Resolute and recent 2016-2017 Asanko Gold drill hole data, as summarised Table 10-10.

Table 10-10: Akwasiso Drilling Statistics (source: Asanko Gold)

RC			DDH	Total RC +DDH
Resolute	Number of holes	148	32	180
	Metres	9,730	5,089	14,819
Asanko	Number of holes	73	7	80
	Metres	6,259	1,888	8,147
Total	Number of holes	221	39	260
	Metres	15,989	6,977	22,966

The distribution of borehole collars at Akwasiso and Nkran Extension, which locates immediately to the south west, is shown in Figure 10-8.

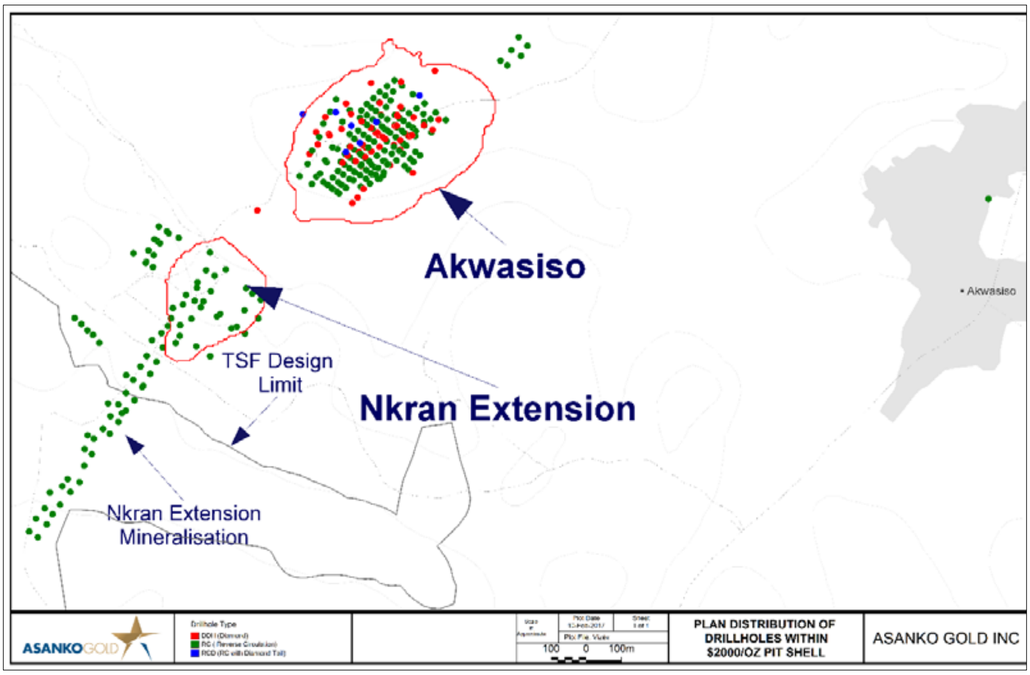


Figure 10-8: Distribution of Borehole Collars at Akwasiso and Nkran Extension (Source: Asanko Gold)

Table 10-11: Asanko Gold has carried out further grade control drilling during 2017 (source: Asanko Gold)

Locality / Pit	Asanko Gold Grade Control Hole Type Meterage			
	RC	RCD	RL	Grand Total
Akwasiso 2017	13,668	0	0	13,668

10.2.8 Nkran Extension

The Asanko Gold drilling statistics for Nkran Extension are shown in Table 10-12 and the distribution of the boreholes above in Figure 10-8 (above).

Table 10-12: Nkran Extension Drilling Statistics (source: Asanko Gold)

RC			DDH	Total RC +DDH
Asanko Gold	Number of holes	56	3	59
	Metres	4,818	698	5,516

10.3 Drilling Procedures

10.3.1 Diamond Drilling

Procedures are discussed for Nkran and Esaase and apply to all other deposits.

10.3.2 Nkran

All historic diamond boreholes (Figure 10-9) drilled by Resolute and PMI were commenced using either pre-collar RC, or HQ core for saprolite and then reduced to NQ when fresh rock was encountered. All rigs were in good working order, and were operated by experienced drillers. The contractors used historically were Burwash Drilling from Vancouver Canada, and Geodrill from Ghana since 2011.

Figure 10-9: Hole NKR11-040 Drilling on the SEside of the Nkran Open Pit (2011) (Source: Spiers 2011)

10.3.3 Esaase

The initial 14 DC drill holes, (HQ and NQ diameters), were completed by Eagle Drilling using a Longyear 38 skid mounted DC drill. All subsequent drilling was completed by Geodrill using UDR650 and UDR900 multipurpose rigs for the RC and DC drilling. DC was oriented by a combination of the spear technique, the 2iC Ezymark orientation device and Reflex ACT II electronic orientation system. Drill hole collars were surveyed by a Coffey Mining surveyor utilising a Thales Promark 3 DGPS unit.

10.3.4 Hole Survey

10.3.4.1 Nkran

All drill holes were laid out using handheld GPS and sighted for direction with a compass. Later, the actual drilled collars were picked up by licenced Ghanaian surveyors using differential GPS from a local base station. Checks of hole positions using a handheld GPS all appear reasonable, given the lower level of accuracy of these units. Holes are surveyed every 30m to 40m down hole using a Reflex magnetic compass and dip unit.

10.3.4.2 Esaase

Drill hole collars were surveyed by a Coffey Mining surveyor utilising a Thales Promark 3 DGPS unit. This unit was validated as returning sub-centimetre accuracy when compared to the topography pickup completed by Coffey Mining using a Geodimeter 610S total station. These instruments have an accuracy of better than 1 cm and are considered conventional.

Drill holes were surveyed on approximately 50m, or less down hole intervals, using a Reflex EZ-Shot®, an electronic single shot instrument manufactured by Reflex of Sweden. These measurements have been converted from magnetic to UTM Zone 30 North values. The factor used to convert between the two grids is -5 degrees.

10.3.5 Core Handling

10.3.5.1 Nkran and Esaase

Core was carefully handled by the drill crew, correctly oriented with regard to the down hole direction and then carefully slipped directly on to pre-marked core trays, (Figure 10-10).

Figure 10-10: Laying Core out into Pre-marked Trays (Source : Spiers, 2011)

10.3.6 Core Boxes

10.3.6.1 Nkran and Esaase

All core boxes were labelled prior to loading with core. All core boxes have the borehole number, box sequence and depths recorded on them in permanent marker. Wooden block markers are inserted by the driller to record depth. The recording of recoveries was the geologist's responsibility.

Typically, the drilling contractor managed approximately 30m to 40m per shift (2 x 10 hour shifts), which is considers to be a reasonable drilling rate.

10.3.7 Core Orientation

10.3.7.1 Nkran and Esaase

For core orientation, a Reflex ACT IID electronic core orientation tool and barrel was used for orienting and marking core (Figure 10-11). The barrel was oriented using the electronic orientation unit prior to the drill run. The full, oriented barrel was then retrieved, the core aligned and the core marked using a bottom hole convention. The local geologists were competent in the use of this device and procedure and take due care. All core is oriented. The down hole direction is marked on the core at the base. If two sections of broken core cannot be matched, then no structural mark-up is made for the lower, (down hole) part of the core run until the next barrel is retrieved and oriented. This avoids erroneous structural measurements being made.

Figure 10-11: Core Orientation Procedures using a Reflex ACT II System (Source: Spiers 2011)

The core is marked with a red permanent marker in the bottom hole position and a directional arrow put on at regular intervals to record the down hole direction. The core is transported carefully back to the core yard. The core is laid out in aluminium boxes that can accommodate 5m of core.

10.3.8 Recoveries

10.3.8.1 Nkran

Core recoveries were recorded once the trays are delivered back to the core facility / yard and recorded in the geological logs. Core recoveries were typically in excess of 95%.

10.3.8.2 Esaase

Core recoveries were typically calculated at the drilling site immediately prior to drilling by qualified technicians and recorded in the geological logs. The core is transferred from the trays and pieced together on a V-rail (angle iron) rack and the recoveries calculated. Core recoveries were typically in excess of 95%.

10.3.9 Core Storage

10.3.9.1 Nkran

The Resolute and PMI Nkran core drilling was originally stored and managed at a coreyard known as the Barracks, adjacent to the NE corner of the Nkran pit. Historically the aluminium core boxes were stacked on the floor (Figure 10-12). Certain core was stored in proper storage racks, and a core yard inventory existed with holes laid out in an organised fashion on the racks. In general, the core yard facilities for logging were adequate, with logging racks for laying out and logging core.

Figure 10-12: Core Boxes Stacked in the old Barracks Core Yard (Source: Asanko Gold)

After geological and structural logging, sampling in preparation for assaying was undertaken. Sampling of the core is routinely taken at 2m intervals down hole, or to geological contacts, and 1m samples are taken in the mineralised, or ore zones. This is a reasonable practice. Given that the geology and mineralised zones are well known at Nkran, the need for routine 2m sampling outside of the mineralised zones has been replaced with sampling 5m either side of the main mineralised zone, and where occasional alteration and veining are recognised.

With the start of reopening the Nkran pit, Asanko Gold has built a new core shed facility (Figure 10-13). This facility has spacious core logging facilities, a dedicated XRF / Spectrometer office, a dedicated core saw / splitter facility, covered core storage on pallets and pallet racking with a forklift, and containerised storage for pulps.

Figure 10-13: New Asanko Gold Core Shack and Logging Facility (Source: Asanko Gold)

10.3.9.2 Esaase

The storage facility at Esaase consists of sheds with elevated racks on concrete floors that are sheltered from wind and rain. The core is stored following geological logging, photography, core cutting and sampling.

10.3.10 Core Photography

10.3.10.1 Nkran

Prior to cutting, all core was routinely photographed by the geologists. The core is laid out in single trays, clearly labelled with borehole ID, and depths. Images are stored in the database. A handheld camera held above the tray being recorded by the geologist was used to photograph the core. This method is now being upgraded to a permanently secured camera dedicated to core photography.

10.3.10.2 Esaase

Prior to logging and cutting, all core was routinely photographed by the geologists. The core is laid out three trays at a time, clearly labelled with borehole ID and depth. Images are stored in the database. A handheld camera held above the tray being recorded by the geologist was used to photograph the core. This method is now being upgraded to a permanently secured camera dedicated to core photography.

10.3.11 Core Cutting and Marking

10.3.11.1 Nkran

Once logged, the core is carefully marked for Sampling (Figure 10-14). The core is then cut using a diamond blade saw.

Figure 10-14: Marking up Core prior to Cutting (Source: Spiers 2011)

For analysis, the left-hand side of the core facing down hole was always taken. This is to avoid bias and keep consistency, and is a good practice. During the core cutting process, all core is handled by several trained technicians.

10.3.11.2 Esaase

The required interval was marked on the core and the sample cut in half by electric diamond blade core saw. The standard protocol is that the cut is made 1 cm to the right in a down hole direction of the orientation line, with the left side being retained and the other half broken up for assay. In the upper Oxide zone, where the core was too friable for diamond saw cutting, the procedure was to dry cut, or cleave the core.

10.4 RC Sampling Procedures

10.4.1 RC Sampling and Logging

10.4.1.1 Nkran and Esaase

RC drill chips were collected as 1m intervals down hole via a cyclone into PVC bags, and then weighed prior to splitting. The collected samples were riffle split using a three tier Jones riffle splitter. A final sample of approximately 3 kg was collected for submission to the laboratory for analysis. All 1m interval samples were analysed. RC chip trays were systematically compiled and logged with all bulk rejects stored at the project site. Unfortunately sun damage through improper storage has destroyed most of the bulk rejects at the Project. A full record of pulp samples is stored in containers on both sites.

Grade Control drilling at Nkran was initiated mid 2015. The sampling methodology is the same as above, although samples are collected on a 1.5m down hole basis, due to the 3m flitch and 6m bench thicknesses. Currently the splitter methodology on the Grade Control rigs is being modified with instalment of the latest Metzke rotary splitters, which generate 3 equal samples (assay, backup, geology) for each sample interval.

10.5 Diamond Core Sampling and Logging

10.5.1 Sampling and Logging

10.5.1.1 Nkran

The sampling review for the project has been thorough. Initially H&S (2010) carried out a detailed review of sampling procedures and protocols; this was followed by SRK's QA/QC review (March 2011) and a later detailed QA/QC analysis of assays and densities in September 2011. This was again updated by the addition of the latest set of QA/QC data from August 2011 to January 2012, as well as that from Dynamite Hill in 2013 and 2014. The QA/QC sampling conducted during 2015 and 2016 has been incorporated into this report. CSA have conducted independent reviews of sample methodologies and QAQC practices applied by Asanko Gold on site.

SRK has observed the onsite sample processing facility at Nkran and preparation procedures (core splitting, panning, logging, sampling and storage). The observation by SRK was that the methodology and procedures used were appropriate and this is detailed in previous NI 43-101 technical reports. Subsequent site visits by CJM and latterly CSA for purposes of the updated MRE made the same conclusions.

Original sampling was predominantly carried out at 1.0m intervals, but sample intervals ranging from 0.25m to 5m have also been utilised.

It should be noted that these sampling intervals are much smaller than the true width of overall mineralised zones, which is variable throughout the deposit, but is typically in excess of 30m.

10.5.1.2 Esaase

The sampling of the core was subject to the discretion of the geologist completing the geological logging. Early in the exploration, nominal 2m intervals samples were taken unless otherwise dictated by geological, or structural features. After December 2006, the sample interval was 1m intervals with the majority (90.7%) of samples submitted to the laboratory as half core and the remaining submitted as whole, or quarter core.

The sampling intervals are significantly smaller than the true width of overall mineralised zones, which is variable throughout the deposit, but is typically in excess of 30m.

After the marking out of the required interval, the core was cut in half by electric diamond blade core saw. The cut is made 1 cm to the right (looking down hole) of the orientation line with the left side being retained and the other half broken up for assay.

In the upper Oxide zone, where the core was too friable for diamond saw cutting, the procedure was to dry cut, or cleave the core.

Core structure orientations were routinely recorded to assist in determining the controls on mineralisation, in establishing a reliable geological model for resource estimation, and to provide additional geotechnical information to determine likely blast fragmentation and pit stability characteristics.

The core is transferred from the trays and pieced together on a V-rail (angle iron) rack and the orientation line (bottom of hole), determined by the orientation tool recorded during drilling, is drawn along the entire length of the assembled core.

Geotechnical logging has recorded percentage core recovery, lithology, weathering and oxidation, rock strength, RQD percentage and rock defects including frequency, orientation, type and characteristics. A set of approximately 28 oriented core HQ3 core holes have been drilled radially outward from within the deposit through depths beyond an assortment of potential pit wall limits.

10.5.2 Drilling Orientation

10.5.2.1 Nkran

Due to the variable orientations of mineralisation in each deposit, a range of drilling orientations were used. All deposits were driller perpendicular to mineralisation wherever possible.

10.5.2.2 Esaase

The vast majority of drill holes in the west dipping mineralisation were collared at an orientation of approximately 100°(UTM). A small number of holes were drilled towards approximately 300°. Water bore holes have been drilled vertically.

10.5.3 Sample Recovery

10.5.3.1 Nkran and Esaase

Sample recovery for RC drilling was noted as very good and averages approximately 34 kg per metre drilled. Bulk sample weights (on a per meter basis) have been recorded in the database for approximately two-thirds of all RC samples drilled. Sample recovery in diamond holes was very good although recoveries for core from the moderate to highly weathered saprolite and highly fractured and brecciated zones returned poor recoveries. In general, core recoveries were in excess of 95%.

10.5.4 Sample Quality

10.5.4.1 Nkran and Esaase

The sampling procedures adopted for drilling are consistent with current industry best practise. Samples collected by diamond coring within the highly weathered zones are of moderate quality, with the remainder being high. Sample recoveries and quality for the RC drilling are high with drilling switching to diamond core once wet samples were noticed.

10.5.5 Factors Influencing the Accuracy of Results

10.5.5.1 Accuracy and Reliability of Results

There is no identified drilling, sampling, or recovery factors that materially impact the accuracy and reliability of the results of the drilling programmes.

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 QA/QC Review

The acquisition of PMI by Asanko Gold in early 2014, leading to the formation of the AGM, necessitated the collation and merging of two exploration databases. These were initially captured into an Acquire database, and more recently (2015) consolidated into a SQL Datashed database.

Data validation of Keegan, and then PMI data was previously conducted by Alpar Kovacs, an independent database Consultant. Kovacs effected the initial merge of the datasets. Subsequently AKG have implemented its own database SOP's, including standard practise Quality Assurance ("QA") and Quality Control ("QC") measures. CJM reviewed the QC for the 2012 and 2014 MRE work, but at that juncture did not validate the database. For the current MRE QA/QC results were reviewed by CSA for the general exploration data, Nkran Pit, Dynamite Hill, Akwasiso and Esaase deposits, inclusive of the 2016 Nkran Grade Control datasets. CJM reviewed the QC for the satellite deposits Abore, Asuadai, Adubiaso, Adubiaso extension, and Nkran Extension.

11.1.1 Nkran Exploration Data

11.1.1.1 Exploration Summary

The Nkran exploration data comprises databases generated by Resolute, PMI, Keegan and Asanko Gold. These have been merged and validated into one centralised (on site) database in Datashed software. The assay database comprises

- Historic assays from a variety of analytical laboratories in Ghana, vis ALS (Kumasi), Precision Labs (Bibiani), SGS (Tarkwa), Inchcape (Tarkwa), Transworld Labs (Tarkwa)
- Current analyses from ALS (Kumasi), Intertek (Tarkwa) and AKG on site assay laboratory

In respect of the historic Resolute and PMI data for Nkran gold assay results from 945 holes (DDH, RC and RCD) were reviewed and conclusions are summarised below:

- No QC for historical samples (analysed at Analabs, Inchcape and ALS (Kumasi - PMI historical -). No duplicates for SGS samples
- Overall insufficient duplicate samples to monitor sample and assay precision
- A minor number of mislabelled, or misidentified standards and blanks do occur
- A small number of incorrect dates occur in the assay batches in the database (future dates, or incorrect years)
- Specifically, SGS blank samples have 7% failures, which may indicate either possible sample contamination, or else mislabelled standards

- Standards show instances of apparent mislabelling and bias, as well as failures, and has been addressed in the database
- The number of duplicates included in the database are low and those that are there have high precision errors (45% for field duplicates), indicating that assay results have a low repeatability. This is interpreted to be due to the presence of very nuggety gold, or else could indicate that the sample sizes are too small. This has been mitigated by the implementation of bottle roll analysis of 2 kg samples for all Grade Control and exploration analyses
- A bias is observed in field duplicates which show a 22% bias to original samples at grades up to 1 g/t Au, however the dataset is small

11.1.2 Asanko Gold Grade Control Dataset

11.1.2.1 Asanko Gold Grade Control Summary

Asanko Gold Grade Control samples were initially analysed at ALS (Kumasi) and since January 2017 at the Asanko Gold Mine laboratory. Initially 50g fire assays were analysed at ALS, and on the inception of production from the main Nkran orebody a 2 kg bottle roll analytical method was introduced to mitigate the locally nuggety nature of the ore. Gold assay results for 2015 and 2016 were reviewed by CSA and conclusions are summarised below:

- The blank samples do not indicate any significant contamination
- Field duplicates have no significant bias, but have a high precision error (55%) which indicates that assay results have a low repeatability. This could be due to the presence of very nuggety gold
- There is an issue with a number of mislabelled or misidentified standards and blanks
- There are some incorrect dates in the assay batches (future dates, or incorrect years)

11.1.3 QA/QC Conclusions and Recommendations

11.1.3.1 Conclusions

- The Asanko Gold QA/QC procedures (Sampling, QA/QC and Analytical Guide Document), stipulate appropriate numbers of standards and blanks to be inserted. CSA noted that some improvement is necessary with respect to field duplicates procedures and quantities, and as to failures and how these should be resolved. Asanko Gold is currently addressing these observations.
- Historical data, (as expected), have little to no associated QC
- It appears that a small number of the QC samples and results have not been included in the master database, and therefore the quantity of QC results reviewed, is in some cases (Akwasiso and Esaase projects) insufficient to

adequately monitor the sampling and assay contamination, accuracy and precision

- There are a number of cases of apparent misidentified standards and blanks in the database which reduce the confidence in the database and data management procedures (rectified by Asanko Gold)
- Apart from the GC database, the quantity of field duplicates is not always sufficient to adequately monitor precision (being rectified)
- Some Laboratory QC results have not always been included (being rectified)
- There are instances of incorrect assay loading templates having been used, resulting in the wrong laboratory being named in the database and incorrect dates (e.g. future dates being used) (rectified)

11.1.4 Recommendations that have been addressed by Asanko Gold:

- Written QA/QC procedures need reviewing and updating to reflect current and best practices, including what constitutes a failure and the mitigating action to be taken
- All data should be included in the master database and any missing data should be merged in
- The apparent mislabelled blanks and standards should be investigated and where possible, corrected in the database
- The proportion of field duplicates should be increased to 5% and biased to mineralised samples
- Any available laboratory QC results should be included in the database and going forward all laboratory QC results should be included
- Due care should be taken to ensure that the correct loading templates are used and that blanks and standards are correctly identified

11.1.5 QP Qualifying Statement

Malcolm Titley, CSA Principal Consultant and QP for Nkran, Akwasiso and Dynamite Hill is satisfied that notwithstanding the issues identified above (which have been rectified by Asanko Gold), the drilling and sampling data available for the MRE is suitable for use and will provide a reliable Mineral Resource Estimate.

Cognisance has been taken of the QA/QC audits undertaken by SRK in 2011 and 2012 (Spiers 2011, SRK Technical Report 2012). In the QP’s opinion there are no significant biases between historic and Asanko Gold QA/QC.

11.1.6 Nkran QA/QC Conclusions

CJM and CSA are both of the opinion that the QA/QC undertaken by the three companies, Resolute, PMI and Asanko Gold is adequate and that the current QA/QC systems in place at Nkran to monitor the precision and accuracy of the sampling and assaying are adequate and should continue to be implemented.

A recommendation that umpire analyses are performed for available samples from the resource areas has been adopted on an ongoing basis. It should be noted that with the initiation of mining Grade Control drilling assays have been conducted by Leachwell bottle roll on 2 kg RC chip splits, as opposed to 50g FA.

11.2 Sampling Practice and Security

- CJM and CSA have reviewed the sampling procedures (see Sections 10.4 and 10.5), preparation and security in transit and on the mine site laboratory. The methodologies applied are standard practice and consistently undertaken
- Grade Control and Exploration RC samples are taken on the site of the drilling rig from a rotary splitter which generates equal aliquots to mitigate any bias. All assays are now standardized, and are conducted by Leachwell bottle roll on 2 kg RC chip splits

Coarse rejects of all samples are kept as a backup for at least three months (Grade Control) and six months (Exploration).

Data validation has been undertaken historically as part of the 2014 MREs in this report have not changed due to there being no material change to the supporting evaluation data. The following is an extract from the 2014 NI 43-101 disclosure, and specifically is applicable to Esaase, Abore, Asuadai and Adubiaso Main deposits. The QP's do not disclaim responsibility for the information set out in this sub-section.

11.2.1 Sample Handling Prior to Dispatch

11.2.1.1 Nkran

Resolute and PMI Sampling

CJM was not able to review the sample submission procedures in practice as utilised by Resolute or PMI personnel. CJM has made use of the observations and findings of SRK utilised for their May, 2012 MRE exercise. CJM, however, did review the written standards and procedures as employed by PMI.

SRK visited the onsite sample processing facility at Nkran and the preparation procedures (core splitting, panning, logging, sampling and storage) employed by PMI personnel. It was SRK's opinion that the methodology and procedures used were appropriate.

According to SRK, original sampling was predominantly carried out at 1.0m intervals, but sample intervals ranging from 0.25m to 5m had also been utilised.

SRK did not carry out any independent sampling of core, or RC samples, but has reviewed the sample quality control protocols introduced and conducted by PMI and SGS in Tarkwa.

Individually bagged core and RC drilling samples were packed in polyweave, or heavy plastic sacks (i.e. 5-10 samples per sack), tied with binding wire and prepared for transport to the laboratory. All samples were firmly secured and locked in a designated sample room at PMI's field office. The company geologist, responsible for core logging and RC sampling, held the only key to the room where samples were secured. The geologist was responsible at all times for their secure shipment to the laboratory. SRK considered that the sample preparation, security and analytical procedures adopted by PMI provide an adequate basis for the current MRE.

Asanko Gold Sampling

CJM and CSA have reviewed Asanko Gold sampling handling prior to dispatch, which is in line with previous protocols. SOP's are in place and monitored. The new Asanko core shed facility has improved the sampling and dispatch process, as well as increasing sample storage facilities.

Esaase

The close scrutiny of sample submission procedures by Asanko Gold and Coffey Mining technical staff, and the rapid submission of samples from drilling for analysis, provides little opportunity for sample tampering. Equally, given the Umpire assaying via an external international laboratory and the regular 'blind' submission of international standards to both the primary and Umpire assay facilities, any misleading analytical data would be readily recognised and investigated. Current Asanko Gold sampling procedures require samples to be collected in staple-closed bags once taken from the rig or core-cutting facility. The samples are then transported to the project camp to be picked up by the laboratory truck. The laboratory truck then takes them to the laboratory directly.

11.2.2 Sample Preparation and Analysis Procedures

Nkran

PMI

PMI procedures are included here as the PMI drilling is still the major informant to the Nkran orebody MRE. In 2014 CJM was not able to review the PMI Sample Preparation and Analysis Procedures due to no sampling or drilling exercises being in process at the time. CJM reviewed the findings of SRK in May 2013 and reviewed the standards and procedures for PMI and shares the opinion of SRK that sample preparation was adequate for the resultant assays to be utilised for MRE. SRK assumed the continuity of processes and procedures, based on their historical exposure to the tenements.

As a result of the advanced nature of the projects, SRK conducted an analysis of the QA/QC and found that results were sufficient for establishing the highest possible degree of confidence in the data. CJM reviewed the QA/QC, as well and shares SRK's opinion.

PMI typically inserted random blank samples into the assay stream. These blanks consistently returned very low assays. Additionally, any samples in which visible gold was noted, during the logging, or in the case of panning RC drill chips, or any samples which returned high gold grades, were routinely submitted for either screened metallic, or a bulk cyanide leach assay. In addition, random pulps and rejects were submitted to other certified labs for checking or confirmation purposes.

According to SRK, comparison of the results from the various different assays and laboratories utilised, indicates a high measure of confidence in the assay data. The assay labs utilised by PMI utilised their own in-house QC programmes. These included standards, replicas, duplicates and blanks. In general, every batch of 50 solutions contained two standards positioned randomly; two replicates positioned at the end of the rack; two duplicates selected randomly and positioned immediately under the original and one blank positioned randomly.

All sampling was carried out under the direct supervision of PMI senior personnel, either the President, VP of Exploration, project manager, or the Chief Geologist. All drill cores from the concessions were geologically and structurally logged, split (sawn), photographed and stored at PMI's field offices, or sampling and storage facility in Nkran.

During a site visit in March 2011, SRK was able to visit the SGS laboratory in Tarkwa. SRK observed the sample preparation area inclusive of sample crushing and pulverisation, drying ovens, dust extraction devices, cupellation laboratory, furnaces and the analytical laboratories. The main comment from the routine inspection was a concern regarding the high levels of dust in the preparation and analytical facility of the very busy laboratory. It should be noted that the laboratories used by PMI are certified, being local divisions of international laboratories such as SGS (Bibiani and Tarkwa) and ALS (Kumasi).

All crushing and grinding was carried out by the analytical laboratory. Sample pulps and coarse reject material was returned to PMI only after completion of both the initial sampanalysis and any additional checks which PMI may have required following receipt of the initial sample assays. All assays were carried out by Fire Assay on 50g samples with an Atomic Absorption (AA) finish, or as otherwise reported.

The quality of analysis at the laboratories was monitored by the use of blanks, standards, duplicates and check assays and re-runs at alternate labs (in this instance ALS Kumasi). Historically, during 2008 to 2009, all analysis were carried out by SGS / Analabs in Bibiani. Recent samples were processed by SGS Tarkwa laboratories.

All core samples were submitted to SGS in Tarkwa and check samples to ALS Commercial Laboratory in Kumasi. All samples were analysed for gold, either by 5g Fire Assay or Screen Metallic Fire Assay with AA Finish (AA26); or for cyanide leach, depending on peculiar features and characteristics of the rock or the drill cuttings. Screen metallic fire assaying is often used for samples suspected of being high grade where coarse gold is anticipated. Remaining samples, expected to represent “waste” or non-ore mineralised are analysed using straight fire assay.

Esaase

RC field duplicate samples were routinely collected to allow assessment of the field sampling error, (or bias) once the laboratory error, determined from analysis of pulp duplicates, has been subtracted. Acceptable reproducibility has been identified during an assessment of RC field duplicate data generated and no distinct bias is evident.

Reference material is retained and stored at the Asanko Gold exploration camp at Tetrem, as well as chips derived from RC drilling, half-core and core photographs generated by diamond drilling, and duplicate pulps and residues of all submitted samples. Assessment of the data indicates that the assay results are generally consistent with the logged alteration and mineralisation, and are entirely consistent with the anticipated tenor of mineralisation.

11.2.3 Quality Assurance and Quality Control

A number of analytical laboratories have been used during exploration and mining operations (Table 11-1).

Table 11-1: Summary of Analytical Laboratories used for the AGM (source: Asanko Gold)

Laboratory	Locality	Period	Accreditation	Au Assay Method	Lower Detection Limit
SGS	Accra	1995	ISO/IEC 17025	Fire assay	0.01 g/t
Inchcape	Obuasi	1995-1997	ISO/IEC 17025	Fire assay	0.01 g/t
Analabs	Nkran Site	1997-1998	ISO/IEC 17025	Fire assay	0.01 g/t
SGS	Bibiani	2009 - 2012	ISO/IEC 17025	Fire assay	0.01 g/t
SGS	Tarkwa	2010-2011	ISO/IEC 17025	Fire assay	0.01 g/t
Min Analytical	Perth	2011-2014	ISO/IEC 17025	Fire assay	0.005 g/t
ALS Kumasi	Kumasi	2006-present	ISO 9001:2000	Fire assay, Leachwell Bottle Roll	0.01 g/t
Trans World	Tarkwa	2009-2010	ISO/IEC 17025	Fire assay	0.01 g/t
Intertec	Tarkwa	2010-present	ISO/IEC 17025	Fire assay, Leachwell Bottle Roll	0.01 g/t
Performance Labs	Bibiani	2010-2012	ISO/IEC 17025	Fire assay	0.01 g/t
Asanko Gold	Nkran site	From 2017	Nil	Leachwell Bottle Roll & Fire Assay	0.01 g/t

Nkran QA/QC Procedures

The quality control methods adopted at Nkran were made by 3 different companies: Resolute, PMI and Asanko Gold.

Table 11-2: Summary of Validated Samples for AGM up to 2014 (source: Asanko Gold)

Prospect	Resolute data		PMI data		Asanko Gold Data	
	Sample	Available/ Validated	Sample	Available/ Validated	Sample	Available/ Validated
Abore	19,941	0%	5,102	99.76%	-	-
Adubiaso	17,823	38%	3,526	99.94%	-	-
Asuadai	8,198	0%	6,906	99.88%	-	-
Dynamite Hill	-	-	10 110	100.00%	5,179	100.00%
Nkran	60,753	13%	23,687	96.23%	-	-

11.2.4 Quality Control Procedures

The quality control procedures adopted by Resolute and the relevant analytical laboratories are listed in point form below.

SGS Accra - used in 1995

Preparation: "The samples have been sorted and dried. Primary preparation has been by jaw crushing the whole sample to -6mm, cone crushed or disk milled to -2mm. A 300g to 1 kg sub-fraction is then pulverized to -200 mesh in labtechnic homogenizing mill".

Method - Au FA: "Have been analysed by firing a 50 gm proportion of the sample. The resulting Noble Metals have been determined by Atomic Absorption"

INCHCAPE Obuasi - used in 1995, 1997

- Preparation: "Dry, Crush, Pulp 2 Kg"
- Extraction: "Fire Assay+HCL-HNO3"
- Method - 309: "30 g Fire Assay - AA"
- Preparation: few batch of the samples were prepared in ANALABS Obotan with unknown details
- Method - F650: "50 g fire assay, Lead collection, Aqua Regia digest, AAS"
- Method - F644: included sample preparation "Drying @105 °C, Ringmill 500g to 1.5 Kg nominal 75 microns" and method "Screen fire assay, 106

micron. Reported results include weight fraction, gold in individual size fractions and calculated gold"

ANALABS Nkran - used in 1997, 1998

- Preparation: "Drying, Jaw crushing to nominal 6mm to 12mm. Sample volume reduction - riffle split. Ringmill <1 kg, nominal 75 microns."
- Method - P625: "Aqua Regia digest, DIBK extraction, AAS, 25g sample", Method - F650 with the same details described previously in ANALABS Bibiani paragraph

Assay Method

The assay method and quality control procedures adopted by PMI and the relevant analytical laboratories are listed in point form below:

- SGS Bibiani - used from 2009
- Method - FAA505: "Gold by fire assay, AAS finish, 50g weight, detection limits 0.01 – 100 ppm"
- Method - ARA155: for As "Aqua Regia Digest 50g-250ml, AAS, detection limit 20¬50000 ppm"
- SGS Tarkwa - used from 2010 to 2012
- Method - FAA505: "Gold by fire assay, AAS finish, 50g weight, detection limits 0.01 – 100 ppm"
- Method - ARA155: for As "Aqua Regia Digest 50g-250ml, AAS, detection limit 20¬50000 ppm"
- Method - ARE155: for Au "AAS after Aqua Regia Digest, DIBK, 50g"
- Method - FAS31K: "Screen fire assay at 106 qm"
- Intertek Tarkwa - used in 2010
- Method - FA51/AAS: "Lead collection fire assay, 50g sample weight, AAS detection limit 0.01 ppm"
- ALS Kumasi - used in 2011
- Method - Au-AA26: "Fire Assay Fusion, 50g, AAS Finish, detection limits 0.01-100 ppm"
- Min Analytical Perth - used from 2011 - 2014
- Method - FA50AAS: "Lead collection fire assay using 50g sample weight, AAS Finish, detection limit 0.005 ppm"
- Performance Laboratories Bibiani - used from 2012 - 2014
- Method - FAS50/AAS: "Fire Assay / 50 gm aliquot, AAS finish"
- Method - BLG/AAS: "Dissolved Au on Solution samples, BLEG, leachwell"
- Quality Control Procedures Asanko Gold

- The assay method and quality control procedures adopted by Asanko Gold for general exploration and the relevant analytical laboratories are listed below:

SGS Tarkwa - from 2014 - 2015

Method - FAA505: "Gold by fire assay, AAS finish, 50g weight, detection limits 0.01 – 100 ppm"

ALS Kumasi – from 2015

Method - Au-AA26: "Fire Assay Fusion, 50g, AAS Finish, detection limits 0.01-100 ppm"

Asanko Gold Nkran site – from 2017

Method - 2kg Bottle roll for 3 hours (with 1 x Leachwell pill), AAS finish, detection limits 0.01 – 100ppm

Esaase

Keegan utilised the following procedures:

- Insertion of 16 (Geostats Standards and CDN Resource Standards) internationally certified standard reference material (5% of samples)
- Insertion of blank material (5% of samples)
- RC field duplicates taken (5% of samples)
- Diamond Core Field duplicates completed by a second split at the 3mm jaw-crushing stage
- Submission of selected Umpire samples to SGS Tarkwa
- Review of the Esaase and the internal laboratory QC data on a batch by batch basis
- The assay quality control procedures applying to the various laboratories are summarised in the following sections:

SGS Tarkwa

The following quality control procedures are adopted by SGS which is part of the global group of SGS laboratories with ISO/IEC 17025 accreditation:

- Cross referencing of sample identifiers (sample tags) during sample sorting and preparation with sample sheets and client submission sheet
- Compressed air gun used to clean crushing and milling equipment between samples
- Barren quartz 'wash' applied to the milling/pulverising equipment at the rate of 1:10
- Quartz washes assayed to determine the level of cross contamination
- Sieve tests carried out on pulps at the rate of 1:50 to ensure adequate size reduction

- Assaying of certified standards at the rate of one per batch of 20
- A minimum of 5% (1:20) of the submitted samples in each batch are subject to repeat analysis
- Blank samples inserted at the rate of approximately 1:30
- Industry recognised certified standards disguised and inserted at a rate of 1:30
- Assaying of internal standards data
- Participation in two international round-robin programmes; LQSi of USA and Geostats of Australia

Transworld Tarkwa

TWL applies most of the QC procedures used by SGS although it only participates in the Geostats round-robin Umpire assay programme and it does not utilise the CCLAS computer system. TWL Tarkwa was acquired by Intertek Minerals Group in October 2008. Intertek Minerals Group includes Genalysis Laboratory Services Pty Ltd of Australia and operates in accordance with ISO/IEC 17025, which includes the management requirements of ISO 9001:2000.

ALS Kumasi

The following quality control procedures are adopted by ALS which is part of the global group:

- ALS Laboratory Group with ISO 9001:2000 accreditation:
- Cross referencing of sample identifiers (sample tags) during sample sorting and preparation with sample sheets and client submission sheet
- Compressed air gun used to clean crushing and milling equipment between samples
- Barren 'wash' material applied to the milling/pulverising equipment at between sample preparation batches
- Quartz washes assayed prior to use to determine the level of cross contamination
- Sieve tests carried out on pulps on a regular basis to ensure adequate size reduction
- Assaying of certified standards at the minimum rate of one per batch (dependant on batch size and assay technique)
- A minimum of one of the submitted samples in each batch are subject to repeat analysis
- Blank samples inserted at the beginning of each batch
- Participation in a number of international round-robin programmes which include CANMET of Canada and Geostats of Australia

Quality Control Analysis

The quality control data analysed by Coffey Mining at Esaase included:

- Standard and blanks (both Field and Laboratory)
- RC Field duplicates
- Laboratory repeats
- Re-assayed pulps; and
- Umpire assaying
- The assay quality control data, as they pertain to resource estimates completed on the basis of data available, were subset into the categories above, and reviewed separately. The quality control data was assessed statistically using a number of comparative analyses for available datasets. The objectives of these analyses were to determine relative precision and accuracy levels between various sets of assay pairs and the quantum of relative error. The results of the statistical analyses are presented as summary plots, which include the following:
  - Thompson and Howarth Plot showing the mean relative percentage error of grouped assay pairs across the entire grade range, used to visualise precision levels by comparing against given control lines
  - Rank % HARD Plot, which ranks all assay pairs in terms of precision levels measured as half of the absolute relative difference from the mean of the assay pairs (% HARD), used to visualise relative precision levels and to determine the percentage of the assay pairs population occurring at a certain precision level
  - Mean vs. % HARD Plot, used as another way of illustrating relative precision levels by showing the range of % HARD over the grade range
  - Mean vs. %HARD Plot is similar to the above, but the sign is retained, thus allowing negative or positive differences to be computed. This plot gives an overall impression of precision and also shows whether or not there is significant bias between the assay pairs by illustrating the mean percentage half relative difference between the assay pairs (mean % HARD)
  - Correlation Plot is a simple plot of the value of assay 1 against assay 2. This plot allows an overall visualisation of precision and bias over selected grade ranges. Correlation coefficients are also used
  - Quantile-Quantile (Q-Q) Plot is a means where the marginal distributions of two datasets can be compared. Similar distributions should be noted if the data is unbiased

Transworld Laboratory, Tarkwa

TWL Duplicate Repeats

At TWL, every 20th sample is duplicated. A duplicate is two separate samples taken from the total pulped sample. Duplicate repeats are analysed in the same batch and are therefore not subject to intra-batch variance. Only assays greater than 10 times the detection level (>=0.1 ppm Au) are included in the assessment and data are divided into drill core (HQ and NQ, 177 assays) and riffle split 1m RC drill chips (461 assays). Results show equivalent means between the duplicate repeats and precision within acceptable limits for both diamond core and RC samples.

TWL Pulp Respray

After initial calibration of the AAS with control standards, the batch is sprayed (the aspirator tube is placed in the DIBK layer and approximately 1 ml is sprayed into the AAS flame). On combustion, the absorbance is measured by the AAS and the strength of the absorbance is proportional to the gold concentration). At the end of spraying, the operator returns to every 10th sample and performs the same operation and this is the Pulp Respray. At the end, control samples are again presented to the AAS to verify that short-term drift has not occurred. Only assays greater than 10 times the detection level (>=0.1 ppm Au) are included in the assessment for a total of 1202 assays. Results show equivalent means between the duplicate repeats and precision well within acceptable limits.

TWL Check Repeats

Check repeats occur where high grade samples are encountered or where the result is out of sequence (e.g., 0.01-0.04-0.02-1.2-0.03: Result 1.2 is out of sequence and would be repeated). A repeat is a second 50 g sample taken from the same kraft envelope as the original analysis (Au1) and is thus different from the duplicate repeat. Check Repeats are analysed later than the original assay (in a different batch) and may therefore be subject to intra-batch variance compared with the original result. Only assays greater than 10 times the detection level (>=0.1 ppm Au) are included in the assessment and data are divided into drill core (HQ and NQ, 265 assays) and riffle split 1m RC drill chips (573 assays). Check Repeat analyses data to September 2007 was available for review. Results show equivalent means between the duplicate repeats and precision within acceptable limits for both diamond core and RC samples

TWL Pulp Re-assay

Only pulp re-assays greater than or equal to 10 times the detection level (0.1 ppm Au) are considered for analysis and these comprise 1,615 riffle split 1 m RC drill chip assays. Results show equivalent means between the duplicate repeats and precision within acceptable limits. TWL Lab Standards and Blanks Analysis six certified standards were inserted by TWL into the sample batches at a rate of one in twenty in addition to preparation blanks and reagent blanks at a similar rate. The supplied database only contains Lab standards analysis received to September 2007. A total of 3,512 standards and blanks assays are available for analysis. Results generally show a positive bias that varies between -0.44% and 3.05%. This positive bias is more evident for higher grade standards.

SGS Laboratory, Tarkwa

SGS Duplicate Second Split

This comprises RC (339) and diamond core (73) field duplicates and is achieved by taking a second split at the 3mm jaw crushing stage of the sample preparation. Results show equivalent means and a high level of precision between the original and the re-assay for both DC and RC samples.

SGS Replicate First Split

These assays represent a random repeat assay with four random repeats completed from each batch of 50 samples. A total of 582 DC and 2,392 RC analyses are available for analysis. Results show equivalent means and an acceptable level of precision between the original and the re-assay.

Lab Standards and Blanks Analysis

4 certified standards were inserted by SGS into the sample batches at a rate of one in twenty in addition to preparation blanks and reagent blanks at a similar rate. The supplied database only contains lab standards analysis received to September 2007. A total of 938 standards and blanks assays are available for analysis. Results show a relative low bias of up to -2.09%.

ALS Laboratory, Kumasi

ALS Duplicate Second Split

This comprises RC (176) and DC (62) duplicates and is achieved by taking a second split at the 3mm jaw crushing stage of the sample preparation. Results show equivalent means and a high level of precision between the original and the re-assay for the diamond core samples. Results for the RC samples demonstrate a high level of precision between the original and the re-assay however the second mean is 7.5% lower than the original assay.

ALS Replicate

These assays represent a random repeat assay of a second sample taken from the original pulp. A total of 223 DC and 892 RC analyses are available for analysis. Results show equivalent means for diamond core however the second mean for the RC samples is significantly lower than the original. Overall levels of precision between the original and the Re-assay are low for both DC and RC samples.

ALS Intra Batch Analysis

These assays represent a random repeat assay analysed in a different assay batch to the first. Results show equivalent means and acceptable precision (although at the lower end) for both RC and DC samples.

A total of 16 Certified Standards and one blank were included in sample batches sent to TWL, ALS and SGS. A total of 11,507 assays were available for analysis. Where identifiable, outliers to the data which are obviously a misplaced standard have been removed from the data before analysis resulting in 9,818 valid standard assays. Results show a moderate positive bias of up to 6.09% for Transworld Laboratories. There is no relationship between grade and bias. One standard shows negative bias of 5.33%.

Blind standards analysis at SGS shows a spread of bias with one standard displaying a significant negative bias of up to -8.41%. In addition, one standard shows a positive bias of 6.93%. Again, there is no relationship between grade and bias. Blind standards analysis at ALS shows a spread of bias from -3.65% to 5.64%. Negative bias is apparent at lower grades and positive bias up to 5.64% is seen in two standards at 2.58 g/t Au and 2.74 g/t Au. For higher grade samples the bias approaches zero.

Esaase Field Duplicates

Field duplicates totalling 1,567, 1163 and 2,802 were sent to TWL, SGS and ALS respectively. DC field duplicates consist of a portion of the “coarse rejects” obtained after the crushing stage. RC field duplicates consist of a second sample split from the reject sample in the field. Only assays returning values greater than ten times the detection limits (>0.1 ppm Au) and less than 5 g/t Au have been considered in the analysis. Results for TWL, SGS and ALS show equivalent means and acceptable precision for both RC and DC samples.

Esaase Assay Resplits (Umpire)

In January and February 2007 a total of 1,197 RC samples were re-split and sent for analysis at SGS Tarkwa (TWL was the primary laboratory for the initial analysis). Only assays >0.1 g/t Au are considered in the analysis and a total of 481 assay pairs are available for analysis. Results show a significantly lower mean (by 15.6%) for analysis completed at SGS (although this is significantly reduces if outliers to the data are removed). SGS Tarkwa has been utilised as a primary laboratory for the project since February 2007 and umpire samples numbering 1,633 have subsequently sent to Genalysis of Perth for umpire analysis. Only assays >0.1 g/t Au are considered in the analysis and a total of 1,572 assay pairs are available for analysis. Results show equivalent assay means for the pairs between ALS and Genalysis and between SGS and Genalysis. The means of the assay pairs between TWL and Genalysis show high bias for TWL, a finding which is supported by Standards analysis. Precision is less than acceptable for all comparisons and this requires investigation.

- Adequacy of Sample Preparation Esaase
- Analytical Laboratories

Preparation and assaying of samples from the Esaase deposit were carried out at three independent laboratories:

- SGS Tarkwa (“SGS”) (from April 2007)
- Transworld Tarkwa (“TWL”) (from October 2006)
- ALS Kumasi (from November 2007)
- Sample Preparation and Analytical Procedure
- Transworld Tarkwa
- The assay method applied by TWL Tarkwa for the Esaase drilling is summarised below. All aspects of sample preparation and analysis were undertaken at TWL Tarkwa:
    - Sample Preparation
    - 3 kg or less of sample is dried, disaggregated, and jaw crushed to 3mm. Sample is pulverised to a nominal 95% passing -75 µm using an LM2 pulveriser. Two pulp samples are taken for analysis and pulp storage

- o Sample Analysis
- o 50 g charge, Fire Assay fusion, lead collection, AAS determination to 0.1 ppm

SGS Tarkwa

The methodology for the 50g fire assay from the SGS Tarkwa laboratory is the same as that completed at TWL. All aspects of sample preparation and analysis were undertaken at SGS Tarkwa. SGS is part of the global group of SGS laboratories with ISO/IEC 17025 accreditation.

ALS Kumasi

The assay method applied by ALS Kumasi for the Esaase drilling is summarised below. All aspects of sample preparation and analysis were undertaken at ALS Kumasi. ALS is part of the global group ALS Laboratory Group with ISO 9001:2000 accreditation.

Sample Preparation:

- 3 kg, or less of sample is dried, disaggregated, and jaw crushed to 2mm with a nominal 70% passing 2 mm
- Sample is pulverised to a nominal 85% passing -75 µm using an LM2 pulveriser. Two pulp samples are taken for analysis and pulp storage.
- Sample Analysis
- 50g charge, Fire Assay fusion, lead collection, AAS determination to 0.1 ppm.

Esaase QA/QC Conclusions

- CJM and CSA are of the opinion that the QA/QC undertaken by Coffey Mining is adequate and that the current QA/QC systems in place at Esaase to monitor the precision and accuracy of the sampling and assaying are adequate and should continue to be implemented. Pertinent conclusions from the analysis of the available QA/QC data include:
  - o Use of Certified Standard Reference material has shown a significant relative low bias for SGS Laboratories, Tarkwa
  - o Use of Certified Standard Reference material has shown a relative high bias for Transworld Laboratories, Tarkwa and this interpretation is supported by the umpire analysis programme
  - o Repeat analyses have confirmed that the precision of sampling and assaying is generally within acceptable limits for sampling of gold deposits
  - o Umpire analysis at Genalysis in Perth has shown a lack of precision between the various laboratories. This is currently unexplained and requires investigation
  - o Adequacy of Procedures

- o Analytical procedures associated with data generated to date are consistent with current industry practise and are considered acceptable for the style of mineralisation identified at Esaase

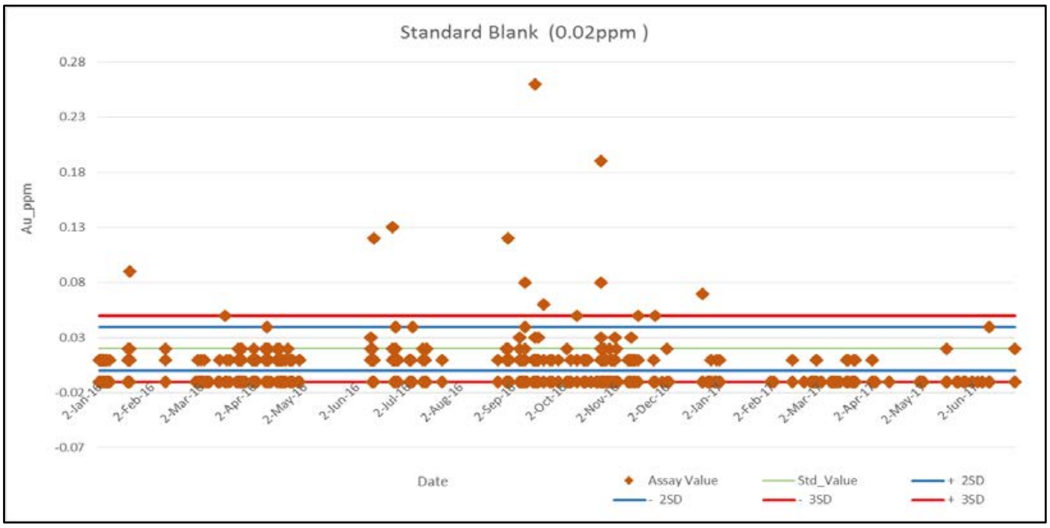
11.2.5 QA/QC for Exploration (From 1st January 2016 – 30th June 2017)

Statistics of QA/QC (source: Asanko Gold)

	QAQC Stats for Exploration		% of QA/QC usage
1	No. of Blanks	1,134	3.573
2	No. of Certified Reference Materials (CRM)	1,830	5.765
3	No. of Duplicates	683	2.152
Total		**3,647**	**11.490**

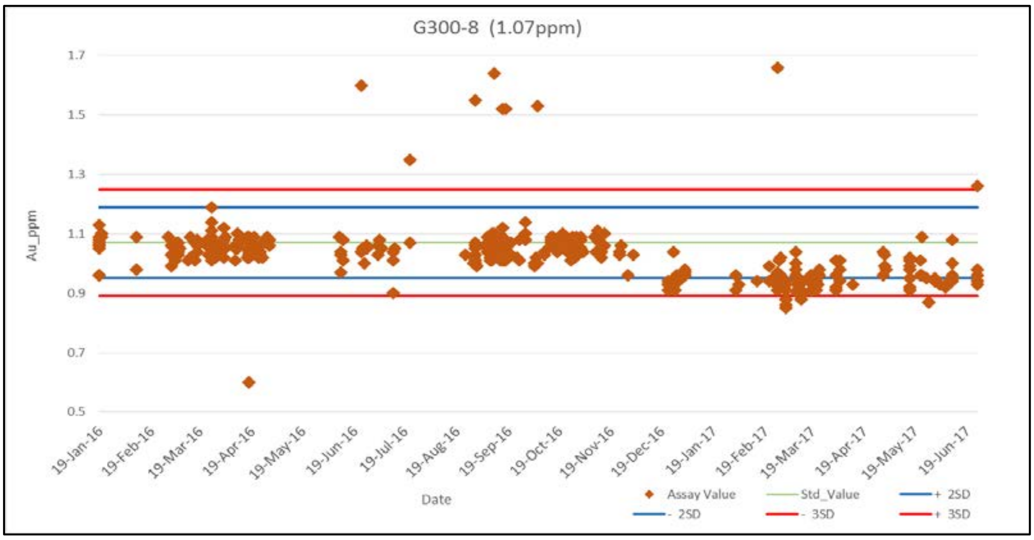
Total Samples submitted to the lab (01 January 2016 – 30 June 2017) = **31,741**

QA/QC Performance Chart

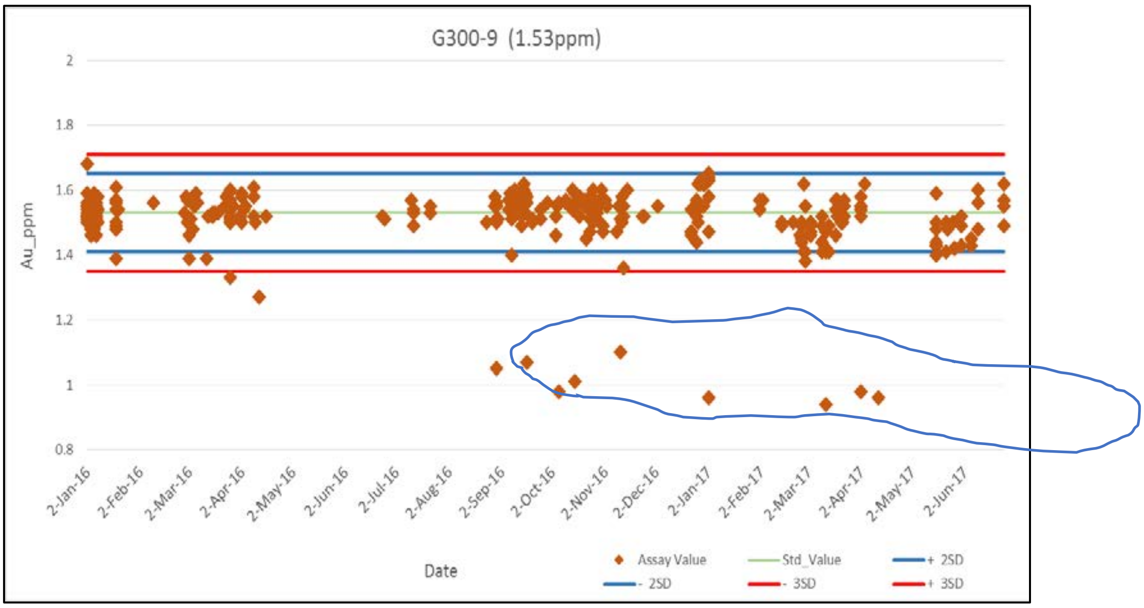


About 80% of standard blanks passed at +/- 2 and 3 standard deviations (SD).

Some of these failures were re-assayed and later passed at 2SD. Those not re-assayed were because they fell within the waste zone.



G300-8 chart above shows erratic distribution of standards along the mean line whilst indicating a few failures. Investigation has shown the Asanko Gold Mine Laboratory had problems with volumes, drying and weighing, which have been rectified now.

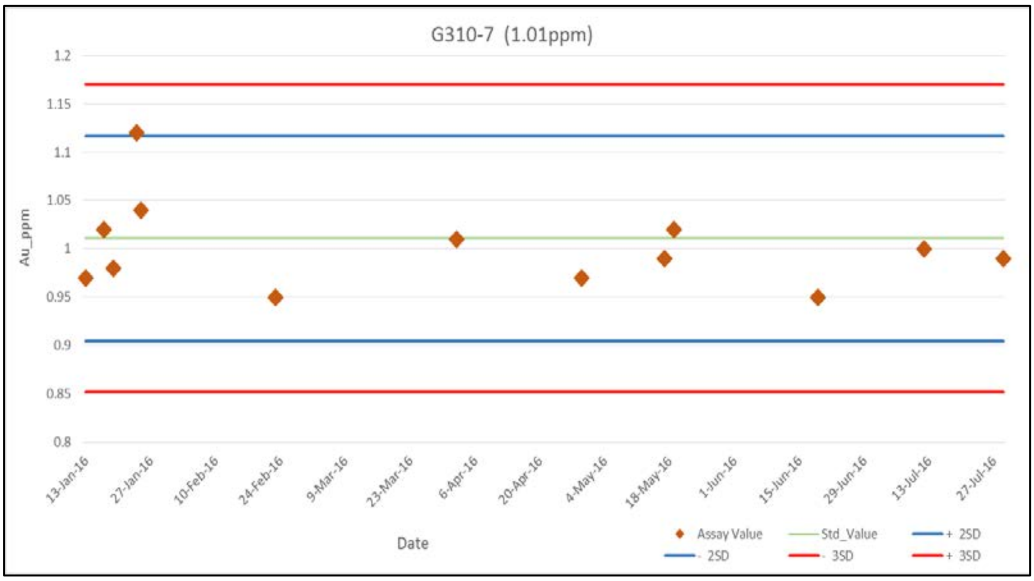


G300-9 shows uniform distribution of standards along the mean line with a few failures.

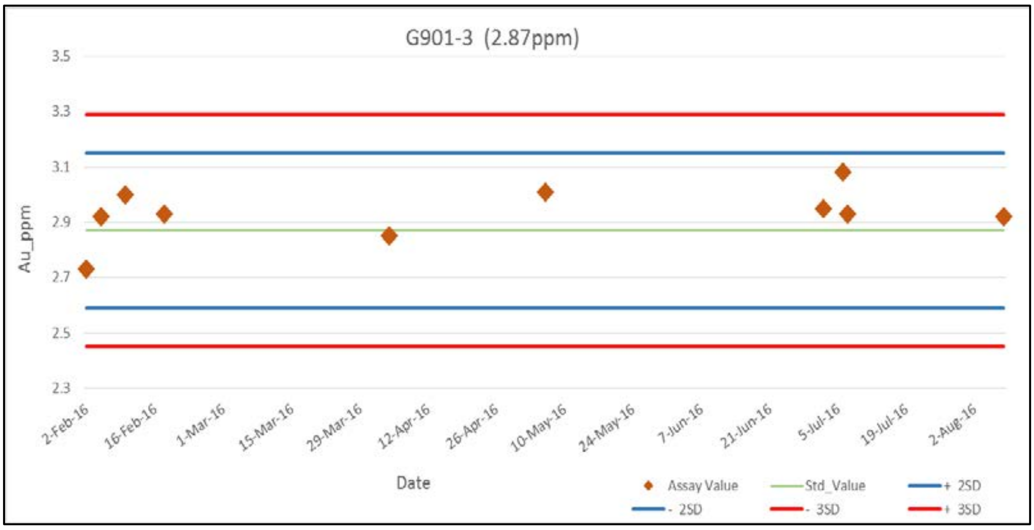
The standards highlighted indicate wrong label of standards centered at 1ppm mean.

This trend is evident from September 2016 to April 2017.

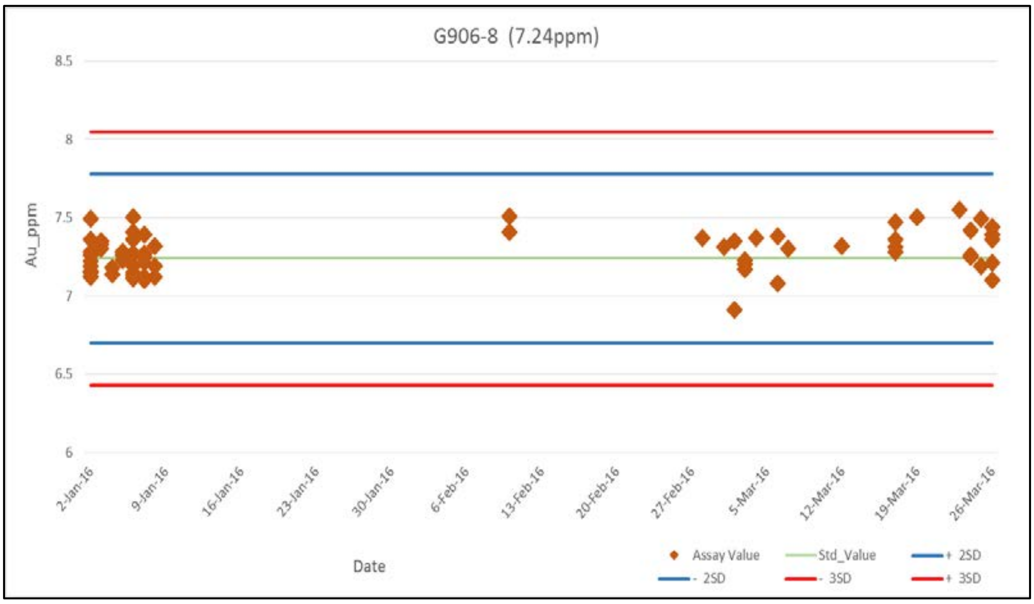
To avoid wrong labeling of standards, Asanko Gold has labelled all CRMs on site.



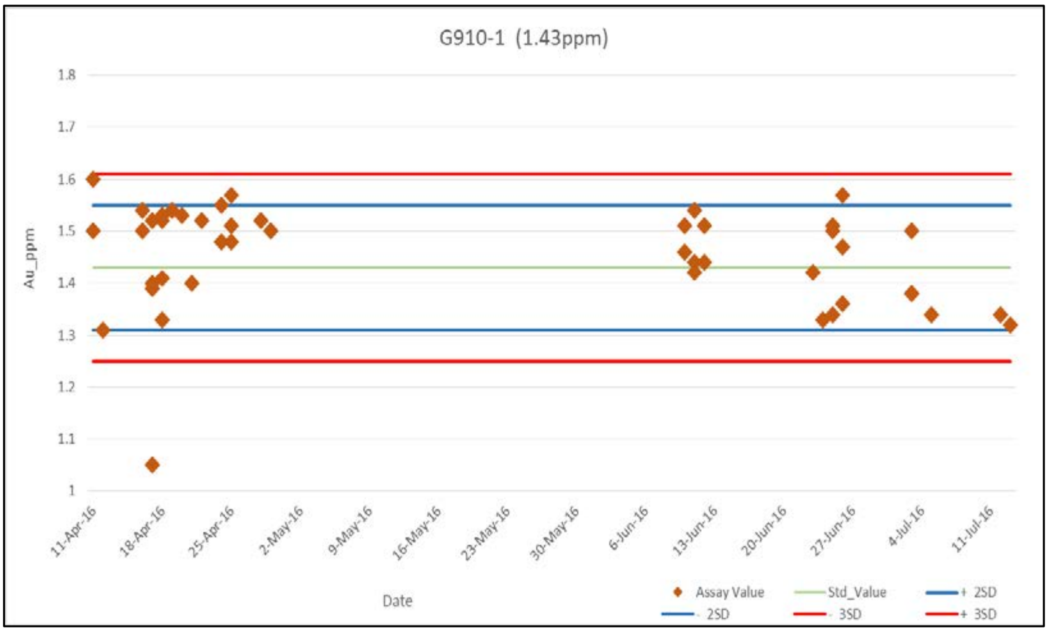
G310-7 All standards passed at +/-2 & 3 standard deviations



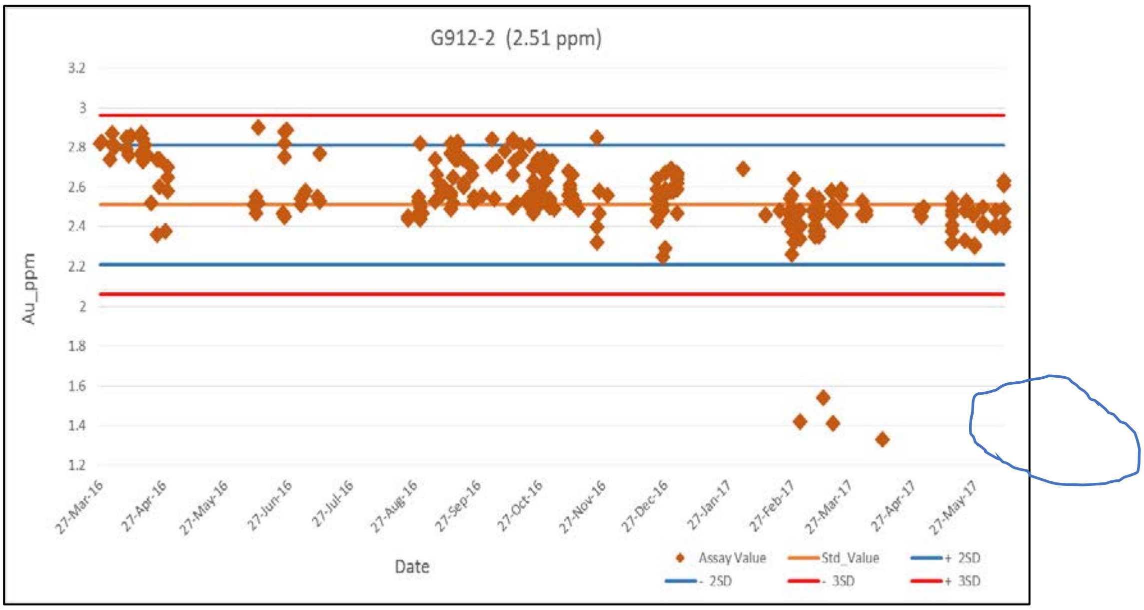
G901-3 All standards passed at +/-2 & 3 standard deviations



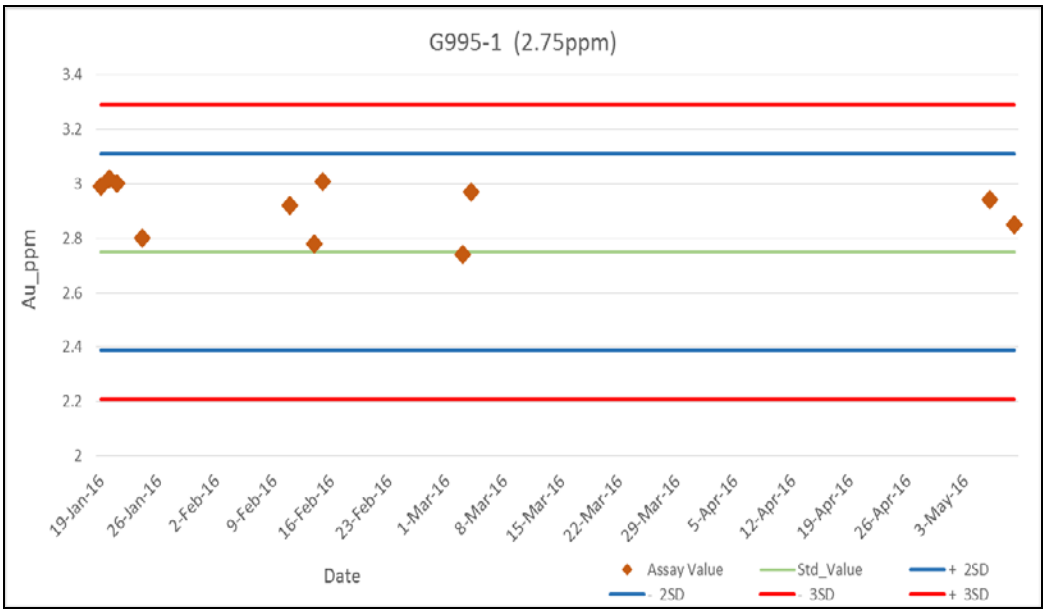
G906-8 shows uniform distribution of standards along the mean line but with intermittent breaks in its usage.



G901-1 One standard analysis failed at -3SD, this was re-assayed and later passed at +2SD.



G912-2 shows random distribution of standard analyses with a high positive bias from March 2016 and gradually moved towards a negative bias from Feb 2017. The highlighted area indicates wrong labelling of standards, which has been rectified.



G995-1 All standard analyses passed at +2SD.

In general, there has been an improvement in the standard performance and monitoring of the standards. From the QC chart above, it can be seen that the month of March 2017 was quiet erratic with some failures and wrong sample labels. These issues have been rectified and the trend improved again from April 2017 to date.

Analysis of Field Duplicate Samples

Chart 1: x=y @ 25g/t

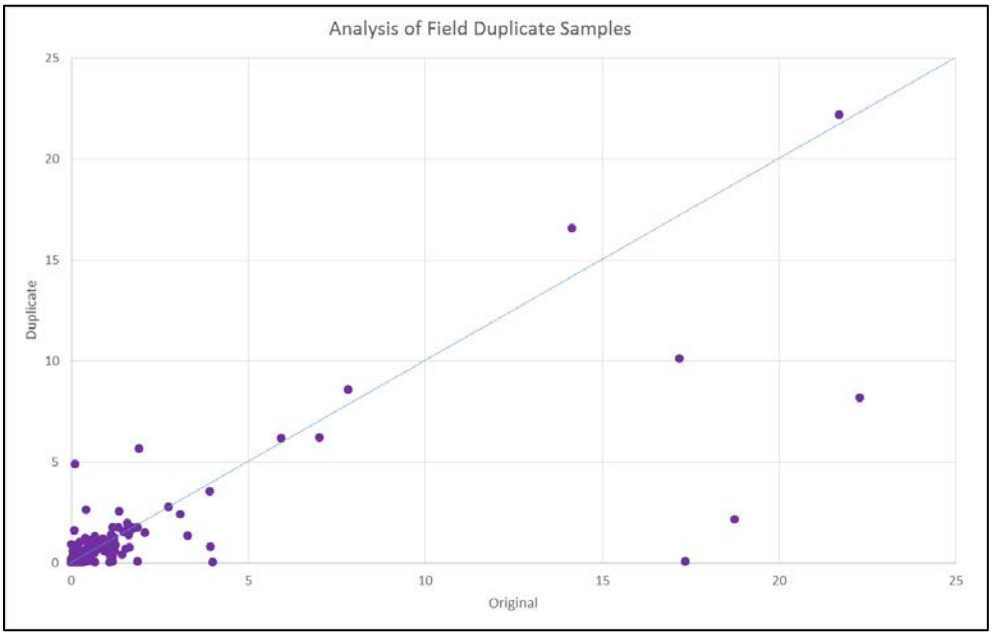


Chart 2 : x=y @ 10g/t

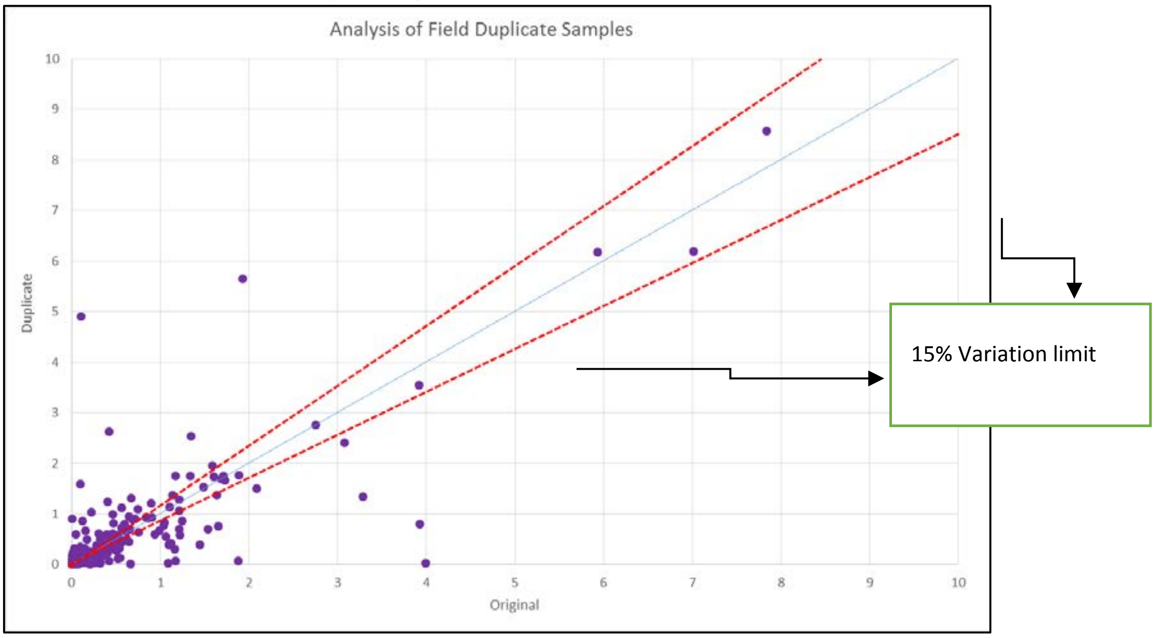


Chart 3: x=y @ 2g/t

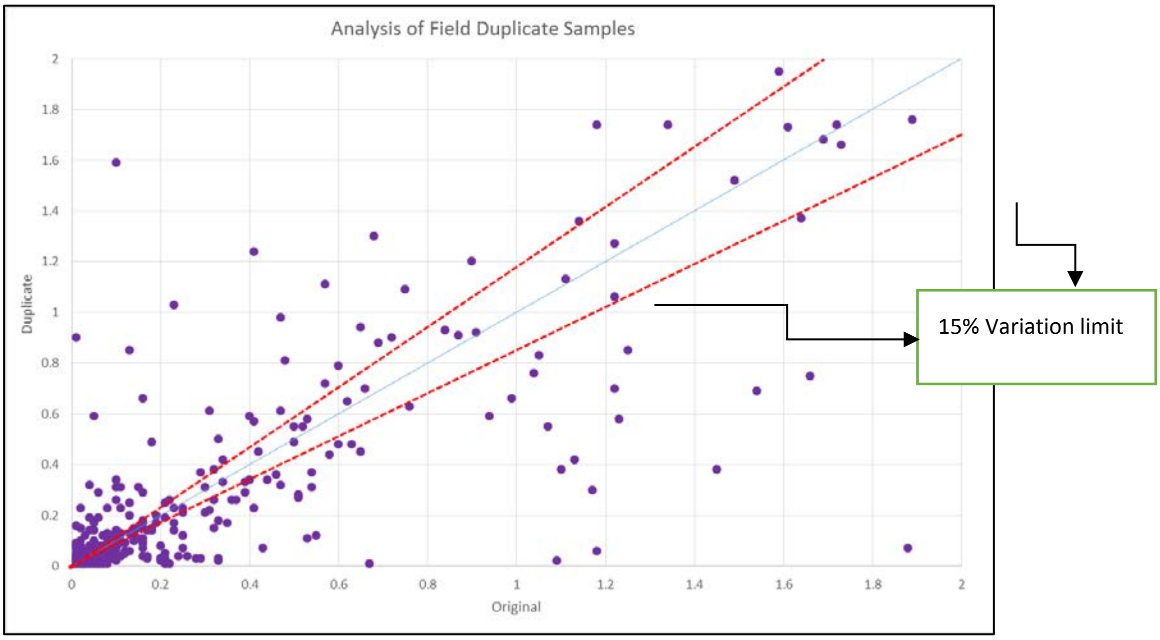
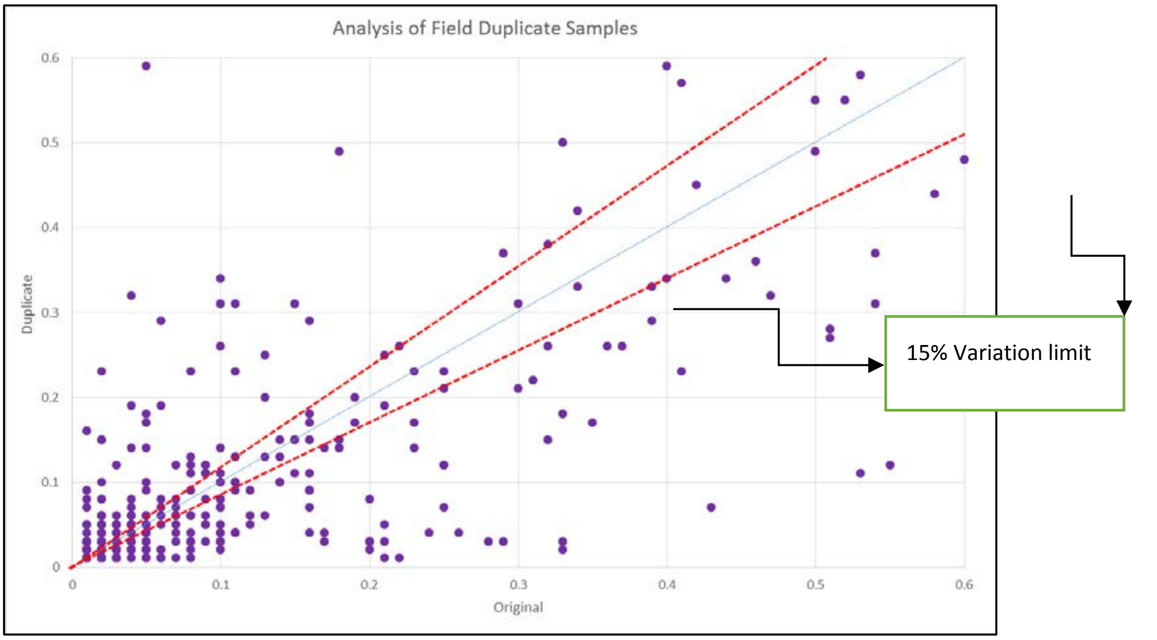


Chart 4: x=y @ 0.6g/t

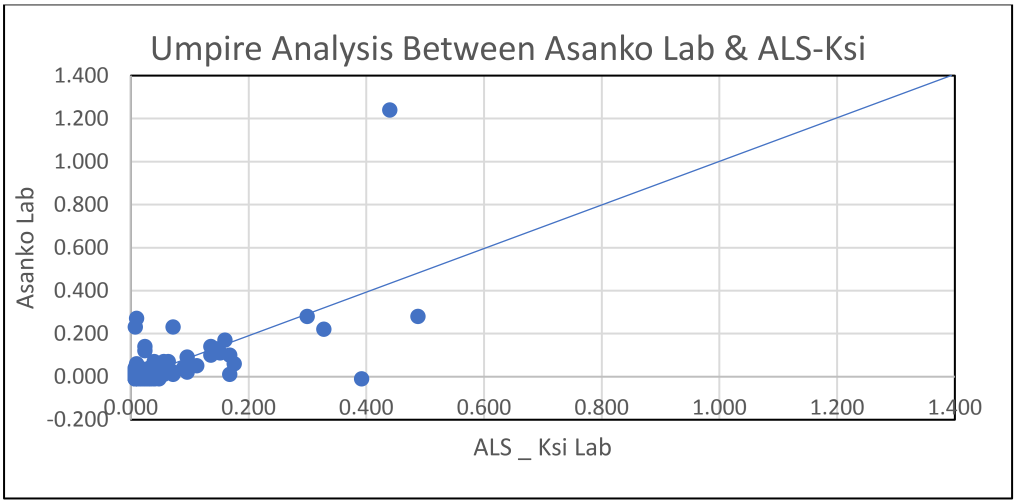


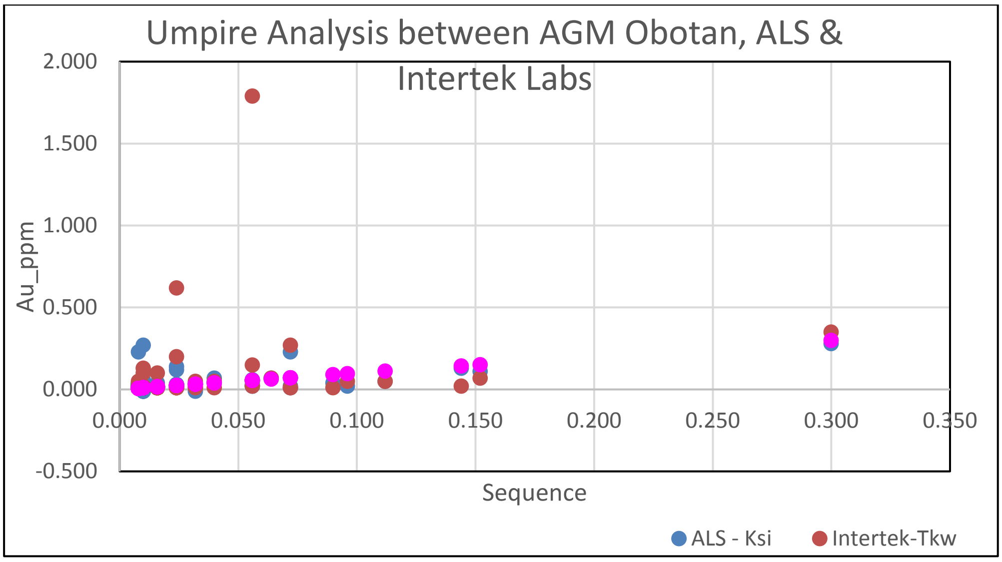
In the above field duplicate analyses, there is some level of precision and repeatability within the 15% variation limit. Out of about 683 field duplicate samples, about 20% reported some inconsistency. From chart 3 & 4, it can be seen that the duplicates correlate well with a cut off of 2g/t and 0.6g/t, with a few outliers.

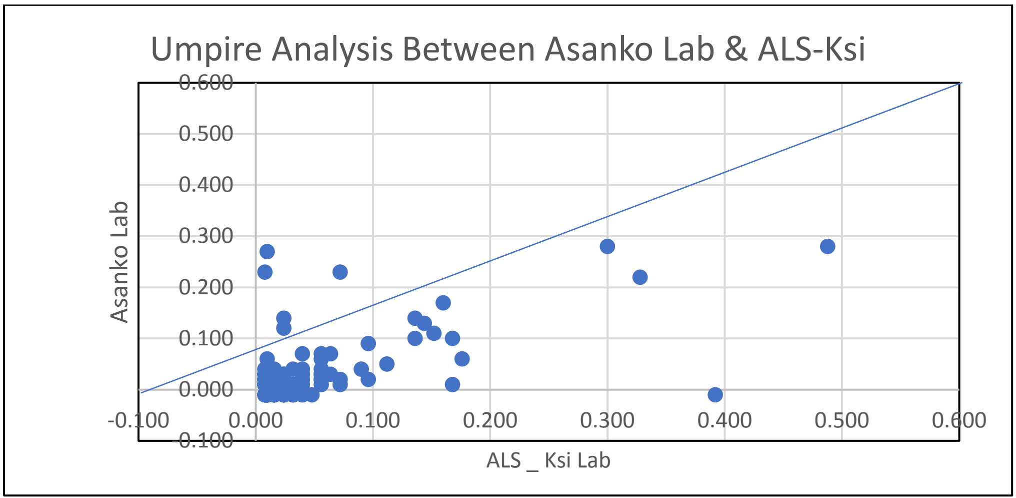
Umpire Analyses

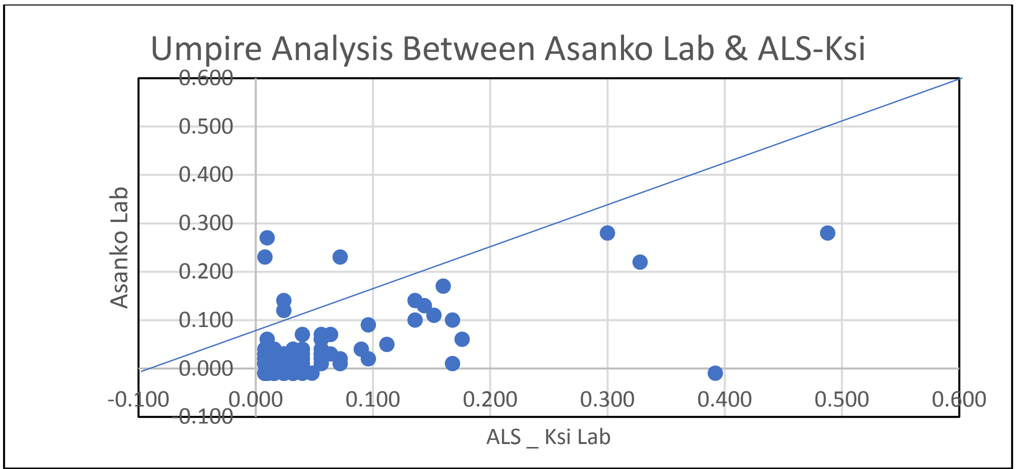
These charts represent umpire assays of 200g of pulverized samples sent to AGM Lab and ALS for a Leachwell bottle roll Analysis since February 2017. The database is still growing, so the analysis of results is a currently restricted. The initial umpire consignments were of low grade sample (waste 0 g/t to low grade 0.6 g/t Au). The current results show ALS a little lower than the AGM laboratory. Intertek and SGS labs in Tarkwa are being added to the consignment list.

The umpire assay procedure adopted by Asanko Gold involves dispatches every month of a range of samples that include ore and waste material. Future dispatches will include a range of different ore grades 0.7 – 1.5 g/t Au, 1.5-2.5 g/t Au, and > 2.5 g/t Au to continue building up the analyses database.









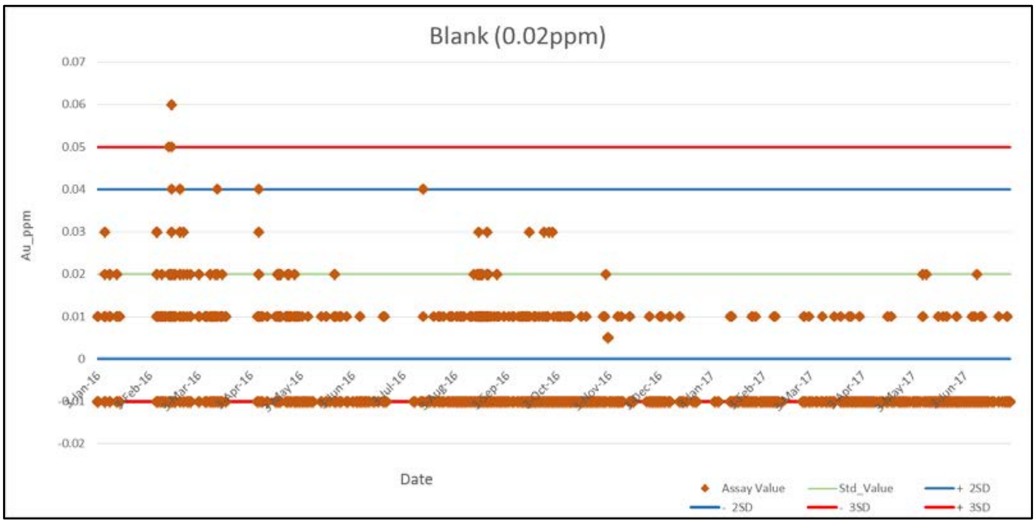
11.2.6 QA/QC for Asanko Gold Grade Control (1 January 2016 – 30 June 2017)

Statistics of QA/QC (source: Asanko Gold)

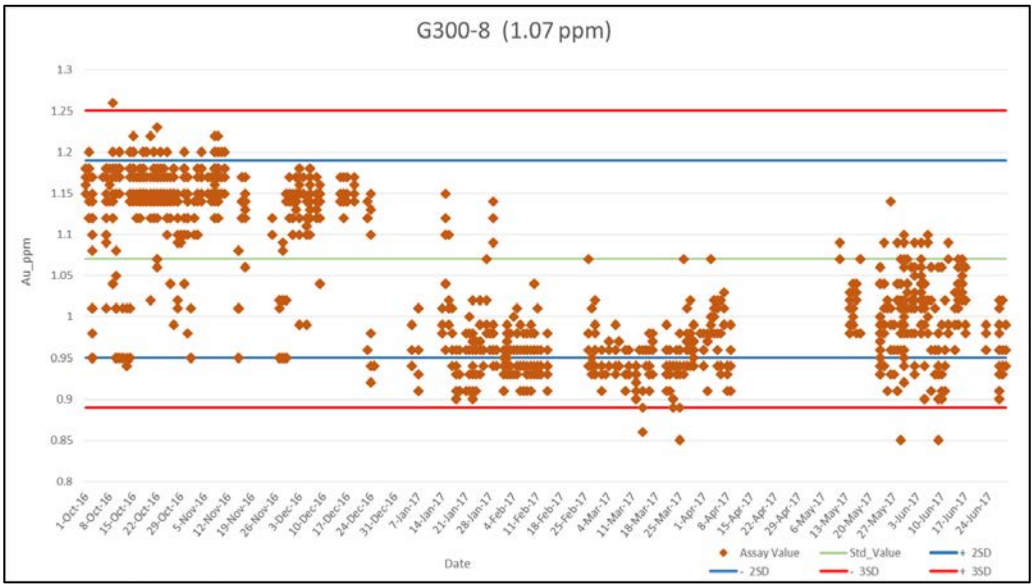
	QA/QC Stats for Grade Control		% of QA/QC usage
1	No. of Blanks	5,852	3.833
2	No. of Certified Reference Materials (CRM)	5,551	3.636
3	No. of Duplicates	4,715	3.088
Total		**16,118**	10.558

Total Samples submitted to the lab (Jan 01, 2016 – June 30, 2017) = **152,667**

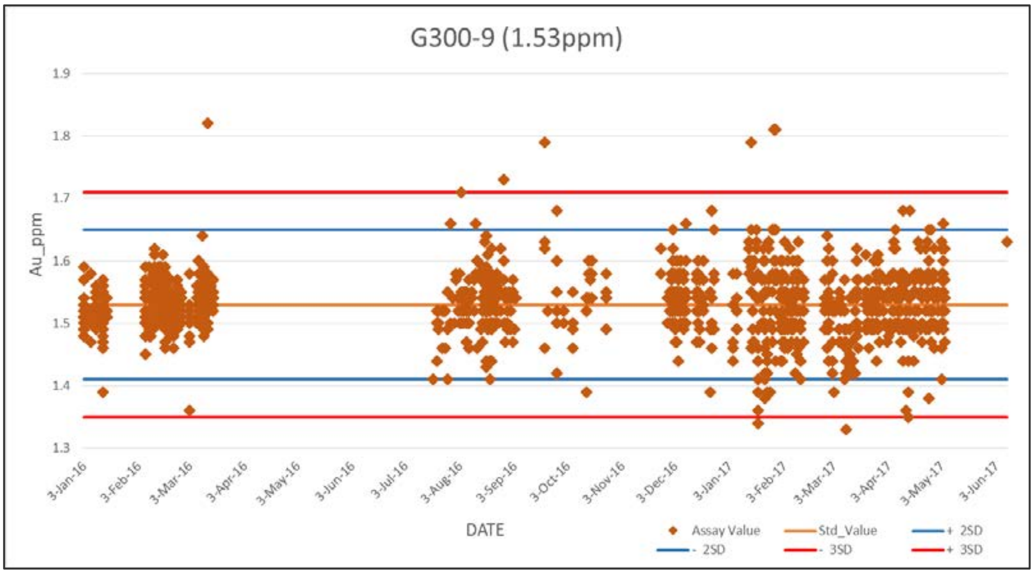
Standard Performance Chart



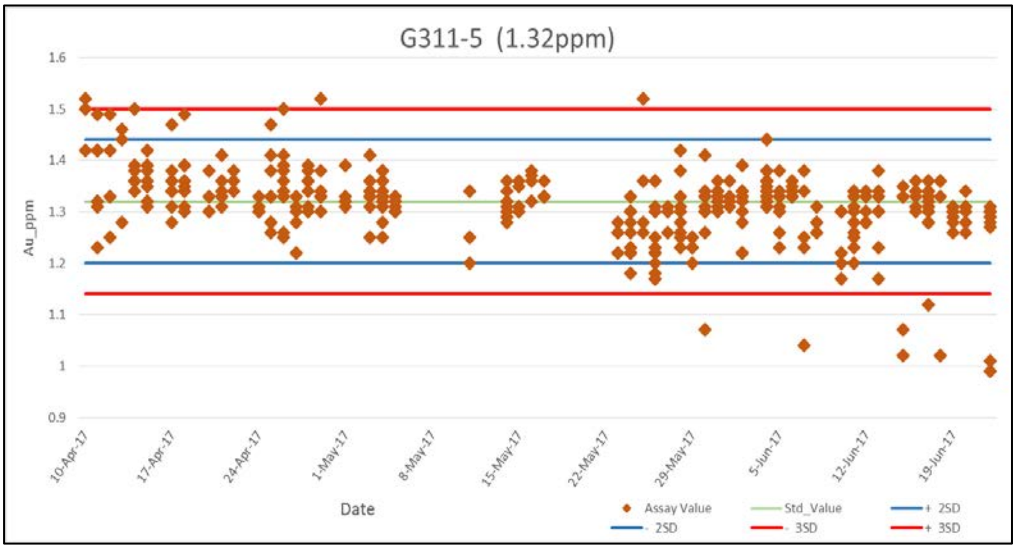
About 98.1% of blanks passed at +/-2&3 standard deviation (SD) with the remaining 1.90%% failing at +3SD. These failures were not re-assayed as it fell within a waste zone.



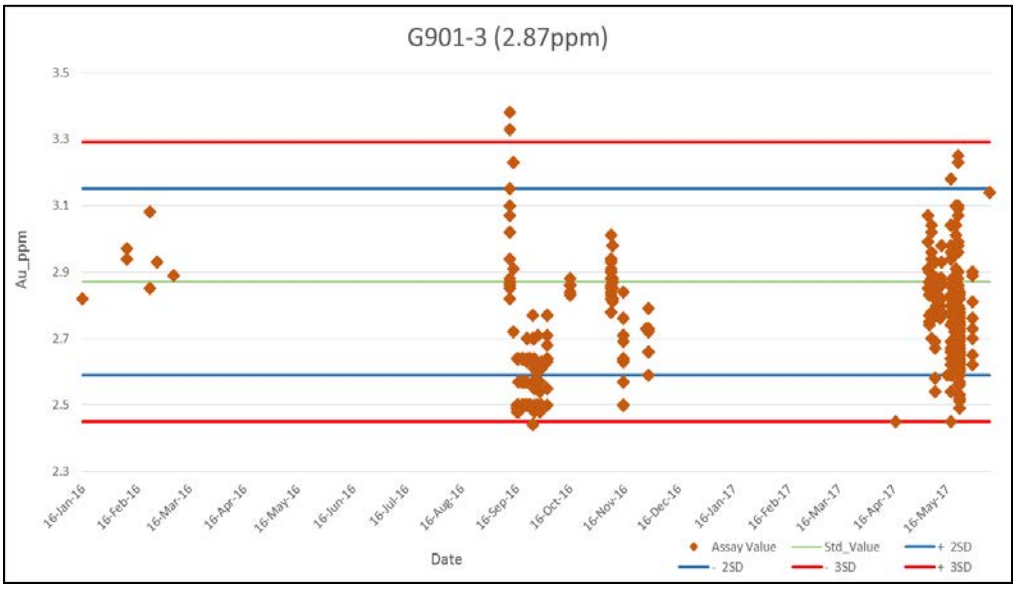
Most of the standards fell within +/-2&3 standard deviation (SD) with a high positive bias in October 2016 but with a clear cut negative bias from January 2017 to date. This however has some few failures. The percentage of failure is 3.8%. The negative and positive biases according to the lab was due to sample volume measurement precision. This was rectified when their attention was drawn to the bias. This was reflected in an improvement in 2017.



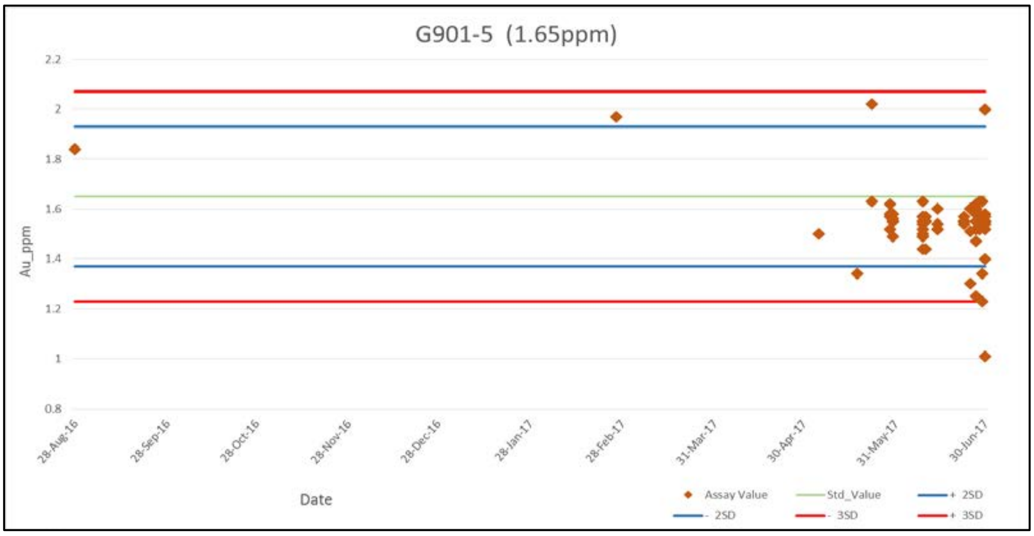
Most of the standards passed at +/-2&3 standard deviation (SD). The percentage of failure is 2.60% . These failures were not re-assayed as it fell within a waste zone.



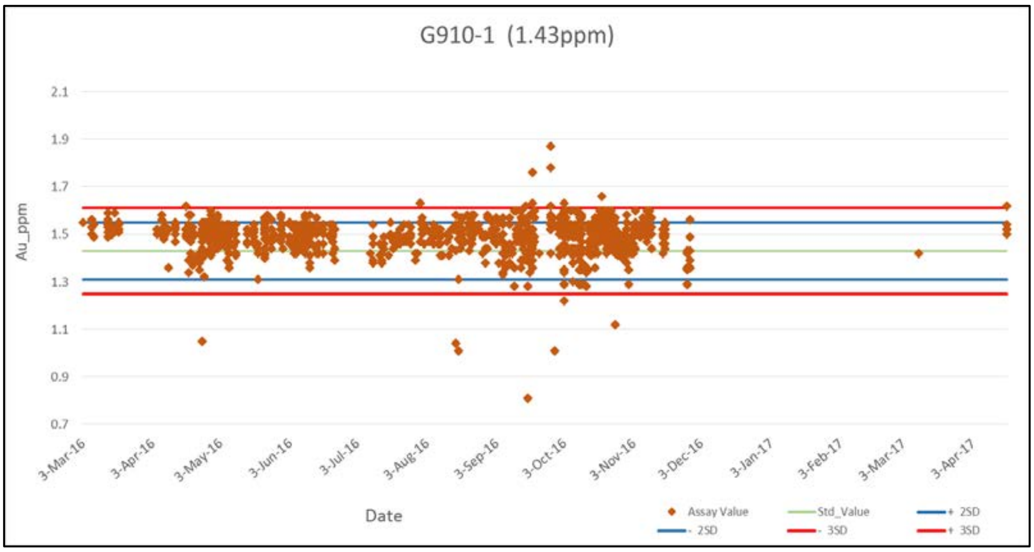
These standards show consistent distribution within +/- 2 standard deviation with some few failures at -3SD. The percentage of failure is 2.96%.



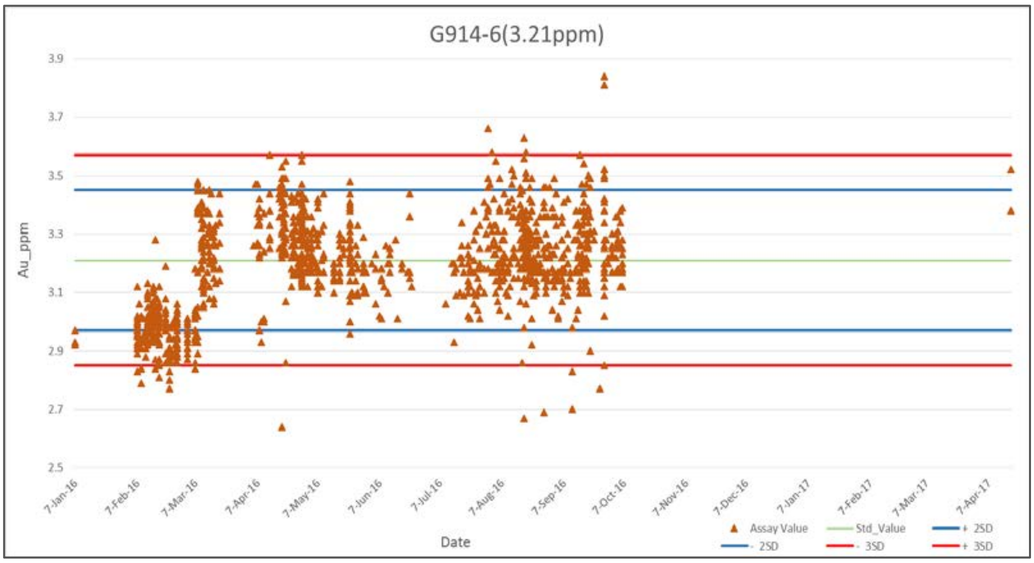
These standards passed at +/- 2&3 Standard deviation with 7.7% failure. The usage of this standard was high during September 2016 & May 2017.



Most of the standards fell within -2SD with a few failed ones. The percentage of failure is 1.7%. Again the usage of this standard was high in June 2017.



Random distribution of standards within +/-2&3 standard deviation (SD) between March & December 2016 with a few outliers. The percentage of failure is 1.77%. This results were from ALS_Ksi.



Standards showing a trend of negative bias from February and gradually to a uniform distribution along the mean line in September 2016. There were however some few failures. The percentage of failure is 3.97%. February 2016 was when the AGM laboratory was setup and hence it's likely they encountered some challenges and this is evident in the bias movement from February – October 2016.

11.2.7 Analysis of field Duplicate Samples

Chart 1: x=y @ 70g/t

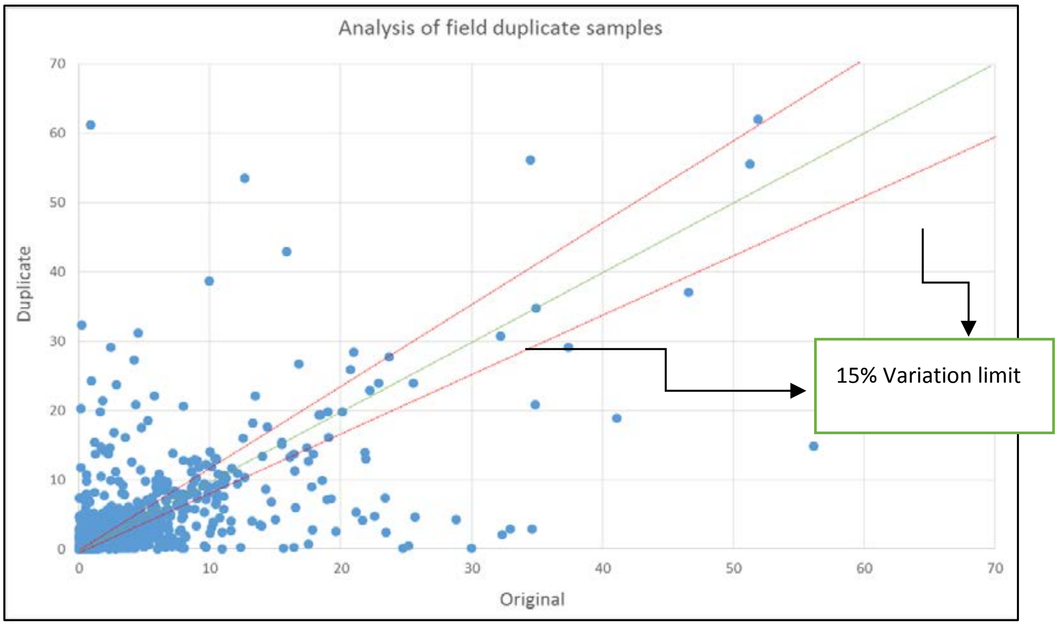


Chart 2: x=y @ 30g/t

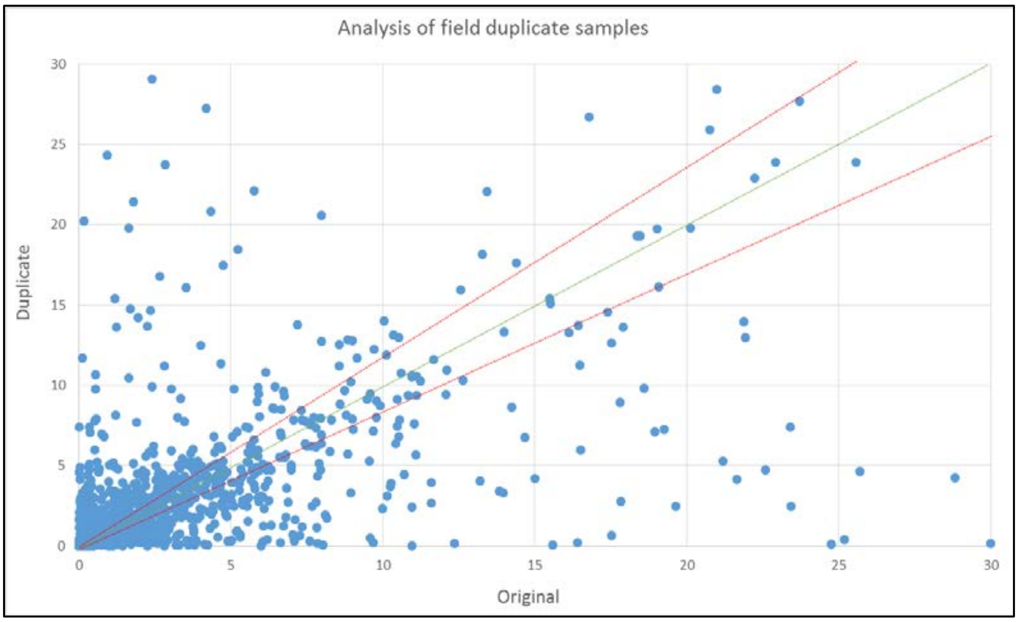


Chart 3: x=y @ 10g/t

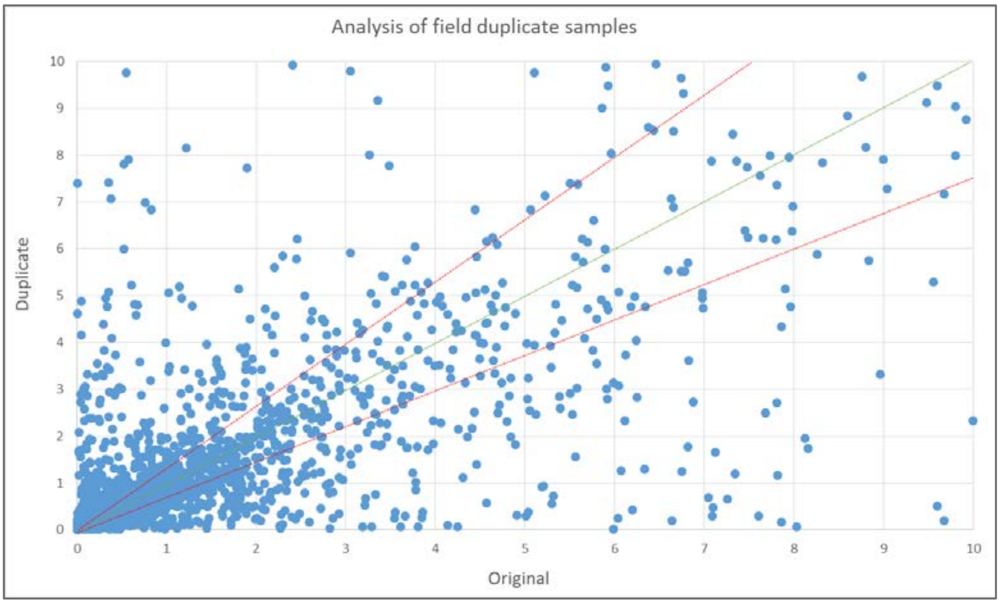
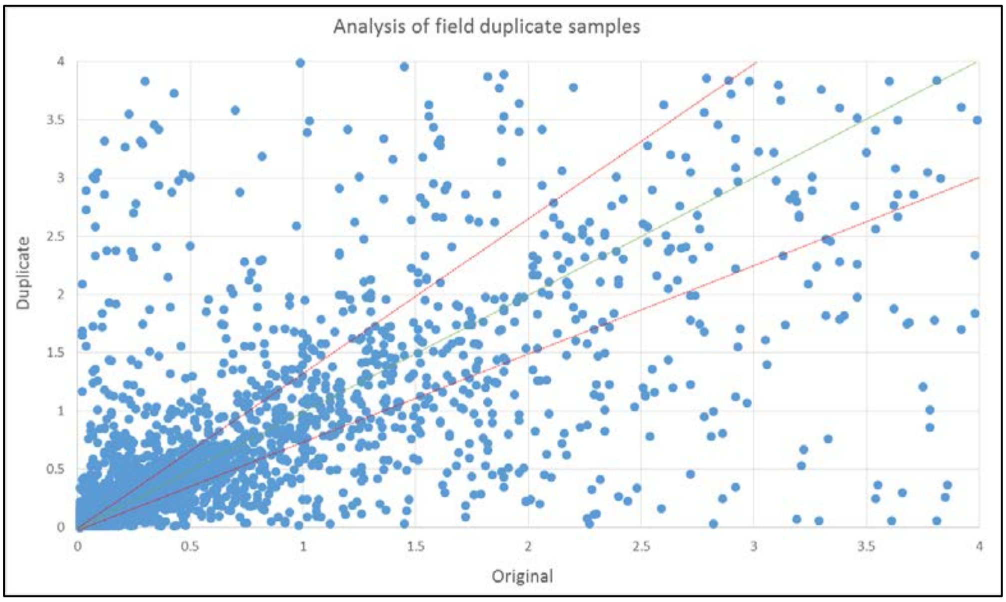


Chart 4: x=y @ 4g/t



In general, a comparison of the duplicate with different cut off content shows a wide scatter between the original and duplicate samples although 50% of the RC drilling duplicates lie outside of the nominal 15% tolerance or the variation limit, the overall trend in the scatterplot depicts a poor precision and its inconsistent.

The method of duplicate sample collection is in the process of being reviewed to ensure consistency and good precision at all ranges, or cut-offs. The current method of duplicate collection is that the splitting is done on the field, labelled as a routine sample and sent to the lab. The RC rigs are all being upgraded to latest Metzke rotary splitters, that will enable better control and representivity of sampling.

11.2.8 Historic Analytical Laboratory Repeat Analyses Graphs (Section 11)

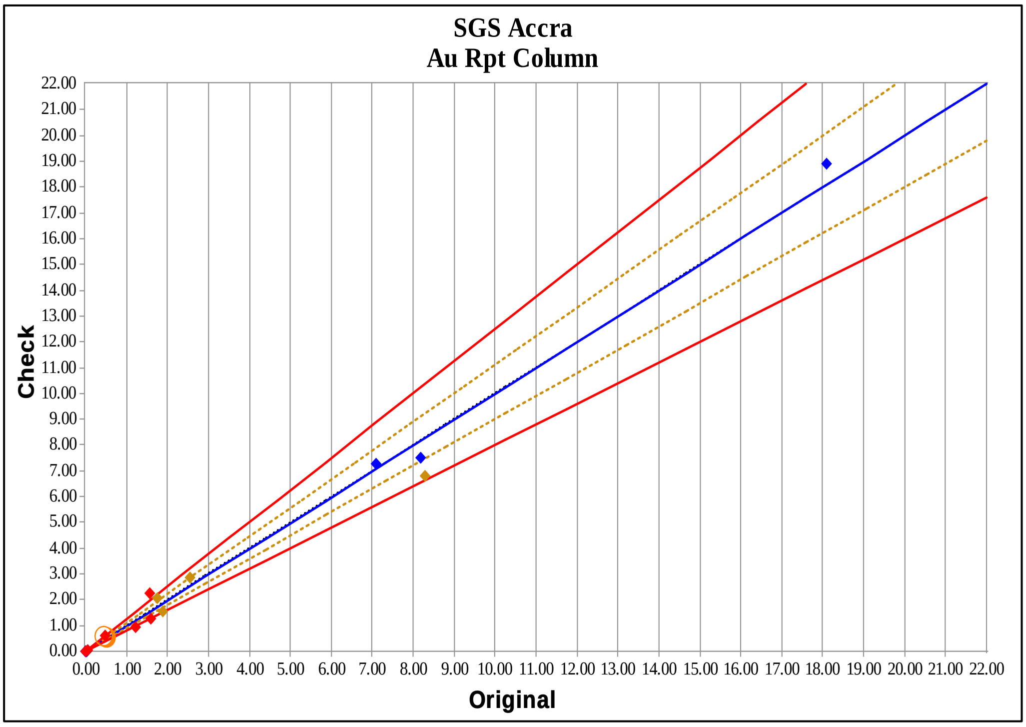


Figure 11-1: SGS Accra Au Rpt Column

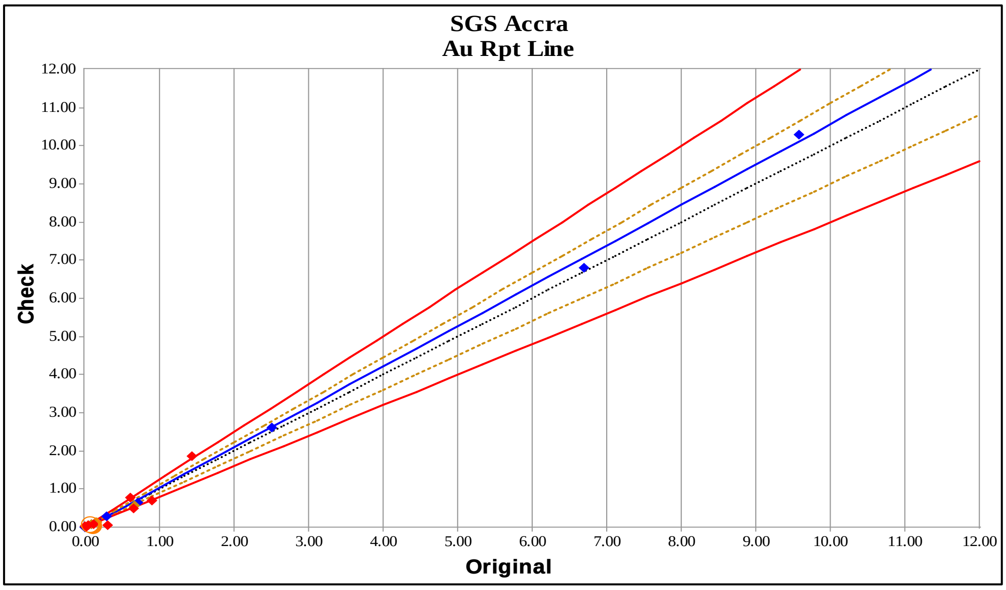


Figure 11-2: SGS Accra Au Rpt Line

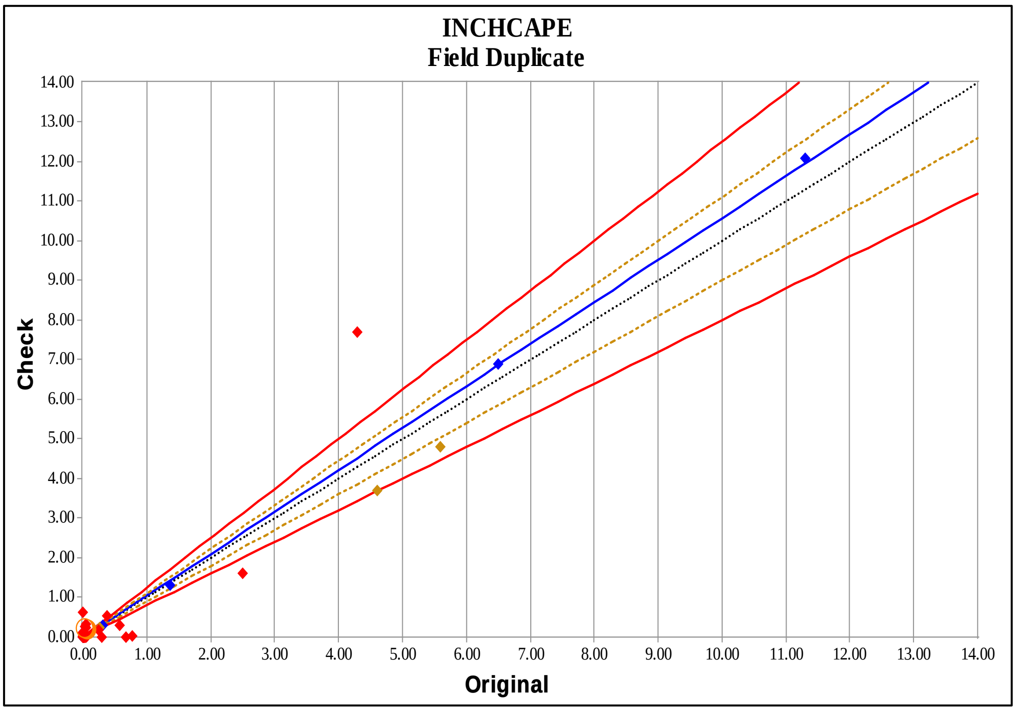


Figure 11-3: INCHCAPE Field Duplicate

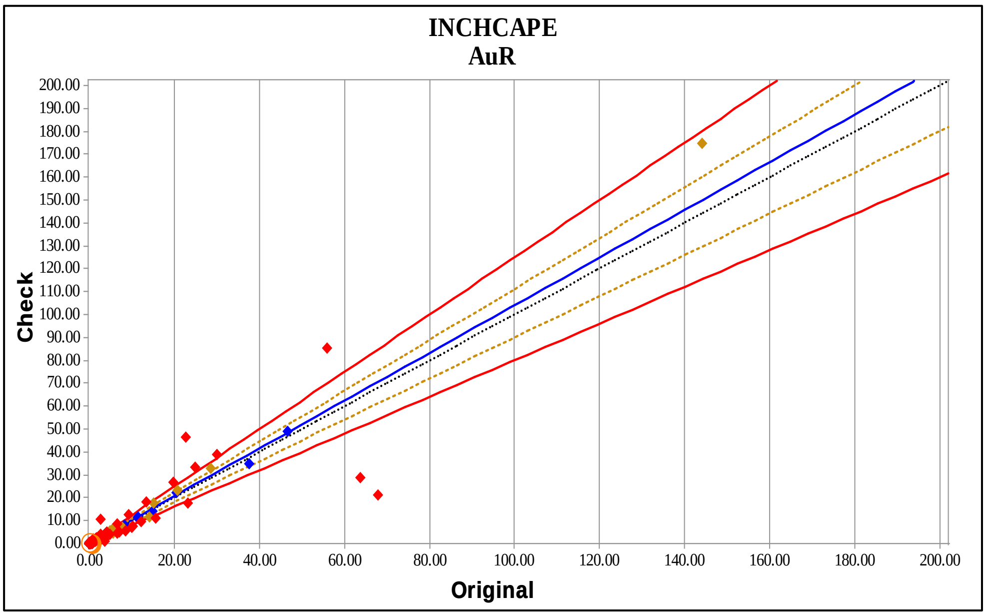


Figure 11-4: INCHCAPE AuR

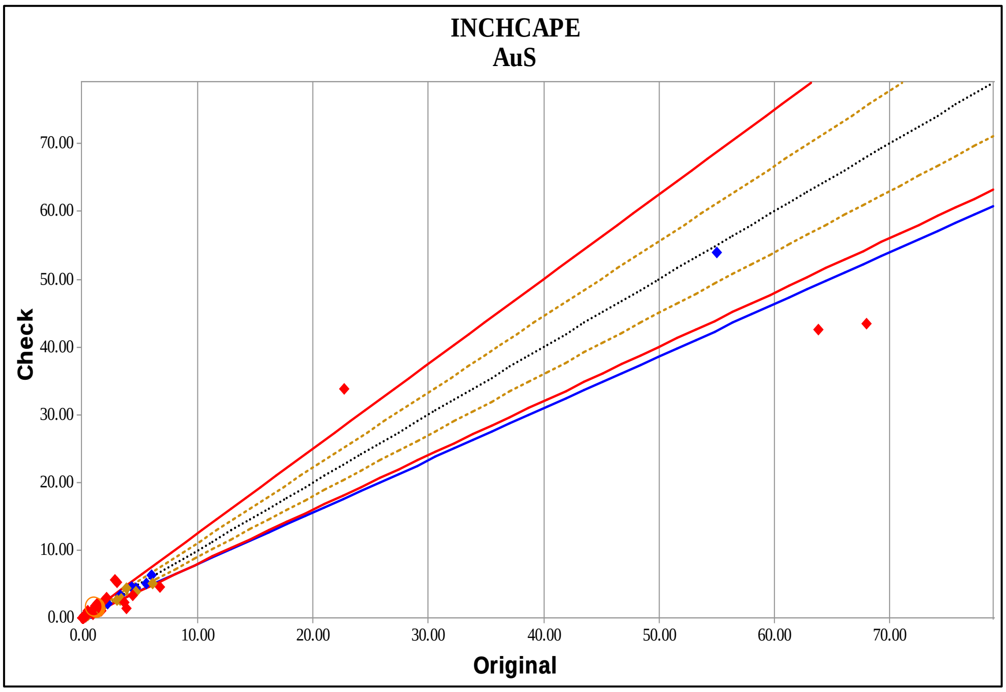


Figure 11-5: INCHCAPE AuS

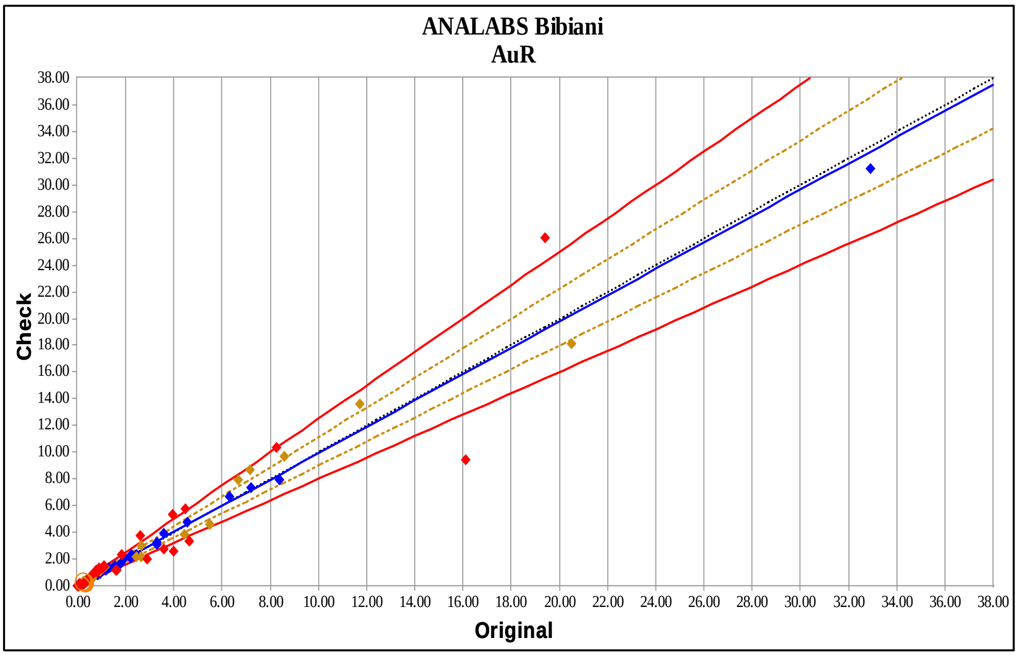


Figure 11-6: ANALABS Bibiani AuR

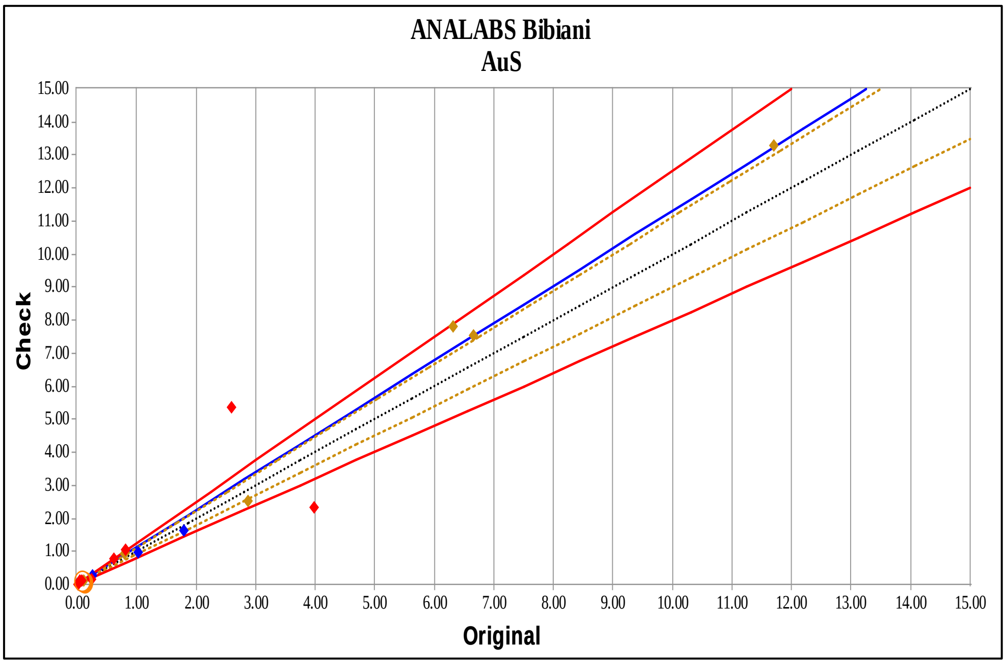


Figure 11-7: ANALABS Bibiani AuS

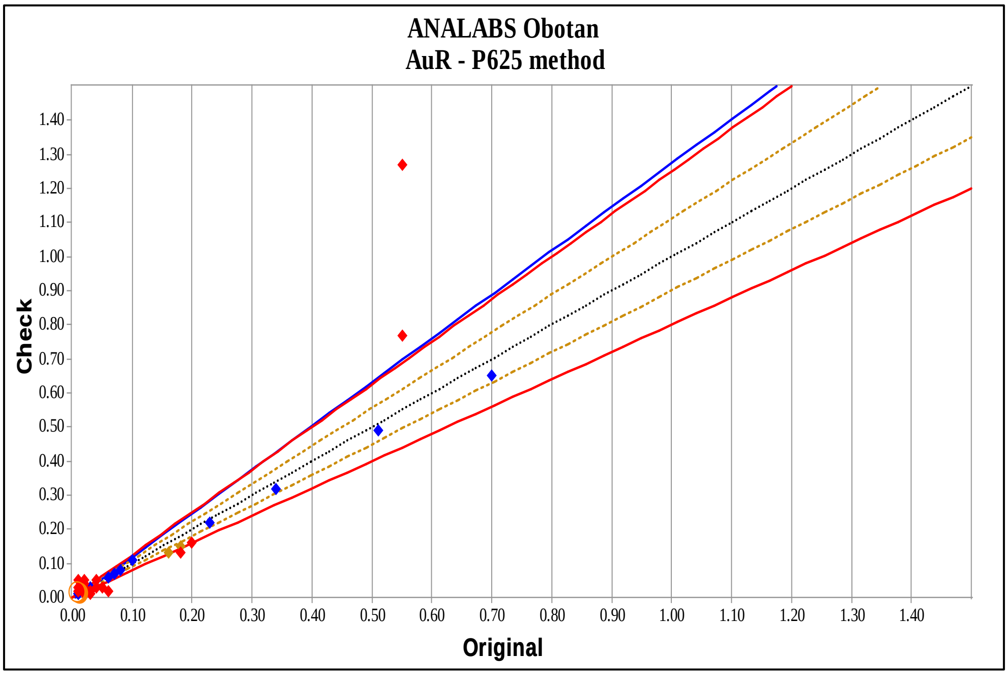


Figure 11-8: ANALABS Obotan AuR – F625 method

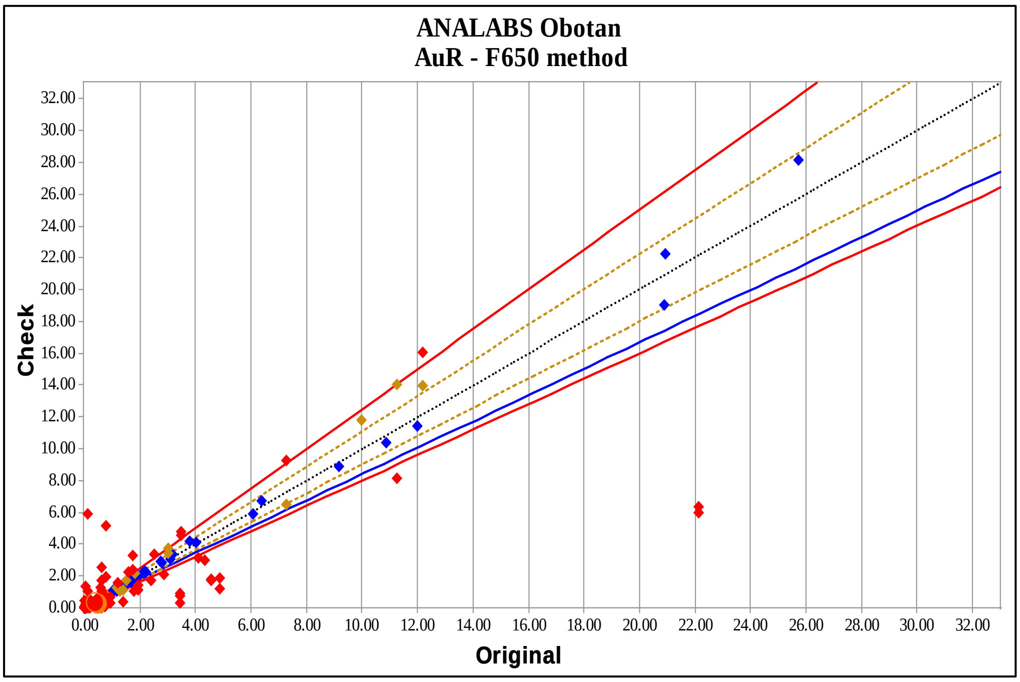


Figure 11-9: ANALABS Obotan AuR – F650 method

12 DATA VERIFICATION

Historic data verification has been undertaken by a number of independent consultants over the periods Keegan and PMI owned the Esaase and Nkran properties respectively, prior to the Asanko Gold takeover of PMI and the merger of the two entities and the commencement of mining from the Nkran pit in 2015. These include SRK (2011, 2012), CJM (2014, 2016) and CSA (2016).

The verification steps undertaken by the earlier Companies are shown below. These verifications are disclosed as being documentation of previous verification, but that it hasn't been relied upon as a proxy for current verification by the current QPs.

The data for the various Obotan and Esaase tenements was based on the available exploration drill hole data and geological models and litho-domains which was provided to CSA and CJM by Asanko Gold. The steps taken for verification of the data relied upon for the current MRE are shown in Section 12.2 for CJM, and in Section 12.3 for CSA. Both QPs are satisfied with the accuracy of the data for the purposes of conducting MRE.

12.1 SRK Data Verification and Site Visits

This disclosure is related to historic data verification by SRK, and hasn't been relied upon upon as a proxy for the verification by the current QP's.

As part of the initial 2011 SRK MRE, Principal Geological Consultant Peter Gleeson conducted two site visits to the Nkran project covering a total period of 12 days in March and June 2011.

The visits included a review of the following:

- Drilling techniques
- Sampling
- Logging procedures
- Structural logging
- Density measurements
- Exploration data
- Resources review
- Geological review of core and pits
- Geochemical sampling
- Data QA/QC
- Data entry
- Laboratory review
- Hold general geological discussions with the site geologist
- Review sections and plans of the resources

During the visit geological inspections of the Nkran pit, Adubiaso pit, Abore pit and Asuadai prospect were undertaken. At the end of the visit, the SGS assay laboratories in Tarkwa were also inspected. After extensive QA/QC work on the resource database undertaken in April 2011 and August 2011, along with a review of previous work by H&S, SRK were of the opinion that no serious issues with regard to any of the historic data exists and that in general, the quality of data is well within industry standards.

An extensive write up detailing the findings of the SRK data verification site visits is included in the 2011 and 2012 SRK MRE 43-101 reports which are filed on SEDAR.

12.2 CJM Data Verification and Site Visits

The Competent Person for the 2014 updated MRE, Charles Muller (CJM) is also a QP for elements of the current technical report. CJM reviewed all available reports and found the previous data verification work to be adequate. In addition to this, Charles Muller personally conducted a number of site visits of the Nkran project area, including 29th September to 2nd October 2012 (Esaase), and 1st to 3rd September 2014, (Nkran general).

During the commissioning of the Nkran mining operation and subsequent production during 2016, CJM conducted site visits in November 2015 and March 2016. These were more directed at auditing and effecting the Grade Control drilling methodology, Grade Control model estimation parameters, and implementation of Minesoft Reconciliation software.

The data (CSV and Datamine files) was reviewed, validated and corrected by CJM prior to commencing the MRE studies. Only the Grade Control RC, RC and diamond drilling sample data were included for use in the estimation process. Checks made to the data prior to estimation included:

- No overlapping intervals
- Down hole surveys at 0m depth
- Consistency of depths between different data tables
- Check gaps in the data
- Detection limits
- Zero value samples
- Missing intercepts

12.2.1 Limitations of Data Verification

Although CJM has relied on certain aspects of the validation of the current data by the relevant qualified personnel of Asanko, CJM does not disclaim responsibility for data accuracy in these respects. Notwithstanding, CJM has completed to their satisfaction adequate reviews and verification works so as to be happy to rely on the data as presented.

These include:

- Verified collar positions of drill holes

- Verified drill hole survey data
- Lab certificate verification
- QA/QC of assay data
- Lithological logging of the drill hole intersections

12.2.2 Adequacy of Data

The CJM Representative and Qualified Person, Charles Muller, personally visited and inspected all of the project areas on numerous occasions over the last four years (2012-2016). He conducted reviews of the data, the available data and geological models. He conducted field visits, as well as inspections of the drill core and the core storage facilities for the all projects.

It is CJM's opinion that the application of the surface drill hole data is adequate for the geostatistical estimation processes employed on the various tenements of both Obotan and Esaase. The data is spatially well represented and of an adequate support level for estimating ore bodies of this nature. The procedures and codes of practice employed by personnel of Asanko, with regard to geological logging, sample preparation and analytical procedures, conform to industry standards and are therefore adequate for use in geological modelling and geostatistical estimation.

12.3 CSA Data Validation and Site Visits

Malcolm Titley CSA Principal Consultant and QP for the Nkran, Akwasiso and Dynamite Hill MREs visited the AGM on three occasions. The first site visit was during the period 1st to 6th September 2016 with a follow-up visits during November 2016 and 12th to 16th January 2017. The focus of the site visits were:

- Review of mining and Grade Control procedures with emphasis on reconciliation between production and the MRE models
- Review of mining and metallurgical plant reconciliation and metal accounting
- Review and validation of Nkran geology and drill hole sample data, in preparation for MRE update
- Site visit to Dynamite Hill deposit and review of drilling, sampling and diamond core
- Review of Akwasiso diamond drill core

12.4 CSA Data Validation

12.4.1 Database Verification

Backups of the Asanko Gold Exploration (Nkran) and Asanko Gold Grade Control ("GC") Structured Query Language ("SQL") databases were received by CSA on the 13th January and 16th January 2017 respectively. An updated copy of the GC database was provided by Asanko Gold on the 21st January to address issues noted with assay batch GC-0033. The new database has not been reviewed as communications from the Asanko data manager noted that this was the only change made. Reviews were completed on the GC SQL backup dated the 16th January 2017.

The databases were validated and checked to ensure that both their structures and the data contained were valid. Checks undertaken are discussed in the following sections.

12.5 Database Structure

The database schema used is the Maxwell Data Schema ("MDS") which has standard constraints, keys and triggers to ensure that only validated data can be loaded. If these constraints, keys or triggers have been edited or removed, invalidated data can be merged into the database, (e.g. overlapping intervals, data that exceeds the maximum depth of the drill hole, etc.).

Standard validation rules in the MDS include the following:

- **Data** is captured in the correct format:
  - Real Number: This is a number such as a drill hole depth, co-ordinate, etc. In some cases, there can be a constraint on a number (e.g. a number which is a percent should be ≤ 100)
  - Date: Set format such as dd/mm/yyyy
  - Text: Usually a comment
  - Library field: A library field (lookup) has a predetermined list of values allowing only those values to be entered in the field (e.g. lithotype codes, or responsible person). This ensures that there is consistency in the database (e.g. a quartz vein is always captured as "Qv" not as Q-V, Qtz V, etc.)
  - **Collar table**: Incorrect co-ordinates (not within known range), unique hole ID's per dataset. Data can only be merged into the database if the drill hole has been entered into the collar table
- **Survey table**: Duplicate entries, survey intervals past the specified maximum depth in the collar table, overlapping intervals and anomalous dips and azimuths are not merged until corrected
- **Geotechnical tables**: Core recoveries and RQD's greater less than 0%, or greater than 120% (Recovery), or 100% (RQD), overlapping intervals, negative widths, geotechnical results past the specified maximum depth in the collar table are not merged until corrected
- **Geology table**: Duplicate entries, lithological intervals past the specified maximum depth in the collar table, overlapping intervals and negative widths are not merged until corrected. Standardised logging codes are required
- **Sampling table**: Duplicate entries, sampling intervals past the specified maximum depth in the collar table, negative widths, overlapping intervals, sampling widths exceeding tolerance levels, missing intervals and duplicated sample ID's are not merged until corrected

- **Assay table**: Missing samples (assay results received, but no samples in database) are imported into an incoming assay table, assay metadata such as detection limits, methods, etc. are captured where possible

AdeptSQL was used to compare database structures against each other and against the MDS to determine whether the database structures were still intact. Changes had been made, but none of these were material with respect to the integrity of the databases. The database comparison reports are available if required and changes noted included:

- Additional tables
- Additional fields in a table
- Field column order within tables

However, it must be noted that the even though the database structures were not materially different from the MDS, constraints could have been removed to merge data and reinstated once data had been merged. Therefore, checks of the data within the databases were also undertaken.

12.6 Data Review

12.6.1 Asanko Gold Exploration Database

The Exploration database is a large database with 62,105 drill holes (to Dec 2016) in the collar file and data were extracted from the database for the various projects using bounding values for easting and northings. These data extractions were validated and exported for downstream work. During the validations, some minor issues were noted and examples included:

- Hole NKR11-059 has alteration records that exceed the depth of the hole in the collar table
- 25 holes that don't have a max depth in the collar table
- Alteration records exist with inconsistent priority data

In addition, it appears that the database doesn't contain all the available data as there were instance where data had been used in resource calculations, or QA/QC reviews which weren't in the master database. This issue of surrogate datasets has been fixed by Asanko.

No in-depth audit or review has been completed as this was not required in the current scope of work, but it is apparent that at some stage constraints have been removed and non-validated data imported into the database. It is recommended that a comprehensive audit of the database is undertaken and any issues resolved. An audit will be undertaken when the full SQL database management system is implemented Q2 2017.

12.6.2 Asanko Gold Grade Control Database

The Asanko Gold Grade Control dataset has minor validation issues with some gaps and missing data which are summarised below. CSA notes that some of these gaps can materially impact the quality of/or confidence in data used for any downstream work and recommends that missing data be located (e.g. density, sample recovery, missing assay results, etc.). All subsequently fixed by Asanko:

- 316 holes have no hole type in the database
- There are 17 holes in the database without a survey record
- 60% of holes in the database have lithology records and 70% of the RC holes have lithology records
- No density data are contained in the database
- No Recovery data are contained in the database
- No Geotechnical data are contained in the database
- 282 holes do not have samples
- No sample weight, or recovery data provided so majority of samples have no recovery information
- 697 samples do not have assay results, which are mostly precollar RC
- ALS Perth was not used as a Grade Control laboratory (turnaround time wouldn't have been quick enough), but is listed in the database as a laboratory used. Asanko Gold's explanation is that the incorrect data loading template was used
- There are instances of the same sample having two different assay values from two labs (sample not a duplicate sample). This was investigated by Asanko Gold and corrected
- Assay Batch table has numerous incorrect dates including future dates

12.7 Database Conclusions and Recommendations

The structures of both databases are intact, but it is apparent that the Nkran database has had these constraints removed and then re-instated, permitting some non-validated data to be loaded. Some minor validation issues were noted when reviewing the data extractions, but overall these were deemed to not materially impact the quality of the data used for resource estimation. Gaps were present and some of these are material to downstream work (e.g. density data). This issue has been remedied by Asanko.

Other recommendations that have been already remedied by Asanko Gold include:

- A comprehensive audit of the Nkran database to identify invalid records and resolve these (completed Q2 2017).
- Resolve the issues noted in the GC database such as future dates in the assay batch table (remedied)
- Ensure that all available data are captured in the master database (remedied)
- Where possible, fill in any gaps identified in the databases (remedied).

13 MINERAL PROCESS AND METALLURGICAL TESTING

13.1 Project Background

The Obotan Gold Project was developed by Resolute during 1996-1997. In 1999 Resolute completed an upgrade study to expand the Obotan Gold Project to treat Oxide ore from the satellite pit Adubiaso and primary ore from the Nkran ore body. After the closure of the Resolute mining operations at Obotan in November 2002, further metallurgical test work campaigns were carried out for the treatment of Obotan ore. The AGM Phase 1 flow sheet was developed based on historical operating data and testing conducted on Obotan samples in 2015. The AGM Phase 1 circuit was based on a typical single stage crushing, SAG and SABC with gravity concentration followed by a CIL plant, and was commissioned in 2016.

The future AGM DFS is to be completed in two stages.

- The first stage, P5M, includes the upgrading of the existing AGM Phase 1 plant from the current throughput of 3.6 Mtpa to 5.0 Mtpa, by means of various equipment upgrades. The increased throughput will be achieved by supplementing the current supply of Nkran Sulphide material with Oxide material, initially sourced from Obotan satellite pits and later from Esaase
- The second stage, P10M, includes the addition of a second 5.0 Mtpa CIL processing facility to take the total processing throughput of the AGM Project to 10 Mtpa. The Esaase Mine will supply Oxide and Fresh material to this stage

13.2 P5M Test Work Summary

13.2.1 Previous Metallurgical Test Work and Historical Operating Data

As per the NI 43-101 Technical Report “Asanko Gold Mine – Phase 2 Pre-Feasibility Study” issued June 2015.

13.2.2 Current Metallurgical Test Work

13.2.2.1 Gravity-CIL Test Work on Nkran Fresh V2 Sample

As part of the AGM Expansion P10M Phase 2 test work campaigns, conducted during July 2015 (see Section 13.3.2.2) and Q4 2016 (see Section 13.3.2.2), two gravity-CIL tests were conducted on the Nkran Fresh V2 composite sample (as per Section 13.3.2.1) at a grind of P_{80} 106 µm. The aim of these tests were to assess the impact of reduced pre-oxidation and CIL residence times on the CIL staged recovery.

The results are summarized in Table 13-1 and Figure 13-1. The following was noted:

- High gravity recovery of ~50 - 60%, in-line with AGM Phase 1 operating data
- Low cyanide consumption of 0.29 – 0.49 kg/t milled
- Leach was essentially complete after 16 hours

The sample showed rapid leach kinetics, which informed the decision to not expand the AGM Phase 1 CIL circuit as part of the P5M expansion. When treating 5.0 Mtpa, pre-oxidation duration is 2 hours while the leach residence time equates to 16 hours when utilizing the infrastructure currently installed.

Table 13-1: Gravity-CIL Test Result on Nkran Fresh V2 sample (source: DRA 2017)

Test ID	Grind µm	Leach Duration hrs	Gravity Feed Calc. g/t Au	Gravity Recovery %	CIL Residue g/t Au	Overall Recovery %	NaCN usage kg/t
JR1889	106	3.5 h Pre-oxidation 24h CIL	1.99	49.5	0.20	90.20	0.29
JR2636	106	2.0 h Pre-oxidation 24h CIL	2.01	59.1	0.12	94.27	0.49

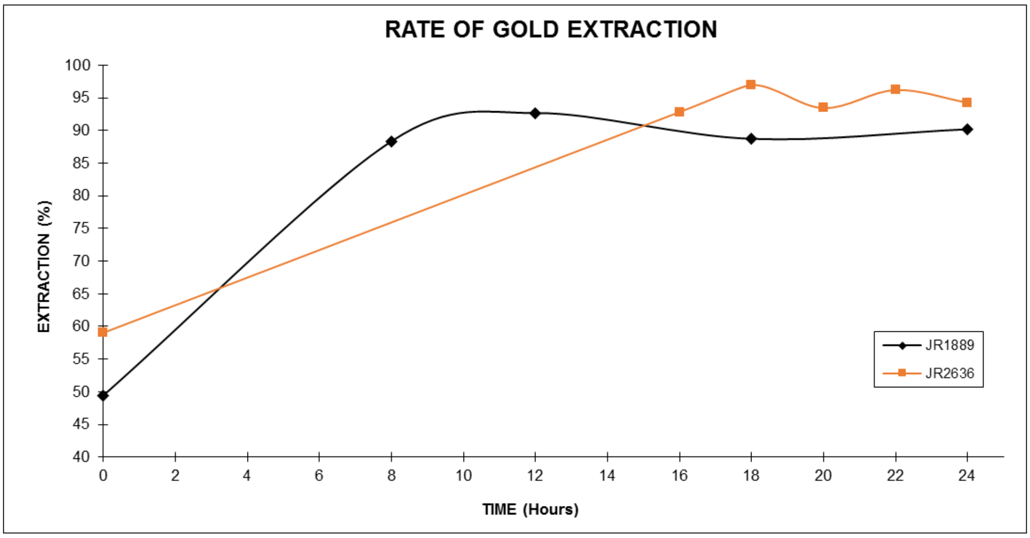


Figure 13-1: Kinetic Gravity-CIL test result on Nkran Fresh V2 sample[1] (source: ALS, 2016 & 2017)

13.3 P10M Test Work Summary

13.3.1 Previous Metallurgical Test Work

The AGM DFS P10M test work was conducted in two test work programmes, Phase 1 and Phase 2. Phase 1 test work was done during 2014 – 2015 to support the AGM Phase 2 PFS.

[1] JR2636 sampling started at 16 hours, while JR1889 sampling started at 8 hours.

Phase 2 test work was done during 2015 - 2016 to finalise the process design, capital and operating costs for the AGM DFS. Both test work phases were undertaken by the Perth based ALS under the management of DRA, and evaluated blends of Esaase and Nkran ores for processing at a central facility.

Prior to the AGM DFS P10M, the Esaase ore was evaluated separately in test campaigns between 2008 and 2013.

13.3.1.1 Esaase Deposit (Phase I to VI)

As per the NI 43-101 Technical Report "Asanko Gold Mine – Phase 2 Pre-Feasibility Study" issued June 2015, the Esaase deposit has been subjected to 6 phases of metallurgical test work aimed at determining comminution, gravity, flotation and leaching parameters.

13.3.1.2 AGM DFS P10M - Phase 1 Test Work Programme

Up to this point, previous test work mainly considered the treatment of Nkran ores in a gravity-CIL circuit, and treatment of Esaase ores in a gravity-flotation-CIL circuit.

This phase of test work conducted during 2014 - 2015, undertaken by the Perth based ALS under the management of DRA was conducted to support the AGM Phase 2 PFS. The objective of this campaign was to evaluate the metallurgical response of various blends of Esaase and Obotan ores at a central processing facility consisting of a gravity-CIL circuit (AGM Phase 1) and a gravity-flotation-CIL circuit (as per the Esaase PFS). It was further investigated if it would prove feasible to dedicate specific ore types to either one of the two processing routes to optimise overall recoveries and operating costs. The scope of work included the following:

- BBWi and Grindmill testing on a blend of 75% Nkran Fresh and 25% Esaase Fresh material
- Gravity separation tests performed on various blends of Nkran Fresh material and Esaase material at grinds of 75 µm and 106 µm
- Sighter flotation tests performed on various blends of Nkran Fresh material and Esaase material at a grind of 75 µm and 106 µm
- A bulk flotation test on a blend of 75% Esaase Fresh and 25% Nkran Fresh with regrind (5 kWh/t) and CIL on the flotation concentrate
- CIL tests performed on various blends of Nkran Fresh material to Esaase material at a grind of 106 micron.

Refer to Table 13-2 for a summary of the results obtained.

Table 13-2: Summary of AGM DFS P10M Phase 1 Test Work Programme (source: DRA 2017)

Test Work Scope	Samples	Tests	Results	Comments
Evaluation of the metallurgical response of a gravity-CIL circuit and a gravity-flotation-CIL circuit when treating blends of Esaase and Obotan ores.	Composite samples of Oxide, Transitional and Fresh ores, produced from individual core samples from the Nkran and Esaase ore bodies.	BBWi and Grindmill testing on 75% Nkran Fresh: 25% Esaase Fresh Sample.	A BBWi of 15.7 kWh/t at 106 µm limiting screen was reported.	Result similar to previous work on Nkran Fresh (BBWi ranging from 11.0 kWh/t to 15.0 kWh/t).
		Gravity separation and sighter, batch flotation tests on blends of Nkran Fresh and Esaase material at P_{80} -75 µm and 106 µm.	Nkran Fresh achieved an average total recovery of 95.1% at P_{80} -75 µm. Blends of Nkran Fresh and Esaase Fresh ore at P_{80} -75 µm achieved overall recoveries ranging from 93.2% to 96.6%. Blends of Nkran Fresh and Esaase Oxide ore at P_{80} -106 µm achieved overall recoveries ranging from 90.6% to 94.7%. Blends of Nkran Fresh and Esaase Transition at P_{80} -106 µm achieved overall recoveries ranging from 81.6% to 92.9%.	The overall recovery for the flotation circuit increased with a finer grind.
		Bulk flotation test on 75% Esaase Fresh: 25% Nkran Fresh at P_{80} -75 µm, followed by a 5 kWh/t regrind and 24h CIL on the flotation concentrate.	A single bulk flotation test at P_{80} -75 µm on a blend of 25% Nkran Fresh: 75% Esaase Fresh material achieved a total Au recovery of 92.7% with a final residue of 0.17 g/t Au from a 2.2 g/t mill feed grade at a 6% mass pull.	The concentrate from this flotation test was used to perform a flotation concentrate CIL testing.

Test Work Scope	Samples	Tests	Results	Comments
		CIL testing on blends of Nkran Fresh and Esaase material flotation concentrate product after applying a 5 kWh/t regrind step.	High solution cyanide concentrations were maintained (1800 ppm to 3000 ppm) in an effort to target high overall recovery. The resulting average cyanide consumption of 12.7 kg/t was noted as being higher than historical Nkran concentrate leach testing at IMO and ALS.	
		CIL tests on blends of Nkran Fresh and Esaase material gravity tailings at P_{80} -106µm.	Esaase Oxide sample achieved a total gold recovery of 94.7% with a final residue of 0.09 g/t Au. Blends of Nkran Fresh and Esaase Oxide achieved total gold recoveries ranging from 81.1% to 93.7% and final residues ranging from 0.15 g/t Au to 0.57 g/t Au. Esaase Transition sample achieved a total gold recovery of 87.4% with a final residue of 0.24 g/t Au. Nkran Fresh and Esaase Transition achieved total gold recoveries ranging from 90.9% to 92.2% with final residues ranging from 0.22 g/t Au to 0.25 g/t Au. Nkran Fresh sample reported a total gold recovery of 94.0% with a final residue of 0.25 g/t Au. Esaase Fresh sample achieved a total gold recovery of 94.1% with a final residue of 0.24 g/t Au. Blend of 50% Esaase Fresh: 50% Nkran Fresh achieved a total gold recovery of 93.5% with a final residue of 0.25 g/t Au.	Esaase Oxide ore responded best with a final residue of 0.09 g/t Au. Nkran Fresh, Esaase Transition, and Esaase Fresh ore achieved residue grades over 0.22 g/t Au with recoveries ranging from 87.4% to 94.1%. Esaase Transition ore achieved similar residues to the Fresh ore. The lower recovery of 87.4% was attributed to a lower head grade for the Transition material.

13.3.2 Current Metallurgical Test Work

13.3.2.1 AGM DFS P10M Phase 2 Test Work Samples

In July 2015, a total of 116 individually bagged samples, packaged in 6 drums, were delivered to ALS's mineral processing facility by AGM.

These Phase 2 test work samples were made up from Nkran and Esaase core material, as shown in the RC core met sample maps detailed in Figure 13-2 and Figure 13-3.

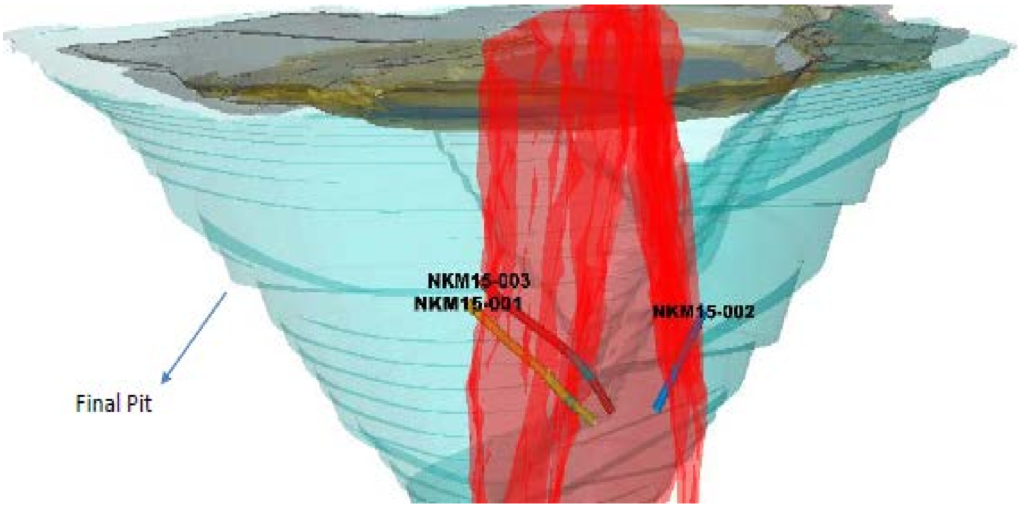


Figure 13-2: Met Sample Map indicating Locations of Nkran Cores used in the Phase 2 Campaign (source: AGM, 2016)

The above samples, together with the various samples tested during the historical AGM Phase 1 and AGM DFS P10M Phase 1 test work campaigns, are representative of the Nkran material to be processed.

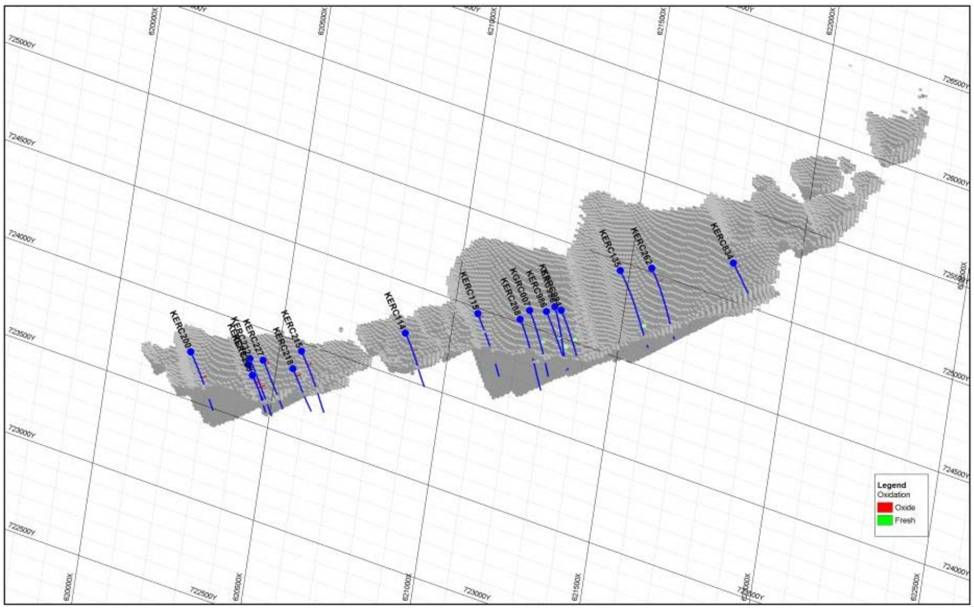


Figure 13-3: Met Sample Map indicating Locations of Esaase Cores used in the Phase 2 Campaign (source: AGM, 2016)

The above samples, together with the various samples tested during the historical Esaase and AGM DFS P10M Phase 1 test work campaigns, are representative of the Esaase material to be processed.

The sample material received was composited to produce individual composite samples of Esaase Oxide, Esaase Fresh, Nkran Fresh, and Nkran Fresh LG as presented in Table 13-3 and Table 13-4.

Table 13-3: Compositing Details on the Phase 2 Test Work Campaign Nkran Fresh Samples (source: DRA 2017)

Nkran Fresh					Nkran Fresh LG				
Hole ID	From (m)	To (m)	Mass (kg)	Au (g/t)	Hole ID	From (m)	To (m)	Mass (kg)	Au (g/t)
NKM15-002	246	247	14.5	2.26	NKM15-002	240	241	14.0	1.68
NKM15-002	247	248	13.8	2.82	NKM15-002	243	244	14.3	1.57
NKM15-002	253	254	13.8	2.56	NKM15-002	256	257	13.9	1.75
NKM15-002	254	255	13.6	2.13	NKM15-002	267	268	15.9	1.8
NKM15-001	248	249	14.5	2.77	NKM15-002	271	272	13.9	1.74
NKM15-001	262	263	13.4	2.39	NKM15-003	313	314	14.3	1.54
NKM15-001	263	264	14.5	2.84	NKM15-003	314	315	7.5	1.94
NKM15-003	284	285	14.0	2.21	NKM15-003	321	322	14.1	2.4
NKM15-003	287	288	14.1	2.78	NKM15-003	322	323	13.8	1.59
NKM15-003	296	297	13.8	3.2					
NKM15-003	300	301	14.3	3.45					

Table 13-4: Compositing Details on the Phase 2 Test Work Campaign Esaase Samples (source: DRA 2017)

Esaase Fresh					Esaase Oxides				
Hole ID	From (m)	To (m)	Mass (kg)	Au (g/t)	Hole ID	From (m)	To (m)	Mass (kg)	Au (g/t)
KGRC007	100	101	6.06	1.56	KERC114	58	59	7.76	1.71
KGRC007	101	102	6.12	1.31	KERC114	59	60	8.26	1.61
KGRC007	102	103	6.12	0.69	KERC193	78	79	7.9	2.12
KGRC007	103	104	5.96	1.09	KERC193	79	80	8.34	1.11
KGRC007	104	105	6.22	2.21	KERC193	80	81	9.02	1.29
KGRC007	105	106	6.40	0.99	KERC193	81	82	8.9	2.75
KGRC007	106	107	6.28	0.83	KERC200	87	88	11	1.94
KGRC007	107	108	6.24	3.83	KERC200	88	89	9.7	2.94
KERC288	105	106	6.52	2.28	KERC200	89	90	10.1	2.28
KERC288	106	107	5.92	0.66	KERC232	90	91	7.28	1.9
KERC288	107	108	6.06	2.24	KERC232	91	92	5.96	1.15
KERC986	91	92	5.54	2.21	KERC232	92	93	7.56	2.4
KERC986	92	93	5.46	0.96	KERC834	17	18	10.44	1.25
KERC986	93	94	5.12	1.37	KERC834	18	19	10.5	1.97
KERC994	94	95	6.04	3.77	KERC227	12	13	8.18	1.28
KERC994	98	99	6.36	3.96	KERC227	13	14	8.74	1.52
KERC994	99	100	6.20	1.92	KERC227	14	15	7.1	2.76
KERC994	102	103	6.20	1.26					
KERC994	103	104	6.42	0.6					
KERC996	123	124	4.50	2					
KERC996	125	126	3.92	0.66					
KERC996	126	127	4.50	6.18					
KERC996	127	128	4.48	0.75					
KERC996	128	129	5.58	1.44					
KERC996	129	130	4.08	0.71					
KERC996	130	131	4.78	1.59					
KERC996	131	132	5.98	0.61					
KERC996	134	135	4.08	1.3					
KERC996	135	136	4.02	1.2					
KERC135	176	177	6.06	1.74					
KERC135	177	178	6.04	3.01					
KERC135	178	179	6.50	1.5					
KERC135	182	183	5.88	6.28					
KERC135	184	185	5.60	0.99					
KERC135	185	186	5.86	3.26					

Sub-samples of each of the master composite samples were submitted for comprehensive head assays, screened fire assay and BLEG. Based on the head assay results received, a third Nkran Fresh sample, Nkran Fresh V2, was prepared by compositing material from Nkran Fresh and Nkran Fresh LG targeting a sample head grade of 2.20 g/t Au, as per the expected LoM average grade. The head assay data is summarized in Table 13-5.

Table 13-5: Phase 2 Test Work Master Composites Head Grades (source: DRA 2017)

Analyte	Units	Esaase Oxide	Esaase Fresh	Nkran Fresh	Nkran Fresh LG	Nkran Fresh V2
Au*	ppm	1.64	2.08	2.60	1.60	2.02
BLEG Au**	ppm	2.50	1.68	-	-	2.11
Ag	ppm	<0.3	0.3	0.6	0.3	0.6
As	ppm	1720	460	4760	3620	4080
C_{TOTAL}	%	0.36	0.90	1.32	1.68	1.56
$C_{ORGANIC}$	%	0.24	0.27	0.33	0.36	0.42
Fe	%	3.78	4.20	4.02	5.02	4.62
Hg	ppm	<0.1	<0.1	<0.1	0.3	-
S_{TOTAL}	%	0.24	0.36	1.08	0.64	0.82
$S_{SULPHIDE}$	%	0.18	0.32	0.78	0.60	0.72

*Average grade from 6 x screen fire assay at 75 µm

**Grade from 5 kg sample intensive cyanidation (BLEG)

13.3.2.2 AGM DFS P10M – Phase 2 Test Work

During July 2015 further metallurgical test work was undertaken to evaluate the combined treatment of Esaase and Obotan ore at a central processing facility, to finalise the process design, capital and operating costs for the AGM DFS.

The AGM DFS Phase 2 test work programme was undertaken by the Perth based ALS under the management of DRA.

The scope of work for this phase of test work was designed to evaluate the opportunity to process gravity tailings from the AGM Phase 1 circuit together with the reground flotation concentrate product from a gravity-flotation circuit in a combined CIL circuit, when treating blends of Esaase and Obotan ores. If feasible, it could result in significant savings in capital requirements for the processing facility.

In addition to the above, further diagnostic test work was conducted on the treatment of Nkran material in a gravity-flotation-CIL circuit.

A brief summary of the Phase 2 test work scope and findings are presented below.

The detail is captured in the DRA DFS report (JGHDP0221-RPT-007) for more information.

- Table 13-6 summarises the findings of the gravity-flotation test work done on Nkran Fresh material at grinds of 150 µm, 106 µm and 75 µm, followed by cyanidation test work on the flotation concentrate product at grinds of 33 µm (no regrind), 22 µm and 20 µm (3 kWh/t regrind energy input) with the aim of reducing cyanide consumption in the concentrate CIL stage
- Cyanidation of Nkran Fresh flotation tailings at P_{80} 106 µm.

Table 13-6: Gravity-flotation and Flotation Concentrate Cyanidation Test Work on Nkran Fresh V2 Material (source: DRA 2017)

Sample	Test Description	Results	Comments
Nkran Fresh V2 composite sample	Gravity-flotation tests at grinds of 150 µm, 106 µm and 75 µm	Gravity recoveries between 46.1% and 62.3%. The following recoveries were achieved when using the Esaase reagent suite: 95.4% at P80 75µm and a 22% mass pull 94.6% at P80 106µm and a 20% mass pull 94.9% at P80 150µm and a 19% mass pull Reagent dosages was adjusted to lower mass pulls; the following recoveries where noted: 96.4% at P80 75µm and a 19% mass pull 91.2% to 96.1% at P80 106µm and a mass pulls between 14% - 17% 93.6% at P80 150µm and a 15% mass pull	Gravity recoveries in agreement with current AGM Phase 1 operating information. High mass pulls on NFV2 sample attributed to mineralogy of the sample The flotation staged recovery on the NFV2 material improved with finer grinds. The finer grind however, also resulted in higher mass pulls
	Cyanidation test work on flotation tailings at 106 µm	3.5h Pre-ox followed by 12h CIL on NFV2 flotation tailings achieved a 32% recovery	

Sample	Test Description	Results	Comments
	Cyanidation test work on flotation concentrate at grinds of: 33 µm (no regrind) 22 µm 20 µm (3 kWh/t energy input)	A bulk flotation test was conducted as comparison to batch test, and achieved a slightly lower mass pull than the batch flotation test (16% vs 17%) Cyanidation leach tests on flotation concentrate achieved gold recoveries between 89.5% to 93.5%, with cyanide consumptions between 1.43 kg/t and 2.24 kg/t being recorded	Flotation concentrate leach cyanide consumptions were lower than the consumptions noted during AGM Expansion Phase 1 campaign. The cyanide consumption remained moderate to high. Regrinding of the concentrate gave 2% to 4% higher CIL staged recoveries but had significantly higher cyanide consumptions

- Refer to Table 13-7 for a summary of the findings of the Bulk gravity-flotation tests conducted on a 50% Esaase Fresh: 50% Esaase Oxide sample, at a grind of 75 µm, followed by cyanidation test work on the bulk flotation concentrate sample, at a regrind energy input of 5 kWh/t to investigate the effects of 24h CIL, 24h leach-CIL, and a 16h direct leach

Table 13-7: Gravity-flotation and Flotation Concentrate Cyanidation Test Work on 50% Esaase Fresh: 50% Esaase Oxide Material (source: DRA 2017)

Sample	Test Description	Results	Comments
50% Esaase Fresh: 50% Esaase Oxide	Bulk gravity-flotation testing at P_{80} 75 µm	High gravity recovery of ~58%, and a flotation staged recovery of 80%. Overall test recovery of 91.5% at a mass pull of 6%.	
	Cyanidation test work on concentrate product after 5kWh/t regrind, P_{80} 16µm. Preg-robbing potential investigated	CIL staged recoveries ranging from 92.6% - 95.1% was achieved with cyanide consumptions between 2.2kg/t and 2.8kg/t in a 24h CIL CIL staged recovery of 92.6% achieved in a 16h direct leach with a 2kg/t cyanide consumption	16h direct leach resulted in 2% lower recovery compared to the 24h CIL Pre-treatment with diesel did not show benefit The test in which carbon was added at the start of the leach reported the highest recovery, eluding to possible preg-robbers in the sample tested
	CIL test work on flotation tailings product, P_{80} 75µm	3.5h Pre-ox followed by 12h CIL on flotation tailings achieved a 55.6% recovery	

- Refer to Table 13-8 for a summary of the findings of the CIL tests performed on a blend of Nkran Fresh gravity tail (post pre-oxidation) and the residual leach slurry from the 50% Esaase Fresh: 50% Esaase Oxide flotation concentrate to investigate the treatment of these streams in a common CIL.

Table 13-8: Cyanidation of Nkran Fresh Gravity Tails and Flotation Concentrate Leach Slurry from 50% Esaase Fresh: 50% Esaase Oxide Material (source: DRA 2017)

Sample	Test Description	Results	Comments
50% Esaase Fresh: 50% Esaase Oxide	Bulk gravity-flotation testing at P_{80} 75 µm	High gravity recovery of ~58%, and a flotation staged recovery of 80%. Overall test recovery of 91.5% at a mass pull of 6%.	
	Cyanidation test work on concentrate product after 5kWh/t regrind, P_{80} 16µm. Preg-robbing potential investigated	CIL staged recoveries ranging from 92.6% - 95.1% was achieved with cyanide consumptions between 2.2kg/t and 2.8kg/t in a 24h CIL CIL staged recovery of 92.6% achieved in a 16h direct leach with a 2kg/t cyanide consumption	16h direct leach resulted in 2% lower recovery compared to the 24h CIL. Pre-treatment with diesel did not show benefit. The test in which carbon was added at the start of the leach reported the highest recovery, eluding to possible preg-robbers in the sample tested
	CIL test work on flotation tailings product, P_{80} 75µm	3.5h Pre-ox followed by 12h CIL on flotation tailings achieved a 55.6% recovery	

- Refer to Table 13-9 for a summary of the findings of Gravity-flotation tests performed on Esaase Oxide and Esaase Fresh samples at a grind of 75 µm, followed by cyanidation test work on the bulk flotation product streams (flotation tail and 5 kWh/t re-ground concentrate) of an Esaase Oxide and Esaase Fresh sample.

Table 13-9: Gravity-flotation Test Work and Cyanidation of Flotation Concentrate on Esaase Oxide and Esaase Fresh Material (source: DRA 2017)

Sample	Test Description	Results	Comments
Esaase Oxide	Bulk and batch gravity-flotation testing at P_{80} 75 µm using the Esaase reagent dosing suite	High gravity recovery of ~56%, and a flotation staged recoveries ranging from 74.7% to 77.2% Overall test recoveries ranging from 89.0% to 89.8% at mass pull between 8% and 17%	Batch test achieved high mass pull at 17% compared to the 8.5% mass pull noted in the bulk tests
	Cyanidation test work on concentrate product after 5kWh/t regrind, P_{80} 16µm.	CIL staged recoveries of 91.2% achieved with cyanide consumption of 2.8kg/t in a 16h direct leach	
	CIL test work on flotation tailings product, P_{80} 75µm	3.5h Pre-ox followed by 12h CIL on flotation tailings achieved a 36.0% recovery	
Esaase Fresh	Bulk and batch gravity-flotation testing at P_{80} 75 µm using the Esaase reagent dosing suite	High gravity recovery of >60%, and a flotation staged recoveries ranging from 77.5% to 84.7% Overall test recoveries ranging from 91.1% to 94.7% at mass pull between 7.2% and 14%	Batch test achieved high mass pull at 14% compared to the 8.5% mass pull noted in the bulk tests

- Refer to Table 13-10 for a summary of the findings of the cyanide consumption optimisation test work on Esaase Oxide, Nkran Fresh and a blend of 33% Esaase Oxide: 67% Nkran Fresh material, at a grind of 106 µm.

Table 13-10: Cyanide Optimisation Test Work (source: DRA 2017)

Sample	Test Description	Results	Comments
Nkran Fresh V2	Cyanide optimisation test work on gravity tailings samples at P_{80} 106 µm	NFV2 sample achieved an staged CIL recovery of 82.8% when consuming 0.24kg/t cyanide	Cyanide consumption could be lowered to 0.35kg/t without negatively impacting on recovery
Esaase Oxide 33% Esaase Oxide: 67% Nkran Fresh V2	Cyanide optimisation test work on gravity tailings samples at P80 106 µm	Esaase Oxide sample achieved an staged CIL recovery of 86.2% when consuming 0.34kg/t cyanide Blend of 33% Esaase Oxide: 67% NFV2 sample achieved an staged CIL recovery of 82.9% when consuming 0.32kg/t cyanide	

- Refer to Table 13-11 for a summary of the findings of the test work to evaluate various potential P10M flowsheet options in which the Esaase and Nkran material streams are combined in either flotation, or CIL circuits.

Table 13-11: Evaluation of Flow Sheet Options for P10M by Combining Esaase and Nkran material in Flotation or CIL Circuits (source: DRA 2017)

Sample	Test Description	Results	Comments
Nkran Fresh V2 Esaase Fresh 50% Esaase Fresh: 50% Esaase Oxide 75% Esaase Fresh: 25% NFV2 60% NFV2: 40% Esaase Oxide	50% EF: 50% EO flotation conc. treated in 24h CIL; while treating NFV2 gravity tails in a 24h CIL	Final residue of 0.24g/t Au and a cyanide consumption of 0.61kg/t	A common CIL could be used to process the Nkran gravity tailings together with the Esaase flotation concentrate leach product, provided that no carbon poisoning due to flotation reagents are recorded.
	50% EF: 50% EO flotation conc. treated in 24h CIL, product combined with NFV2 gravity tail in 23h CIL	Final residue of 0.25g/t Au and a cyanide consumption of 0.62kg/t	
	50% EF: 50% EO flotation conc. treated in 16h direct leach; product combined with NFV2 gravity tail in 23h CIL	Final residue of 0.24g/t Au and a cyanide consumption of 0.67kg/t	
	75% EF: 25% NF flotation concentrate treated in 24h CIL; while treating NFV2 gravity tails in 24h CIL	Final residue of 0.17g/t Au and a cyanide consumption of 0.50kg/t	
	75% EF:25% NF flotation conc. treated in 18h direct leach; product blended with NFV2 gravity tail in 23h CIL	Final residue of 0.17g/t Au and a cyanide consumption of 0.50kg/t	
	Esaase Fresh flotation conc. treated in 18h direct leach; product combined with 60% NFV2: 40% EO gravity tail in 23h CIL	Final residue of 0.18g/t Au and a cyanide consumption of 0.87kg/t	

- Refer to Table 13-12 for a summary of the findings from the sequential triple contact carbon adsorption CIP and equilibrium carbon loading test work on a stream consisting of a flotation concentrate CIL tail from the Esaase Fresh sample and the gravity tailings product of a 60% Nkran Fresh: 40% Esaase Oxide sample.

Table 13-12: Sequential Triple Contact Carbon Adsorption CIP and Equilibrium Carbon Loading Test Work (source: DRA 2017)

Sample	Test Description	Results	Comments
Blend of Esaase Fresh flotation concentrate 18h direct leach product and the gravity tailings of a 60% Nkran Fresh V2: 40% Esaase Oxide sample	A sequential triple contact CIP test to determine the Fleming Kinetic adsorption constants k and n	Fleming's K-constant: 129.02 Fleming's n-constant: 0.76 Loaded carbon Au content: 1 562g/t Au	The k-value of 129 is lower than the typical average of 150. Fast kinetics are represented by a rate constant >200, while very slow kinetics are represented by a rate constant of <100. The below average k-value is most likely caused by fouling of the carbon surface area and pores by flotation reagents such as PAX and MIBC.
	Equilibrium carbon loading test work to determine the equilibrium carbon gold loading capacity for the sample tested.	Equilibrium carbon Au loading at 1.0ppm solution concentration: 2 793 g/t Au Equilibrium carbon Au loading at 0.5ppm solution concentration: 2 377 g/t Au Equilibrium carbon Au loading at 0.1ppm solution concentration: 2 921 g/t Au	The results indicate low gold adsorption characteristics with an equilibrium gold loading on carbon of 2 793 g/t Au at a 1.0 ppm Au solution; likely caused by fouling of the carbon by flotation reagents Theoretical equilibrium carbon loadings ranges from 8 000 g/t Au to 10 000 g/t Au. It is noted that the test duration was only 48h; it is possible that the low loadings are due to insufficient time allowed for equilibrium to be reached.

- Refer to Table 13-13 for a summary of the findings from the cyanide detoxification test work on Esaase flotation concentrate CIL and leach tailings.

Table 13-13: Cyanide Detoxification Test Work on Esaase Flotation Concentrate Cyanidation Products (source: DRA 2017)

Sample	Test Description	Results	Comments
50% Esaase Fresh: 50% Esaase Oxide flotation concentrate tail Esaase Oxide flotation concentrate 16h leach tail Esaase Fresh flotation concentrate 16h leach tail	Cyanide detoxification test work	SO_2 consumptions between 4.1g/g N_{WAD} and 5.9 g/g N_{WAD} was noted $CuSO_4.5H_2O$ consumptions between 75mg/L and 79mg/L was noted Lime consumptions between 0.26g/g SO_2 and 0.43g/g SO_2 was noted Cyanide in effluent was reduced to below 5mg/L from feed concentrations ranging from 780mg/L to 1030mg/L at a pH of 8.5	

In addition to the above test work, a number of tests were further conducted to assess the feasibility of installing 8 Mtpa flotation capacity as part of the AGM DFS P10M. Different flowsheets were considered mainly involving flotation of Nkran material and Esaase material in separate flotation circuits, after which different combinations of feed streams into the existing CIL circuit were tested.

The detail of these tests are excluded from this report as it pertains to a flotation circuit, while the selected flowsheet includes the treatment of gravity tailings in a CIL circuit (see Section 13.3.3.2) Refer to the AGM DFS report JGHDP0221-RPT-007 for more detail.

13.3.3 Addendum to Current Metallurgical Test Work

The initial flowsheet envisaged for the P10M DFS, was the addition of a 5.0 Mtpa gravity-flotation circuit (as per the PFS and Section 13.3.1.2 test work).

During the early stages of the DFS, the flowsheet was updated to utilize a common CIL circuit to treat the reground flotation concentrate and the AGM Phase 1 gravity tailings, in order to save on capital expenditure. However, the test work described in Section 13.3.2.2 above, indicated the following:

- High flotation mass pulls on Nkran Fresh V2 sample tested which achieved best recoveries when milled to 80% -75µm
- Moderate to high cyanide consumptions in the reground flotation concentrate CIL circuit
- Cyanide optimisation test work reduced cyanide consumptions to 0.35kg/t without impacting on recovery
- Sequential triple contact CIP test and equilibrium carbon loading test work indicated that poor management of the flotation circuit with excessive reagent addition would negatively impact on the achievable gold loadings and result in high solution losses in a common CIL circuit

A comparison between a gravity-flotation and gravity-CIL circuit was completed using cyanide consumptions based on Section 13.3.2 and Esaase gravity-CIL recoveries as per Nkran (assumed at the stage of the evaluation). The exercise indicated that, if the Esaase material were to respond similar to the Nkran material in a gravity-CIL circuit, then the gravity-CIL circuit would produce gold ounces at a lower cost. In addition, further reasoning towards a CIL circuit for the AGM P10M were:

- Successful operation of the AGM Phase 1 plant and the experience gained from operating the existing circuit
- Familiarity of the Ghanaian workforce with CIL circuits compared to flotation circuits
- Potential savings on reagent holding costs, reduced insurance and operating spares holdings

Based on the above, it was decided to further evaluate the inclusion of a gravity-CIL circuit for the treatment of the Esaase material as part of the AGM P10M DFS.

13.3.3.1 AGM DFS P10M – Phase 2 Test Work Campaign Addendum

This additional test work was conducted during Q4 2016 and was undertaken by Perth based ALS under the management of DRA.

The Phase 2 campaign addendum work was aimed at investigating the Esaase Sulphide material recovery potential in a gravity-CIL circuit similar to the existing Phase 1 circuit, and to further investigate the effect of gravity recovery and pre-loaded carbon on the CIL staged recovery. The preg-robbing potential of the Esaase Sulphide material was once again investigated.

The Phase 2 test work campaign addendum test work was conducted on an Esaase Fresh composite sample (Section 13.3.2.1).

A brief summary of the Phase 2 addendum test work scope and findings are presented below.

The detail is captured in the DRA DFS report (JGHDP0221-RPT-007) for more information.

- Refer to Table 13-14 for a summary of the findings from the test work conducted to evaluate the effect of different gravity recovery methods on the staged CIL recovery of the Esaase Fresh ore at a grind of 106 µm.

Table 13-14: Evaluation of Gravity Recovery Methods on Staged CIL Recovery of Esaase Fresh Material (source: DRA 2017)

Sample	Test Description	Results	Comments
Esaase Fresh	Gravity-CIL test work at P_{80} 75 µm and P_{80} 106 µm, applying double gravity recovery followed by mercury amalgam Gravity-CIL test work at P_{80} 106 µm, applying single gravity recovery followed by mercury amalgam Gravity-CIL test work at P_{80} 106 µm, applying single gravity recovery followed by intensive leach Whole ore leach at P_{80} 106 µm	Gravity recoveries in excess of 60% Gravity-CIL tests at 106 µm achieved overall recoveries between 94.8% to 95.2% when subjected to a 3.5h pre-oxidation and 24h CIL, with an average cyanide consumption of 0.33 kg/t Gravity-CIL tests at 75µm achieved overall recoveries from 94.90% to 95.60% when subjected to a 3.5h pre-oxidation and 24h CIL. An average cyanide consumption of 0.38 kg/t was noted Whole ore leach test at a grind of 106 µm achieved an overall recovery of 91.4%, at a cyanide consumption of 0.47 kg/t The double gravity recovery steps achieved roughly 10% higher gravity recovery, compared to a single gravity recovery step. The single stage recovery test consumed 0.48 kg/t cyanide compared to 0.33 kg/t for the double gravity recovery tests The single stage gravity recovery step followed by mercury amalgam achieved similar overall gold recoveries at similar cyanide consumptions, when compared to the single stage gravity recovery step followed by an intensive leach	Whole ore leach test result highlighted the requirement for an extensive gravity recovery circuit on Esaase Fresh material

- Refer to Table 13-15 for a summary of the findings from the gravity-CIL test work conducted to evaluate the effect of using pre-loaded carbon in CIL. An Esaase Fresh ore sample at a grind of 106 µm was used. Tests were conducted by addition of pre-loaded carbon at the start of CIL and 2 hours after cyanide addition. Further tests were done to assess the impact of the low cyanide concentrations in the pre-oxidation solution.

Table 13-15: Evaluation of the Effect of Pre-loaded Carbon on the Staged CIL Recovery of Esaase Fresh Material (source: DRA 2017)

Sample	Test Description	Results	Comments
Esaase Fresh	Investigation into effect of pre-loaded carbon on staged CIL recovery, at a grind of P80 106 µm	Gravity-CIL tests using pre-loaded carbon added at the start of CIL, and 10ppm cyanide in the pre-oxidation solution: Overall recoveries from 91.7% to 93.1% when subjected to a 2h pre-oxidation followed by a 24h CIL. Residues ranged from 0.11 g/t Au to 0.14 g/t Au Average cyanide consumption of 0.40kg/t Gravity-CIL tests using pre-loaded carbon added after 2 hours of cyanidation: Overall recovery of 89.6% when subjected to a 2h pre-oxidation followed by a 24h CIL Residue value of 0.13 g/t Au Cyanide consumption of 0.47 kg/t	Final leach solution grades increased from less than detection to 0.015 g/t Au

-

- Refer to Table 13-16 for a summary of the findings from the test work to investigate the preg-robbing potential of Esaase Fresh ore. Gravity-cyanidation tests were conducted at a grind of 106 µm with no carbon addition, carbon addition at start of cyanidation, and carbon addition after 2 hours of cyanidation.

Table 13-16: Evaluation of the Preg-Robbing Potential of Esaase Fresh Material (source: DRA 2017)

Sample	Test Description	Results	Comments
Esaase Fresh See Section 13.3.2.1	Investigation into preg-robbing potential of sample at a grind of P_{80} 106 µm	Gravity-CIL tests achieved overall recoveries from 92.8% to 93.4% when subjected to a 2h pre-oxidation followed by a 24h CIL. Residue values ranging from 0.11 g/t Au to 0.12 g/t Au was recorded. An average cyanide consumption of 0.44 kg/t was recorded. Gravity-leach test achieved an overall recovery of 91.0% when subjected to a 2h pre-oxidation followed by a 24h leach. A residue value of 0.14 g/t Au and a cyanide consumption of 0.17 kg/t was recorded.	Although the leach residue grade increased from the CIL residue grades (by 0.02g/t), is bordering on the determination limit of the assaying method used, and likely within experimental variation. Any preg-robbing occurring, is countered by the addition of carbon.

13.3.3.2 Conclusion: AGM DFS P10M – Phase 2 Test Work Addendum

Refer to Table 13-17 below for a summary of the main input parameters used in the trade-off study to evaluate the treatment of Esaase material in a gravity-flotation and gravity-CIL circuit.

Table 13-17: Esaase Circuit Evaluation Input Parameters (source: DRA 2017)

Parameter	Gravity-Flotation	Gravity-CIL
LoM Ore Tonnage	37,026,870 t	37,026,870 t
Esaase Sulphides	*84.0 %*	*84.0 %*
Esaase Transitional	*12.7 %*	*12.7 %*
Nkran Sulphides	*3.3 %*	*3.3 %*
Head Grades	1.40 g/t Au	1.40 g/t Au
Esaase Sulphides	*1.37 g/t Au*	*1.37 g/t Au*
Esaase Transitional	*1.50 g/t Au*	*1.50 g/t Au*
Nkran Sulphides	*1.90 g/t Au*	*1.90 g/t Au*
Gravity Recovery	51.2 %	51.2 %
Esaase Sulphides	*52.0 %*	*52.0 %*
Esaase Transitional	*47.0 %*	*47.0 %*
Nkran Sulphides	*50.2 %*	*50.2 %*
Solid Residue Grade	0.16 g/t Au[2]	0.12 g/t Au
Esaase Sulphides	*Evaluated as a blend*	*0.10 g/t Au*[3]
Esaase Transitional		*0.22 g/t Au*
Nkran Sulphides		*0.10 g/t Au*
Discounted Recovery (includes solution and carbon losses)	87.5 %	90.6 %
Electricity supply cost	US$ 0.15 / kWh	US$ 0.15 / kWh
Target grind	80% - 75µm	80% - 106µm
Mass pull	9.5 %	Not applicable
Concentrate target grind	90% - 20µm	Not applicable
Cyanide consumption	Concentrate CIL	Gravity Tail CIL
Esaase Sulphides	0.59 kg/t milled	0.72 kg/t milled
Esaase Transitional	0.48 kg/t milled	0.53 kg/t milled
Nkran Sulphides	0.73 kg/t milled	0.42 kg/t milled
Operating Cost	US$ 10.86 /t milled	US$ 9.74 /t milled
Production Cost	**US$ 275 /oz produced**	**US$ 238 /oz produced**

[2] Includes flotation tailings and concentrate leach residue
[3] Value assumed at time of evaluation based on AGM Phase 1 operation data on Nkran Fresh ore.

Based on the above evaluation, the Phase 2 addendum test work was motivated.

During the addendum test work, the Esaase Fresh material achieved recoveries ranging from 93.4% to 95.1% in a CIL circuit, with residues varying from 0.105g/t Au to 0.115g/t Au with a sample head grade of 2.2 g/t Au tested. Cyanide consumptions between 0.51 kg/t to 0.56 kg/t was recorded.

It can be concluded that the assumptions made in the evaluation as per Table 13-17 was validated by the addendum test work.

13.4 Recovery Assessment for the AGM DFS Project

- Test work as per Section 13.3.2 and Section 13.3.3 informed the decision to move from a gravity-flotation circuit to a gravity-CIL circuit for the P10M circuit.
- The AGM P5M processing plant recovery estimated describes the expected recovery when treating a blend of Nkran ores together with Esaase Oxide material in an upgraded Phase 1 gravity-CIL circuit, while the AGM P10M processing plant recovery estimate describes the expected recovery when treating Esaase material in an additional 5 Mtpa gravity-CIL circuit – similar to the upgraded Phase 1 plant.

13.4.1 Information Sources

- The assessment was based on the following information:
- The mine production schedules indicating plant gold feed grade per ore type as supplied by mining team
- Historical Resolute operating data
- Current AGM Phase 1 operating data
- Various metallurgical test work campaigns:
  - Obotan DFS testing at ALS in 2012
  - Obotan Variability testing at SGS Booysens in 2014/2015
  - Obotan GRG test work and modelling report
  - Esaase Fresh gravity-CIL testing as part of AGM DFS Phase 2 Addendum test work at ALS in 2016
  - Dynamite Hill testing at Metallurgy Pty Ltd in 2014

13.4.2 LoM Mining and Plant Feed Profile

The following LoM average plant feed grades apply:

- P5M gravity-CIL plant: 1.65 g/t Au
- P10M gravity-CIL plant: 1.46 g/t Au

Refer to Table 13-18 for a summary of the plant feed profile expected for the AGM DFS Project.

Table 13-18: AGM DFS Plant Feed Profile Summary (source: DRA 2017)

Ore Type	P5M Gravity-CIL Plant[4]		P10M Gravity-CIL Plant	
	kt	%	kt	%
Obotan Oxide	3,887	6.6%	3	0.0%
Obotan Transitional	2,500	4.3%	18	0.0%
Obotan Fresh	32,338	55.3%	1,434	3.2%
Esaase Oxide	4,666	8.0%	10,776	24.4%
Esaase Transitional	2,081	3.6%	4,505	10.2%
Esaase Fresh	13,054	22.3%	27,484	62.2%

13.4.3 Basis of Recovery Estimate

13.4.3.1 Gravity-CIL Recovery Estimate for Nkran Ore

The Nkran ore gravity-CIL recovery estimate was based on modeled gravity circuit recovery, final CIL residue grades as determined from test work, historical operating data, as well as current AGM Phase 1 operating data.

Gravity Recovery Estimate

The gravity recovery estimate for each Nkran ore type is summarized in Table 13-19.

Table 13-19: Nkran Gravity Recovery Data used in the AGM DFS Recovery Estimate (source: DRA 2017)

Ore Type	Head Grade g/t Au	Oz. % of LoM	Gravity Recovery %	Information Source
Nkran Fresh	1.79	37.5 %	54.2 %	
Nkran Main Pit	1.85	28.4 %	55.0 %	Phase 1 operating data
Other[5]	1.61	9.0 %	51.7 %	Dynamite hill testing ALS 2012 Benchmarking
Nkran Oxide	1.31	3.2%	37.3%	
Combined[2]	1.31	3.2%	37.3%	Historical Resolute data ALS 2012 Benchmarking
Nkran Transitional	1.65	2.6%	48.5%	

[4] AGM Phase 1 circuit upgraded to 5.0 Mtpa

[5] Satellite pits: Akwasiso, Adubiaso Main and Extension, Dynamite hill, Abore, Asuadai, Nkran extension

Ore Type	Head Grade g/t Au	Oz. % of LoM	Gravity Recovery %	Information Source
Combined[2]	1.65	2.62%	48.5%	Historical Resolute data ALS 2012 Dynamite hill 2014 Benchmarking

CIL Recovery Estimate – Nkran Fresh

Operational data from the initial months of operation of the AGM Phase 1 plant was used as basis of the Nkran Fresh residue value used in the recovery estimate. The operational data was compared to the test work data available on the Nkran Sulphide material.

It can be noted from Figure 13-4 that the 85th percentile point of the operational data (March 2016 – June 2016) corresponded well to the pre-2015 test work data available. Based on this, the Nkran Fresh CIL residue grades were calculated using the pre-2015 test work data trend line.

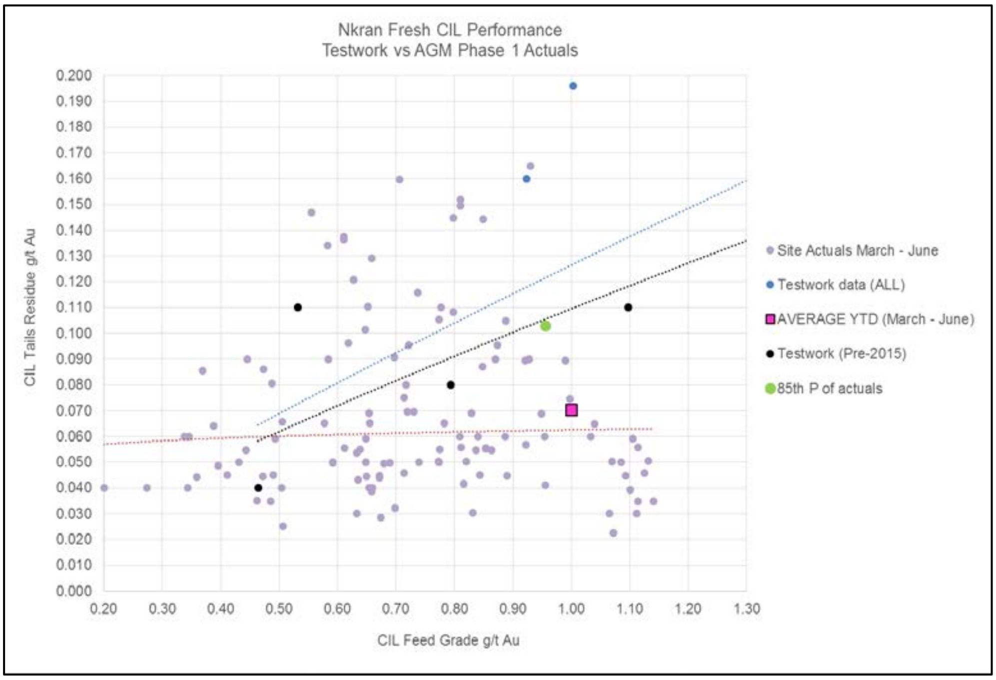


Figure 13-4: Nkran Fresh CIL Residue Grade as a Function of CIL Feed Grade (source: DRA 2017)

Refer to Table 13-20 for a summary of the Staged CIL recoveries per ore type, as applied in the recovery evaluation.

Table 13-20: Nkran Staged CIL Recovery Data Used in the AGM DFS Recovery Estimate (source: DRA 2017)

Deposit	CIL Feed g/t Au	% of LoM Oz.	Staged CIL Recovery	Information Source
Nkran Fresh	0.82	37.5 %	88.6 %	
Nkran Main Pit	0.83	28.4 %	88.7 %	Pre-2015 test work. In-line with current operational info.
Other[6]	0.78	9.0 %	88.4 %	Dynamite hill testing ALS 2012 testing Benchmarking
Nkran Oxide	0.82	3.2 %	88.3 %	
Combined[6]	0.82	3.2 %	88.3 %	Historical Resolute data ALS 2012 testing Dynamite hill 2014 testing Benchmarking
Nkran Transitional	0.85	2.6 %	88.6 %	
Combined[6]	0.85	2.6 %	88.6 %	Historical Resolute data ALS 2012 testing Dynamite hill 2014 testing Benchmarking

13.4.3.2 Gravity-CIL Recovery Estimate for Esaase Ore

The Esaase ore gravity-CIL recovery estimate was based test work conducted as per Section 13.3.3, as well as bench marked data from Nkran material.

Gravity Recovery Estimate

The gravity recovery estimate for each Nkran ore type is summarized in Table 13-21 below.

[6] Satellite pits: Akwasiso, Adubiaso Main and Extension, Dynamite hill, Abore, Asuadai, Nkran extension

Table 13-21: Esaase Gravity Recovery Data Used in the AGM DFS Recovery Estimate (source: DRA 2017)

Esaase Material	Head Grade g/t Au	% of LoM Oz.	Gravity Recovery%	Information Source
Fresh	1.46	36.7 %	55.0 %	Benchmarking
Oxides	1.41	13.5 %	48.7 %	ALS 2015 test work
Transitional	1.60	6.5 %	43.0 %	ALS 2015 test work

CIL Recovery Estimate – Esaase Fresh

Phase 2 addendum test work data at a grind of 106 µm was compared against operational data from the AGM Phase 1 plant, as per Figure 13-5.

It was noted that, the Phase 2 addendum test work data point normalized to the Nkran Fresh gravity recovery of 55%, agreed with the Nkran Fresh pre-2015 CIL Feed vs CIL residue test work correlation.

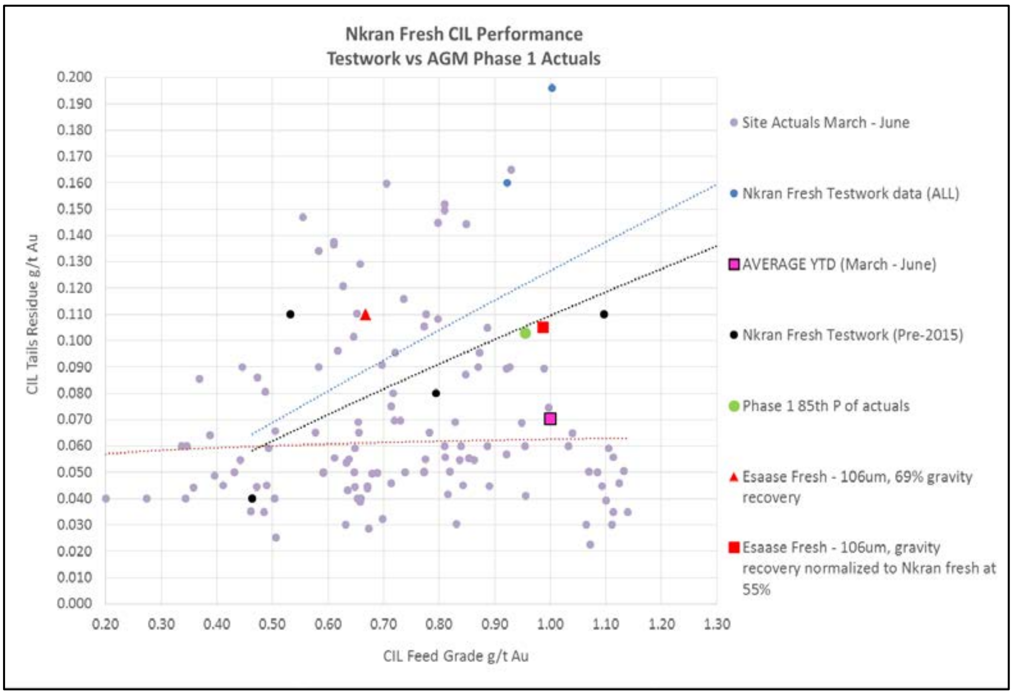


Figure 13-5: Esaase Fresh CIL Residue Grade as Compared to Nkran Fresh data (source: DRA 2017)

The expected Esaase Fresh average LoM head grade ranges between 0.87 g/t Au and 2.28 g/t Au for the AGM Expansion Project CIL circuits. When applying a 55% gravity recovery, a CIL feed grade of 0.39 – 1.03 g/t Au is calculated. Variability test work on Nkran Fresh samples at SGS, during 2014/2015, as part of the AGM Phase 1 campaign also reported CIL residue values ranging between 0.04 g/t Au and 0.08 g/t Au for head grades between 1.36 g/t Au and 1.78 g/t Au. Gravity recoveries ranging between 55% and 66% were reported.

Based on the above bench marking, the Nkran Fresh pre-2015 test work correlation was applied to determine the CIL residue value applied to the Esaase Fresh material in the recovery evaluation. A minimum residue value of 0.08 g/t Au was applied to the lower grade Esaase Fresh material.

CIL Recovery Estimate – Esaase Oxides

The staged CIL recovery of the Esaase Oxide material was based on test work conducted at ALS in 2015. A staged CIL recovery of 87.9% was applied in general, with a minimum solids residue of 0.09 g/t for the lower grade material.

CIL Recovery Estimate – Esaase Transitional

Due to a lack of sufficient test work data available on the processing of Esaase Transitional material in a gravity-CIL circuit at a grind of 106 µm, solids residue values as per Esaase Fresh material (based on correlation of pre-2015 test work data) was applied to the Esaase Transitional material.

It is noted that the Esaase Transitional material contributes a total of 6.5% of the AGM DFS LoM tonnage.

13.4.3.3 Basis of Recovery Estimate Summary

A summary of the basis for the recovery estimate of the AGM DFS is presented in Table 13-22.

Table 13-22: Summary of the Data Used to Derive Gravity-CIL Recovery Estimates (source: DRA 2017)

<table>
<tr><th>Ore Type</th><th>Gravity Stage Recovery (%)</th><th>Maximum CIL Stage Recovery</th><th>Minimum CIL Tails Residue Grade (g/t Au)[7]</th></tr>
<tr><td>Nkran Oxide</td><td>37.3%</td><td>88.3 %</td><td>0.10</td></tr>
<tr><td>Nkran Transition</td><td>48.5%</td><td>88.6 %</td><td>0.10</td></tr>
<tr><td>Nkran Fresh</td><td>54.2%</td><td>88.3 %</td><td>0.09</td></tr>
<tr><td>Esaase Oxide</td><td>48.7%</td><td>87.9 %</td><td>0.09</td></tr>
<tr><td>Esaase Transition</td><td>43.0%</td><td>88.9 %</td><td>0.10</td></tr>
<tr><td>Esaase Fresh</td><td>55.0%</td><td>87.8 %</td><td>0.08</td></tr>
<tr><td colspan="4">Recovery Estimation Criteria</td></tr>
<tr><td colspan="4">Gravity recovery estimate derived from current operating data as well as GRG test work and modelling, and weighted as per the various deposits in the mine blend.
CIL Solids residues per ore type derived from current operational data and test work, and weighted as per the various deposits in the mine blend.
106 µm target grind.
100 ppm terminal cyanide concentration in CIL circuit.</td></tr>
<tr><td colspan="4">Discount Factors</td></tr>
<tr><td>CIL Carbon fines losses</td><td colspan="3">Based on 40 g/t carbon at 50 g/t Au.</td></tr>
<tr><td>Solution Au losses</td><td colspan="3">Based on 45% solids in CIL tailings and 0.01 g/L Au in solution.</td></tr>
<tr><td>Commissioning</td><td colspan="3">0.01 % over LoM for P5M; 0.02 % over LoM for P10M.</td></tr>
</table>

7 Due to the level of analytical accuracy of a solids tailings grade of <0.1 g/t Au, these values need to be substantiated by an overall monthly gold balance.

13.4.4 LoM Recovery Estimate for AGM DFS

The ore blends and plant production profiles detailing the expected feed blends and LoM recovery for each of the AGM DFS are presented in Table 13-23 and Table 13-24. It is noted that the residue values from the Project 10M test work campaign compared to the current residues of the existing AGM operation of treating gravity tailings in a CIL circuit. These test work residues ranging between 0.10 g/t Au to 0.12 g/t Au are optimal by industry standards. On this basis there was no reason to determine any effect of deleterious elements and the impact it would have on the economic extraction, on the samples tested.

Table 13-23: AGM P5M Gravity-CIL Plant LoM Recovery Estimate (source: DRA 2017)

		LoM	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029
Mill Feed Tonnage	kt	58,525	3,950	5,012	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	3,713	850
Obotan Oxide	%	*6.6%*	*4.6%*	*36.9%*	*2.6%*	*0.7%*	*2.9%*	*0.4%*	*0.2%*	*3.0%*	*3.0%*	*0.0%*	*13.2%*	*15.1%*	*0.0%*
Obotan Transitional	%	*4.3%*	*0.1%*	*2.0%*	*1.7%*	*0.0%*	*2.8%*	*0.0%*	*1.0%*	*0.0%*	*1.6%*	*0.0%*	*40.8%*	*0.0%*	*0.0%*
Obotan Fresh	%	*55.3%*	*95.4%*	*61.1%*	*63.6%*	*67.3%*	*62.3%*	*67.6%*	*64.8%*	*60.0%*	*58.4%*	*60.0%*	*6.0%*	*0.0%*	*0.0%*
Esaase Oxide	%	*8.0%*	*0.0%*	*0.0%*	*32.0%*	*29.7%*	*12.8%*	*7.8%*	*2.8%*	*2.0%*	*1.1%*	*1.6%*	*1.8%*	*2.3%*	*0.0%*
Esaase Transitional	%	*3.6%*	*0.0%*	*0.0%*	*0.0%*	*2.2%*	*12.4%*	*4.6%*	*6.1%*	*7.0%*	*3.6%*	*2.5%*	*0.5%*	*3.7%*	*0.0%*
Esaase Fresh	%	*22.3%*	*0.0%*	*0.0%*	*0.0%*	*0.1%*	*6.8%*	*19.6%*	*25.1%*	*27.9%*	*32.3%*	*35.9%*	*37.7%*	*78.8%*	*100.0%*
Mill Feed Grade	g/t Au	1.65	2.03	1.78	1.60	1.56	1.79	1.49	1.52	1.71	1.62	1.63	1.52	1.56	1.74
Gravity Recovery	%	52.5%	53.6%	49.7%	52.3%	52.4%	51.8%	53.3%	53.4%	53.0%	53.6%	54.2%	49.8%	52.1%	55.0%
CIL Feed Grade	g/t Au	0.78	0.94	0.90	0.77	0.74	0.86	0.69	0.71	0.80	0.75	0.75	0.76	0.75	0.78
CIL Residue Grade[8]	g/t Au	0.10	0.11	0.11	0.09	0.09	0.10	0.09	0.09	0.10	0.09	0.09	0.09	0.09	0.10
Undiscounted Recovery	%	94.2%	94.6%	94.0%	94.1%	94.1%	94.3%	93.8%	94.1%	94.4%	94.4%	94.4%	94.1%	94.2%	94.5%
Recovery Discount	%	0.5%	0.6%	0.7%	0.7%	0.6%	0.7%	0.8%	0.8%	0.7%	0.7%	0.7%	0.8%	0.8%	-13.9%
Discounted Recovery	%	93.7%	94.0%	93.3%	93.2%	93.5%	93.6%	93.0%	93.3%	93.7%	93.6%	93.7%	93.3%	93.4%	108.4%[9]

8 Due to the level of analytical accuracy of a solids tailings grade of <0.1 g/t Au, these values need to be substantiated by an overall monthly gold balance.

9 Includes gold lock-up recovered in last month of operations.

Table 13-24: AGM P10M Gravity-CIL Plant LoM Recovery (source: DRA 2017)

		LoM	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029
Mill Feed Tonnage	kt	44,219				3,450	5,000	5,000	4,890	5,000	5,000	5,000	5,000	5,000	879
Obotan Oxide	*%*	*0.0%*				*0.0%*	*0.0%*	*0.0%*	*0.0%*	*0.1%*	*0.0%*	*0.0%*	*0.0%*	*0.0%*	*0.0%*
Obotan Transitional	*%*	*0.0%*				*0.0%*	*0.0%*	*0.0%*	*0.4%*	*0.0%*	*0.0%*	*0.0%*	*0.0%*	*0.0%*	*0.0%*
Obotan Fresh	*%*	*3.2%*				*0.0%*	*0.0%*	*2.3%*	*18.8%*	*8.0%*	*0.0%*	*0.0%*	*0.0%*	*0.0%*	*0.0%*
Esaase Oxide	*%*	*24.4%*				*92.1%*	*53.1%*	*41.8%*	*22.7%*	*14.0%*	*11.7%*	*4.0%*	*4.4%*	*0.8%*	*0.0%*
Esaase Transitional	*%*	*10.2%*				*7.5%*	*30.4%*	*10.8%*	*11.3%*	*15.8%*	*8.8%*	*6.2%*	*1.2%*	*0.7%*	*0.0%*
Esaase Fresh	*%*	*62.2%*				*0.5%*	*16.5%*	*45.1%*	*46.8%*	*62.2%*	*79.5%*	*89.8%*	*94.3%*	*98.5%*	*100.0%*
Mill Feed Grade	g/t Au	1.46				1.48	1.38	1.34	1.46	1.59	1.39	1.42	1.44	1.61	1.85
Gravity Recovery	%	52.16				48.3%	47.8%	50.8%	51.8%	51.6%	53.3%	54.2%	54.6%	54.8%	55.0%
CIL Feed Grade	g/t Au	0.70				0.76	0.72	0.66	0.71	0.77	0.65	0.65	0.65	0.72	0.83
CIL Residue Grade[10]	g/t Au	0.09				0.09	0.09	0.09	0.09	0.09	0.08	0.08	0.08	0.09	0.10
Undiscounted Recovery	%	94.1				93.8%	93.7%	93.6%	94.1%	94.2%	94.2%	94.2%	94.4%	94.5%	94.5%
Recovery Discount	%	0.8				5.2%	0.9%	0.9%	0.8%	0.8%	0.9%	0.8%	0.8%	0.7%	-13.1%
Discounted Recovery	%	93.3				88.3%	92.9%	92.7%	93.3%	93.5%	93.4%	93.4%	93.6%	93.7%	107.6%[11]

10 Due to the level of analytical accuracy of a solids tailings grade of <0.1 g/t Au, these values need to be substantiated by an overall monthly gold balance.

11 Includes gold lock-up recovered in last month of operations.

14 MINERAL RESOURCE ESTIMATE

Qualified Persons from CJM Consulting ("CJM") and CSA Global ("CSA") compiled the components of the MRE's, in compliance with the definitions and guidelines for the reporting of Exploration Information, Mineral Resources and Mineral Reserves in Canada, "the CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines" (2014). These MRE also adhere to the Rules and Policies of the National Instrument 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1 and Companion Policy 43-101CP.

Furthermore, the Mineral Resource classifications are consistent with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of 2012 as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and the Mineral Council of Australia ("JORC").

On a global basis, both CJM and CSA are satisfied that the MRE globally reflects the ore bodies based on the available data. Suitably experienced and qualified geologists, surveyors and other Mineral Resource practitioners employed by Asanko were responsible for the capture of the drill hole information and geological information. The QP's do not disclaim responsibility for the technical data and information captured.

For the purposes of this disclosure CJM has estimated the MRE for the AGM projects Esaase, Abore, Asuadai, Adubiaso, Adubiaso Extension and Nkran Extension, and CSA estimated the MRE for Nkran and Dynamite Hill, and has overseen the Akwasiso MRE (April 2017). All other estimates are as at 31 December 2016. All gold grade estimation was completed using Ordinary Kriging ("OK"). This estimation approach was considered appropriate based on a review of various factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style and geometry of mineralisation.

14.1 Effective Date of Mineral Resource

The effective date of the MRE's is 31 December 2016 with the exception of Akwasiso which is 30 April 2017.

14.1.1 Cautionary Note about Mineral Resources

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. MRE's do not account for mine ability, selectivity, mining loss and dilution. These MRE's include Inferred Mineral Resources that have a lower leavel of confidence than Indicated Mineral Resources and as such have not been converted to Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to the Indicated category through further exploration. The Company advises investors that while the term "Inferred Mineral Resources" is recognised and required by Canadian regulations, the SEC does not recognise it. Under Canadian rules, estimates of Inferred Mineral Resources may not form the basis of economic studies and they have not been used in the 2017 DFS or this NI 43-101 to estimate Mineral Reserves.

As per the May 2014 CIM Mineral Definition Standards, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with further evaluation, considering Inferred Mineral Resources are commonly direct extensions of higher confidence Mineral Resources. All Inferred Resources reported are constrained by a $1500/Oz Au pit shell and as such, are considered to have a reasonable prospect of eventual economic extraction.

14.2 Assumptions, Parameters and Methods used for MRE's

The Asanko Gold Mine deposits are all located along major shear corridors at the intersection with cross-cutting faults. The mineralisation is directly related to the structural setting and interactions with specific lithological units. Interpreted 3D geological models delineated based on lithological-structural domains form the basis for the MRE. Host lithologies are primarily isoclinally folded basin sedimentary units regionally overprinted by greenschist facies metamorphism and small granitic intrusions. Gold mineralisation is directly associated with quartz veins distributed preferencially within these units. The gold in all deposits is free-milling and non-refractory.

Within each domain, mineralisation and waste proportions were delineated using an Indicator Kriging ("IK") methodology. In all CJM estimations a 0.3 g/t cut-off was used to flag data as mineralised or waste. Above 0.3 g/t is assigned a value of 1 for ore, and below 0.3 g/t a value of zero for waste. Internal waste (values below 0.3 g/t) of 2m and less were also considered a part of the mineralised zone and flagged 1. The 1 and 0 values were then estimated into a block model using ore controlling orientations and relationships observed and measured from drill core and 3D modelling. Each domain has specific probabilities that are related to expected mineralisation relationships. The unconstrained areas outside main lithological domains, (where lithological boundaries do not constrain the mineralisation), were estimated using the same probabilistic approach. In most cases, these were part of the secondary mineralisation which was distinctly different from the main mineralisation.

All geological and structural inputs into the 3D geological models were collected by Asanko personnel. The Nkran, Akwasiso, Dynamite Hill, Adubiaso Extension and Nkran Extension geological models have been updated from the 2014 43-101 and the Esaase, Abore, Adubiaso and Asuadai models are the same. Model updates are the result standard best practice updates to the existing knowledge base resulting from the addition of new understanding based on reinterpretation of results.

A summary of key criteria for drilling, sampling and geology are tabulated below Table 14-1. These criteria and the estimation parameters have been consistently applied to all the MRE's in this report.

Table 14-1: Confidence Levels of Key Criteria for Drilling, Sampling and Geology

Items	Discussion	Confidence
Drilling Techniques	RC/Diamond - Industry standard approach.	High
Logging	Standard nomenclature and apparent high quality.	High
Drill Sample Recovery	Diamond core and RC recovery adequate.	High
Sub-sampling Techniques and Sample Preparation	Industry standard for both RC and Diamond core.	High
Quality of Assay Data	Quality control conclusions outlined in Section 14. Some issues were identified. Recent improvements were noted.	Moderate
Verification of Sampling and Assaying	Dedicated drill hole twinning to reproduce original drill intercepts.	High
Location of Sampling Points	Survey of all collars with adequate downhole survey. Investigation of available downhole survey indicates expected deviation.	High
Data Density and Distribution	Core mineralisation defined on a notional 40 mE by 30 mN drill spacing with a small area drilled at 20 mE by 20 mN. Other areas more broadly spaced to approximately 80 mN.	Moderate to High
Database Integrity	Minor errors identified and rectified	High
Geological Interpretation	The broad mineralisation constraints are subject to a large amount of uncertainty concerning localised mineralisation trends as a reflection of geological complexity. Closer spaced drilling is required to resolve this issue.	Moderate
Rock Dry Bulk Density	DBD measurements taken from drill core, DBD applied is considered robust when compared with 3D data.	High below top of transition, moderate in oxide material

The following aspects, or parameters are considered from a geostatistical aspect:

- Number of samples used to estimate a specific block:
  - Measured: at least 9 drill holes within variogram range and a minimum of 27 one meter composited samples
  - Indicated: at least 4 drill holes within variogram range and a minimum of 12 one meter composite samples
  - Inferred: 1 drill hole within search range

- Distance to sample (variogram range):
  - Measured: at least within 60% of variogram range
  - Indicated: within variogram range
  - Inferred: further than variogram range and within geological expected limits
- Lower confidence limit (blocks):
  - Measured: less than 20% from mean (80% confidence)
  - Indicated: 20 to 40% from mean (80 to 60% confidence)
  - Inferred: more than 40% (less than 60% confidence)
- Kriging efficiency:
  - Measured: more than 40%
  - Indicated: 10 to 40%
  - Inferred: less than 10%
- Deviation from lower 90% confidence limit, (data distribution within resource area considered for classification):
  - Measured: less than 10% deviation from mean
  - Indicated: 10 to 20%
  - Inferred: more than 20%
- Regression Slope:
  - Measured – 90%
  - Indicated - 60% to 90%
  - Inferred – less than 60%

The definition of a Measured MRE requires that the nature, quality, amount and distribution of data are such as to leave the CP with no reasonable doubt that the tonnages and grade of the mineralisation can be estimated to within close limits and any variation within these limits would not materially affect the economics of extraction. CJM and CSA reviewed the spatial distribution of the relevant drill hole data, the robustness of the kriged model and the parameters for the area classified into Measured Resource. Both CJM and CSA concur that the geological models are well understood and defined in the Measured areas, and further drilling would not affect any material changes to the tonnages and grades.

The definition of Indicated Mineral Resource states that there should be sufficient confidence for mine design, mine planning, or economic studies. CJM and CSA are of the opinion that there is sufficient confidence in the estimate of the Indicated Resource areas to allow the appropriate application of technical and economic parameters and enable an evaluation of economic viability.

The following numeric resource category codes were assigned into the block models, based on the categorisation criteria listed above:

Measured Resource: RESCAT = 1

Indicated Resource: RESCAT = 2

Inferred Resource: RESCAT = 3

Unclassified Resource: RESCAT = 0

MR classification areas, as well as the grades associated with MR classification, as well as the grade versus tonnage curves are presented in each deposit MRE.

14.3 Density

For the AGM Phase 1 all density measurements for the AGM Phase 1 deposits were taken by means of the water immersion method. A large volume of data was collected by both Resolute and PMI over a range of rock types from half-core samples (667 total - Nkran 269, Adubiaso 129, Abore 184 and Asuadai 85; over a period of about 15 years). This data was viewed as being of sufficient quality and reliability for use in the resource estimation conducted by SRK in May of 2012 and is currently viewed by CJM as being of industry standard practice. The database continues to increase as the Asanko mining operations progress. CSA are of the opinion that current data is adequate for the MRE and that Asanko continue to build the database, especially with respect to oxide densities (saprolite vs saprock).

PMI sampling data included 664 density measurements performed by SGS Bibiani and SGS Tarkwa on samples of DC. For solid core, densities were determined by water immersion with samples wrapped in cling film and coated in hairspray, or beeswax to prevent water absorption. For broken, crumbly, or clay core intervals, densities were determined by a volumetric method. The samples are weighed in air and weighed in water, while suspended beneath a balance. The density was then calculated from these measurements.

The density results used in the SRK May, 2012 and the relevant weathered oxidation state, (as used in that resource estimate) are summarised in Table 14-2 below. These density results were also used by CJM in the October 2014 and December 2016 MRE. The fresh rock densities used for the Nkran pit have been adjusted to reflect ongoing monthly tests conducted during mining operations.

The density measurements originally were dominated by un-weathered and saprolite samples, but the evaluation of Esaase and commencement of mining fresh rock at Nkran has improved the fresh rock sampling. On average across all project areas, measurements tend to show approximately 20% lower densities in the saprolite when compared to transitional samples, and 30% lower densities when compared to fresh measurements.

It is the view of CJM and CSA that additional work should continue to be conducted to validate the historical data and to assist in increasing the tonnage confidence in ongoing MRE exercises.

Table 14-2: Bulk Densities Summarised by Deposit and Oxidation State

Project Area	Oxidation State	No Samples	Density (t/m^3)		
			Minimum	Average	Maximum
Esaase	Saprolite - Oxide	579	1.66	2.29	3.01
	Transitional	2,394	1.56	2.42	3.81
	Unweathered -Fresh	9,765	1.12	2.78	4.34
Nkran DD Core	Saprolite - Oxide	5	1.47	1.68	1.82
	Unweathered - Fresh	10	1.94	2.10	2.29
Nkran in Pit	Saprock - Oxide	-	-	-	-
	Transitional	-	-	-	-
	Unweathered - Fresh	1,289	2.11	2.65	2.96
Abore	Saprolite - Oxide	6	1.72	1.85	1.98
	Transitional	6	2.14	2.42	2.57
	Unweathered - Fresh	21	2.08	2.67	2.87
Adubiaso	Saprolite - Oxide	16	1.52	1.97	2.20
	Transitional	17	2.19	2.40	2.57
	Unweathered - Fresh	19	2.47	2.68	3.49
Asuadai	Saprolite - Oxide	-	-	-	-
	Transitional	-	-	-	-
	Unweathered - Fresh	4	2.71	2.75	2.83
Akwasiso	Saprolite - Oxide	8	1.13	1.70	2.55
	Transitional	3	1.48	2.18	2.76
	Unweathered - Fresh	33	2.50	2.73	2.87
Dyanite Hill	Saprolite – Oxide	26	1.48	1.76	2.21
	Transitional	7	2.01	2.33	2.61
	Unweathered - Fresh	54	2.44	2.73	2.82
Nkran Extension	Saprolite – Oxide	-	-	-	-
	Transitional	-	-	-	-
	Unweathered - Fresh	3	2.70	2.78	2.75

14.4 Determination of Mineral Resource disclosure cutoff grade and pit shell constraints

Asanko Gold has determined for the purposes of the current mineral resource disclosures to use a cutoff grade of 0.5 g/t Au within a US$1500/oz Au price shell. This is a change from the last disclosure (PR 5 June 2017) where a 0.5 g/t cutoff within a US$2,000/oz pit shell was used to constrain resources.

The Company has adopted a more conservative approach to the pit shell constraint, which underpins the QP's judgement with respect to 'reasonable prospects for eventual economic extraction'.

The AGM cutoff grade is determined from the analysis of actual mining, metallurgical processing and G & A operating costs since steady state production in April 2016 till the end of May 2017. The paylimit or breakeven grade for the AGM currently is 0.45 g/t Au. Table 14-3 summarises the parameters used.

Table 14-3 : AGM paylimit calculation. (Source : Asanko Gold May 2017)

Period	Av Opex $/t	Au price Au $/g	Paylimit Au g/t
Apr 16 - May 17	17.50	39.15	0.45

On the basis of this analysis, the Company has taken a more conservative approach to the mineral resource disclosure, and has applied a lower cutoff grade of 0.5 g/t Au.

The application of the $1500/oz Au shell as a constraint is in line with current disclosure practice for gold producers, who range from US$1300-US$1550/oz for mineral resource constraints).

It should be noted that the adjustment to 0.5 g/t Au within a US$1500/oz Au price shell does not impact any of AGM Mineral Reserves, as they are calculated at US$ 1300/oz Au with modifying factors such as dilution and ore losses during mining.

14.5 Nkran Mine MRE (CSA Global)

CSA Global undertook a revised MRE on Nkran in January 2017 (CSA Global Technical Report Nkran February 2017). Malcolm Titley is the QP for this MRE. All of the Tabulations and figures in this section are sourced from the CSA Global Technical Report Nkran February 2017.

The resource estimation methodology comprised the following procedures:

- Asanko geologists created geological surfaces and volumes using 3D implicit and explicit modelling using MicromineTM software. These interpretations were based on logged lithologies, logged oxidation state and chemical Au assays

- Define resource domains for the GC model
- Define resource domains for the IK model
- Data compositing and declustering for geostatistical analysis, variography and validation
- Application of top cuts based on geostatistical analysis
- Construct block model following Kriging Neighbourhood Analysis ("KNA")
- Grade interpolation using OK for the GC and IK models
- Combine GC and IK models
- Resource classification, validation and reporting
- Technical resource report on the MRE
- This estimation approach was considered appropriate based on review of several factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style and geometry of mineralisation
- CSA is satisfied that the MRE globally reflects the ore body based on the available data

14.5.1 Drill Hole Database Loading

CSA was provided an Access Database for Nkran containing collar, downhole survey, lithology, mineralogy, structural, vein, alteration and assay data. This information was exported in CSV format. An excel sheet with dry bulk density data was received separately. The drill data was imported into SQL and Datamine Studio RMTM software for validation.

14.5.1.1 Database Validation

Data was loaded into a SQL database which has constraints and triggers, ensuring that only validated data was included in the database. During the validation process issues were highlighted and corrected where possible. Exports of the clean, verified data were provided to the resource geologists for the MRE.

The list below includes the validation and checks completed:

- Collar table: Incorrect coordinates (not within known range), duplicate holes
- Survey table: Duplicate entries, survey intervals past the specified maximum depth in the collar table, overlapping intervals, abnormal dips and azimuths
- Geotechnical table: Overlapping intervals, missing collar data, negative widths, geotechnical results past the specified maximum depth in the collar table

- Geology, Sample and Assay tables: Duplicate entries, lithological intervals past the specified maximum depth in the collar table, overlapping intervals, negative widths, missing collar data, missing intervals, correct logging codes, duplicated sample ID's, missing samples (assay results received, but no samples in database), missing analyses (incomplete,or missing assay results)
- QA/QC material: A QA/QC (Quality Assurance, Quality Control) report is generated in which results of the standards (CRMs), blanks and duplicates are reviewed, (includes Client QA/QC material and lab checks where applicable).

Due to the large dataset at Nkran, which is comprised of both resource definition drilling and grade control drilling, the datasets were provided in four parts, with each part assigned a "GROUP" code in Datamine™ to ease identification:

- Part one comprised Collar, Survey and Assay data for resource definition holes drilled by PMI (GROUP=1)
- Part two comprised Collar, Survey and Assay data for resource definition holes drilled by Resolute Mining (GROUP=2)
- Part three comprised Collar, Survey and Assay data for grade control holes drilled by Resolute Mining (GROUP=3)
- Part four comprised Collar, Survey and Assay data for grade control data drilled by Asanko (GROUP=4)

In total, there are 42,297 drill holes in the collar file and all of them had coordinate data. The number of holes in each "GROUP" are in Table 14-4 below.

Table 14-4 : Number of Borehole Collars in Nkran Database

Group	Number of Collars
1 (PMI Resource Development)	115
2 (Resolute Resource Development)	830
3 (Resolute Grade Control)	35,923
4 (Asanko Grade Control)	5,429

Within the raw assay files, there are 5,328 absent Au values, 29,976 negative Au values, 6 zero Au values coded 'NSR' and 29,160 zero Au values coded 'BDL'.The breakdown of number of samples by GROUP is given in Table 14-5.

Table 14-5 : Number of Assays in the Nkran Database

Group	Total Number of Assays	Number of Absent Au Data	Number of Negative Au Data	Number of Zero Au Data
1 (PMI)	23,260	775	4,272	453
2 (Resolute)	54,165	233	2,343	538
3 (Resolute GC)	311,913	3,597	0	28,175
4 (Asanko GC)	108,743	723	23,361	0

The absent values were left as absent during the data load. The negative values were reset to half the detection limit to a value of 0.005 g/t Au for GC data (GROUP=3 or GROUP=4), or 0.0025 g/t Au for resource definition data (GROUP=1 or GROUP=2). The reason for this is that the dominant detection limit in the resource definition assays is 0.005 g/t Au, whereas the dominant detection limit in the GC data (dominated by Resolute drilling), is 0.01 g/t Au.

The 6 zero Au values coded 'NSR' were reset to absent. The 29,160 zero Au samples coded 'BDL' were set to half detection limit, using the logic described above.

The CSA data load validations gave the following results:

- 1 collar in the GROUP=1 dataset with no assays
- 36 collars in the GROUP=2 dataset with no assays
- 70 collars in the GROUP=4 dataset with no assays

Following desurveying, sampling gaps (1,215) were identified in the drill holes. These sampling gaps totalled approximately 22,185m in length, the majority of which (21,913m, or 99%) are in the GROUP=4 dataset. These sampling gaps have not been filled for use in the current MRE, as it has been assumed that these gaps represent absent intervals rather than waste. The total length of these gaps approximates 12% of the total length of the GROUP=4 samples, and 3.5% of the length of the overall Nkran dataset

14.5.2 Rotation from UTM to Local Grid

Data was transformed from the UTM WGS 84 30N grid to a local grid, to orientate the mineralisation in a North-South orientation. This transformation ensured that the resource block model did not require rotating to appropriately honour the dominantly NE geology and mineralisation trend. The data rotation parameters are shown in Table 14-6. A plan view of drill hole samples with Au >0.3 g/t and geology wireframes in the local grid is presented in Figure 14-1.

Table 14-6 : Nkran Data – Rotation Parameters

Grid	Rotation	X (Easting)	Y (Northing)	Z (Elevation)
Original UTM centroid	-37˚around Z	611,700	700,500	0
Local Grid centroid		5,000	20,000	1,000

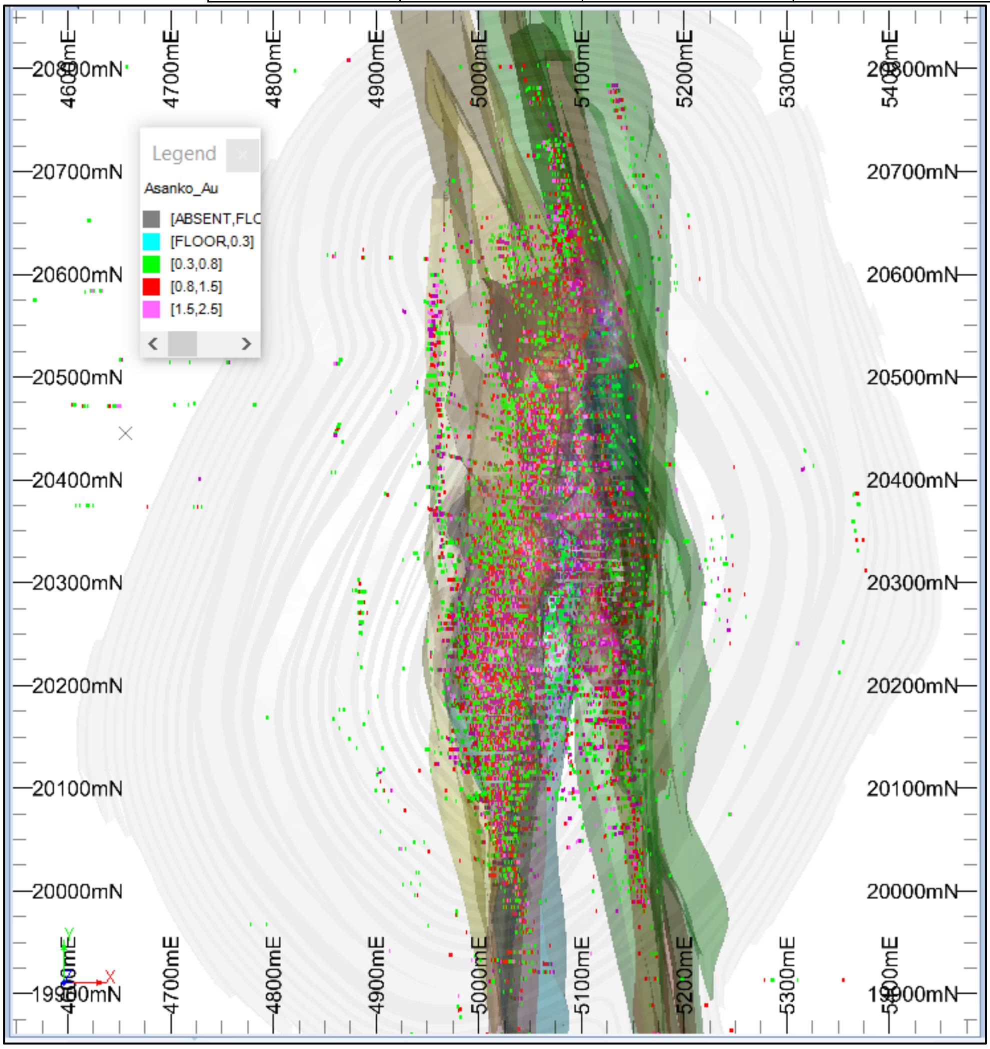


Figure 14-1: Plan view – Nkran drill hole samples >0.3 g/t Au and geology wireframes plotted on the Local Grid (Source : CSA Global 2017)

14.5.3 Geological Interpretation

14.5.3.1 Lithology

The Nkran project is located along a major shear zone cross cut by secondary fault structures. Deposit scale mineralisation is directly controlled by the structural setting and specific lithological units. The 3D lithological and structural models provide the geological boundaries for the MRE.

During 2014, a geology and structure model was developed for Nkran based on information from a re-logging programme of 35 PMI drill holes. The re-logging captured lithology, alteration, structure, mineralisation and veining. The re-logging identified five key structures which separate the lithologies and control distributionof mineralisation within the orebody. The shears / structures have been modelled as planes when in reality they are zones of intense deformation up to several metres in width. At this time the geology of the Nkran deposit consisted of south plunging sandstone dominated western unit, a central granitic intrusion and and a eastern breccia unit with five key structures controlling mineralisation.

This work has been superceded by a 2016 programme by Asanko Gold which included the relogging of 72 diamond boreholes that inform the MRE. In this interpretation a vein and alteration paragenesis was completed by Brett Davis of Orefind Pty Ltd. The ensuing interpretation provided a better definition of the various lithological units, structural controls and continuity of mineralisation, alteration and understanding of mineralising events and deformation history.

The 2016 geology modelling exercise identified the host lithologies several sandstone units (2000, 2100, 2200), two granite units (DGR01, DGR02) and northern and southern easter breccia units (5000, 5100). Unmineralised country rock of intercalated siltstones and shales and an eastern sandstone unit (2350) round out the geology being mined in the pit and these are shown below in Figure 14-2. These units are separated by an anastomosing network of controlling structures which act as the conduits for the mineralising fluids. The modelled wireframes provide the basis of the mineralisation domains.

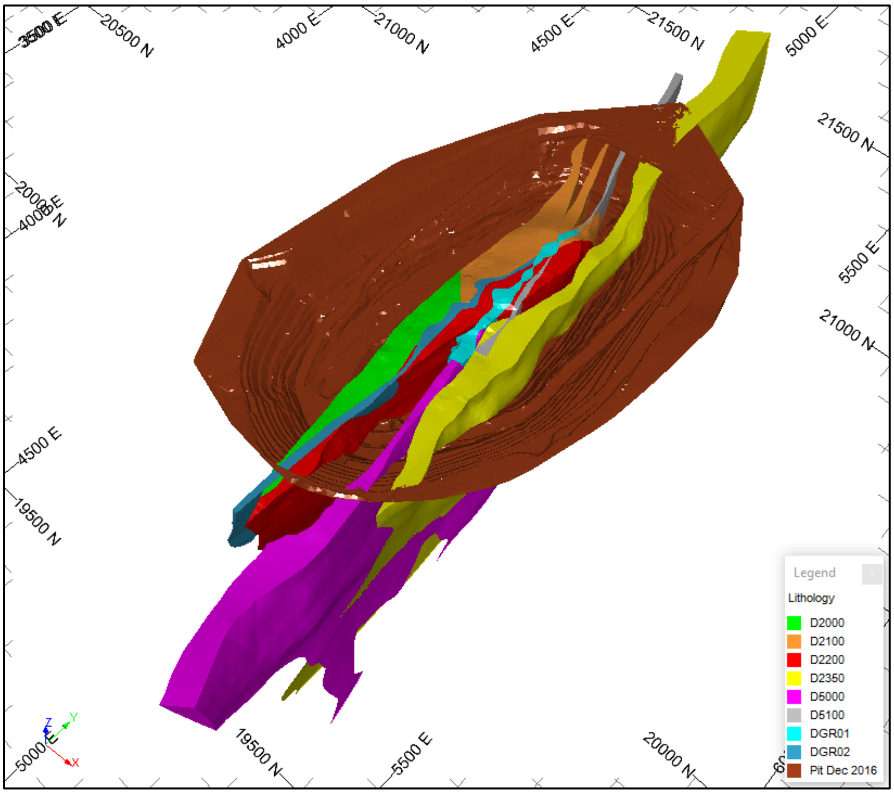


Figure 14-2: 3D view of the Nkran geological domains within the December 2016 Pit Shell (local grid) Source : CSA Global 2017

The host lithologies are bounded to the west and east by the Western Bounding Shear Zone ("WBSZ") and Eastern Bounding Shear Zone ("EBSZ"). The WBSZ and EBSZ are characterized by wide (50m +) zones of strongly sheared phyllite, carbonaceous shale and siltstone. All the mineralisation occurs between these broad, bounding zones in favourable host lithologies, which in some instances are bounded by internal shear zones.

The sandstone, granite, and breccia units are the preferrencial host of the mineralisation because of rheological contrast with the siltstone and shale country rock around them. These units are harder and more massive making them prone to deforming in a brittle fashion during the late stage mineralising events. This brittle deformation causes dilation which is filled with quartz veins and gold mineralisation. The siltstone and shale intercalated units are finer grained and contain many slip planes. During the mineralising deformation events they tend to slip on these planes without brittle deformation or dilation occurring.

The gold mineralisation occurs within quartz-carbonate veins within these preferential host units. The anastomosing network of shear zones separating the lithological units acts as the conduit for mineralised fluids which is emplaced in the host units during the late mineralising brittle deformation events.

The level of geological understanding has increased with the re-opening of the Nkran pit, grade control drilling and inpit mapping. As a consequence, Asanko has continued to progress the accuracy and underseranding of the geological and structural controls on mineralisation. (Asanko Gold in-house information).

14.5.3.2 Weathering

Weathering profiles for the bottom of complete oxidation and the top of fresh material have been modelled by Asanko geologists. Drill hole data was flagged by these weathering wireframes and coded based on the oxidation state as described in Table 14-7.

Table 14-7 : Weathering Codes

Rock	Description	Comment
1	Oxide	Material between topography and bottom of complete oxidation.
2	Transitional	Material between bottom of complete oxidation and top of fresh.
3	Fresh	Material below top of fresh

In general, the weathering surfaces are broadly parallel to the topographical profile, although weathering tends to be deeper within zones of mineralisation and tend to parallel the footwall to the mineralisation where the footwall approaches the surface (CJM, 2014; See Figure 14-3).

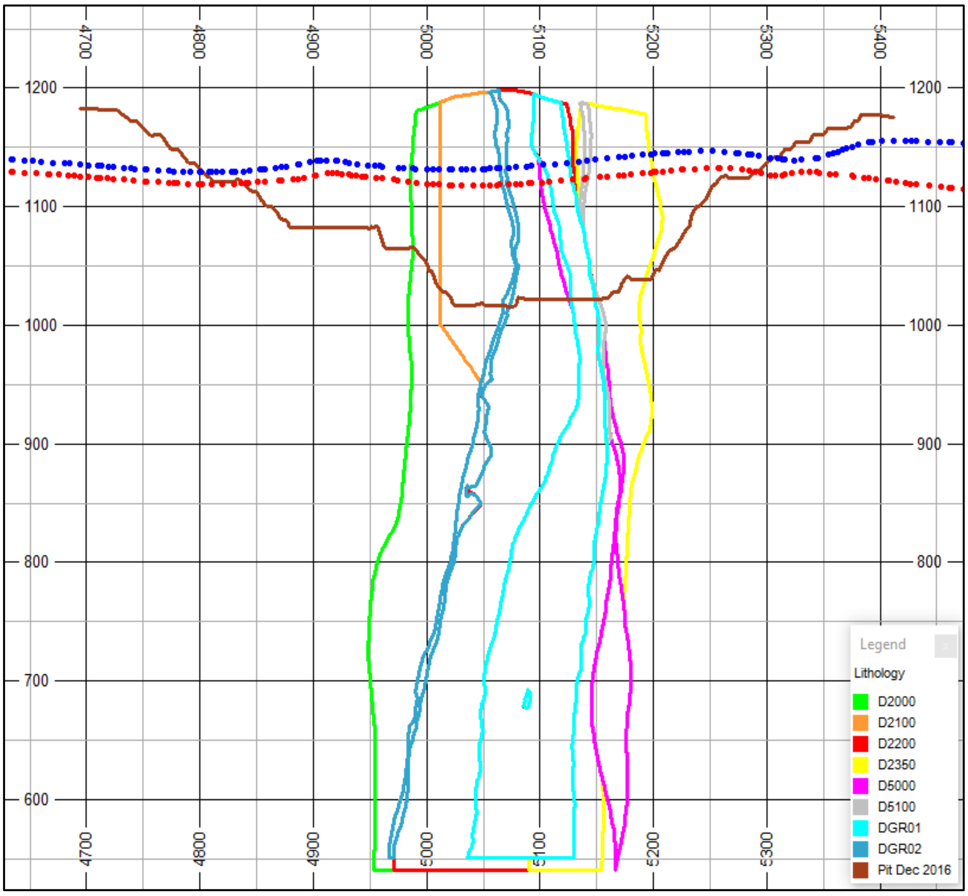


Figure 14-3: Section view of the Nkran weathering profiles and geological domains (local grid). Note : Bottom of complete oxidation (dotted dark blue line); Top of fresh (dotted red). Source : CSA Global Feb 2017

14.5.4 Mineralisation

The following section was modified after CJM (2014). CSA followed the same methodology to define mineralisation domains for the grade estimation of material below the area supported by Grade Control Data, and consider this a valid approach.

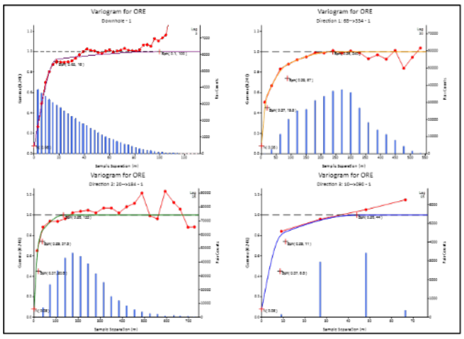
The main lithological units, within specific fault blocks, form the basis for delineating geological domains. Within the domains the mineralised and waste volumes have been defined using an IK method.

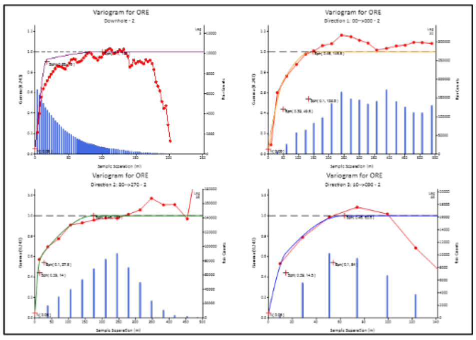
A grade compositing process in Datamine called CompSE was used to generate 'mineable' intercepts – that is, a set of intercepts that meets set minimum length, grade and dilution criteria. The minimum grade used to delineate mineralisation from waste was 0.3 g/t Au. The minimum true width used was 5m, in line with the SMU size of 5 x 5 x 5m (X x Y x Z) modelled. Intercepts that met the CompSE criteria were assigned a value of 1 and intercepts that did not, were assigned a value of zero. The 1 and 0 values are then estimated into a block model (5 x 5 x 5m) using mineralised orientations and relationships observed and modelled from data. Only resource development data was used to define the mineralisation volumes.

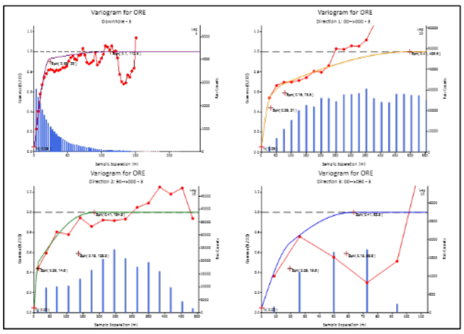
The indicator estimates produce a value between 0 and 1 which is then used as a probability for establishing if a cell is mineralised, or not. Specific probabilities are selected for each domain that represent expected mineralisation volumes. The probability threshold chosen for the main (steep) mineralisation was 0.30, which was chosen as being most representative of the interpreted continuity of the mineralisation.

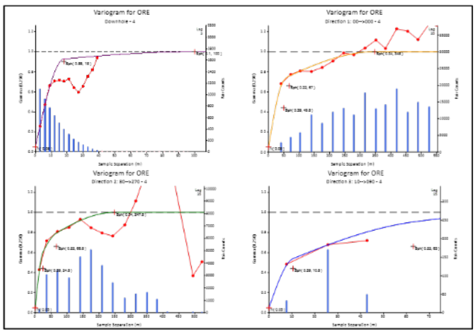
A secondary IK run was completed to estimate shallower flat-lying structures that have been identified during mining. Shallower structures do not extend into the Low Grade / Waste DW domain. The probability threshold used was 0.80. The models were combined with the steep structures taking priority over the shallower. Additional drilling / pit mapping is recommended to delineate the shallower structures further.

Data was flagged using the geological wireframes, and a process was run whereby samples that fell within a block that exceeded the chosen probability threshold was used to estimate grade. A comparison was completed between the composites back-flagged by blocks, and those that met the criteria set out in the CompSE process to assess how much internal dilution was included and how much mineralisation was excluded.









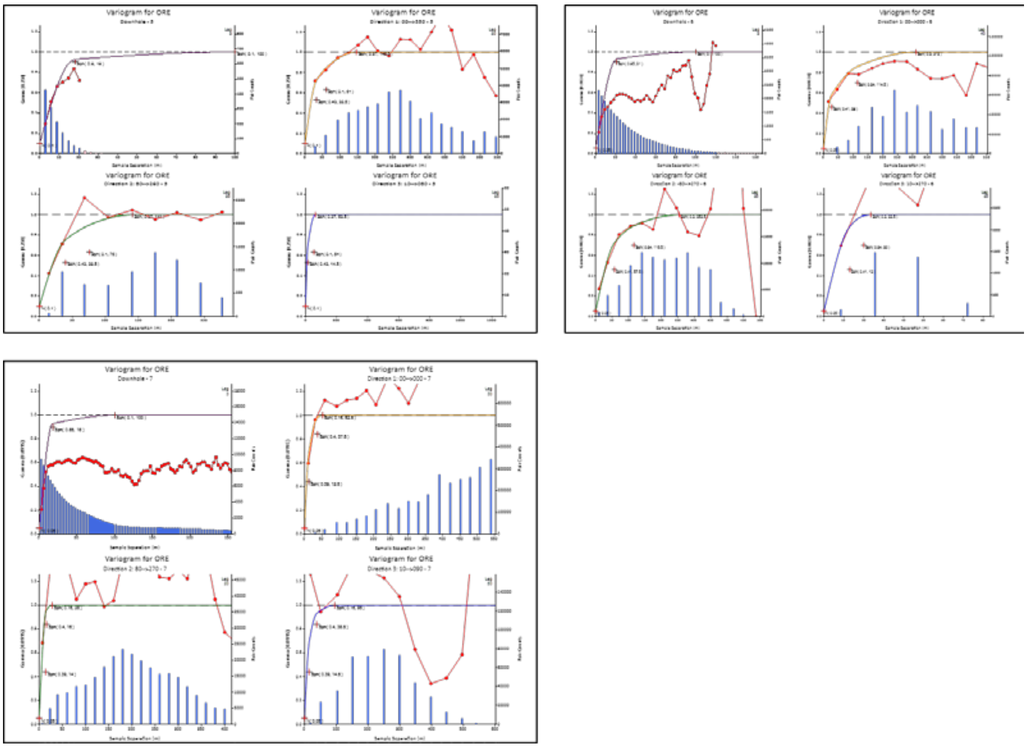


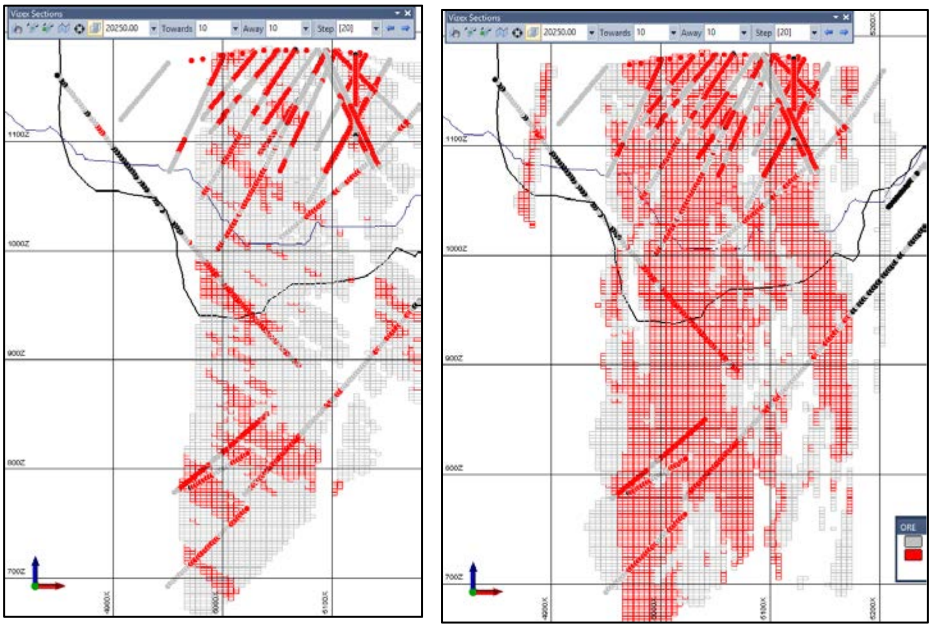
Figure 14-4: Experimental and variogram models for each ESTZON for the indicator variable. (Source CSA Global, 2017)

Table 14-8: Variograms Parameters for Indicator Variable (Source : CSA Global 2017)

ESTZON	Isatis Orientation (ZYX)	Nugget	Structure 1		Structure 2		Structure 3	
			Partial Sill	Range	Partial Sill	Range	Partial Sill	Range
1	80	0.08	0.37	19.5	0.29	87	0.26	240
	0			20.5		37.5		133
	80			8.5		11		44
2		0.05	0.39	49.5	0.1	132.5	0.46	146.5
				14		27.5		175
				14.5		54		63.5
3		0.05	0.39	31	0.15	78.5	0.41	498.5
				14		135.5		184.5
				19.5		58.5		63.5
4		0.05	0.39	49.5	0.22	67	0.34	348
				24.5		66.5		175
				10.5		55		63.5
5		0.1	0.43	32.5	0.1	61	0.37	146.5
				39.5		76		141
				14.5		54		63.5
6		0.05	0.39	49.5	0.1	132.2	0.46	312
				38		197.5		253.5
				8		13.5		23.5
7		0.05	0.39	12.5	0.4	29	0.16	74.5
				14		27.5		175
				14.5		54		63.5

Table 14-9: Search Neighbourhood Parameters for the Indicator Variable per estimation Domain. (Source : CSA Global 2017)

ESTZON	Search Ranges			Composites		
	1	2	3	Min	Max	Max per Drill Hole
1	120	66	22	6	9	3
	73	87	30			
3	249	92	30			
4	174	87	30			
5	73	70	30			
6	156	126	11			
7	37	87	30			



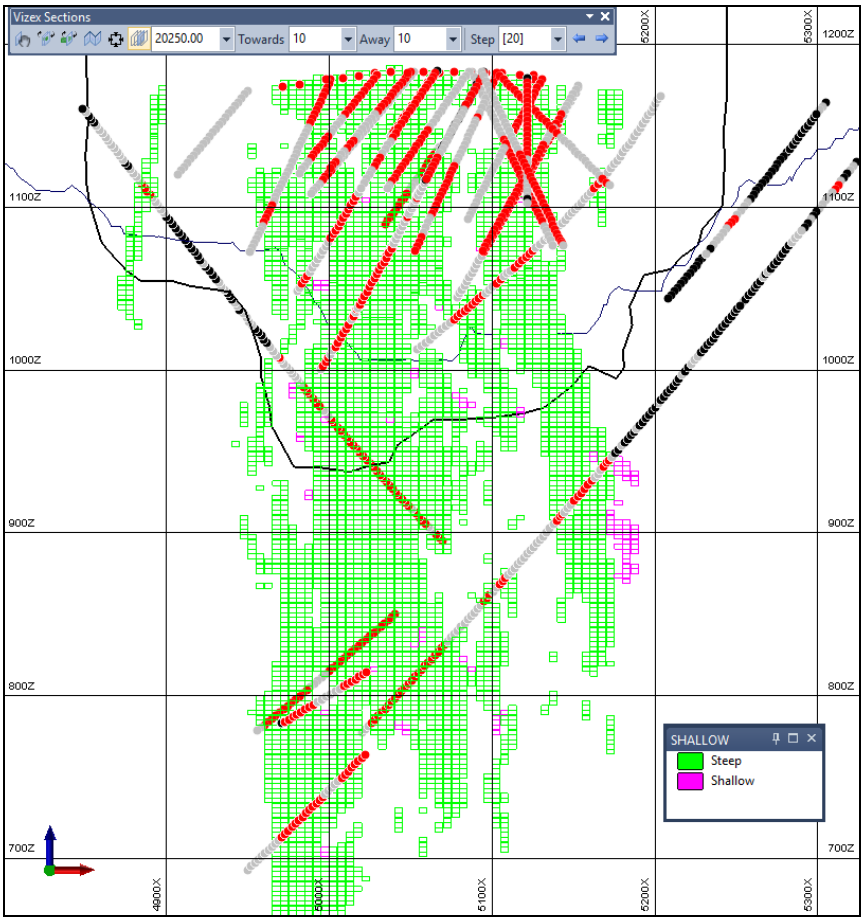


Figure 14-5: Cross sections showing mineralisation volume for steep (top left), shallow (top right) and combined (bottom). (Source : CSA Global 2017)

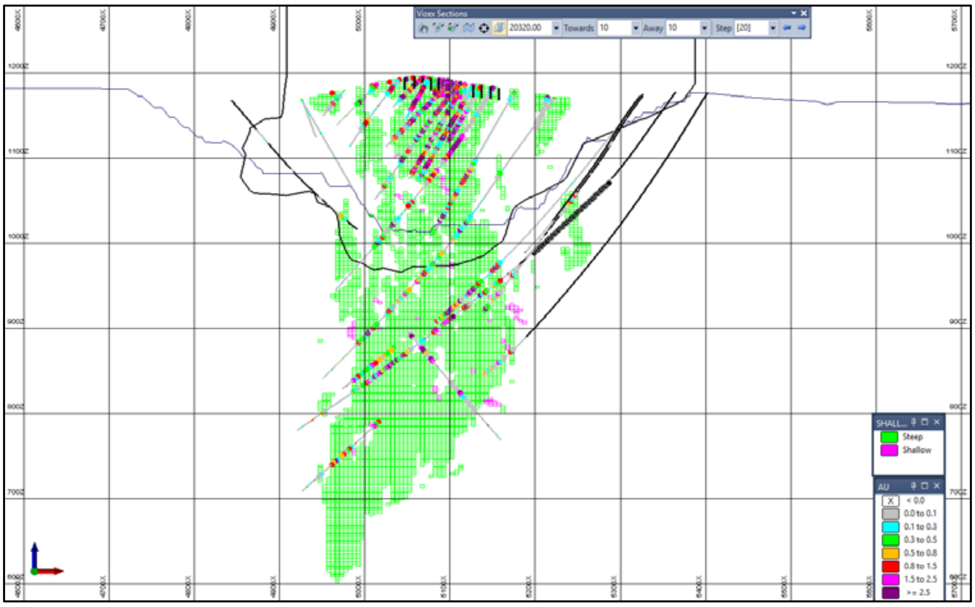


Figure 14-6: Cross section showing volume model and backflagged exploration data. (Source : CSA Global 2017)

14.5.5 Sample Domaining

Before undertaking the estimate, the data was first analysed. Drill hole samples were statistically reviewed, and variograms were calculated to determine spatial continuity for Au.

The statistical analysis was carried out by CSA Global using Datamine StudioRMTM, Supervisor v8.4TM and GeoAccess ProfessionalTM software packages.

14.5.6 Domain Coding

Drill hole coding is a standard procedure which ensures that the correct samples are used in classical statistical and geostatistical analyses, and grade interpolation. For this purpose, the litho-structural volumes, within which the mineralised portions were delineated using IK, have been defined into eight geological domains, called MINZON. An additional domain for all material outside these defined domains was created.

Statistical analysis including comparisons of grade distributions and contact analysis resulted in several domains being combined. The domains as used in the resource estimation (ESTZON) and how they relate to MINZON are summarised in Table 14-10.

Table 14-10: Domain Codes used in the January 2017 CSA Global Resource Estimation. (Source CSA Global 2017)

Wireframe	Domain (MINZON)	Estimation Domain (ESTZON)	Description
D2000	2000	1	Broad Sandstone - Mineralised
D2100	2100	2	Sandstone - Mineralised
D2200	2200	2	Sandstone – Mineralised
D2350	2350	6	Interbedded silts and shale – Not Mineralised
D5000	5000	3	Sandstone East Lode – Mineralised
D5100	5100	4	Sandstone East Lode North - Mineralised
DGR01	3001	5	Granite - Mineralised
DGR02	3002	5	Granite - Mineralised
DW	9000	7	Sandstone - Waste
-	9999	Not Estimated	Outside wireframe

14.5.6.1 Domain 2000/2100/2200/2350 (Sandstone / Siltstone)

Domain 2000 (and 2100, 2200 and 2350) represent the alteration sericite-albite within the broad sandstone units. Mineralisation is not continuous within the domains, two vein orientations have been identified, relating to the two phases of mineralisation; steep W-dipping, NNE-striking ductile quartz veining that is parallel to bedding and in places foliation, shallow NE-dipping, NW-striking brittle veins that cross cut foliation and bedding. The sandstone unit has been strongly folded and forms the shallow south plunging synformal keel of a steep westerly dipping fold hinge, which has been offset by dextral D2 faulting and shearing (CJM, 2014).

14.5.6.2 Domain 5000 (East Lode) / 5100 (East Lode North)

Domain 5000 / 5100 is controlled by the Discovery Structure. The domain is characterised by a steep W-dipping, N-striking ductile fabric, which is in places overprinted by a barren quartz breccia, which contains clasts of previously mineralised material. Grade and continuity along strike are variable. Due to a break in the mineralisation along the northing, the domain has been split into the East Lode (5000), and East Lode North (5100) (CJM, 2014).

14.5.6.3 Granite Domains GR01 and GR02

Two granitic bodies have been identified within the pit. These shapes have been used to define the mineralisation domains GR01 and GR02. Mineralisation hosted within the granite domains typical occurs in thin, 2 cm to 30 cm thick discontinuous near horizontal veins arrays. Mineralisation is sporadic and appears not to be associated with vein density, thickness, or degree of alteration. This makes it is impossible to model up individual lodes. The granite bodies are bound by major structures, e.g. Defender / Discovery and County / Discovery. On these margins, higher grades do occur, but are difficult, with the current level of drilling to define distinct lodes or domains. Historically mineralisation was hard to mine in fresh, due to the short-range nature of the continuity of the veins. Interaction of the granite nose with the intersection of the Discovery / County structures hosts reasonable grade and mineralisation, although multiple vein arrays in numerous orientations make it difficult to model (CJM, 2014).

14.5.7 Naïve Statistics

The MRE was constructed in a two-step approach. The first step was the estimation of a GC model, using grade control and exploration data in an area defined by a wireframe below the December 2016 pit surface, Figure 14-7. The second step involved an IK model, using only exploration data. The final model comprised of the combination of the GC and IK models, where the GC model supersedes IK model blocks in the GC area.

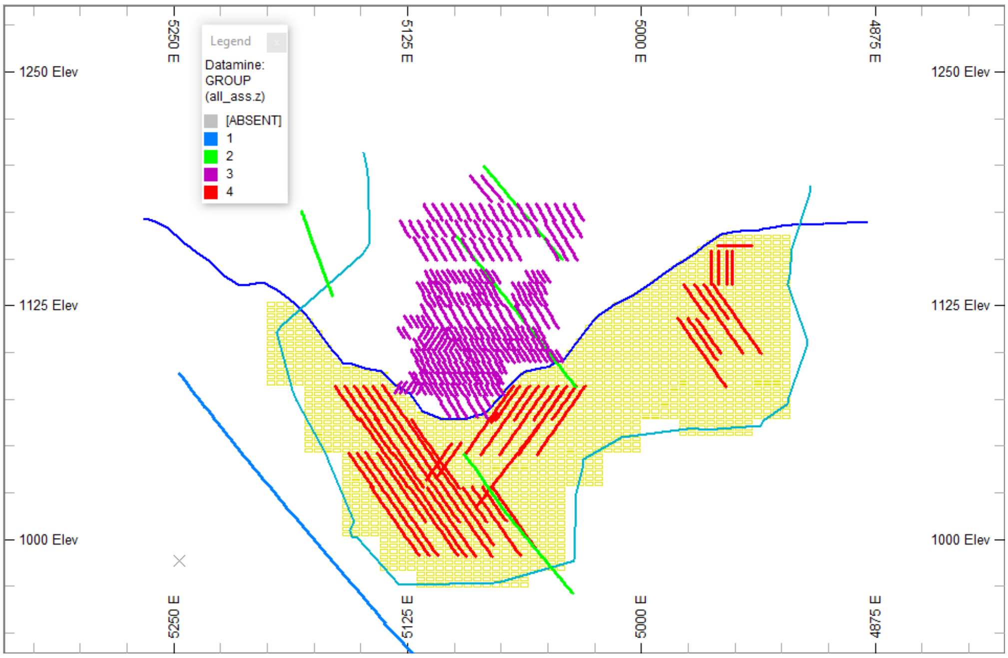


Figure 14-7: Cross Section view 20535 mN (local grid), of the GC area. (Source : CSA Global 2017)

Drill holes coloured by 'GROUP', GC block model coloured in gold, upper bounding surface shown in dark blue, lower bounding 'GCAREA' surface shown in light blue.

14.5.8 Summary of Drilling GROUP codes

1 (PMI Resource Development)

2 (Resolute Resource Development)

3 (Resolute Grade Control)

4 (Asanko Grade Control)

The lower surface was designed based on the approximate base of the GC data, and was digitised in section view, prior to wireframing. The upper bounding surface was a topography wireframe supplied by Asanko (dtopo), which is a topographic surface of the area as at end 2002, when mining by Resolute finished at Nkran.

CSA Global used only GROUP=3 data within 30m of the topography surface (which approximates the extent of the search neighbourhood), to inform the GC estimate.

The second step involved an IK model, using all GC and exploration data. The final model comprised of the combination of the GC and IK models, where the GC model overwrites IK model blocks above the "base of GC" wireframe.

14.5.8.1 GC Model

The GC model had a single domain for estimation. The naïve statistics for this domain is given in Table 14-11 and shown in Figure 14-8.

Table 14-11: Naïve Statistics – GC Model (length weighted). (Source : CSA Global 2017)

Variable	Value
Number	276,905
Minimum	0.0025
Maximum	1,052
Mean	0.81
Std Dev	4.84
CoV	5.99

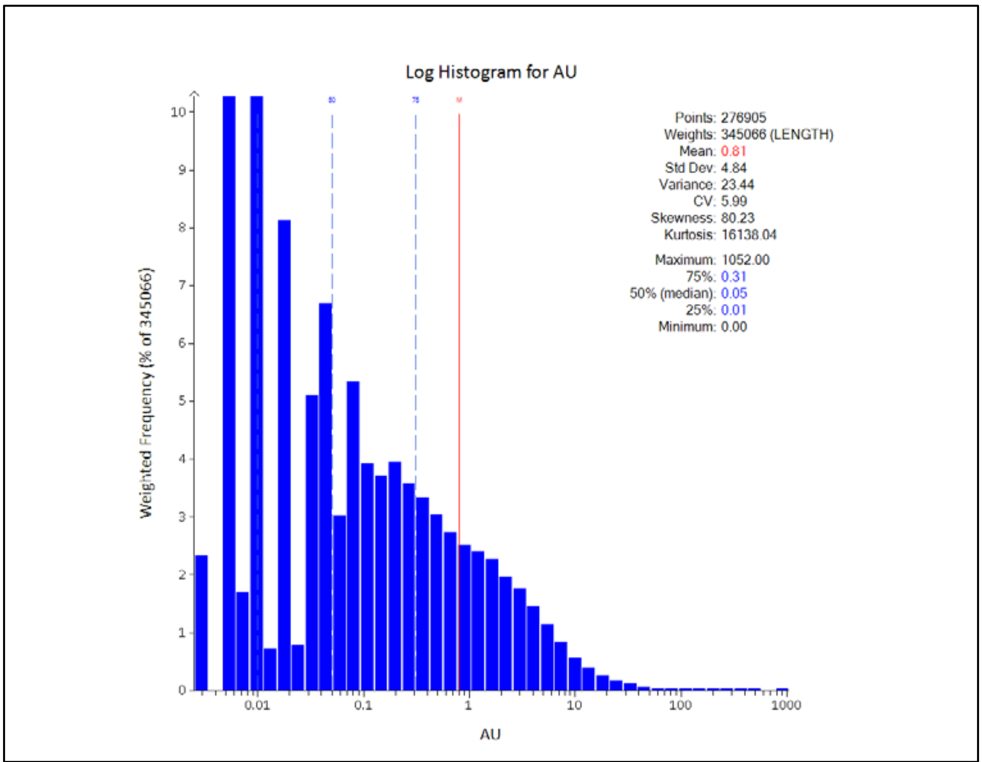


Figure 14-8: Log Histogram of GC model estimation domain naïve statistics. (Source : CSA Global 2017)

14.5.8.2 *IK Model*

The naïve statistics for the estimation domains for the IK grade estimate are given in Table 14-12.

Table 14-12: Naïve Statistics – IK Model. (Source : CSA Global 2017

DOMAIN	1	2	3	4	5	6	7
Number	4,136	2,965	1,916	722	171	314	1,007
Minimum	0.003	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	364.00	390.00	131.00	153.00	29.50	37.80	192.80
Mean	2.38	1.88	1.19	1.49	1.76	0.66	1.01
Median	0.55	0.31	0.25	0.28	0.41	0.06	0.07
Std Dev	9.18	11.44	5.14	6.59	3.98	2.57	6.88
CV	3.86	6.09	4.33	4.42	2.27	3.92	6.79

14.5.9 Sample Compositing

Assays that fall within the modelled mineralisation envelopes were selected from the database and were down-hole composited to 3m prior to statistical review, top-cutting, variography and grade estimation. 3m was chosen given approximately 20% of the exploration samples within the geological domains were sampled at lengths > 1m (Figure 14-9).

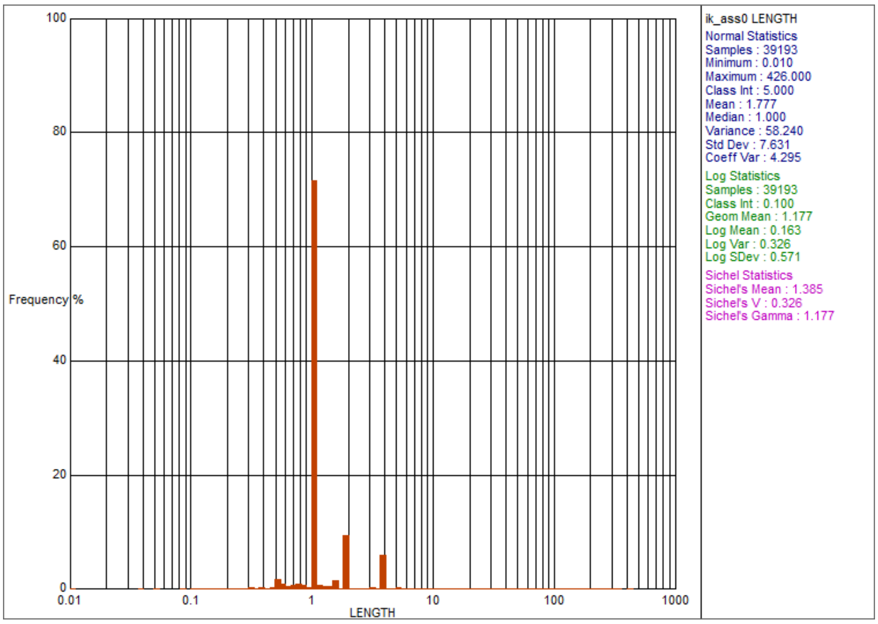


Figure 14-9: Log Histogram overlay of sample lengths for samples within geological domains (exploration drilling only). (Source : CSA Global 2017)

For the GC model, 3m composites were also used. Although the dominant GC sampling interval is 1m, it was felt that due to the large amount of 1.5m samples present, a 1m composite length would result in a large amount of sample splitting. A statistical review demonstrated that 3m was the best compromise for the dominant sampling intervals of 1m and 1.5m. A composite length of 3m reduces sample splitting and is close to the mining bench height. Refer Figure 14-10

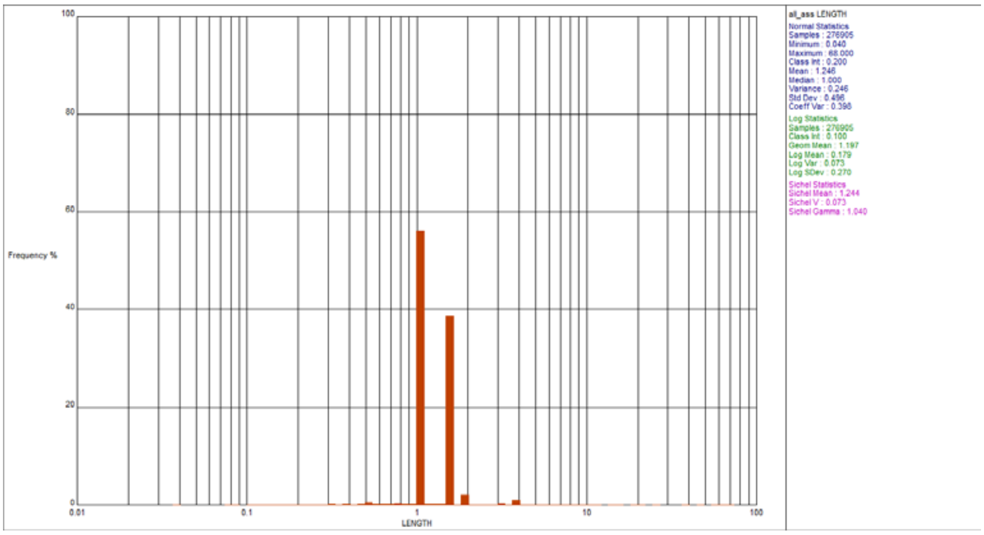


Figure 14-10: Log Histogram overlay of sample lengths for samples within geological domains, GC and exploration drilling combined. (Source CSA Global 2017)

Composites that were less than 50% of the composite length (<1.5m) were excluded from the geostatistical analysis and grade estimation to limit any potential bias in the sample support during kriging.

14.5.10 Statistical Analyses

14.5.10.1 Summary Statistics – Composites

14.5.10.1.1 GC Model

The GC model estimation domain composite statistics are given in Table 14-13 and shown in Figure 14-11.

Table 14-13: Composite Statistics GC Model. (Source CSA Global 2017)

Parameter	Value
Number	116,041
Minimum	0.0025
Maximum	541.96
Mean	0.80
Std Dev	3.37
CV	4.19

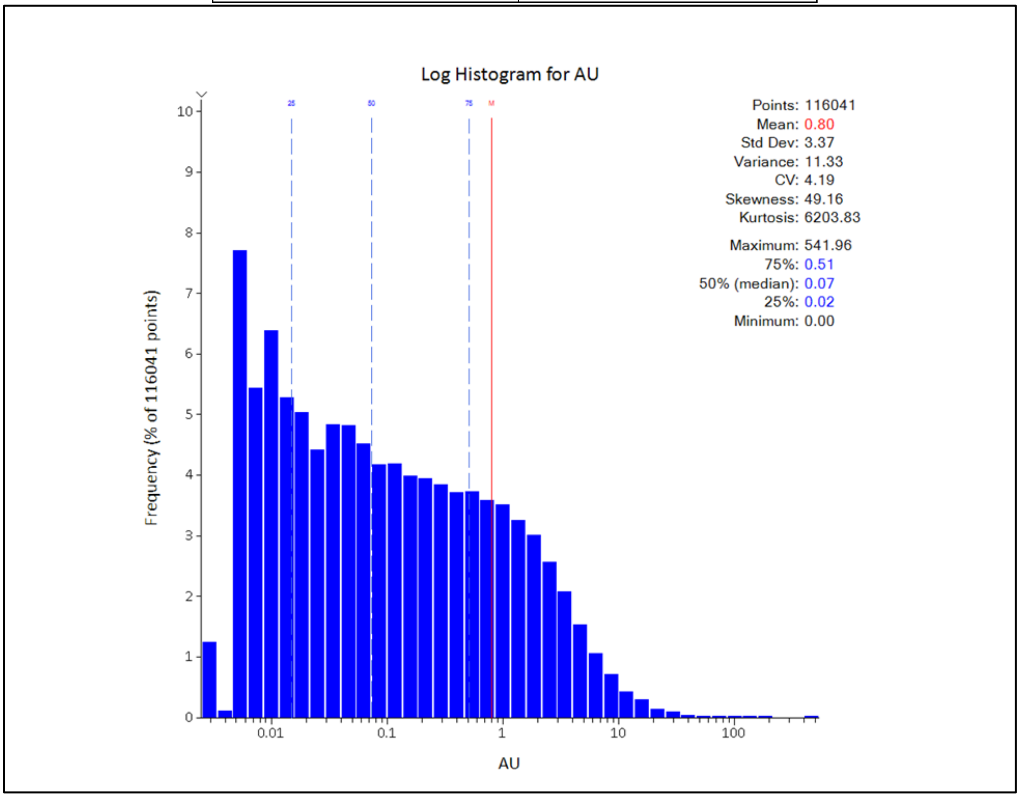


Figure 14-11: Log Histogram of GC model estimation domain composite statistics. (Source : CSA Global 2017)

14.5.10.1.2 IK Model

The IK model estimation domain composite statistics are given in Table 14-14 and Figure 14-12.

Table 14-14: Composite Statistics IK Model. (Source : CSA Global 2017)

ESTZON	1	2	3	4	5	6	7
Number	2,758	3,270	2,184	746	310	317	1,279
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	70.00	35.00	54.26	71.17	32.75	11.93	40.00
Mean	2.07	1.62	1.67	2.29	1.80	0.56	1.25
Std Dev	4.35	3.16	3.44	4.96	3.00	1.14	3.31
CV	2.10	1.95	2.06	2.17	1.66	2.04	2.65

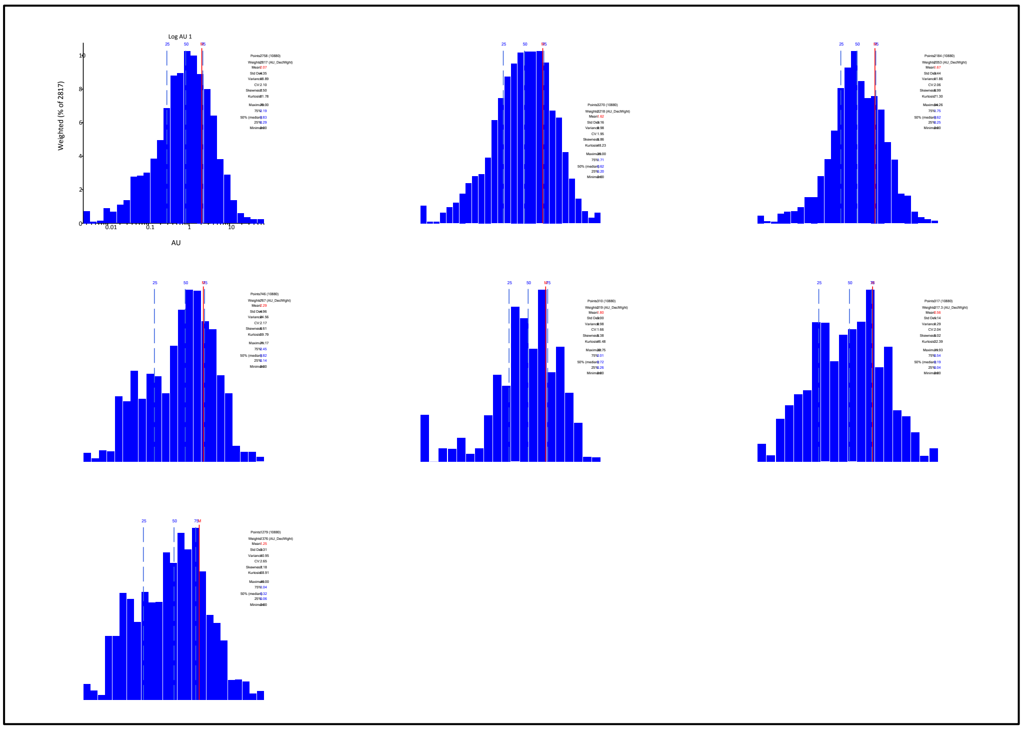


Figure 14-12: Log Histograms of IK model estimation domains composite statistics. (Source : CSA Global 2017)

14.5.11 Grade Cutting

Grade cutting (top cutting) is generally applied to data used for grade estimation to reduce the local high grading effect of anomalous high grade samples in the grade estimate. In cases where individual samples would unduly influence the values of surrounding model cells, without the support of other high-grade samples, top cuts are applied. These top-cut are quantified per the statistical distribution of the sample population.

Cutting strategy was applied based on the following:

- Skewness of the data
- Probability plots
- Spatial position of extreme grades
- Histograms and probability plots were reviewed for Au within the estimation domains for both the GC and IK models to determine the top-cuts

14.5.12 GC Model

The uncut and top-cut statistics are shown in Table 14-15. A total of fourteen samples were greater than the top-cut value of 90 g/t and were reset to the top-cut value.

Table 14-15: Top-cut Statistics GC Model. (Source : CSA Global 2017)

	Uncut	Top-Cut
Number	116,041	116,041
Minimum	0.0025	0.0025
Maximum	541.96	90
Mean	0.80	0.79
Std Dev	3.37	2.77
CV	4.19	3.48

The associated log histogram plots for the uncut and top-cut Au in the GC model estimation domain are shown in Figure 14-13.

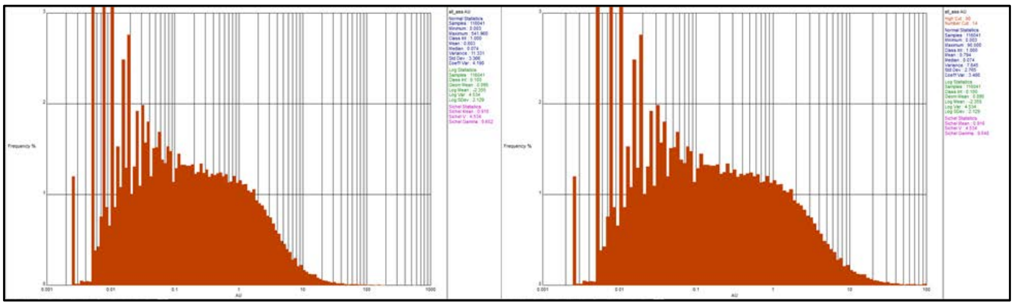


Figure 14-13: Log Histogram Uncut (left) and Top-Cut (right) – GC Model. (Source : CSA Global 2017)

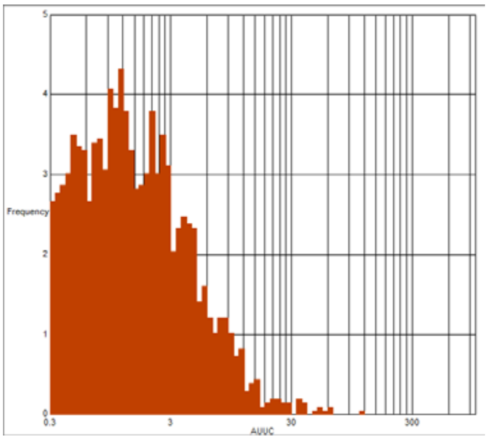
14.5.13 IK Model

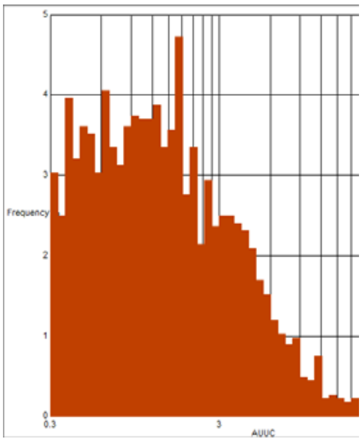
Top cuts used for estimation domains for the IK model are presented in Table 14-16. Top cuts were applied within the neighbourhood within 3m, the composite remains uncut. The top cut is applied for samples beyond 3m of the block centroid.

Table 14-16: Top-cut Statistics IK Model. (Source : CSA Global 2017)

ESTZON	Top-Cut
1	70
2	80
3	70
4	50
5	37
6	-
7	40

The associated log histogram plots for uncut Au in the estimation domain are shown in Figure 14-14, focussing on grades greater than 0.3 g/t Au to show the top end of the distribution.





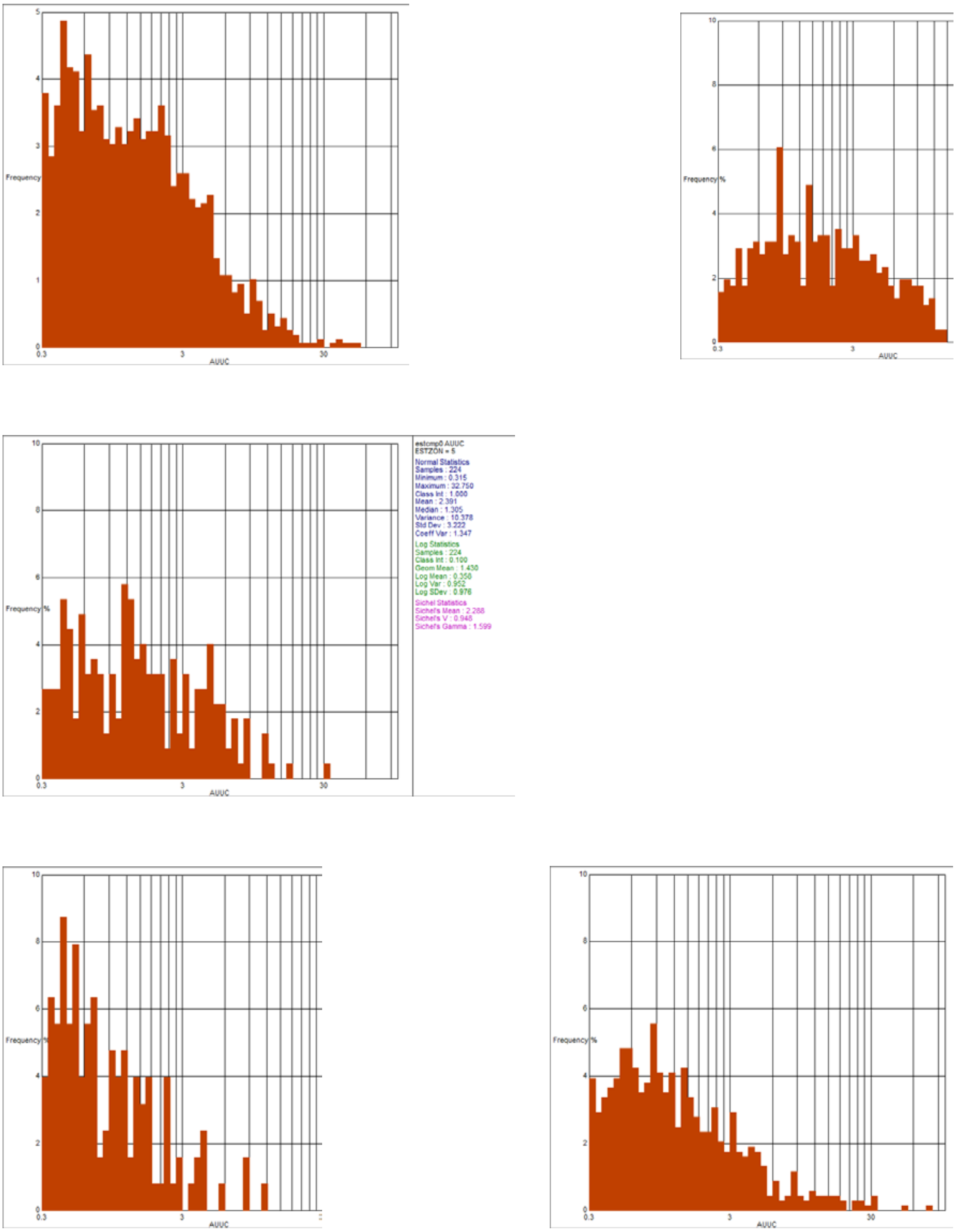


Figure 14-14: Log Histograms Uncut per estimation domain, showing distribution above 0.3 g/t for clarity –IK model. (Source : CSA Global 2017)

14.5.14 Density

Density data provided by Asanko was reviewed by CSA. The density dataset consisted of point density data, measured from in-pit grab samples, and billets of drill core. In-situ dry bulk density was estimated using the water immersion method. Drill core billets were de-surveyed with collar and survey data, to locate them in 3D space. In-pit grab samples were generally located by GPS in most cases, allowing them to be modelled in 3D space.

Density samples were flagged by the weathering surfaces, prior to analysis. Due to historic mining well below the depth of weathering and oxidation, most samples resided in the fresh domain.There were insufficient samples in the oxide and transitional domains to estimate a value with any confidence. Median and mean densities for fresh sandstone, the dominant ore host rock, were in the region of 2.67 to 2.69, with low variance (Figure 14-15). An in-situ dry bulk density of 2.68 t/m^3 was selected for all fresh material. Densities for transitional material were set at ~75% of the fresh material, 2.00 t/m^3, and oxide material set at ~65% of the fresh material, 1.72 t/m^3. It is important to note that only 1% of the remaining resource is oxide and transitional, so the accuracy of these in-situ dry bulk densities is not significant.

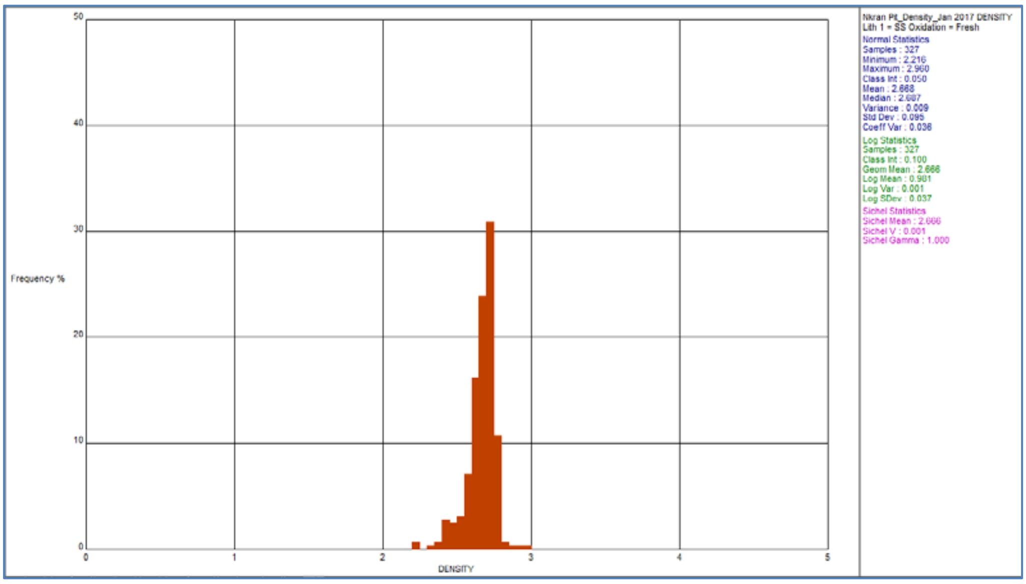


Figure 14-15: Normal histogram of in-pit density data for fresh sandstone. (Source : CSA Global 2017)

14.5.15 Variography

Variograms were modelled for Au on 3.0m declustered composites, with outliers excluded where considered necessary. Nuggets were obtained from the downhole variograms, where the lag was set equal to the composite length of 3.0m. Normal scores transform was used for modelling the variograms prior to back-transforming for the grade estimation.

14.5.16 GC Model

The GC grade estimate utilised soft boundaries between the geological domains, as the domain boundaries were produced from wide spaced resource development drilling and were not accurate as a GC level. Additionally, the tight spacing of the GC drilling – nominally 5m x 10m is the best indicator of grade domain boundaries. Notwithstanding the soft boundary assumption, it was felt that the geology envelopes provided adequate domaining of the GC data to define variograms for each geological envelope. The GC model semi-variograms were well structured, with low to moderate nuggets (6% to 36%) and moderate ranges. The variograms were back transformed prior to grade estimation and are presented in Figure 14-16. The variogram parameters are presented in Table 14-17.

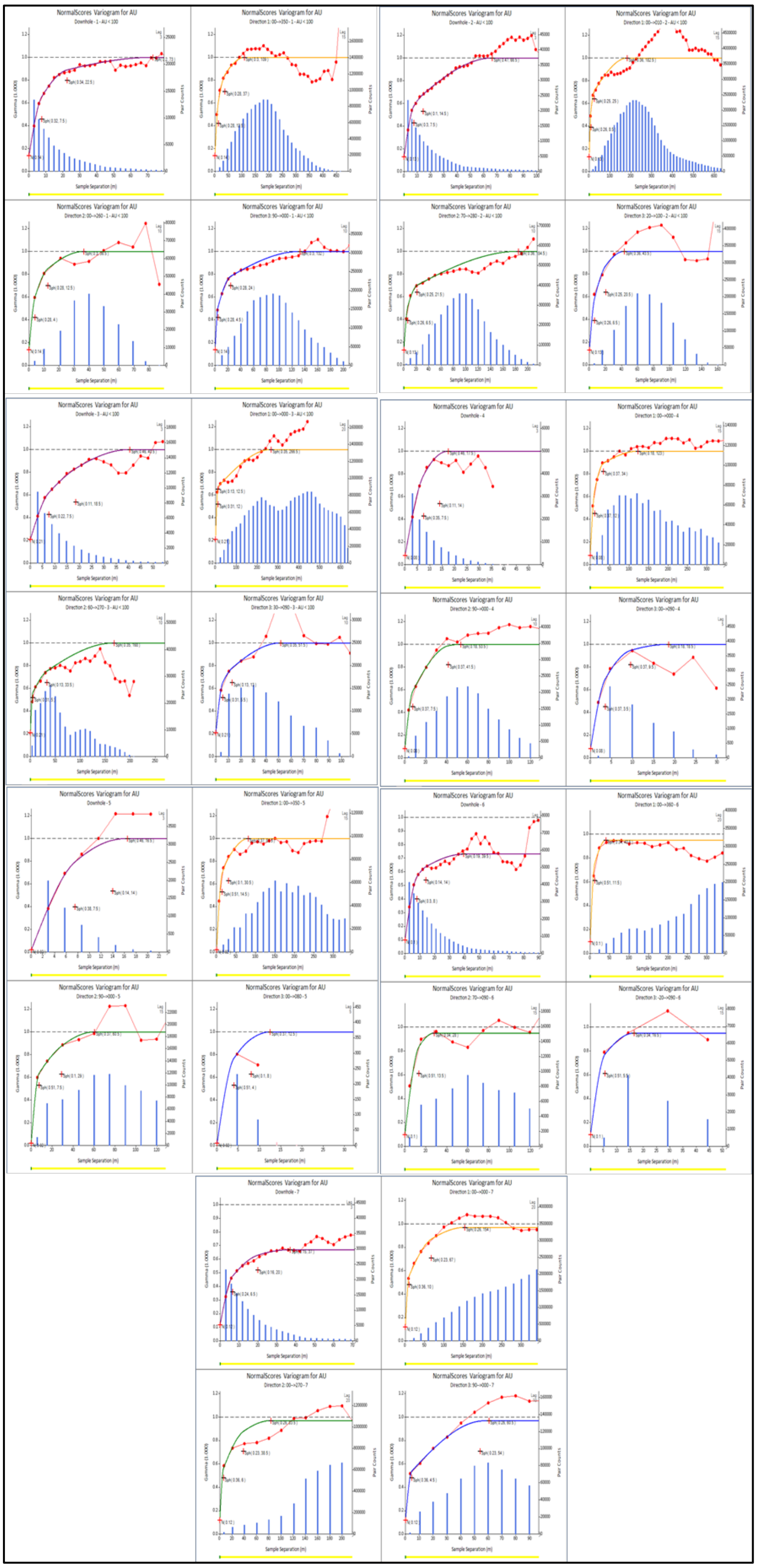


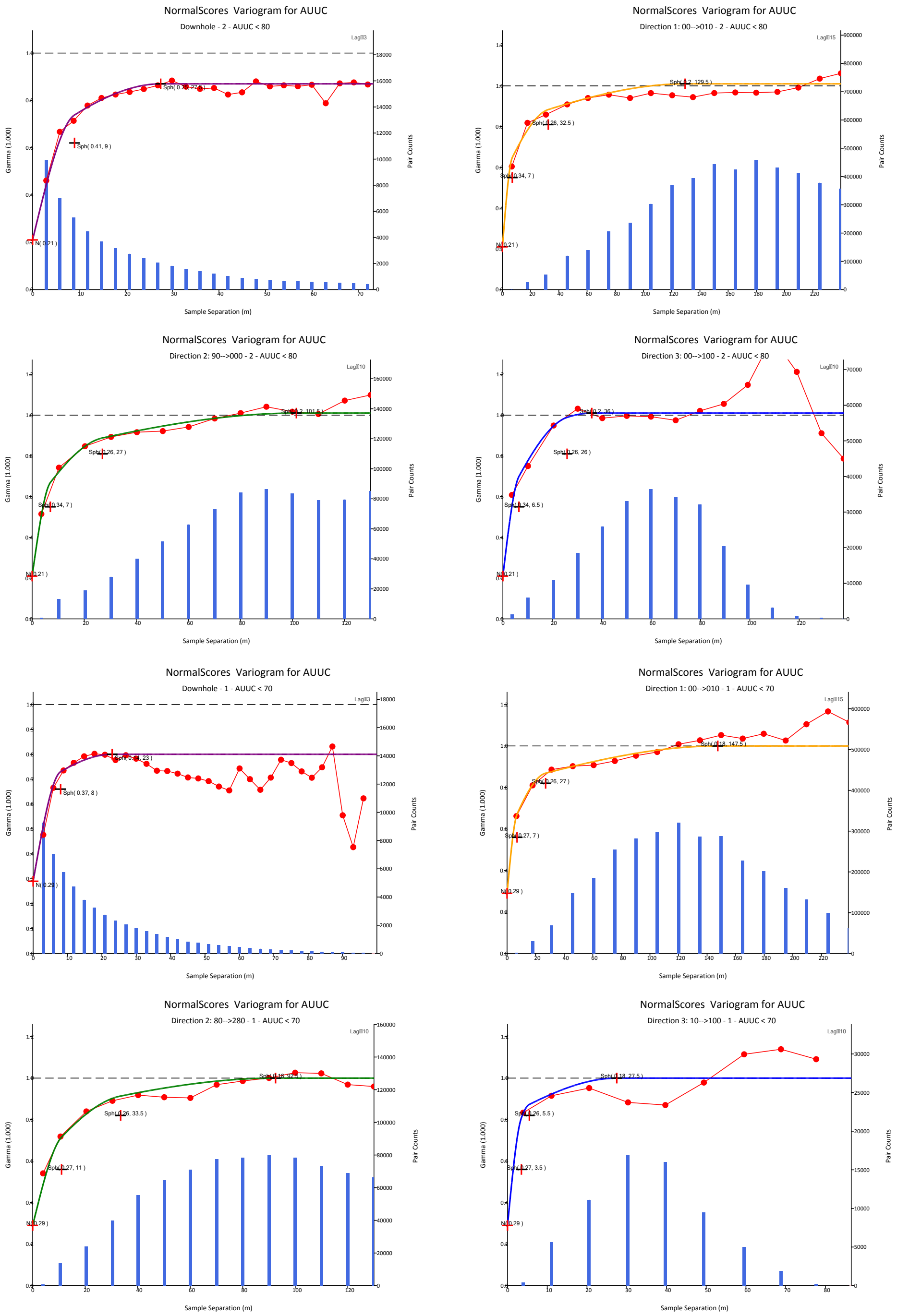
Figure 14-16: Variograms used for Au g/t estimation – GC model. (Source : CSA Global 2017)

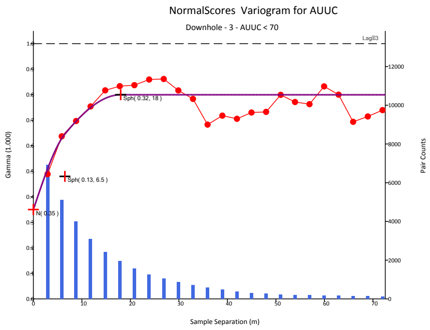
Table 14-17: Variogram Parameters GC Model. (Source : CSA Global 2017)

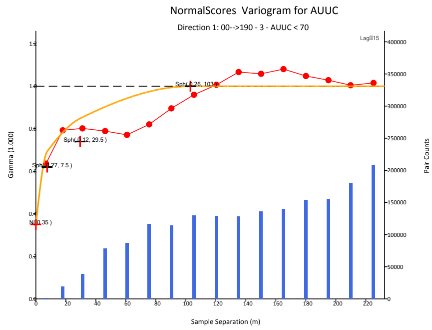
Domains	Estzon	Datamine Orientation (ZXZ)	Nugget	Structure 1		Structure 2		Structure 3	
				Partial Sill	Range	Partial Sill	Range	Partial Sill	Range
D2000	1	0	0.25	0.41	12.5	0.21	37	0.13	109
		0			4		12.5		36.5
		-100			4.5		24		132
D2100 D2200	2	100	0.26	0.39	8.5	0.2	25	0.15	182.5
		70			6.5		21.5		184.5
		180			6.5		20.5		43.5
D3001 D3002	3	90	0.36	0.25	12	0.11	12.5	0.28	266.5
		60			5		33.5		168
		180			5.5		13		51.5
D5000	4	90	0.17	0.57	12	0.2	34	0.06	123
		90			7.5		41.5		53.5
		180			3.5		9.5		18.5
D5100	5	80	0.06	0.78	14.5	0.08	30.5	0.09	81.5
		90			7.5		29		60.5
		180			4		8		12.5
D2350	6	90	0.26	0.67	11.5	0.07	40.5		
		110			13.5		28		
		180			5.5		16.5		
D9000	7	0	0.3	0.54	10	0.11	67	0.04	154
		0			6		38.5		83.5
		-90			4.5		54		60.5

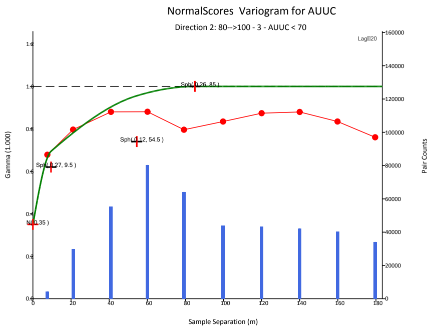
14.5.17 IK Model

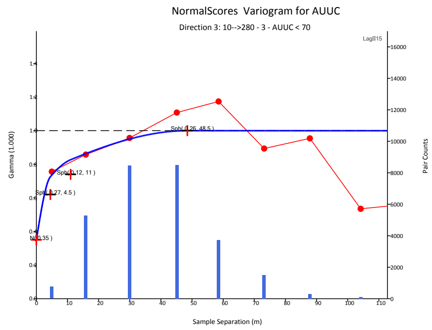
For each of the estimation domains, the semi-variograms were well structured, with a moderate to high nugget and moderate ranges. The variograms were back transformed prior to estimation. The normal score variograms are presented in Figure 14-17. The back transformed variogram parameters used in the estimate are detailed in Table 14-18.

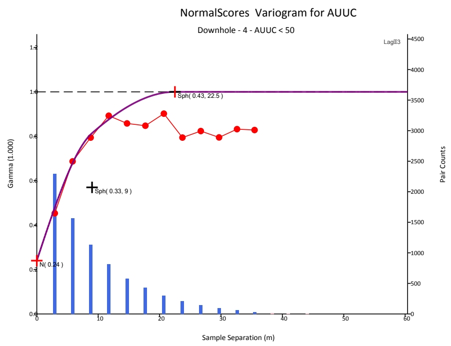


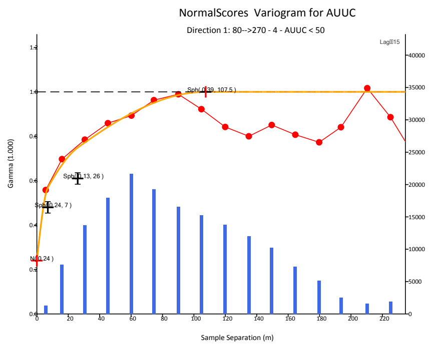

NormalScores Variogram for AUUC
Downhole - 3 - AUUC < 70
Gamma (1.000)
Pair Counts
Sample Separation (m)

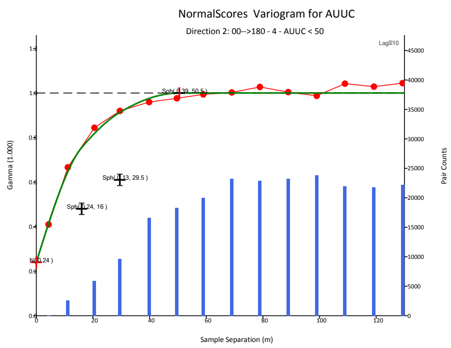

NormalScores Variogram for AUUC
Direction 1: 00-->190 - 3 - AUUC < 70
Gamma (1.000)
Pair Counts
Sample Separation (m)

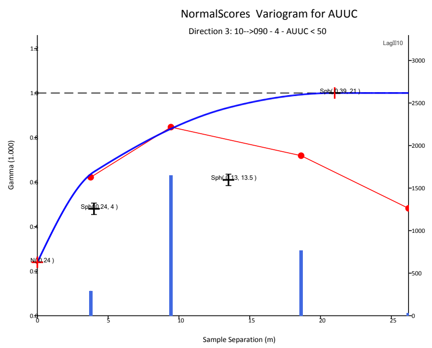

NormalScores Variogram for AUUC
Direction 2: 80-->100 - 3 - AUUC < 70
Gamma (1.000)
Pair Counts
Sample Separation (m)

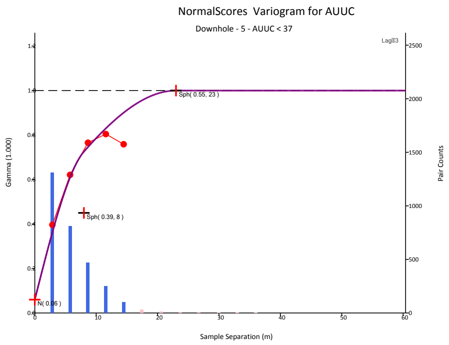

NormalScores Variogram for AUUC
Direction 3: 10-->280 - 3 - AUUC < 70
Gamma (1.000)
Pair Counts
Sample Separation (m)

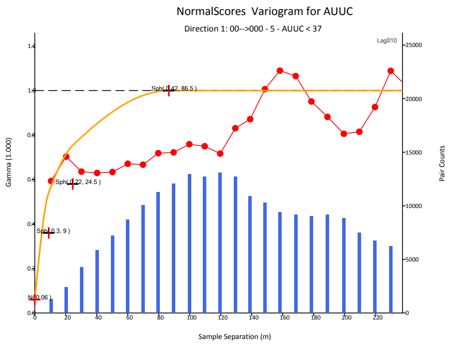

NormalScores Variogram for AUUC
Downhole - 4 - AUUC < 50
Gamma (1.000)
Pair Counts
Sample Separation (m)

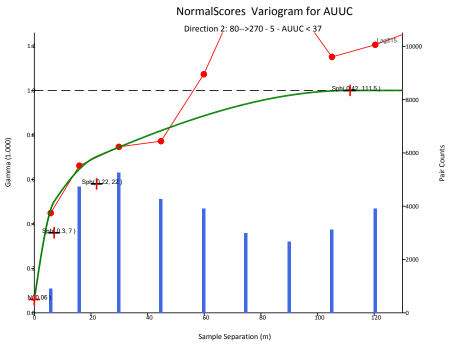

NormalScores Variogram for AUUC
Direction 1: 80-->270 - 4 - AUUC < 50
Gamma (1.000)
Pair Counts
Sample Separation (m)

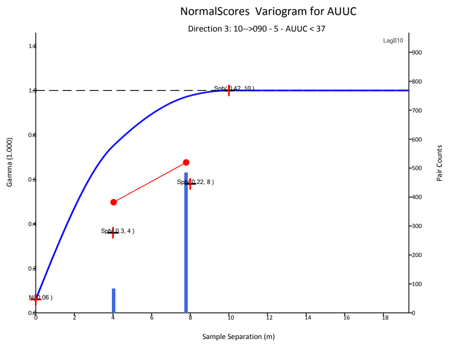

NormalScores Variogram for AUUC
Direction 2: 00-->180 - 4 - AUUC < 50
Gamma (1.000)
Pair Counts
Sample Separation (m)

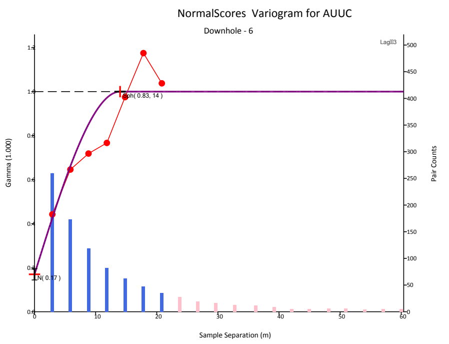

NormalScores Variogram for AUUC
Direction 3: 10-->090 - 4 - AUUC < 50
Gamma (1.000)
Pair Counts
Sample Separation (m)

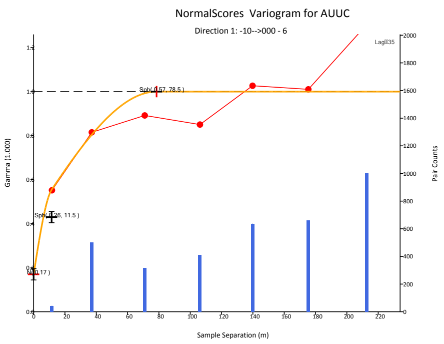

NormalScores Variogram for AUUC
Downhole - 5 - AUUC < 37
Gamma (1.000)
Pair Counts
Sample Separation (m)

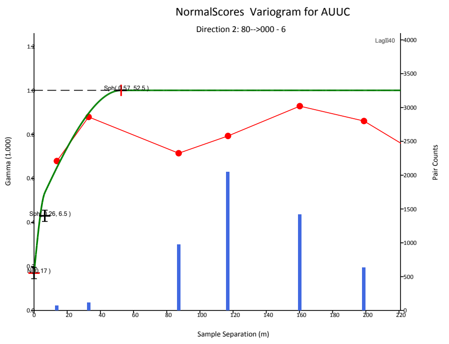

NormalScores Variogram for AUUC
Direction 1: 00-->000 - 5 - AUUC < 37
Gamma (1.000)
Pair Counts
Sample Separation (m)

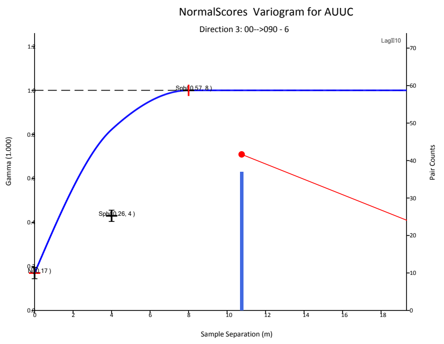

NormalScores Variogram for AUUC
Direction 2: 80-->270 - 5 - AUUC < 37
Gamma (1.000)
Pair Counts
Sample Separation (m)


NormalScores Variogram for AUUC
Direction 3: 10-->090 - 5 - AUUC < 37
Gamma (1.000)
Pair Counts
Sample Separation (m)


NormalScores Variogram for AUUC
Downhole - 6
Gamma (1.000)
Pair Counts
Sample Separation (m)


NormalScores Variogram for AUUC
Direction 1: -10-->000 - 6
Gamma (1.000)
Pair Counts
Sample Separation (m)


NormalScores Variogram for AUUC
Direction 2: 80-->000 - 6
Gamma (1.000)
Pair Counts
Sample Separation (m)


NormalScores Variogram for AUUC
Direction 3: 00-->090 - 6
Gamma (1.000)
Pair Counts
Sample Separation (m)

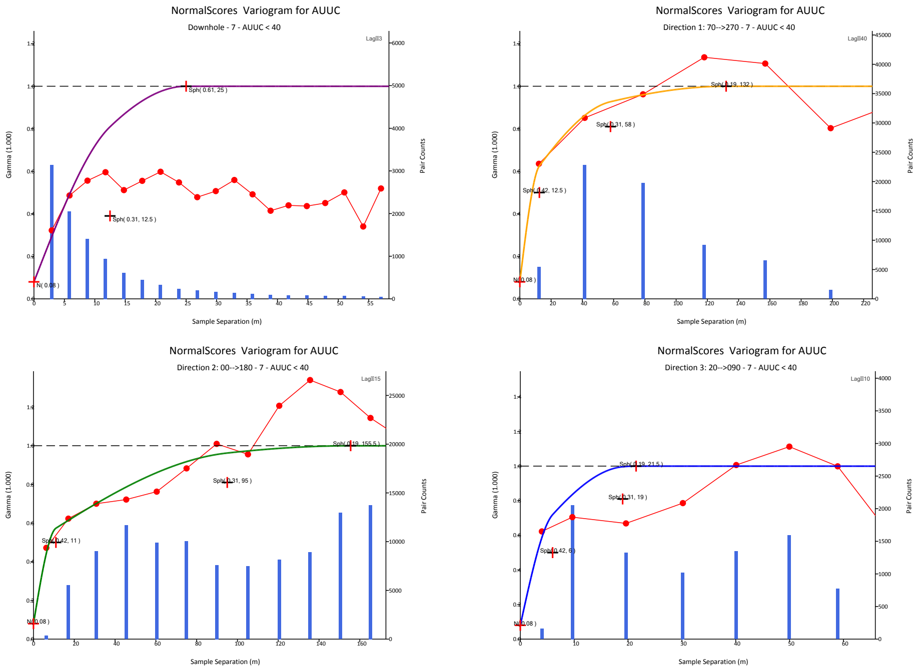


Figure 14-17 Normal score variograms used for Au g/t estimation per domain – IK model. (Source : CSA Global 2017)

Table 14-18: Variograms Parameters IK Model. (Source : CSA Global 2017)

ESTZON	Isatis Orientation (ZYX)	Nugget	Structure 1		Structure 2		Structure 3	
			Partial Sill	Range	Partial Sill	Range	Partial Sill	Range
1	80	0.44	0.3	7	0.17	27.5	0.09	147.5
	0			11		33.5		92.5
	80			3.5		5.5		27.5
2	80	0.37	0.39	7	0.16	32.5	0.08	129.5
	0			7		27		101.5
	90			6.5		26		36
3	-100	0.54	0.26	7.5	0.09	29.5	0.12	103
	0			9.5		54.5		85
	80			4.5		85		48.5
4	-180	0.37	0.28	7	0.13	26	0.22	107.5
	-80			16		29.5		50.5
	0			4		13.5		21
5	90	0.11	0.46	9	0.22	24.5	0.21	86.5
	0			7		22		111.5
	80			4		8		10
6	90	0.27	0.35	11.5	0.37	78.5		
	10			6.5		52.5		
	90			4		8		
7	-180	0.16	0.59	12.5	0.19	58	0.06	132
	-70			11		95		155.5
	0			6		19		21.5

14.5.18 Kriging Neighbourhood Analysis

Kriging Neighbourhood Analysis ("KNA") was completed using Supervisor v8.4TM software, adopting the relevant variogram models for the estimation domains.

KNA on the 3m composites was used to optimise the parent cell sizes for both the GC and IK models and to determine the optimal theoretical estimation and search parameters during kriging.

The following was reviewed for each of the variables per selected domain:

- Slope and Kriging Efficiency ("KE") statistics for a well-informed block for different block sizes
- On choosing a block size, optimum minimum and maximum samples were chosen. The maximum was set at the lowest number of samples from which consistently good slopes and KE could be derived. The minimum was defined as the lowest minimum from which moderate to good statistics could be derived
- On choosing the minimum / maximum samples, search ellipse ranges were defined. The quality of the statistics were least sensitive to this parameter. The ranges chosen approximated the ranges of the first structure of the variogram
- Negative weights were reviewed at each stage to ensure the parameters chosen were not leading to excessive negative weights
- Discretisation was defined
- Maximum number of samples allowed per each individual drill hole, per estimate, was set
- The KNA results show that the search parameters and block sizes selected for both the GC and IK models, respectively, are suitable for use in the MRE and adequately take drill spacing, geology and practicality into account
- The search ranges in the IK model are larger than that of the GC model. This reflects the continuity shown in the data analysis and variography. Smoothing of grades in the areas of closely spaced drill hole data will be reduced by limiting the maximum number of samples used in the estimate

14.5.19 GC Model

The number of composites used for the Au grade estimation is presented in Table 14-19. The modelled variogram parameters together with the selected estimation panel size and number of samples was used to determine the appropriate search ellipse for the primary search pass. This is also presented in Table 14-19.

The low maximum number of samples was introduced in an attempt to lower the smoothing effect. Total average number of samples used for each ESTZON ranged from 8 to 8.9 for ESTZONS 1 to 6, and 4.3 for ESTZON 7. For ESTZON 1 to ESTZON 5, 90% or more of the volume was estimated using the maximum number of 9 samples. For ESTZON 6, this proportion dropped to 70%, and for ESTZON 7, this dropped to only 34%.

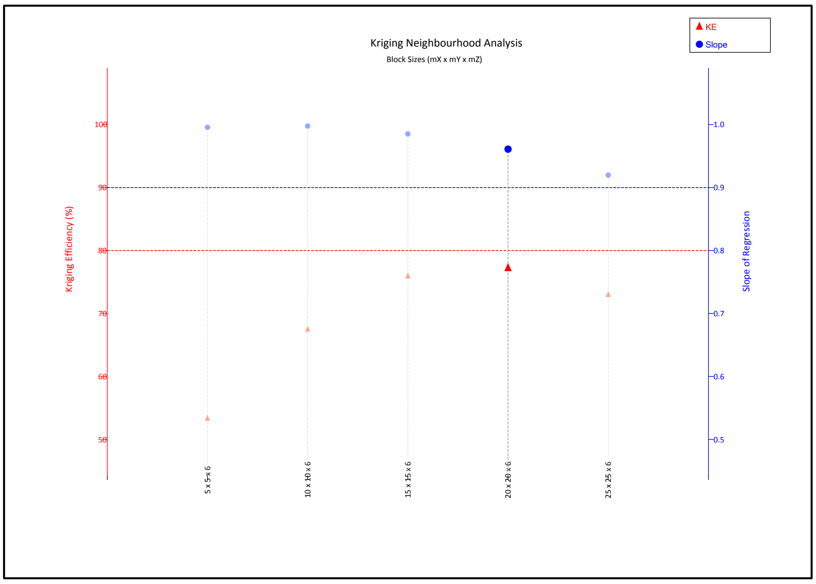
The low minimum number of samples was introduced to allow blocks on the edge of the GC area to be estimated, but by having maximum number of samples per drillhole set to 1, at least 3 informing holes were still needed.

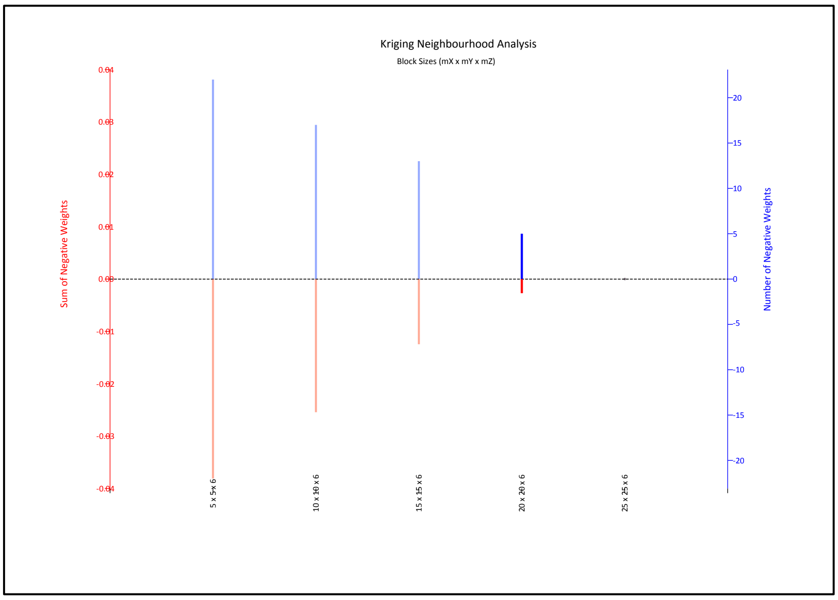
Table 14-19: Search Neighbourhood Parameters for Au – GC Model. (Source : CSA Global 2017)

Estzon	Search Ranges (SVOL1)			Search Ranges (SVOL2)			Composites	
	1	2	3	1	2	3	Min	Max
1	25	25	10	50	50	20	3	9
2	40	30	15	80	60	30	3	9
3	20	20	10	40	40	20	3	9
4	25	25	10	50	50	20	3	9
5	20	20	10	40	40	20	3	9
6	25	20	10	50	40	20	3	9
7	15	15	5				3	9

14.5.20 IK Model

The plots with the selected estimation parameters for the main domain of the IK model are shown in Figure 14-18.





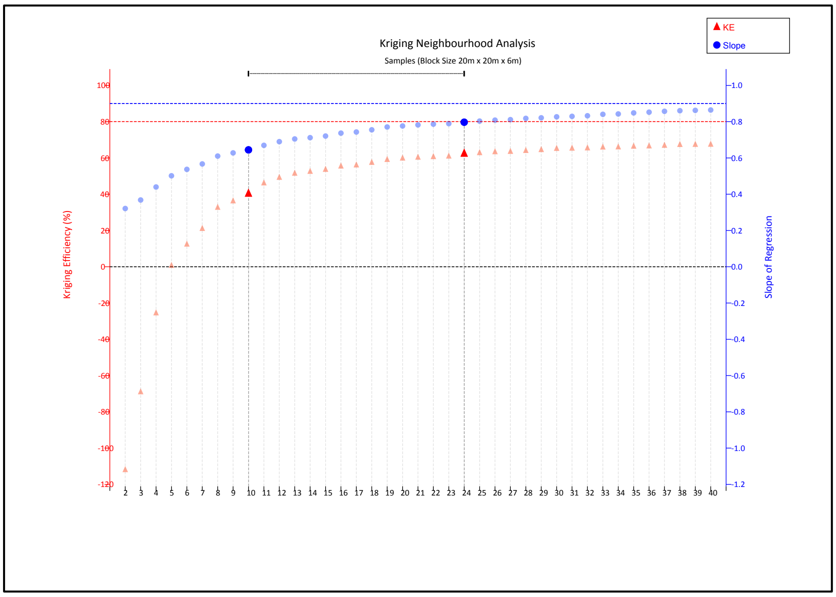


Figure 14-18: KNA block size, samples results – IK model. (Source : CSA Global 2017)

The number of composites used for the Au grade estimation per domain are presented in Table 14-20. The modelled variogram parameters together with the selected estimation panel size and number of samples was used to determine the appropriate search ellipse for the maximum search ranges. These are also presented per domain in Table 14-20.

Table 14-20: Search Neighbourhood Parameters for Au per estimation Domain – IK Model. (Source : CSA Global 2017)

ESTZON	Search Ranges (SVOL1)			Search Ranges (SVOL2)			Search Ranges (SVOL3)			Composites		
	1	2	3	1	2	3	1	2	3	Min	Max	Max per Drill hole
1	74	46	14	148	93	28	148	93	28	9	24	3 (No max for SVOL 3)
2	65	51	18	130	102	36	130	102	36			
3	52	43	24	103	85	49	103	85	49			
4	54	25	11	108	51	21	108	51	21			
5	43	56	5	87	112	10	87	112	10			
6	39	26	4	79	53	8	79	53	8			
7	36	43	10	71	87	20	71	87	20			

14.5.21 Block Model

The model was cut to below the topographic surface. The volume model was built to 5 mN by 5 mE by 3 mRL, with sub-celling to 1.5m in the RL (topography, depletion surfaces). A parent cell estimation prototype was overlain and used for the grade estimate of the IK model. The GC model was joined to the IK model after grade estimation. The model prototypes parameters, including cell dimensions and model extents, are shown in Table 14-21. A list of the final block model attributes is displayed in Table 14-22.

Panel sizes for the grade estimations were based on the following:

- Results of Kriging KNA
- The density of the drilling grids
- The geometry of the mineralisation
- The mining parameters

Table 14-21: Nkran – Block Models Dimensions. (Source : CSA Global 2017)

Prototype	Axis	Origin	Model Extent (m)	# Blocks	Block dimension (m)
MRE Model	Easting (X)	4200	5800	320	5
	Northing (Y)	19300	21300	400	5
	Elevation (Z)	542	1208	222	3

Table 14-22: Nkran Final Block Model Attributes. (Source : CSA Global 2017)

Attribute	Description
IJK	Unique block identifier
XC, YC, ZC	Centre block point X, Y and Z
XINC, YINC, ZINC	Block size along X, Y and Z
MINZON	Geological domain (2000,2100,2200,2350,3001,3002,5000,5100,9000)
ROCK	State of oxidation. Oxide (1), transitional (2) and fresh (3)
TOPO	Flagged block above (0) or below topography (1)
MINED	Flagged mined blocks; Mined = 1 = mined; Mined=0 = unmined

Attribute	Description
GCAREA	GC model area = 1 Exploration area = 0
AU	Estimated Au grade (g/t)
ESTZON	Estimation domain (1 to 7)
SG	Specific Gravity
SHALLOW	Shallow =0 = Steep mineralisation Shallow = 1 = shallow mineralisation
RESCAT	Measured = 1, Indicated=2, Inferred=3, Unclassified=9
XMORG, YMORIG, ZMORIG	Model origin X, Y and Z
NX, NY, NZ	Number of blocks X, Y and Z

A comparison of the wireframe volumes to the block model volume for each of the resource zones is shown in Table 14-23 below. The comparison shows that the resolution of the block models sub-celling is satisfactory.

Table 14-23: Volume Comparison between Mineralisation Wireframes and Block Model. (Source : CSA Global 2017)

Domain	Wireframe Volume	Block Model Volume	Difference %
D2000	14,202,664	14,158,125	0.31%
D2100	11,804,009	11,793,225	0.09%
D2200	16,631,011	16,579,800	0.31%
D2350	53,267,616	52,847,850	0.79%
DGR01	8,493,656	8,471,325	0.26%
DGR02	2,467,241	2,455,200	0.49%
D5000	10,814,388	10,790,250	0.22%
D5100	2,271,649	2,271,525	0.01%
D9000	444,423,127	441,522,375	0.65%
Total	564,375,361	560,889,675	0.62%

14.5.22 Grade Estimation

14.5.22.1 Methodology

14.5.22.1.1 Grade Control Model

Estimation of Au grade was carried out using OK into 5m x 5m x 3m parent cell panels. Zonal control with soft boundaries between mineralisation zones was used during the grade estimation. For each estimation domain, a loop was run setting all data to the current ESTZON, ensuring the relevant variograms and search parameters were used per ESTZON, while using soft data boundaries between the domains.

Grade estimation was completed using up to 2 search passes. The second pass was 2x the volume of the first search and used to estimate the grade of any blocks not estimated in the first pass (98% of cells were estimated in the first pass). Only one search pass was used for domain seven, the low grade un-constrained exterior domain, to ensure that cells on the outside of the GC model were not extrapolated beyond the variogram range of the GC data.

The mineralised areas were estimated using dynamic anisotropy. This process allows the rotation angles for the search ellipsoid to be defined individually for each panel in the block model, so that the search ellipsoid is aligned with the axes of mineralisation. The rotation angles were interpolated into the model cells. The dip and dip direction defining the dominant anisotropy were derived from strings digitised perpendicular to the strike of the mineralisation. These strings were converted to points that contained the true dip and dip direction of the mineralisation by stratigraphic domain.

14.5.23 IK Model

Estimation of Au grade was carried out using OK into parent cell panels. Zonal control with a hard boundary between mineralisation and waste was used during the grade estimation. The estimation domain was assigned a unique ESTZON number, corresponding to the ESTZON field in the input composite data.

A three-phased search pass was applied and the orientation of the search ellipsoid was aligned to the modelled variography. This process involves the estimation being performed three times, where two expansion factors are used. During each individual estimation run this factor increases the size of the search ellipse used to select samples. This method ensures that blocks which are not estimated and populated with a grade value in the first run, are populated during one of the subsequent runs.

The mineralised areas were estimated using dynamic anisotropy. This process allows the rotation angles for the search ellipsoid to be defined individually for each cell in the models, so that the search ellipsoid is aligned with the axes of mineralisation. This therefore requires the rotation angles to be interpolated into the model cells, which in turn requires a set of angles as the input data file for interpolation. The dip and dip direction of the major axis of anisotropy were defined by digitising strings in section perpendicular to the strike of the mineralisation. These strings were converted to points that contained the true dip and dip direction of the mineralisation and stratigraphy.

Section 14.10 outlines the sample search neighbourhood informed by KNA.

14.5.24 Visual Validation

The block models were visually reviewed section by section to ensure that the grade tenor of the input data was reflected in the block models. Figure 14-19 and Figure 14-20 show a cross section and long section view of the GC block model for Au g/t. Generally, the estimates compare well with the input data. The grades in the composites align with the corresponding grades in the block models.

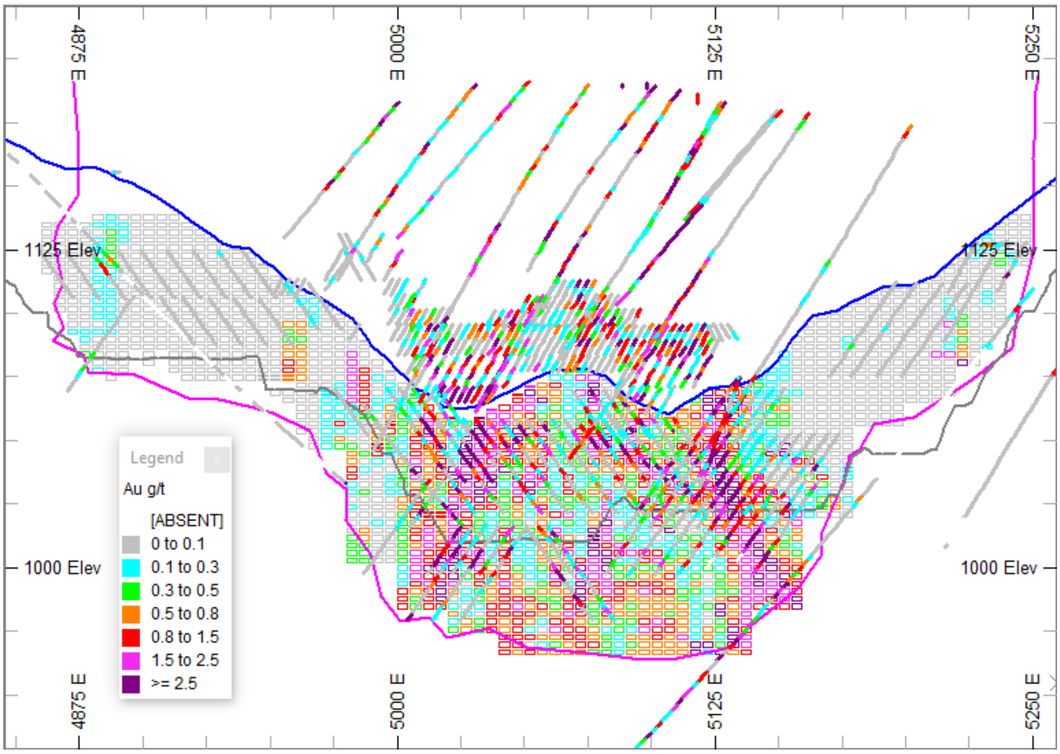


Figure 14-19: Cross section view 20,290mN, looking north – GC Model and composites. (Source : CSA Global 2017)

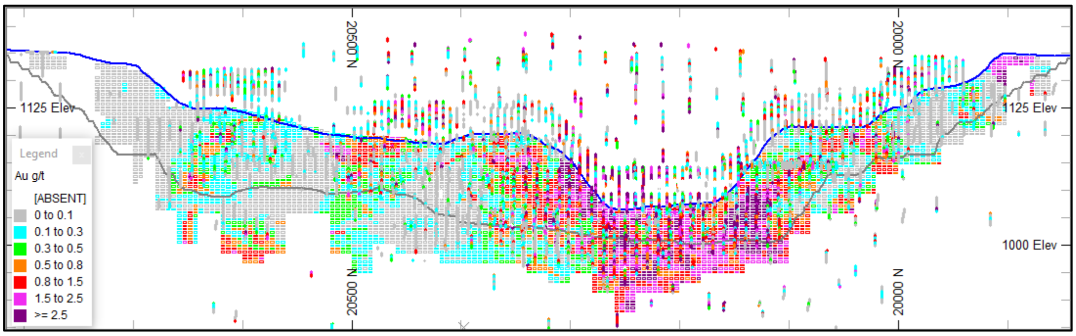


Figure 14-20: Long section view 5,035mE, looking east – GC Model and composites. (Source : CSA Global 2017)

Figure 14-21 and Figure 14-22 show a cross section and plan view of the IK block model for Au g/t. Generally, the estimates compare well with the input data. The grades in the composites align with the corresponding grades in the block models.

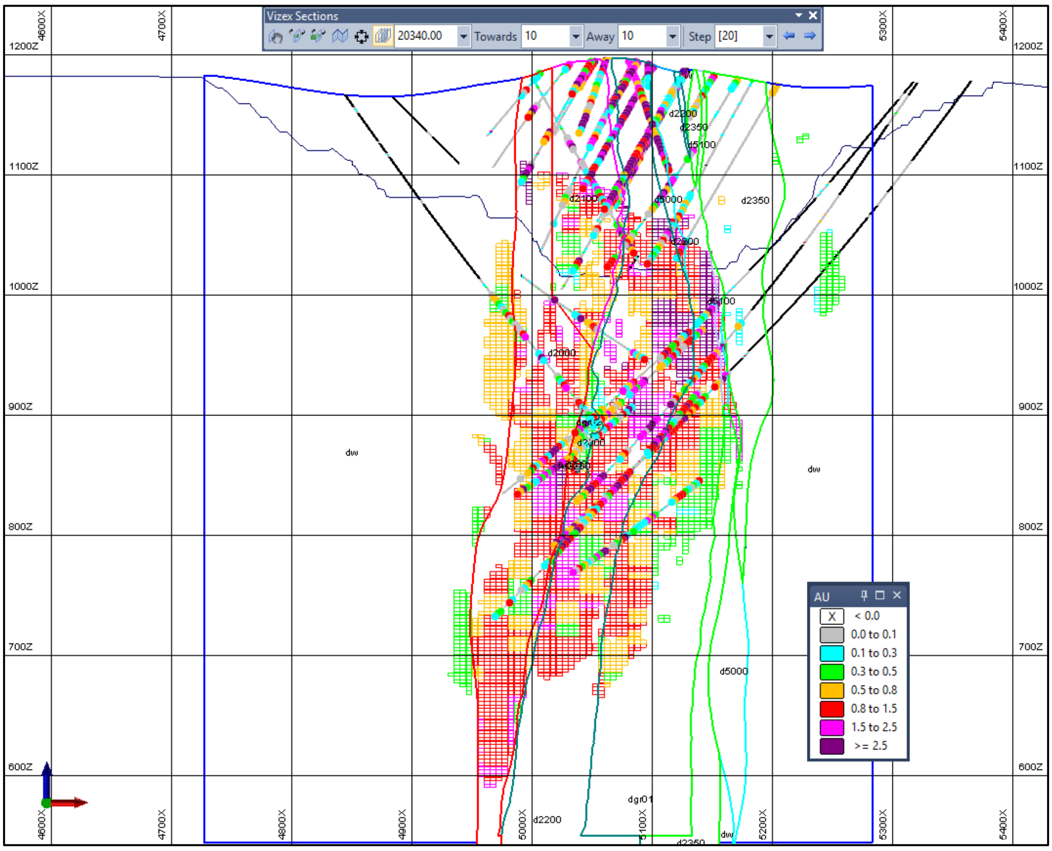


Figure 14-21: Section view looking North – IK Model and composites. (Source : CSA Global 2017)

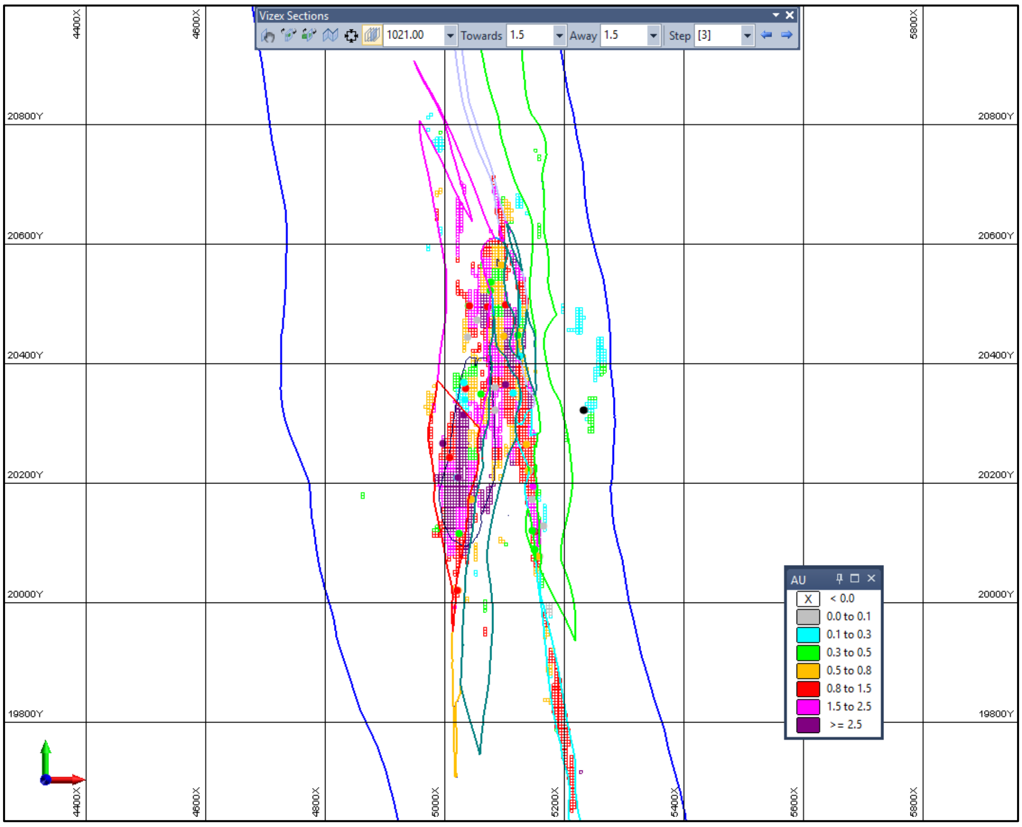


Figure 14-22: Plan view – IK Model and composites. (Source : CSA Global 2017)

14.5.25 Model Validation

Validation of the GC and IK block models were completed by comparing input and output means, comparison of input and output histograms, swath plots, and visual inspection of cross sections through the deposits.

14.5.26 Statistical Validation

14.5.26.1 GC Model

Due to data clustering effects in the GC model which included extrapolated waste blocks and overlapping drill hole tails, a wireframe was constructed to isolate the area of concentrated GC drilling representing around 40% of the total GC model volume (at a 0.3 g/t Au cut-off). The validation area was chosen in the southern end Figure 14-23 and contains approximately 7.5 Mt of material (at zero cut-off).

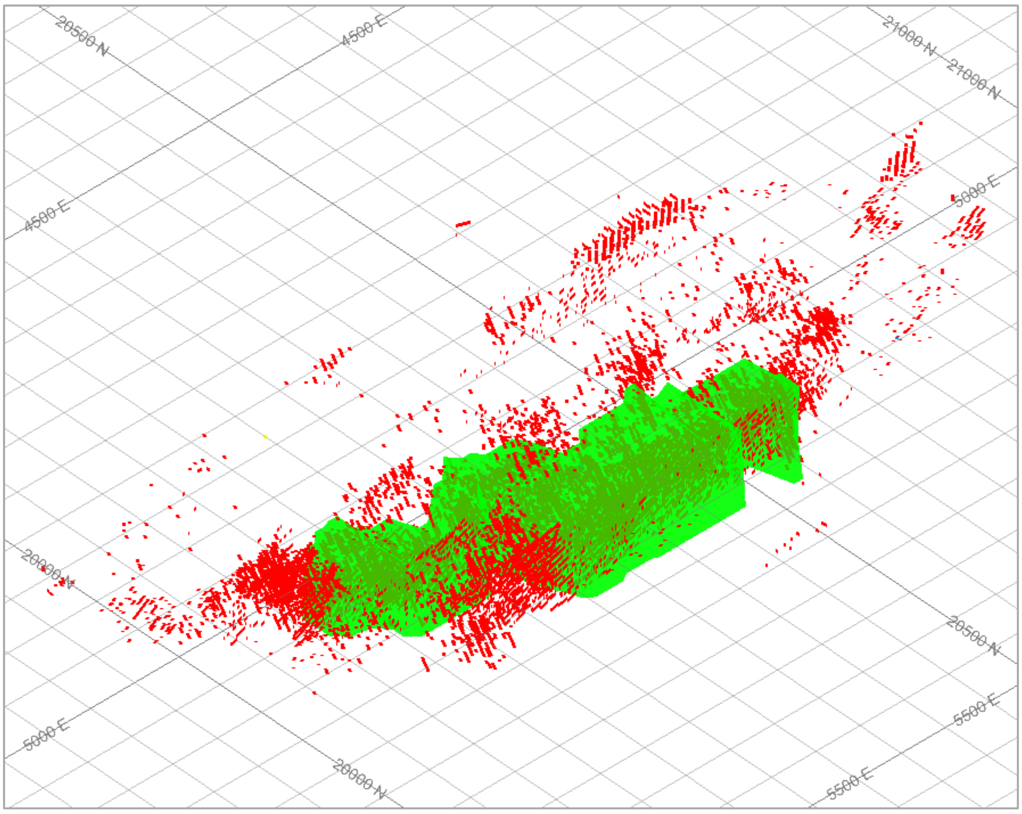


Figure 14-23: Oblique view (view to the NW) of the GROUP=4 GC data, filtered for Au>=0.3 g/t (red), and the test wireframe (green). (Source : CSA Global 2017)

Mean grade comparisons between the block model and the data within the test area is presented in Table 14-24.

Table 14-24: Mean grade comparison for Au g/t - GC model comparison area. (Source : CSA Global 2017)

Data	Variable	Count	Min.	Max.	Mean	Std Dev	Variance	CV
Composites Naïve	AU	15,685	0.0025	90	1.49	3.94	15.50	2.65
GC Model	AU	32,340	0.01	32.40	1.46	1.74	3.02	1.20
Difference relative to the input composite mean grade					-2.20%			

14.5.26.2 Declustering

Irregular sampling of a deposit, most commonly through infill drilling, or drilling in multiple orientations, causes clustering. Clustering results in a disproportionate distribution grades (usually high grades from the infill drilling) in the dataset used for statistical analysis. Mixed populations in the histogram can create a bias when comparing the drill hole sample distribution with the block model distribution (which is declustered) and distort the calculated mean grades and variance.

Different ways of declustering data each give different results. These include interactive filtering, polygonal declustering, nearest neighbour declustering and cell-weighted declustering.

The method used for geostatistical analysis and validation for the Nkran MRE update is cell-weighted declustering, since all samples are considered when determining the average. This method involves placing a grid of cells over the data. Each cell that contains at least one sample is assigned a weight of one. That weight of one is distributed evenly between the samples within each cell.

The OK grade estimation process is a very efficient way of data clustering, therefore declustering before grade estimation is not necessary. Declustering of the input data does give a good indication of the global mean. It is used in the validation of the estimate (comparison of the means). Declustering was applied to remove any bias due to drill spacing prior to validation of the IK model. The declustering parameters for the IK model are presented in Table 14-25.

Table 14-25: Declustering Parameters IK Model. (Source : CSA Global 2017)

Model	Cell size (m)		
	X	Y	Z
IK	40	40	40

14.5.26.3 Results

The global statistics of Au g/t within the IK model were reviewed and the results are reported below in Table 14-26. All estimated block grades are included. The mean grades in the estimated model block parent cells were compared to the raw, as well as the declustered, top-cut composite data.

Generally, the model validates well, showing <1% difference between the declustered composites and the block estimates, globally. This is within expected parameters.

Table 14-26: Declustered Mean Grade Comparison for Au g/t - IK Model. (Source : CSA Global 2017)

	Field	Count	Min.	Max.	Mean	Std Dev	Variance	CV
Composites Naïve	Au	3,648	0.00	70.00	1.64	3.59	12.86	2.18
Composites Declustered		3,648	0.00	70.00	1.42	3.20	10.21	2.24
IK Model		283,960	0.04	11.73	1.41	1.21	1.46	0.85
Difference [(Composite Declustered Grade – IK Model Grade)/IK Model Grade]					%	-1%		

14.5.26.4 Swath Plots

Swath plots were created as part of the validation process, by comparing the parent block grades of the models and input composites (declustered and top cut) in spatial increments. These plots display northing, easting and elevation slices throughout the deposit. Figure 14-24, Figure 14-25 and Figure 14-26.

The plots show that the distribution of block grades honours the distribution of input composite grades. The degree of smoothing is appropriate and accounts for volume variance effects, where block grades should be smoother than point grades.

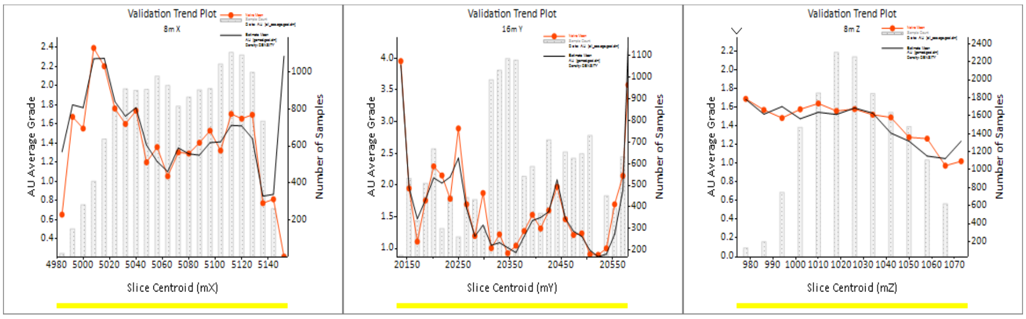


Figure 14-24: GC mode comparison areal - Swath plot by 8m easting, 16m northing, and 8m bench for Au g/t. (Source : CSA Global 2017)

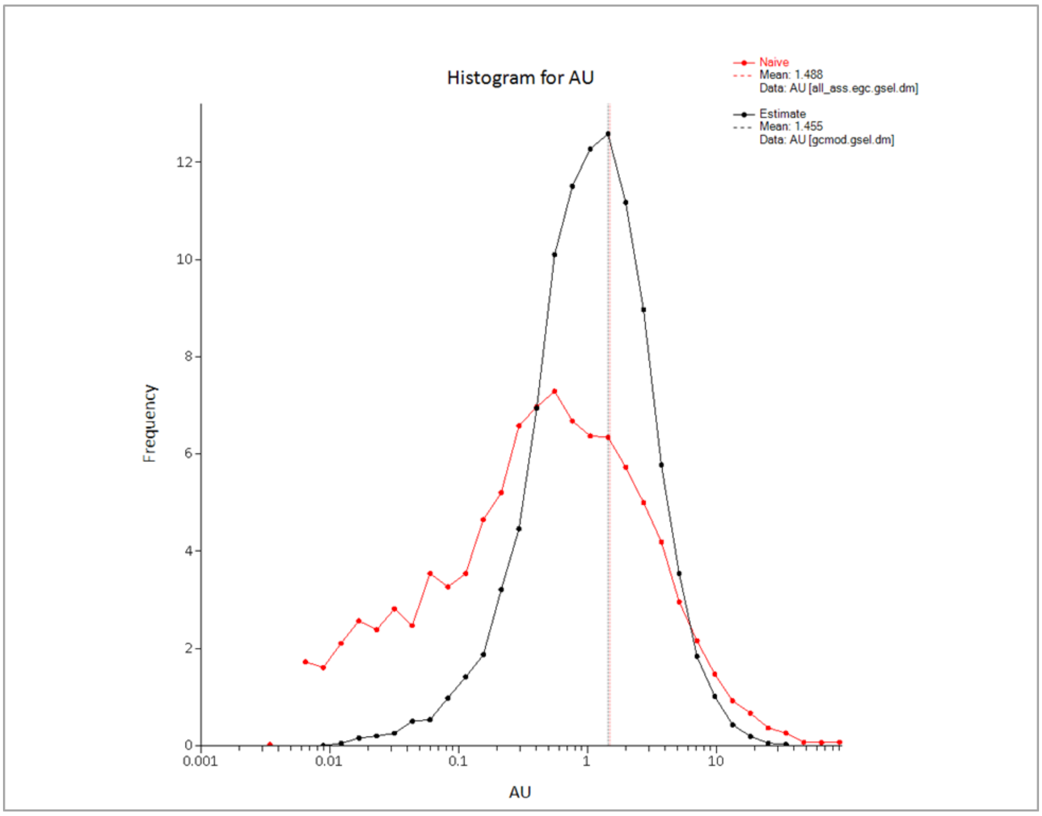


Figure 14-25: Lognormal histogram of input composite data, red, versus output block model, black, for the GC test area. (Source : CSA Global 2017)

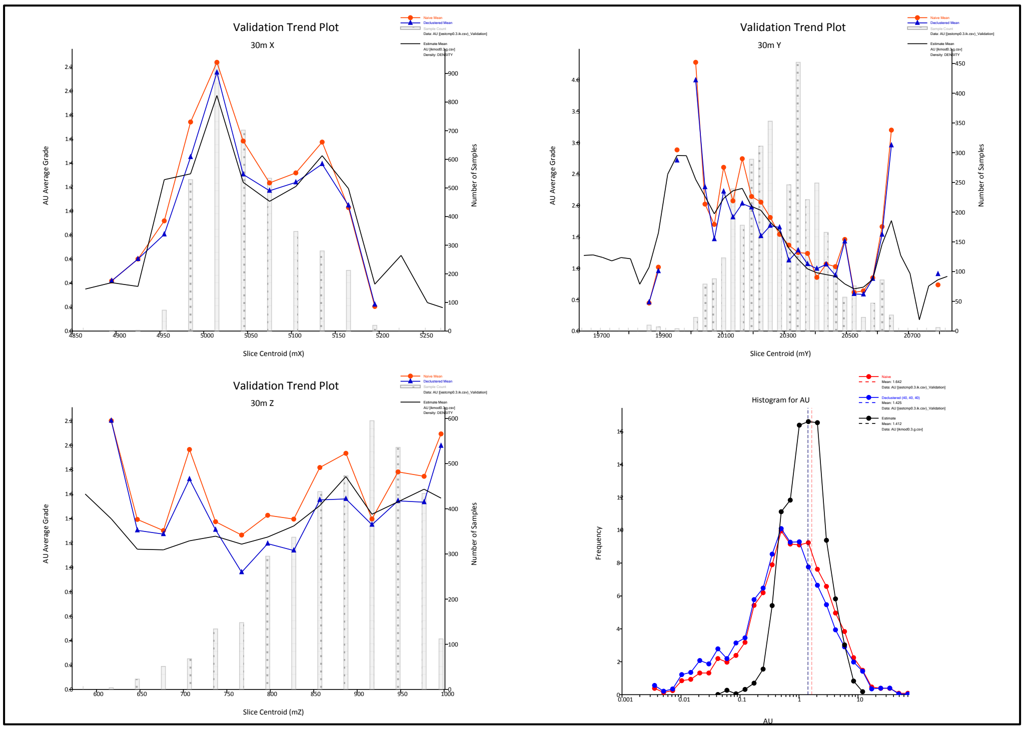


Figure 14-26 IK model - Swath plot by 30m easting, 30m northing, and 30m bench for Au g/t. (Source : CSA Global 2017)

14.6 Mineral Resource Classification

The Mineral Resource has been classified as Measured Mineral Resources, Indicated Mineral Resources and Inferred Mineral Resources based on CIM guidelines adopted for disclosure via NI 43-101. The classification level is based upon an assessment of geological understanding of the deposit, geological and mineralisation continuity, drill hole spacing, quality control results, search and interpolation parameters and an analysis of available density information.

The Nkran deposit shows good continuity of mineralisation within well-defined geological constraints. Drill holes are located at a nominal spacing of 25m on 25m sections extending out to 50m on the peripheries of the deposit. The drill spacing is sufficient to allow the geology and mineralisation zones to be modelled into coherent wireframes for each domain. Reasonable consistency is evident in the orientations, thickness and grades of the mineralised zone.

Measured Mineral Resources were classified only in the area supported by GC drilling and production data (GCAREA=1). Indicated Mineral Resources were informed by Slope statistics and average distance of samples and generally is defined where drilling is approximately 40m x 40m. The remaining material above a wireframe surface nominally based on a US$1500 gold price Whittle pit shell was classified as Inferred Mineral Resources.

A summary of the classification codes applied in the model are shown in Table 14-27 and Figure 14-27 and Figure 14-28 show the final classified block model in plan view and 3D view.

Table 14-27: RESCAT Field and Description. (Source : CSA Global 2017)

RESCAT	Description
1	Measured Mineral Resource
2	Indicated Mineral Resource
3	Inferred Mineral Resource
9	Mineralisation beneath the USD1500 surface, or not estimated – Waste material

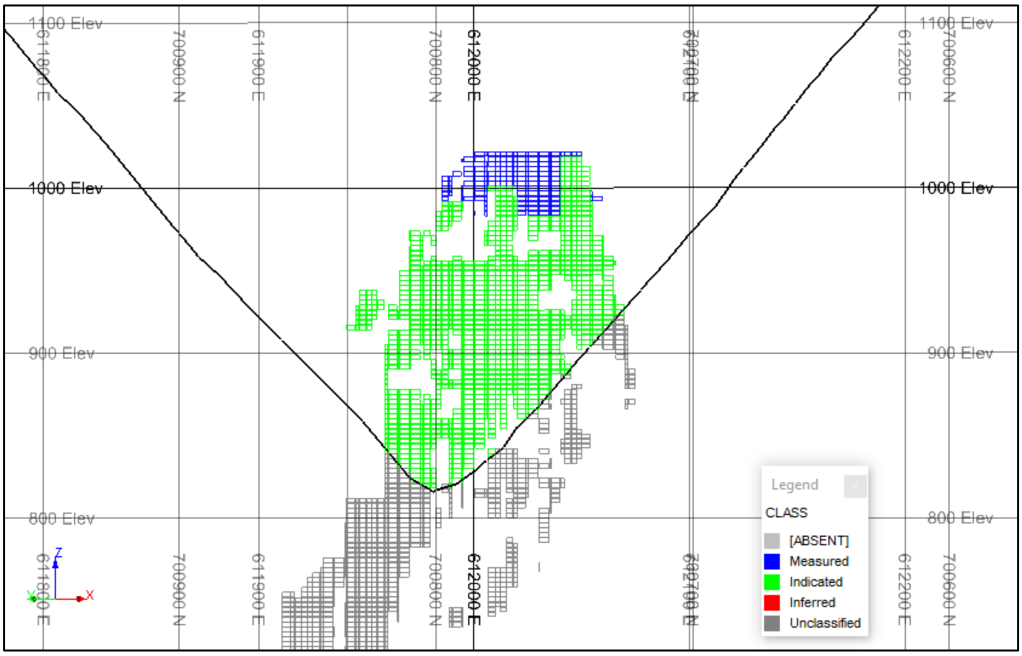


Figure 14-27: Nkran Section view of classified grade model, constrained within nominal US$1500/oz Au pit shell. (Source : CSA Global 2017)

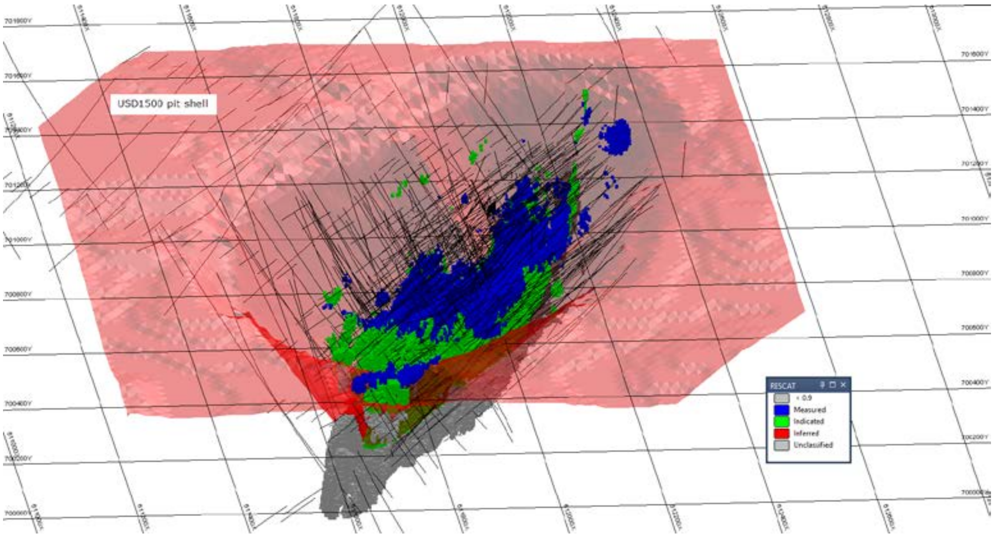


Figure 14-28: 3D view of classified grade model, view towards NW. Nominal US$1500 pit shell shown in red (Source : CSA Global 2017)

14.7 Mineral Resource Reporting

The total Mineral Resource estimate is shown in Table 14-28 as at 23rd January 2017. The MRE compiled by CSA has been classified and is reported as Measured, Indicated and Inferred Mineral Resources based on CIM Definition Standards May 2014.

Table 14-28: Nkran Deposit – Mineral Resource Estimate, reported at a 0.5 g/t Au cut-off, 23rd January 2017

Nkran Main	0.5 g/t cutoff constrained to $1500/oz Au shell					
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Measured	Oxide	88,623	0.09	1.65	4,689	0.005
	Transition	57,525	0.06	1.23	2,282	0.002
	Fresh	4,776,765	4.78	1.82	279,679	0.280
Total Measured		**4,922,913**	**4.92**	**1.81**	**286,650**	**0.287**
Indicated	Oxide	78,045	0.08	2.65	6,660	0.007
	Transition	174,975	0.17	2.28	12,805	0.013
	Fresh	22,838,927	22.84	1.88	1,381,242	1.381
Total Indicated		**23,091,947**	**23.09**	**1.89**	**1,400,706**	**1.401**
Total Measured and Indicated		**28,014,860**	**28.01**	**1.87**	**1,687,356**	**1.687**
Inferred	Oxide	-	-	-	-	-
	Transition	-	-	-	-	-
	Fresh	303,108	0.30	1.88	18,282	0.018
Total Inferred		**303,108**	**0.30**	**1.88**	**18,282**	**0.018**

Notes:

- Reported from 3D block model with Au grade estimated by Ordinary Kriging with 20m x 20m x 6m parent block size below the GC area, and 5m x 5m x 3m parent block sizes within the GC area.
- Based on data available as at 17th January, 2017. Reported using a 0.5 g/t gold cut-off.
- The model is reported above a surface nominally based on the Whittle shell from US$1500/oz gold price pit optimisation run.
- Due to rounding, the totals may not represent the sum of all components.
- The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.

The grade versus tonnage curves for the Measured and Indicated Mineral Resource categories are shown in Figure 14-29 for Measured, Indicated and Inferred Mineral Resource categories in Figure 14-30.

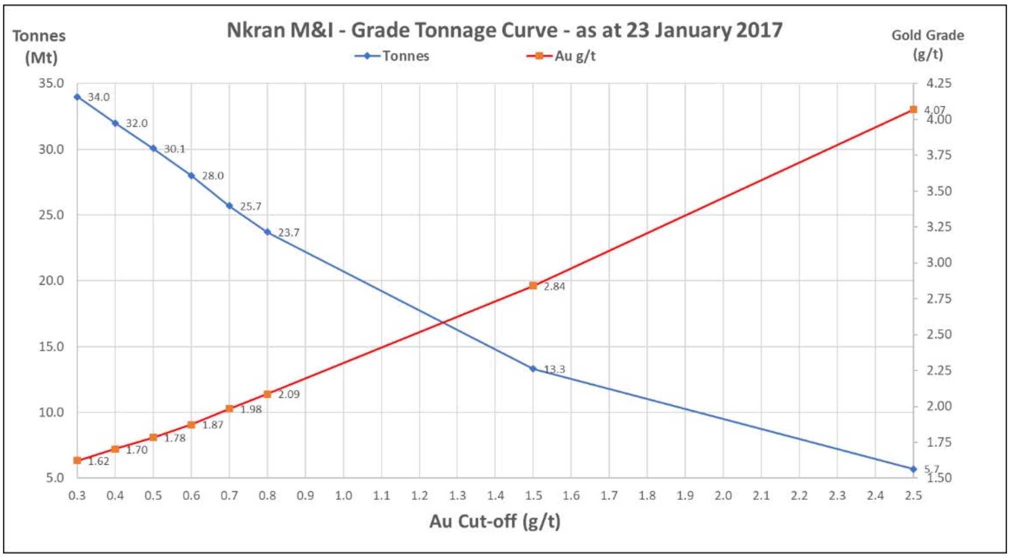


Figure 14-29: Nkran $1500 Grade Tonnage Curve – Measured and Indicated

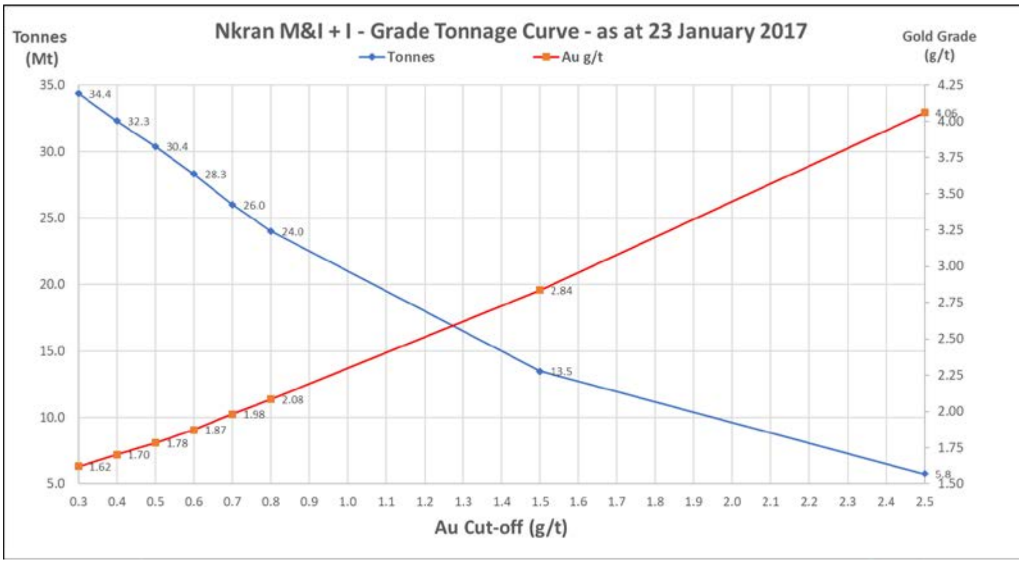


Figure 14-30: Nkran Grade Tonnage Curve – Total Resource

14.8 Previous Mineral Resource Estimates

The Nkran deposit was previously estimated and reported in July 2014 at a 0.8 g/t Au cut-off by CJM. This is presented alongside the current MRE which is reported at a 0.5 g/t Au cut-off for the purposes of comparison (Table 14-29).

Figure 14-31 presents a waterfall chart which attributes the causes of changes in metal and tracks them from the July 2014 MRE to the current January 2017 MRE.

Table 14-29: Nkran Deposit Comparison of CSA January 2017 MRE & CJM July 2014 MRE

Model	Cut-Off Au g/t	Resource Category	Tonnes Mt	Grade Au g/t	Content Au Koz
CSA Model January 2017	0.5	Measured	5.50	1.68	297
		Indicated	24.57	1.81	1,427
		Total M&I	**30.07**	**1.78**	**1,724**
		Inferred	-	-	-
CJM Model July 2014	0.8	Measured	13.24	2.55	1,086
		Indicated	25.8	2.23	1,851
		Total M&I	**39.04**	**2.34**	**2,937**
		Inferred	7.06	2.34	532
% Difference (cf CJM Model)		Measured	-58%	-34%	-73%
		Indicated	-5%	-19%	-23%
		Total M&I	-23%	-24%	-41%
		Inferred	-100%	-100%	-100%

Note: The CJM July 2014 MRE was not constrained by any economical parameters.

Since the July 2014 MRE the following changes have occurred:

- The CJM July 2014 MRE was reported using a proportion field and did not factor regularisation to 5 x 5 x 3m. In discussions with site geologists and engineers, it has been agreed that it is more appropriate to report the regularised 5 x 5 x 3m model to account for dilution.
- There have been some small changes to the density due to additional density data collected during mining for fresh rock and oxide being divided into oxide and transitional.
- Depletion from mining up to 31 December 2016
- The January 2017 CSA MRE has been constrained within a nominal US$$1500 shell to satisfy the requirement for a MRE having reasonable prospects for eventual economic extraction. This is substantiated by deeper drilling that exists under the Nkran US$1500/oz shell (Indicated resource) that is excluded from the MRE.
- Changes to the mineralisation, geology and weathering models as a resulting of infill drilling and re-interpretation due to grade control and infill drilling completed.
- Lowering of the MR cut-off grade from 0.8 g/t Au to 0.5 g/t Au. This is substantiated by the analysis of the actual operating costs for the Nkran Pit from the start of steady state production in April 2016 to May 30 2017, which indicate a break even grade of 0.45 g/t Au.
- Change in estimation methodology whereby the area supported by grade control drilling was subset and estimated using OK (using grade control and exploration data). The remainder of the resource was estimated using

exploration data only, using a combination of IK to define the mineralisation volume and OK to estimate grade into mineralisation blocks.

14.9 Dynamite Hill (CSA Global)

CSA Global undertook the updated Dynamite Hill MRE in December 2016. All figures and tabulations in this section are referred from the CSA Global Dynamite Hill Technical Report Jan 2017.

The Dynamite Hill deposit was initially discovered in 2013 through a trenching programme investigating a weak soil anomaly. The deposit primarily sits under lateritic cover on top of a topographic high, which has largely masked the geochemical signature. There are no significant historical workings associated with Dynamite Hill, apart from some small adits (limited to 10 m penetration) and alluvial gold mining in the valleys below the deposit. A drilling programme followed on from encouraging trench results which successfully intercepted the main ore body. Subsequent drilling programmes delineated the ore body for which a MRE was completed by CJM Consulting ("CJM") in 2014. An updated MRE (CSA Global) was completed in Q4 2016 post an infill drilling campaign to tighten up the open cast pit design.

Geology and mineralisation has been presented in Section 7.

14.9.1 Drilling and Sampling

A total of 185 holes for 21,572m have been drilled since 2013. Since the previous MRE completed in 2014, 56 additional holes for 5,176m have been drilled Q4 2016 to tighten up the grade distribution model. The drilling was completed by Asanko and is summarised in Table 14-30.

Table 14-30: Dynamite Hill Database Summary of Drilling as at 17th January 2017

Year Completed		RC	DDH	RCD	FC	Total
2013	Number of holes	86	3	1	5	95
	Metres	10,446.00	408.10	249.30	168.50	11,271.90
2014	Number of holes	32	2	-	-	34
	Metres	4,503.00	620.60	-	-	5,123.60
2016	Number of holes	56	-	-	-	56
	Metres	5,176.00	-	-	-	5,176.00
Grand Total	Number of holes	174	5	1	5	185
	Metres	20,125.00	1,028.70	249.30	168.50	21,571.50
RC – Reverse Circulation; DDH – Diamond Drill Hole; RCD – Reverse Circulation with Diamond Drill Tail; FC – Face Channel Sample						

14.9.2 Location of Data Points

Topography accurate to ± 2m was generated in several stages for the entire group of Asanko concessions by Photosat Information Ltd. using stereo pairs of IKONOS satellite images collected in December 2007 and July 2008. These images were orthorectified using surveyed control points including drill hole collar points available during the survey (CJM, 2014).

Drill holes were down hole surveyed on approximately 50m, or less downhole intervals, using a Reflex EZ-Shot®, an electronic single shot instrument manufactured by Reflex of Sweden. These measurements have been converted from magnetic to UTM Zone 30 North values. The factor used to convert between the two grids is -5 degrees (CJM, 2014).

14.9.2.1 Rotation from UTM to Local Grid

Data was transformed from the UTM WGS 84 30N grid to a local grid, to orientate the mineralisation in a North-South orientation. This transformation ensured that the resource block model did not require rotating to appropriately honour the dominantly NE geology and mineralisation trend. The data rotation parameters are shown in Table 14-31. A drill hole location plan in the local grid, coloured by year of completion, is presented in Figure 14-31.

Table 14-31: Dynamite Hill Data Rotation Parameters

Grid	Rotation	X (Easting)	Y (Northing)	Z (Elevation)
Original UTM centroid	-37˚around Z	616,000	706,550	0
Local Grid centroid		5,000	20,000	0

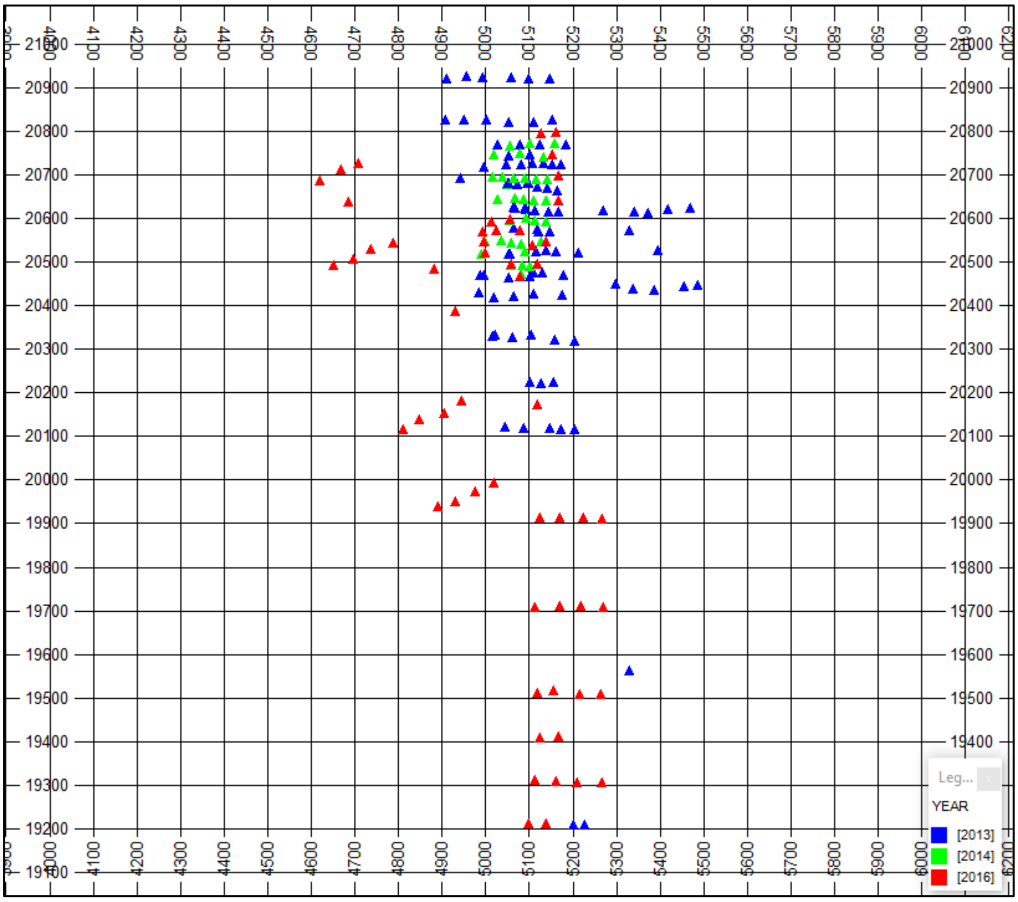
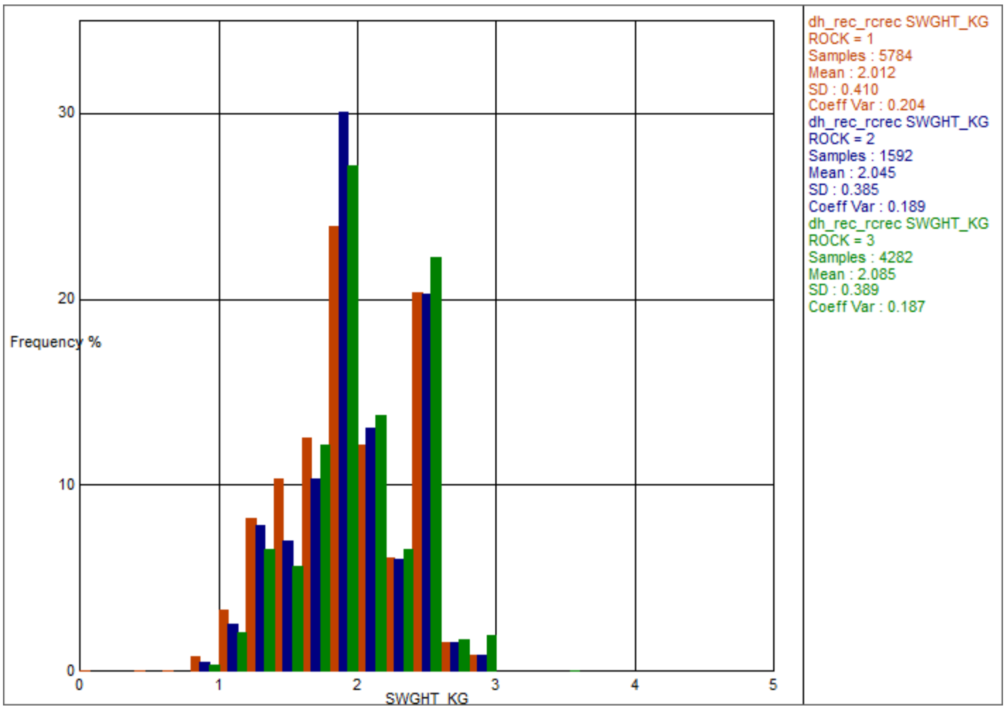


Figure 14-31: Plan view – Dynamite Hill drill hole locations plotted on the Local Grid

14.9.2.2 Drill Sample Recovery and Quality

Sample recovery for RC drilling was noted as very good and averaged approximately 34 kg per metre drilled. Within the Dynamite Hill database, there were 124 RC holes available for review. Analyses of these showed a mean sample weight per rock oxidation state material of 2 kg (Figure 14-32).



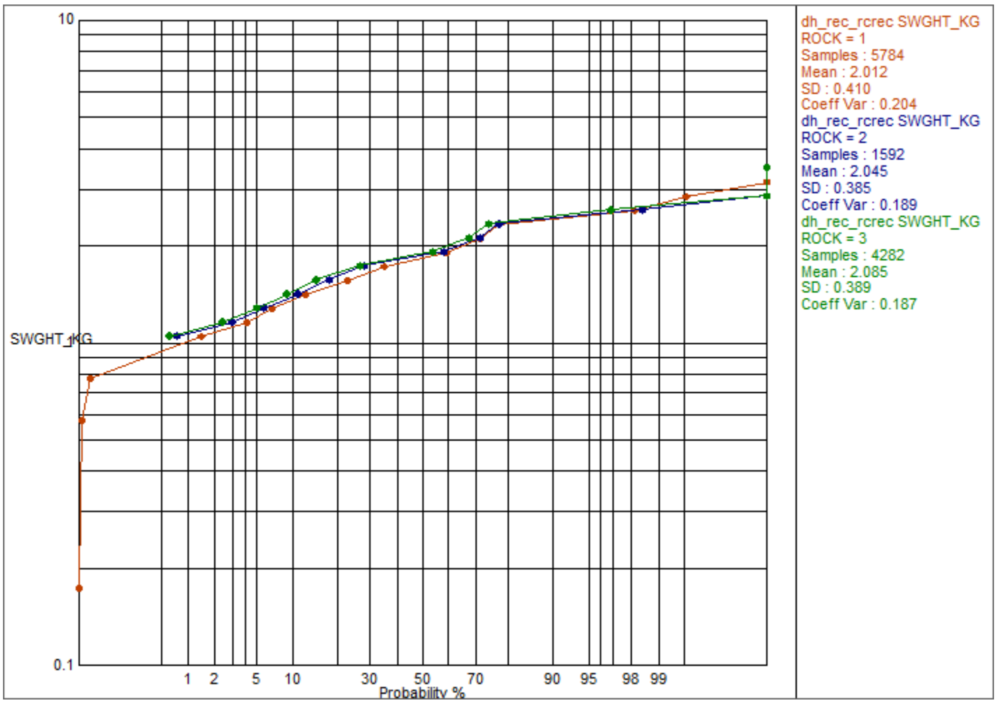


Figure 14-32: Histogram and log probability plots - RC sample weights per rock oxidation state

Oxide (ROCK=1), Transitional (ROCK=2), Fresh (ROCK=3).

There were three DDH holes available for review of the core sample recovery. It should be noted that only two of these (DYDD14-004 and DYDD14-005) were in the database used for the MRE. The third DDH (DYDD14-006) was not in the database received as at 17th January 2017 and was not used in the MRE. However, this drill hole recorded no assay values and as such does not materially impact the MRE.

Sample recovery in DDH was very good in fresh rock averaging 99.6 % (Figure 14-33). Core recovery from the moderate to highly weathered saprolite and highly fractured and brecciated zones averaged 92.8 % (Figure 14-33). Asanko utilises HQ3 drilling to minimise the core loss in the weathered and transition zones.

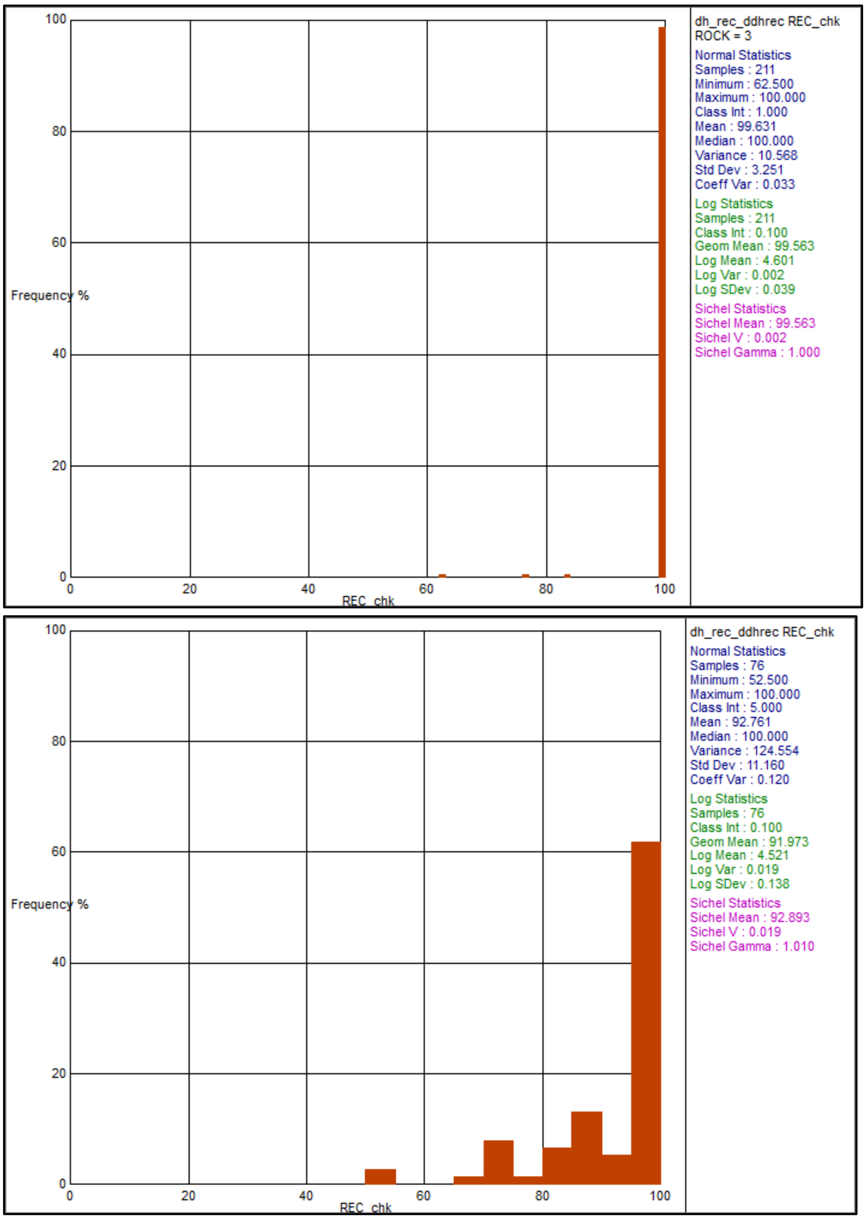


Figure 14-33: Histogram plots - DDH percentage core recovery

Fresh Rock (top) and Transitional Rock + Oxide Rock (bottom).

A brief review of available cut drill core by CSA during the site visit in January 2017, verifies the recoveries noted in the database and indicates reasonable overall drilling sample recovery.

The sampling procedures adopted for drilling are consistent with current industry best practise. Samples collected by diamond coring within the weathered zones are of moderate quality, with the remainder being good quality. Sample recoveries and quality for the RC drilling are high with drilling switching to diamond core once wet samples were observed.

14.9.2.3 Sampling and Logging

The following section was modified after CJM (2014).

RC drill chips were collected as 1 m intervals downhole via a cyclone into PVC bags, and then weighed prior to splitting. The collected samples were riffle split using a three tier Jones riffle splitter. A final sample of approximately 3 kg was collected for submission to the laboratory for analysis. All 1m interval samples were analysed. RC chip trays were systematically compiled and logged with all bulk rejects stored at the Asanko exploration camp in Obotan.

The sampling of the Diamond Drill Hole ("DDH") core was subject to the discretion of the geologist completing the geological logging. Nominally 1m intervals were taken. After marking out the required interval, the core was cut in half by an electric diamond blade core saw. The cut is made 1 cm to the right (looking downhole) of the orientation line with the left side being retained and the other half broken up for assay. In the upper oxide zone, where the core was too friable for diamond saw cutting, the procedure was to dry cut or cleave the core.

Core structure orientations were routinely recorded to assist in determining the controls on mineralisation, in establishing a reliable geological model for resource estimation, and to provide additional geotechnical information to determine likely blast fragmentation and pit stability characteristics.

The core is transferred from the trays and pieced together on a V-rail (angle iron) rack and the orientation line (bottom of hole), determined by the orientation tool recorded during drilling, is drawn along the entire length of the assembled core.

Geotechnical logging has recorded lithology, weathering and oxidation, mineralogy and structure.

The lithology and oxidation codes used for the current MRE are shown in Table 14-32 and Table 14-33.

Table 14-32: Dynamite Hill Database - Logged Lithology Codes

LITH1	LITHCODE	# Intervals	Description
BX	-1	2	Breccia
GR	1	53	Granite
GWKE	2	4,225	Greywacke
LAT	3	299	Laterite
NG	-1	8	No geology (no sample)
NoCore	-1	6	No core
NR	-1	24	No recovery
NS	-1	3	Not sampled
OVB	3	8	Overburden
PHYL	4	1,649	Phyllite
QFP	1	546	Quartz feldspar porphyry
QV	5	32	Quartz veining
QZVN	5	445	Quartz veining
Rd	-1	2	Duricrust/Laterite
Rl	-1	6	Lower Saprolite
Ru	-1	3	Upper Saprolite
SAP	3	4,050	Saprolite
SH	4	126	Shale
SI	4	373	Interbedded Silt/Sand
SR	-1	5	Silica Flooded Unit
SS	2	376	Sandstone
ST	4	26	Siltstone

Table 14-33: Dynamite Hill Database - Logged Oxidation Codes

LITH1OX	OXCODE	# Intervals	Description
	-1	659	Absent
CW	1	488	Completely weathered
FR	3	4,523	Fresh
HW	1	3,760	Highly weathered
MOX	2	68	Moderately oxidised
MW	1	1,784	Moderately weathered
OX	1	4	Oxidised
SW	2	967	Slightly weathered
TRANS	2	4	Transitional
WOX	2	10	Weakly oxidised

14.9.2.4 Data Spacing and Orientation in Relation of Geological Structure

Drilling has been completed on sections which are generally 25m by 25m apart. The drill holes mostly have an azimuth of 50˚ which is nominally perpendicular to the granite and main zone of mineralisation.

14.9.2.5 Data Excluded, Data Storage and Database Closure Date

All data provided in the Dynamite Hill database was used in the updated MRE.

The drill hole data is stored in a proprietary industry standard site based DataShed database. The drilling database used for the current MRE was closed on 17th January 2017.

14.9.2.6 QP Conclusions Regarding Data Quality

Overall, no significant issues were identified in the Dynamite Hill dataset. It is the QP’s opinion that all samples and geological data are fit for use in the MRE.

14.9.2.6.1 Density

Specific gravity measurements for the Dynamite Hill deposit were estimated using the water immersion method. The density is calculated with the following formula:

$$\text{Density } = \frac{\text{Weight in air}}{\text{Weight in air } - \text{ Weight in water}}$$

The In Situ Dry Bulk Densities (“BD”) used by CJM in the 2014 MRE are presented in Table 14-34. These are BD results as determined by SRK in May 2012 (CJM, 2014).

Table 14-34: In Situ Dry Bulk Densities as per CJM MRE (2014)

Oxidation state	Number of samples	In Situ Dry Bulk Density (t/m^3)		
		Minimum	Average	Maximum
Oxide	26	1.48	1.76	2.21
Transitional	7	2.01	2.33	2.61
Fresh	54	2.44	2.73	2.82

Asanko provided CSA with a total of 21 BD samples for review and use in the current MRE. These were flagged by oxidation state. A single measurement of 2.21 t/m^3 within the oxidised material was identified as an outlier and removed from the dataset.

The results are shown in Table 14-35 and Figure 14-34 below.

The average BD value per oxidation state was used in the MRE.

Table 14-35: Dynamite Hill - In Situ Dry Bulk Densities

Oxidation state	Number of samples	In Situ Dry Bulk Density (t/m^3)		
		Minimum	Average	Maximum
Oxide	12	1.50	1.72	1.96
Transitional	3	2.01	2.24	2.41
Fresh	6	2.44	2.64	2.80

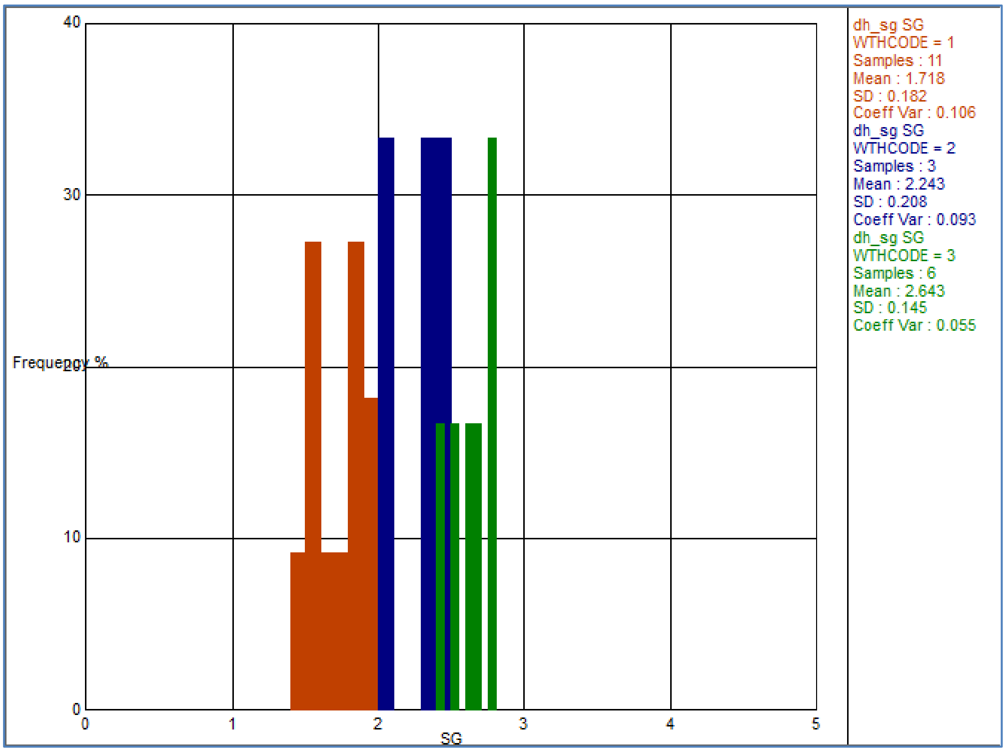


Figure 14-34: Histogram plot - In Situ Dry Bulk Density

Additional dry bulk density data should be collected routinely during grade control and/or mine production and reviewed to build up a useful bulk density database of values that can be used to improve the confidence of the tonnage factors for the Dynamite Hill deposit. The methodology and measurements should be verified and standardised in the MRE.

14.9.3 Geological and Mineralisation Modelling

Asanko geologists created geological and mineralisation surfaces and volumes through cross sectional interpretations and implicit modelling using MicromineTM software. These interpretations were based on logged lithologies, logged oxidation state and chemical Au assays.

The main two modelled sedimentary sequences are an interbedded sandstone and wacke sequence containing thick phyllite interbeds. A granitic intrusion was modelled along the main sedimentary sequence contact and parallels the main Nkran regional shear zone. The lithological codes for building the geological units are shown in Table 14-36, with a 3D view of the modelled units shown in Figure 14-35

Table 14-36: Geological Modelling Lithology Codes

Lithology Code	Modelled Geological Unit
GR/QPF	Granite
GWKE/SS	Sandstone
ST/SI/SH/PHYL	Phyllite

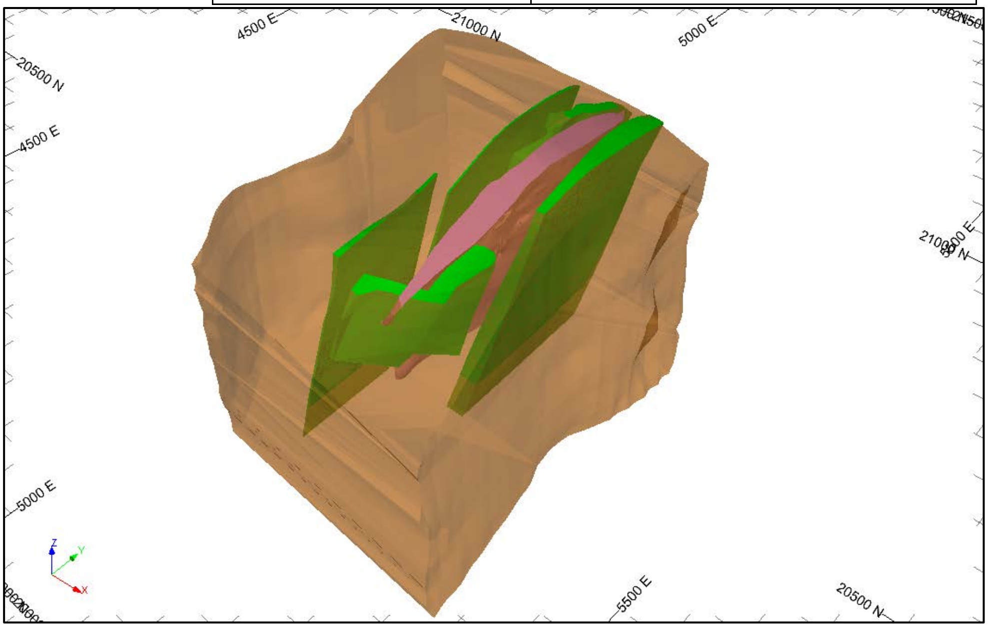


Figure 14-35: 3D view of the Dynamite Hill geological domains (local grid), Sandstone (brown), phyllite (green) and granite (pink)

The modelled weathering profiles comprise a bottom of oxidation ("boco") surface and a top of fresh ("tofr") surface Figure 14-36.

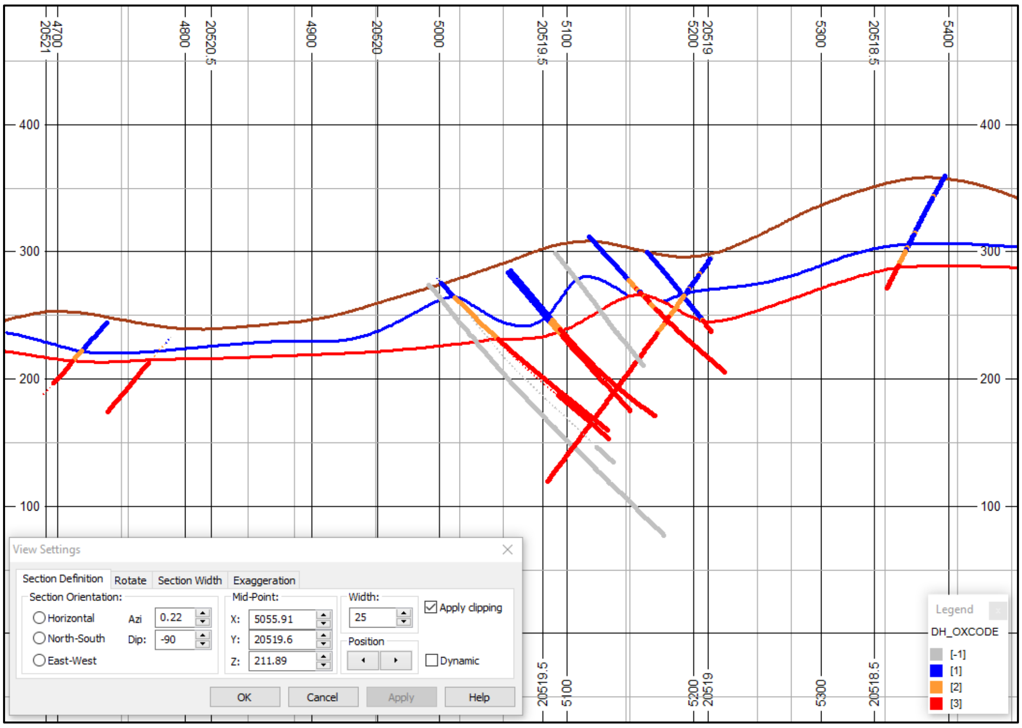


Figure 14-36: Section view of the Dynamite Hill weathering profiles and drill holes (local grid),

Oxide (blue), transitional (orange) and fresh (red).

The mineralisation envelope was delineated using 0.3 g/t Au as a cut-off. Figure 14-37 shows an example of an interpreted cross section with mineralisation and granite. Mineralisation follows the granite interpretation and continuity, but is not totally constrained by it.

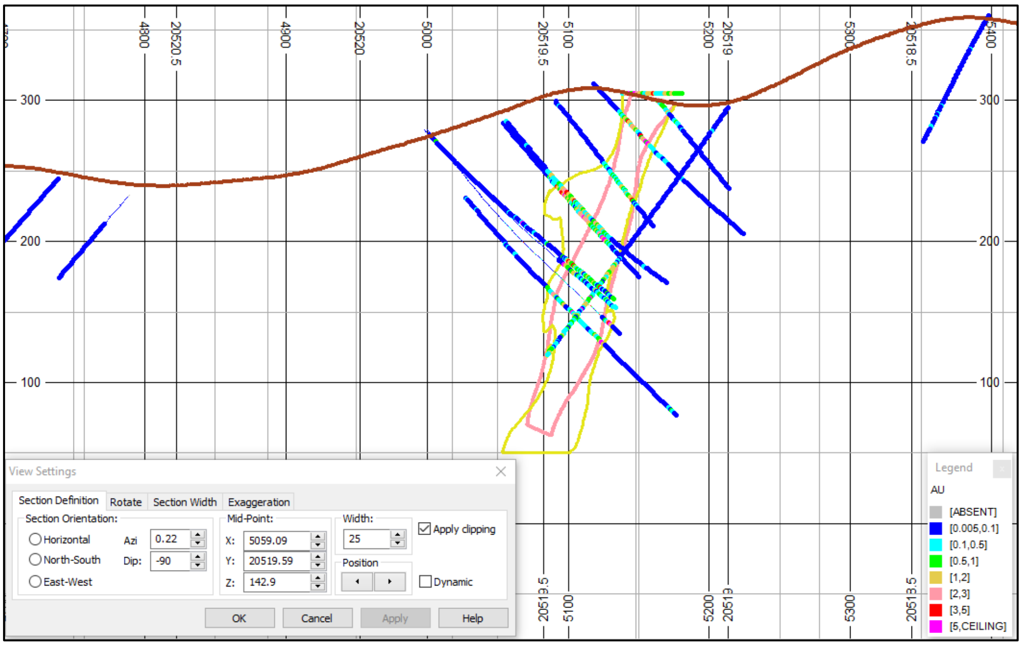


Figure 14-37: Section view of the Dynamite Hill mineralisation and drill holes (local grid), mineralisation shown in yellow, Granite in pink.

14.9.3.1 Statistical Analysis

Before undertaking the estimate, the data was first analysed to understand how the estimate should be accomplished. Drill hole samples were statistically reviewed, and variograms were calculated to determine spatial continuity for Au.

The statistical analysis was carried out by CSA Global using Datamine StudioRMTM, Supervisor v8.4TM and GeoAccess ProfessionalTM software packages.

14.9.3.2 Boundary Analysis

Boundaries are either classified as 'hard' or 'soft'. Where hard boundaries are abrupt, they generally represent a sharp geological contact such as the edge of a quartz vein on its host rocks and where the boundary marks the margin of metal grade. A soft boundary is a gradational one, and represents a gradual reduction in grade, for example as one would find in the alteration zone of a copper porphyry system.

It is important to understand the nature of the boundaries between domains. If domain boundaries are gradational, then data from the adjacent domains should be used during estimation (soft boundary). If there are abrupt boundaries then estimation should be restricted to only use the data within that domain (hard boundary).

Contact analysis for Au g/t between the modelled mineralisation and waste were carried out to assess the nature of the domain boundaries by graphing the average grade with increasing distance from the domain boundary. The average grades can be calculated by incrementally expanding the wire frames, or manually by coding the samples based on distance from the domain contact, as was done in this instance. The contact analysis results for the Dynamite Hill deposit is shown in Figure

14-38. Based on the results of the boundary analysis between mineralisation and waste, the boundary was interpreted to be hard.

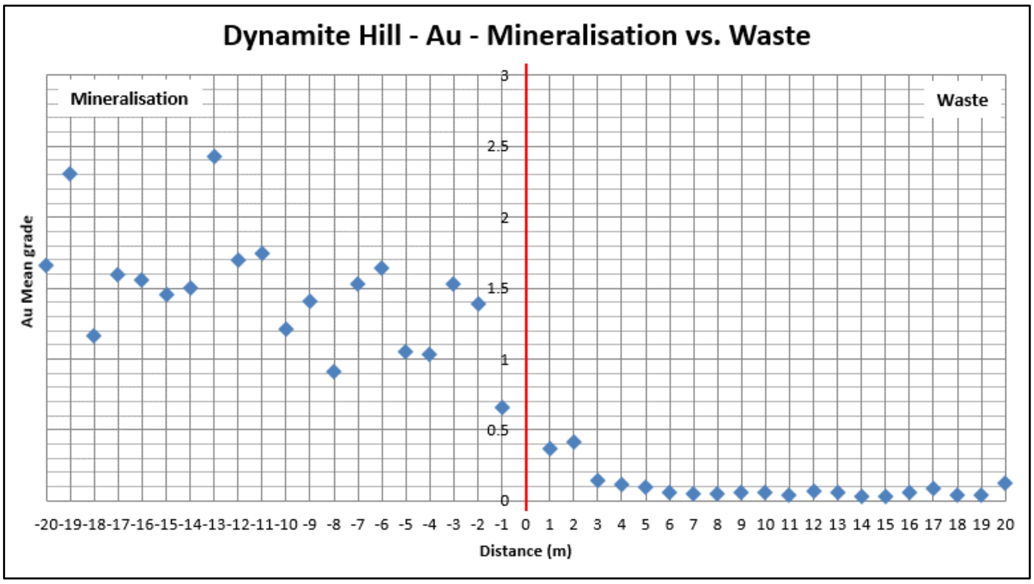


Figure 14-38: Mineralised boundary test graph for Dynamite Hill – Au g/t mineralisation versus waste

Additional contact analysis was carried out to assess the nature of the domain boundaries within the mineralised volume between the weathering profiles. These are shown in Figure 14-39 and Figure 14-40. Based on the results of the boundary analysis for these profiles, the boundaries were interpreted to be soft.

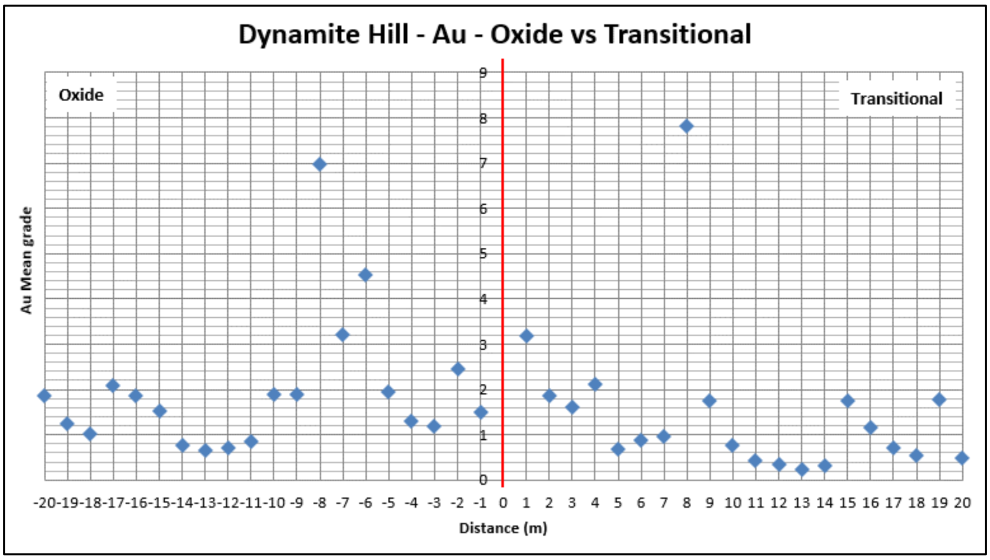


Figure 14-39: Mineralised boundary test graph for Dynamite Hill – Au g/t Oxide versus Transitional

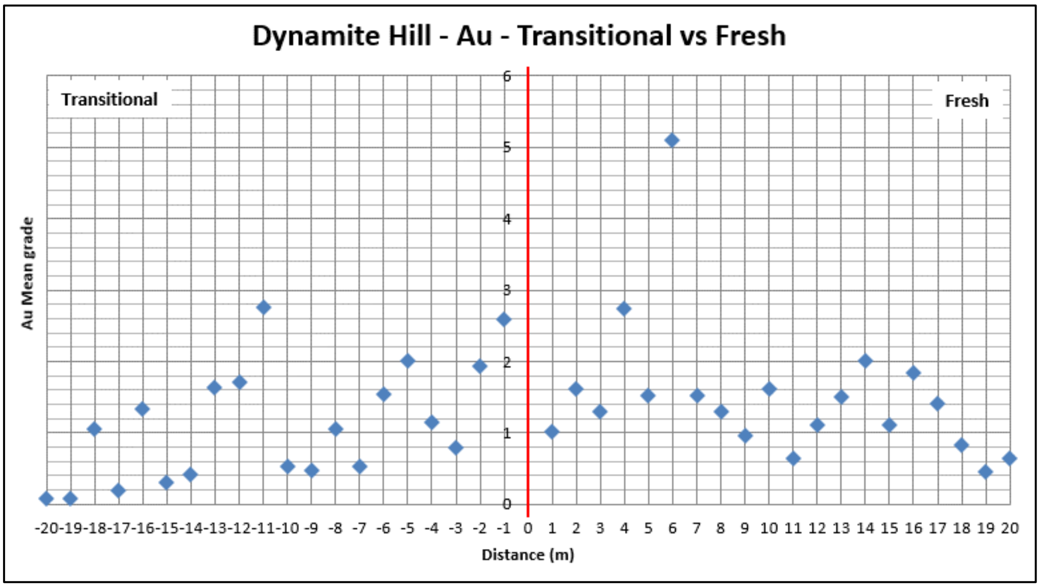


Figure 14-40: Mineralised boundary test graph for Dynamite Hill – Au g/t Transitional versus Fresh

Based on visual review and the results of the boundary analysis, a single mineralisation domain was selected for estimation with a hard boundary against the waste.

14.9.3.3 Naïve Statistics

Drill hole coding is a standard procedure which ensures that the correct samples are used in statistical and geostatistical analyses, and grade interpolation. The mineralised envelope was used to select drill hole samples. The samples were coded by geological domain and oxidation state.

A summary of the domain codes, used to distinguish the data during geostatistical analysis and estimation, is shown Table 14-37 below.

Table 14-37: Data Field Flagging and Description

Field	Code	Description
ROCK	1	Oxide
	2	Transitional
	3	Fresh
	9	Other
GEOL	100	Sandstone
	200	Phyllite
	300	Granite
	999	Other
MINZON	10	Mineralised
	99	Waste
DOMAIN	1000	Mineralised oxide
	2000	Mineralised transitional
	3000	Mineralised fresh
	9999	Other

The naïve statistics, per MINZON, are given in Table 14-38 and shown in Figure 14-41.There are some isolated high values within the waste domain, however, the sample populations are clearly distinct.

Table 14-38: Naïve Statistics - MINZON

Parameter	MINZON 10 (mineralisation)	MINZON 99 (waste)
Number	2,953	15,855
Minimum	0.005	0.005
Maximum	107.28	12.64
Mean	1.48	0.06
Standard Deviation	4.55	0.31
Coefficient of Variation	3.09	5.55

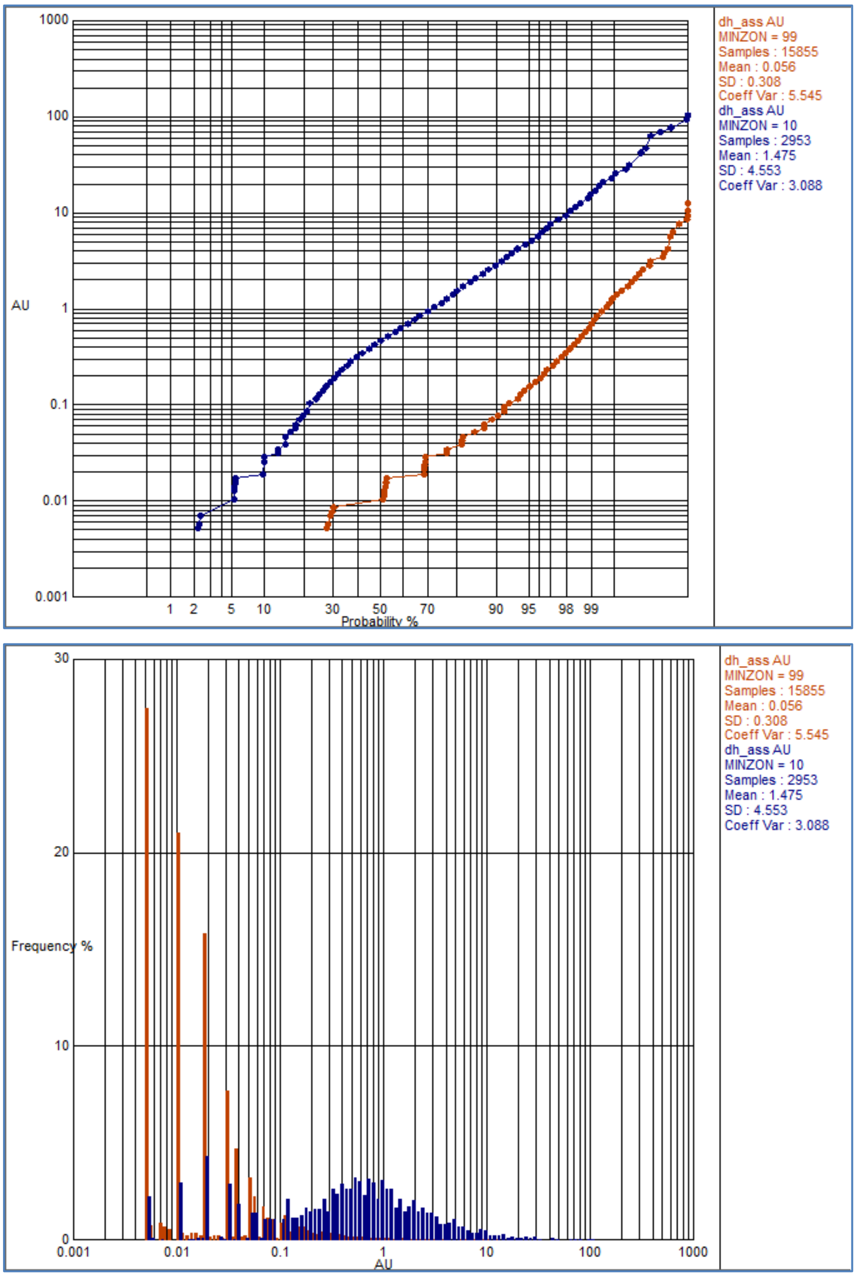


Figure 14-41: Log probability plot (left) and Log Histogram (right) overlays of Mineralisation (MINZON=10) versus Waste (MINZON=99)

14.9.3.4 Compositing

Assays that fall within the modelled mineralisation envelope were down-hole composited to 1m prior to statistical review, top-cutting, variography and grade estimation. Sampling was undertaken at other sampling lengths, with the dominant sampling lengths being 1m and 2m. The dominant as well as the mean length within the mineralisation envelope is 1m. As such, compositing to 1m was selected as the most appropriate for use in estimation Figure 14-42.

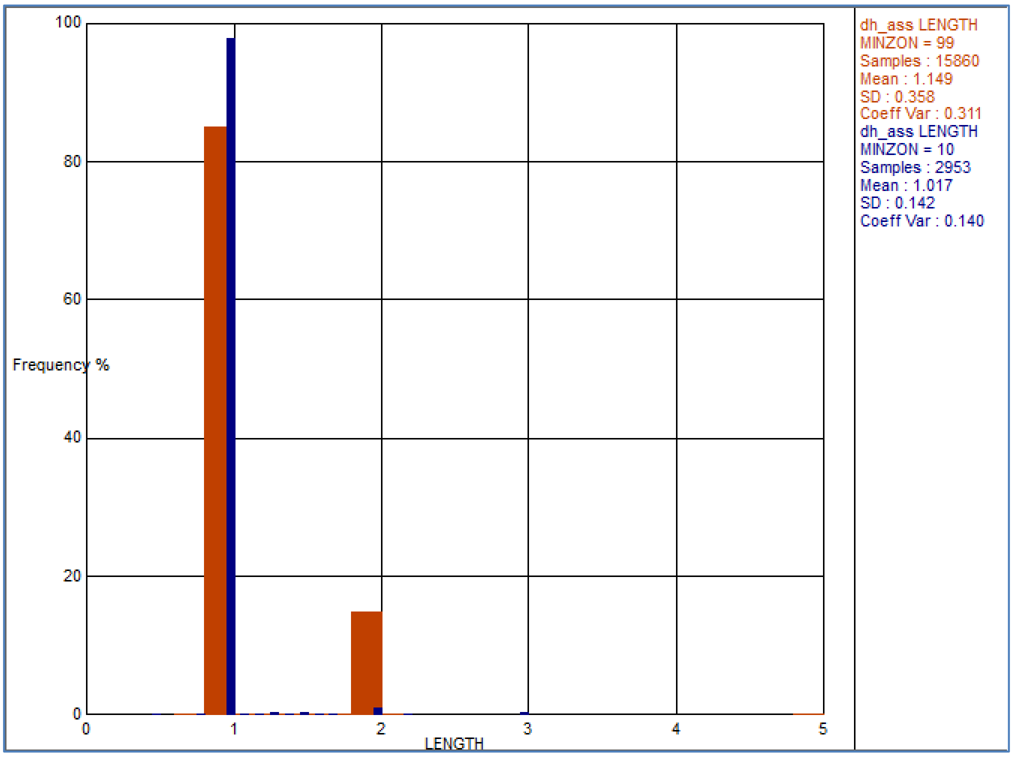


Figure 14-42: Histogram overlay of sample lengths for Mineralisation (MINZON=10) versus Waste (MINZON=99)

Composites that were less than 50% (<0.5m) of the composite length were excluded from the geostatistical analysis and the estimate. This will limit any potential bias in the sample support during kriging. Three samples within MINZON 10 were removed.

A total of 3,005 composites were used in the statistical analysis and resource estimation. The descriptive analyses for the estimation domain, ESTZON 10 (which is equal to MINZON 10) are shown in Table 14-39 and Figure 14-43.

Table 14-39: Composite Statistics – ESTZON 10

Parameter	ESTZON 10 (Mineralisation)
Number	3005
Minimum	0.005
Maximum	107.28
Mean	1.47
Standard Deviation	4.52
Coefficient of Variation	3.07

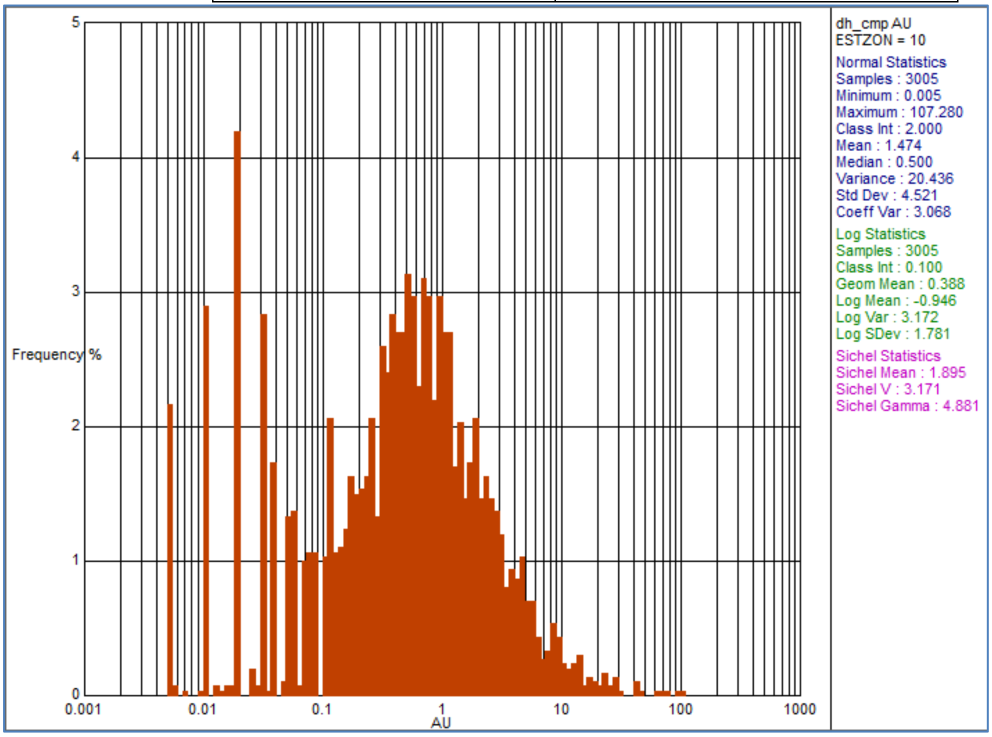


Figure 14-43: Log Histogram of ESTZON 10

14.9.3.5 Top-cut Analysis

Grade cutting (top cutting) is generally applied to data used for grade estimation in order to reduce the local high grading effect of anomalous high grade samples in the grade estimate. In cases where individual samples would unduly influence the values of surrounding model cells, without the support of other high-grade samples, top cuts are applied. These top-cut are quantified according to the statistical distribution of the sample population.

Cutting strategy was applied based on the following:

- Skewness of the data
- Probability plots
- Spatial position of extreme grades

Histograms and probability plots were reviewed for Au within the estimation domain to determine the top-cut. The uncut and top-cut statistics are shown in Table 14-40. A total of nine samples were greater than the top-cut value of 30 g/t and were reset to the top-cut value.

Table 14-40: Top-cut Statistics – ESTZON 10

Parameter	Uncut	Top-Cut
Number	3,005	3,005
Minimum	0.005	0.005
Maximum	107.28	30.00
Mean	1.47	1.37
Standard Deviation	4.52	3.06
Coefficient of Variation	3.07	2.23

The associated log histogram plots for the uncut and top-cut Au in the estimation domain are shown in Figure 14-44.

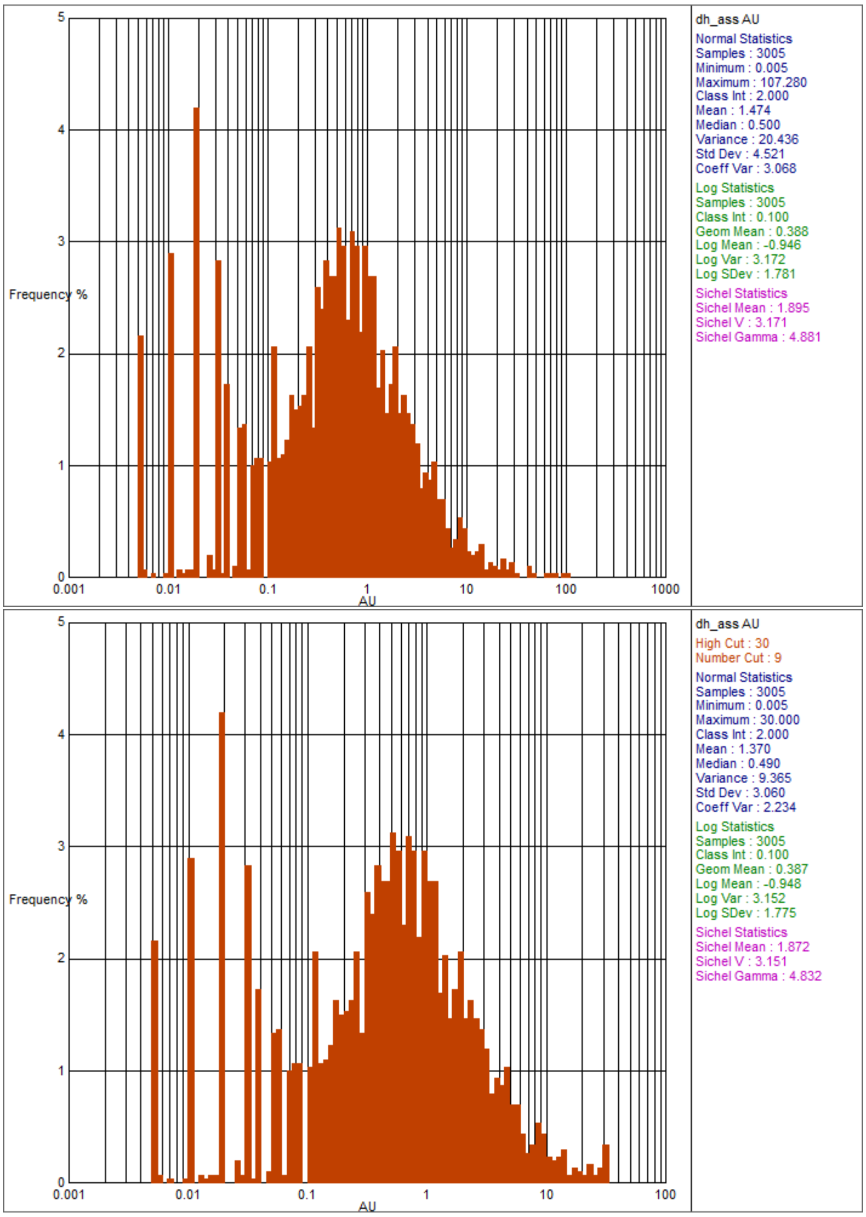


Figure 14-44: Log Histogram Uncut (top) and Top-Cut (bottom) – ESTZON 10

14.9.3.6 Variography

The variograms were modelled for Au on 1.0m composites within the estimation domain. Nuggets were obtained from the downhole variograms, where the lag was set equal to the composite length of 1.0m. Normal scores transform was used for modelling the variograms.

The semi-variograms were well structured, with a high nugget and moderate ranges. The variogram was back transformed prior to estimation and is presented in Figure 14-45. The variogram parameters are detailed in Table 14-40.

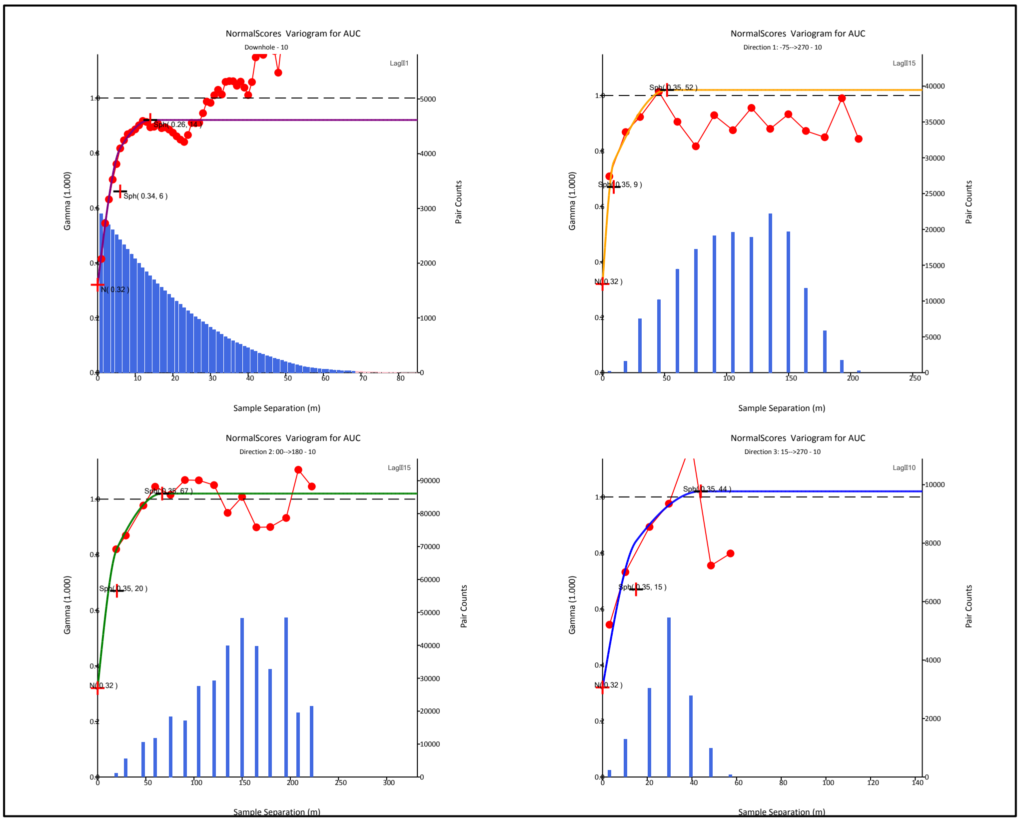


Figure 14-45: Variogram used for Au g/t estimation – ESTZON 10

Table 14-40: Variogram Parameters – ESTZON 10

Grade	Datamine Orientation (ZXZ)	Nugget	Structure 1		Structure 2	
			Partial Sill	Range	Partia l Sill	Range
Au	-90˚	0.49	0.32	9	0.18	52
	75˚			20		67
	-90˚			15		44

14.9.4 Block Model and Grade Estimation

14.9.4.1 Summary

Estimation of Au grade was carried out using OK into parent cell panels. Grade was estimated into all estimation domain blocks (ESTZON 10), using available data within the mineralisation domain. The parameters used for grade estimation are summarised in Table 14-41. These are discussed in the sections below.

The MRE was completed by CSA using the Datamine StudioRMTM software package.

Table 14-41: Dynamite Hill Estimation Parameters Summary

Attribute	Description
Parent cells (block sizes X, Y, Z)	15 mN x 15 mE x 3 mRL
Minimum number of samples	8 (4 on search pass 3)
Maximum number of samples	26 (16 on search pass 3)
Search ranges	45 m x 60 m x 20 m
Search Range multiplier	Pass 2 – 2x; Pass 3 – 3x
Discretisation	5 x 5 x 3
Maxkey	4
Estimation method	Ordinary Kriging

14.9.4.1.1 Block Modelling

The model was cut to below the topographic surface. A waste model prototype with parent cells of 15 mN by 15 mE by 3 mRL, with sub-celling to 5 mN by 5 mE by 3 mRL, was created. A mineralisation model prototype with parent cells of 5 mN by 5 mE by 3 mRL, with no sub-celling, was also created. The mineralisation model prototype was joined to the waste model prototype before grade estimation. This methodology was used to ensure that the volume of the mineralisation and waste zones are adequately represented and that there are no blocks larger than 5 mN by 5 mE by 3 mRL within the mineralisation domain. However, the grade estimation was done into the 15 mN by 15 mE by 3 mRL parent cells of the joined master model prototype. The waste and grade model prototypes parameters, including cell dimensions and model extents, are shown in Table 14-42.

Panel sizes for grade estimation (15 mN by 15 mE by 3 mRL) were based on the following:

- Results of KNA
- The density of the drilling grids
- The geometry of the mineralisation
- The mining parameters

Table 14-42: Dynamite Hill – Block Model Dimensions

Prototype	Axis	Origin	Model Extent (m)	# Blocks	Block dimension (m)
Waste	Easting (X)	4,480	1,200	80	15
	Northing (Y)	20,000	1,200	80	15
	Elevation (Z)	2	375	125	3
Mineralisation	Easting (X)	4,480	1,200	240	5
	Northing (Y)	20,000	1,200	240	5
	Elevation (Z)	2	375	125	3

14.9.5 Kriging Neighbourhood Analysis ("KNA")

KNA on the 1m composites was used to optimise the parent cell sizes and to determine the optimal theoretical estimation and search parameters during kriging.

The following was reviewed for each of the variables per selected domain:

- Slope and KE statistics for a well-informed block for different block sizes
- On choosing a block size (15m x 15m x 3m, X x Y x Z), optimum minimum and maximum samples were chosen. The maximum was set at the lowest number of samples from which consistently good slopes and KE could be derived. The minimum was defined as the lowest minimum from which moderate to good statistics could be derived
- On choosing the minimum/maximum samples, search ellipse ranges were defined. The quality of the statistics were least sensitive to this parameter. The ranges chosen approximated the ranges of the first structure of the variogram
- Negative weights were reviewed at each stage to ensure the parameters chosen were not leading to excessive negative weights
- Discretisation was defined at 5 x 5 x 3 (X x Y x Z)
- Maximum number of samples allowed per each individual drill hole, per estimate, was set to 4

The KNA results show that the search parameters and block size selected are suitable for use in the MRE and adequately take drill spacing, geology and practicality into account. The plots with the selected estimation parameters are shown in Figure 14-46.

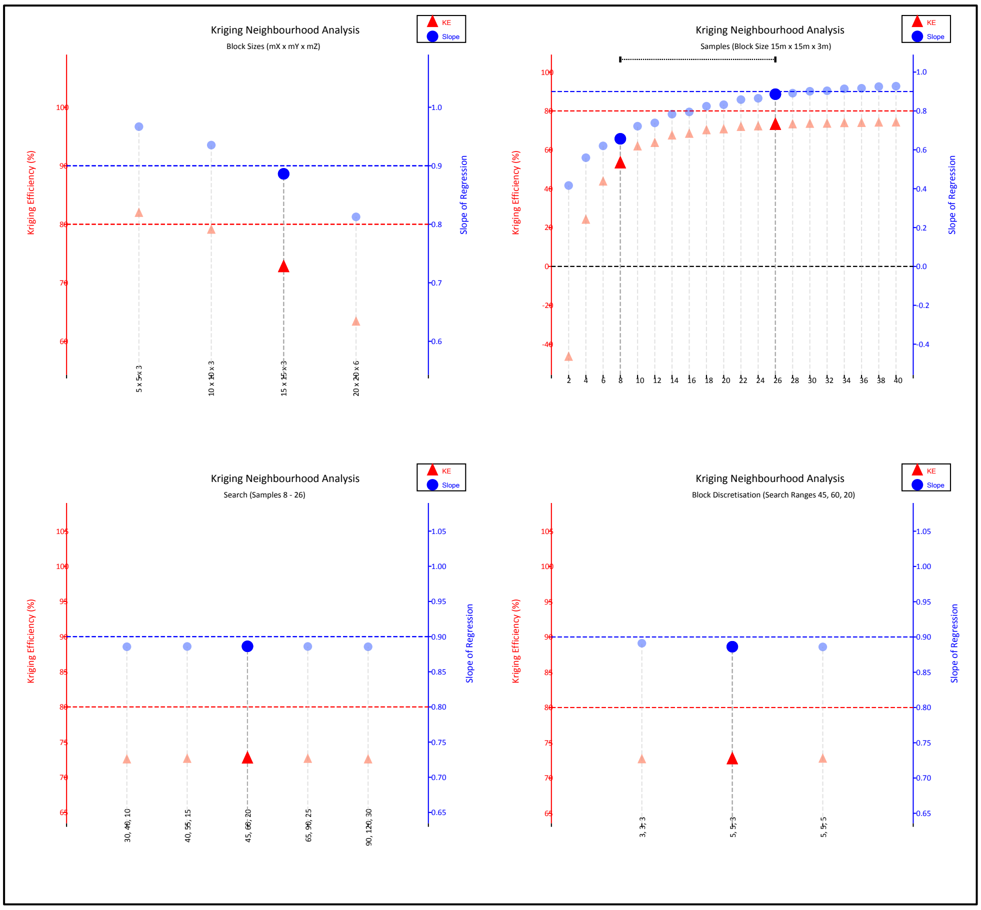


Figure 14-46: KNA block size, samples, search and discretisation results

The number of composites used for the Au grade estimation is presented in Table 14-43. The modelled variogram parameters together with the selected estimation panel size and number of samples was used to determine the appropriate search ellipse for the primary search pass. This is also presented in Table 14-43.

Table 14-43: Number of composites and search neighbourhood parameters for Au

Rotation (Datamine ZXZ)			Search Ranges (SVOL1)			Search Ranges (SVOL2)			Search Ranges (SVOL3)			Composites	
Strike	**Dip**	**Plunge**	**1**	**2**	**3**	**1**	**2**	**3**	**1**	**2**	**3**	**Min**	**Max**
-90˚	75˚	-90˚	45	60	20	90	120	40	135	180	60	8	26

14.9.5.1 Grade Estimation

Estimation of Au grade was carried out using OK into parent cell panels. Zonal control with a hard boundary between mineralisation and waste was used during the grade estimation. The estimation domain was assigned a unique ESTZON number, corresponding to the ESTZON field in the input composite data. The grade estimation domain for the Dynamite Hill estimate is ESTZON = 10.

A three-phased search pass was applied and the orientation of the search ellipsoid was aligned to the modelled variography. This process involves the estimation being performed three times, where two expansion factors are used. During each individual estimation run this factor increases the size of the search ellipse used to select samples. This method ensures that blocks which are not estimated and populated with a grade value in the first run, are populated during one of the subsequent runs (Pass 1 76%, Pass 2 20% and Pass 3 4%).

The mineralised areas were estimated using dynamic anisotropy. This process allows the rotation angles for the search ellipsoid to be defined individually for each cell in the models, so that the search ellipsoid is aligned with the axes of mineralisation. This therefore requires the rotation angles to be interpolated into the model cells, which in turn requires a set of angles as the input data file for interpolation. The dip and dip direction of the major axis of anisotropy were defined by digitising strings in section perpendicular to the strike of the mineralisation. These strings were converted to points that contained the true dip and dip direction of the mineralisation and stratigraphy (fields SANGLE1_F and SANGLE2_F in the search parameter files).

14.9.5.2 Validation

Validation of the block model was completed by comparing input and output means. Several techniques were used for the validation. These included visual validation of block grades, global grade comparisons and swath plots.

14.9.5.3 Visual Validation

The block model was visually reviewed section by section to ensure that the grade tenor of the input data was reflected in the block model (example shown in Figure 14-47. Figure 14-48 shows a 3D view of the block model for Au g/t. Generally, the estimates compare well with the input data. The grades in the composites align with the corresponding grades in the block models.

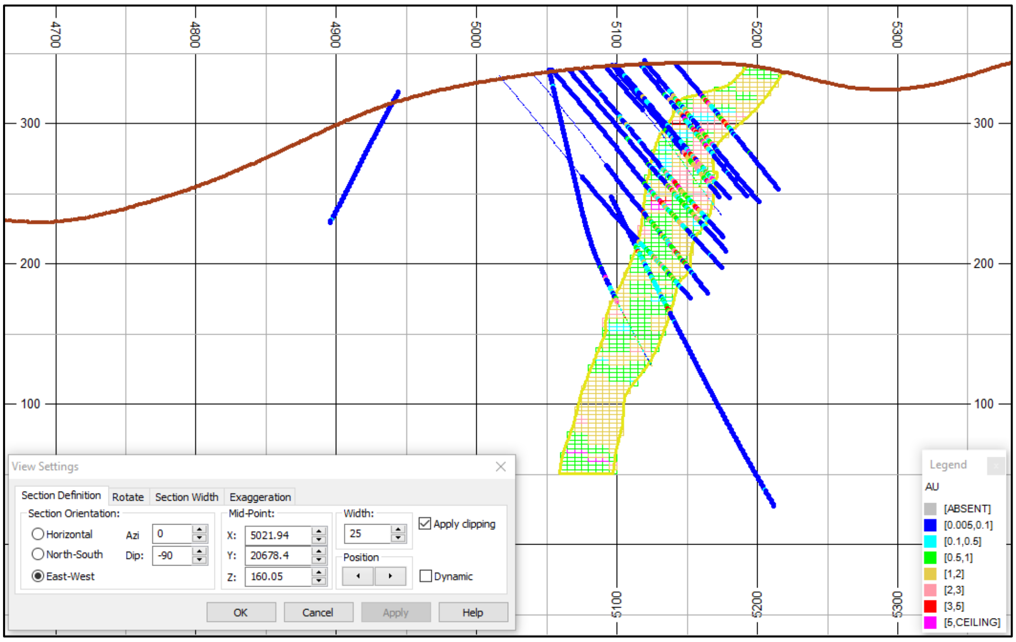


Figure 14-47: Section view – Grade Model and composites

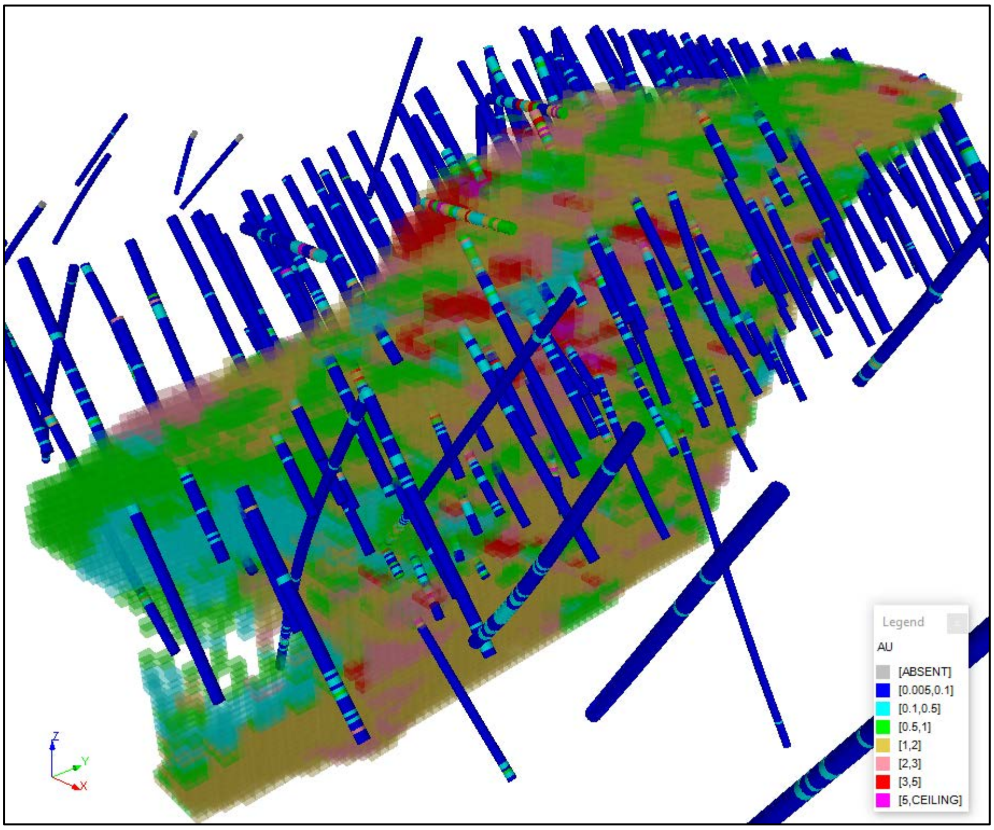


Figure 14-48: 3D view – Grade Model and composites

14.9.6 Statistical Validation

14.9.6.1 Declustering

Irregular sampling of a deposit, most commonly through infill drilling or drilling in multiple orientations, causes clustering. Clustering results in a disproportionate distribution grades (usually high grades from the infill drilling) in the dataset used for statistical analysis. Mixed populations in the histogram can create a bias when comparing the drill hole sample distribution with the block model distribution (which is declustered) and distort the calculated mean grades and variance.

Different ways of declustering data each give different results. These include interactive filtering, polygonal declustering, nearest neighbour declustering and cell-weighted declustering.

The method used for geostatistical analysis and validation for the current MRE update is cell-weighted declustering, since all samples are considered when determining the average. This method involves placing a grid of cells over the data. Each cell that contains at least one sample is assigned a weight of one. That weight of one is distributed evenly between the samples within each cell.

The OK grade estimation process if a very efficient way of data clustering, therefore declustering before grade estimation is not necessary. Declustering of the input data does give a good indication of the global mean. It is used in the validation of the estimate (comparison of the means). Declustering was applied to remove any bias due to drill spacing prior to validation. The declustering parameters are presented in Table 14-44.

Table 14-44: Declustering Parameters

Cell Size (m)			Block Model Origin		
X	Y	Z	X	Y	Z
15	15	3	4,480	20,000	2

14.9.6.2 Results

The global statistics of Au g/t were reviewed and the results are reported below in Table 14-45.

All estimated block grades are included. The mean grades in the estimated model block parent cells were compared to the raw, as well as the declustered, top-cut composite data.

Generally, the model validates well, showing <0.5% difference between the declustered composites and the block estimates. This is well within expected parameters.

Table 14-45: Declustered Mean Grade Comparison for Au g/t

	Field	Count	Min.	Max.	Mean	Standard deviation	Variance	Coefficient of Variance
Composites Naïve	AUC	3,005	0.005	30.00	1.37	3.06	9.37	2.23
Composites Declustered	AUC	3,005	0.005	30.00	1.34	2.99	8.91	2.23
Model	AU	36,146	0.10	7.21	1.34	0.78	0.61	0.58
Difference [(Composite Declustered Grade – Model Grade)/Model Grade]					+0.2%			

14.9.6.3 *Swath Plots*

Swath plots were created as part of the validation process, by comparing the model parent block grades and input composites (declustered and top cut) in spatial increments. These plots display northing, easting and elevation slices throughout the deposit. Figure 14-49.

The plots show that the distribution of block grades honours the distribution of input composite grades. There is a minor degree of smoothing evident, which is to be expected from the estimation method used, with block grades showing lower overall variance. The general trend of the composite grades is reflected in the block model.

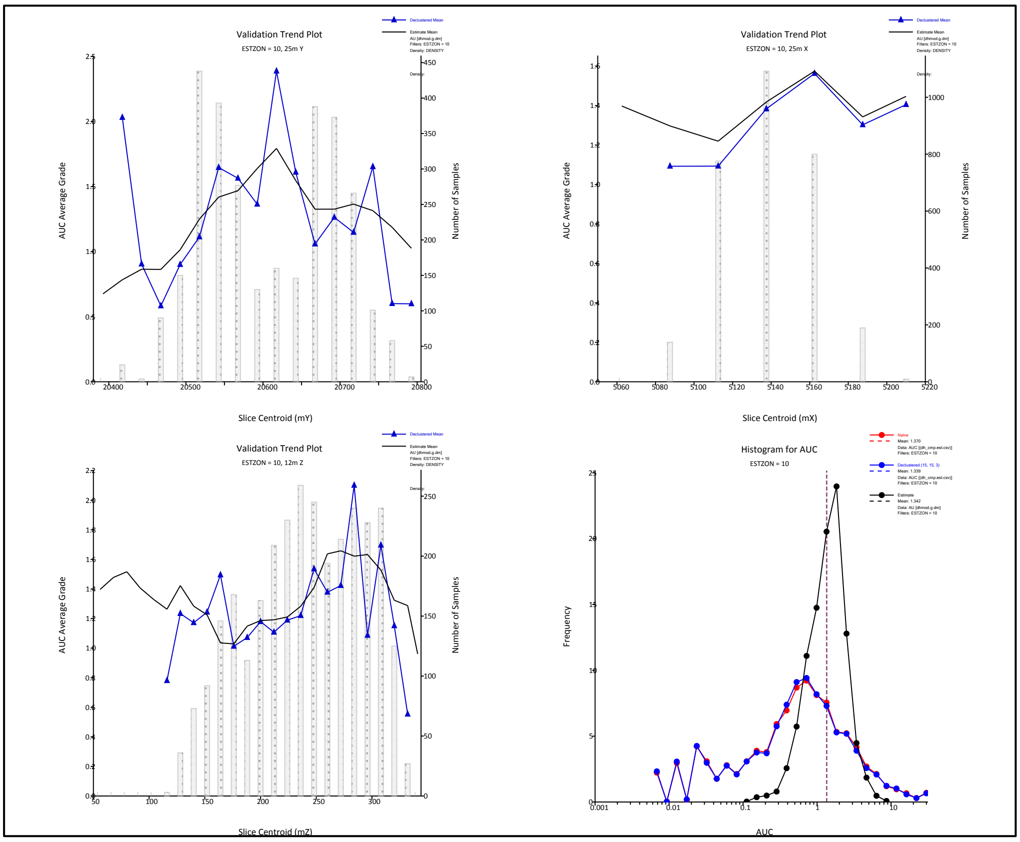


Figure 14-49: Swath plot by 25m easting, 25m northing, and 12m bench for Au g/t

14.9.7 Mineral Resource Classification

The MR has been classified as Indicated Mineral Resources and Inferred Mineral Resources based on recommended NI 43-101 guidelines. The classification level is based upon an assessment of geological understanding of the deposit, geological and mineralisation continuity, drill hole spacing, quality control results, search and interpolation parameters and an analysis of available density information.

The Dynamite Hill deposit shows good continuity of mineralisation within well defined geological constraints. Drill holes are located at a nominal spacing of 25m on 25m sections extending out to 50m on the peripheries of the deposit. The drill spacing is sufficient to allow the geology and mineralisation zones to be modelled into coherent wireframes for each domain. Reasonable consistency is evident in the orientations, thickness and grades of the mineralised zone.

Validation of the historical drill holes, particularly in relation to the exact collar locations and assay results, and the availability of QA/QC information, has allowed for the classification of IMR.

A summary of the classification codes applied in the model are shown in Table 14-46, Figure 14-50 and Figure 14-51 show the classified block model in plan and 3D view.

Table 14-46: RESCAT Field and Description

RESCAT	Description
1	Measured Mineral Resource
2	Indicated Mineral Resource
3	Inferred Mineral Resource
9	Unclassified – Estimated Material outside a Whittle $1500 gold price and 2016 operating costs and recoveries pit shell, as well as all waste material not estimated

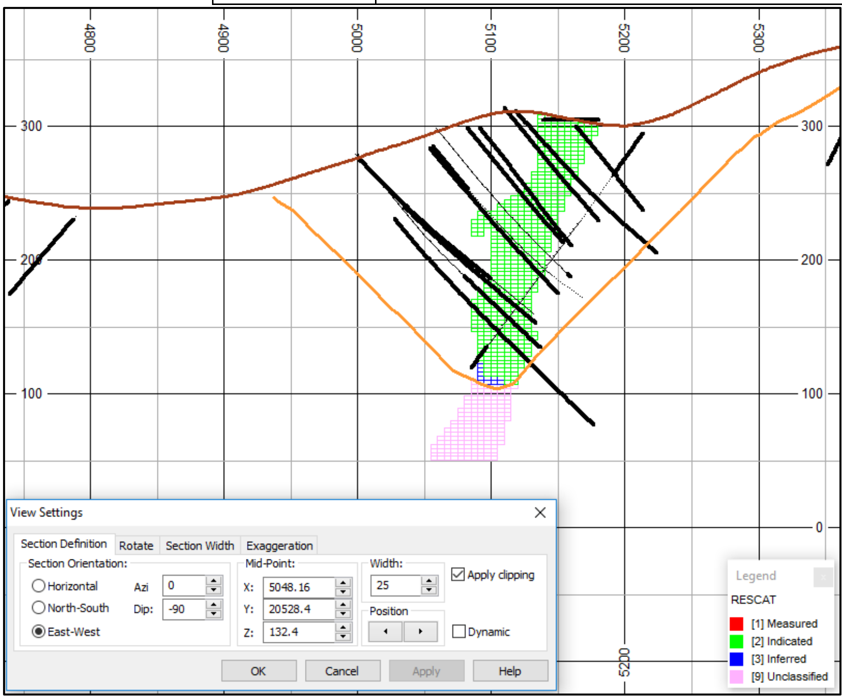


Figure 14-50: Section view of classified grade model. The Whittle pit shell (orange) is based on a $1500/oz Au gold price and 2016 operating costs and recoveries.

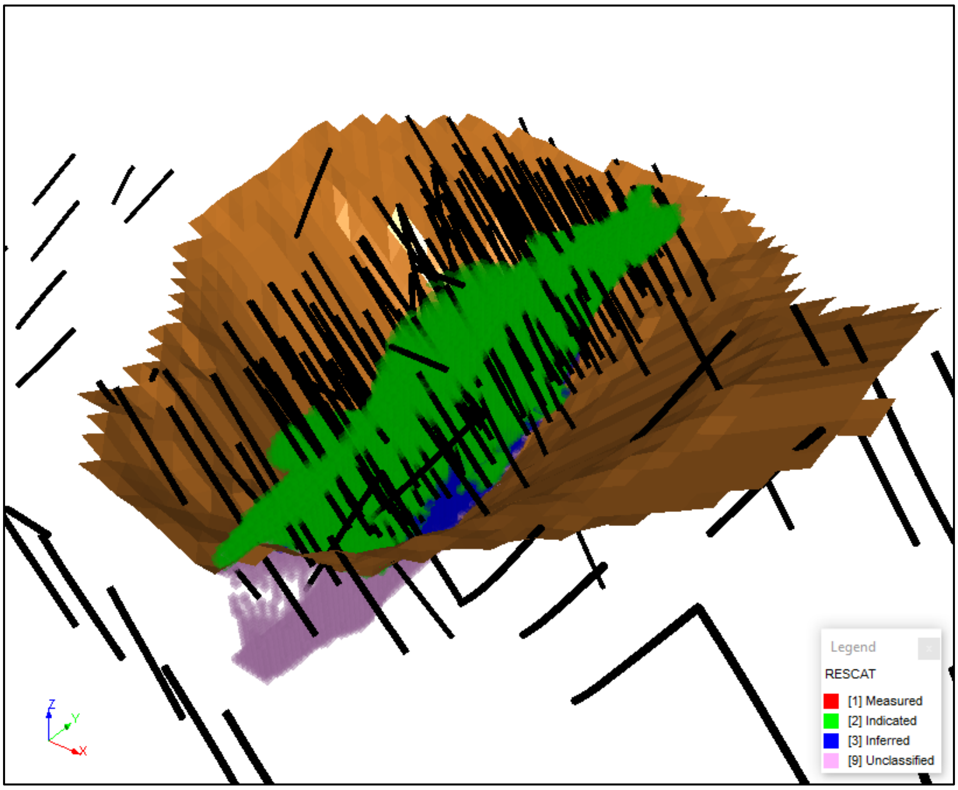


Figure 14-51: 3D view of classified grade model. The Whittle pit shell (brown) is based on a $1500/oz gold price and 2016 operating costs and recoveries

14.9.8 Mineral Resource Statement

The total MRE is shown in Figure 14-46 as at 23rd January 2017. The MRE compiled by CSA has been classified and is reported as IMR (Measured, Indicated and Inferred Mineral Resources) based on recommended NI 43-101 guidelines.

Table 14-47: Dynamite Hill Deposit – MRE reported at a 0.5 g/t Au cut-off 23rd January 2017

Dynamite Hill	0.5 g/t cutoff constrained to $1500/oz Au shell				
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Measured	Oxide	0.00	-	-	-
	Transition	0.00	-	-	-
	Fresh	0.00	-	-	-
Total Measured		**0.00**	**-**	**-**	**-**
Indicated	Oxide	0.83	1.51	40,560	0.041
	Transition	0.27	1.48	13,056	0.013
	Fresh	2.30	1.46	108,202	0.108
Total Indicated		**3.41**	**1.48**	**161,818**	**0.162**
Total Measured and Indicated		**3.41**	**1.48**	**161,818**	**0.162**
Inferred	Oxide	0.00	-	-	-
	Transition	0.00	-	-	-
	Fresh	0.21	1.60	10,665	0.011
Total Inferred		**0.21**	**1.60**	**10,665**	**0.011**

Notes:

- Reported from 3D block model with Au grade estimated by Ordinary Kriging with 15m x 15m x 3m panel size.
- Based on data available as at 17th January, 2017. Reported using a 0.5 g/t gold cut-off.
- The MRE is reported within a Whittle $1500 gold price and 2016 operating costs and recoveries pit shell.
- Due to rounding, the totals may not represent the sum of all components.
- The MR's are stated as in situ dry tonnes. All figures are in metric tonnes.

The grade versus tonnage curves for the Indicated and Inferred Mineral Resource categories are shown in Figure 14-52.

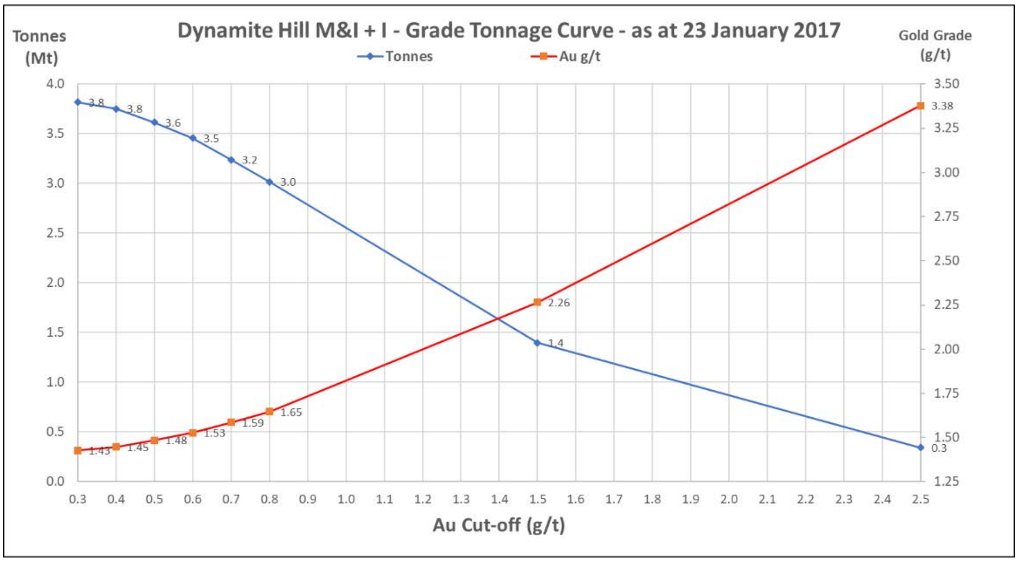


Figure 14-52: Dynamite Hill Grade Tonnage Curve

14.9.9 Comparison with Previous Estimates

The January 2017 MRE is reported at various cut-off in Table 14-48. The Dynamite Hill deposit was previously estimated and reported in April 2014 at various cut-offs by CJM Table 14-49.

Since then the following changes have occurred:

- Infill drilling completed in 2016
- Changes to the mineralisation, geology and weathering models as a resulting of the infill drilling
- Changes in the Bulk Density applied to the MRE
- Change in estimation methodology

Table 14-48: Dynamite Hill Deposit – MRE reported at various Au g/t cut-off's, January 2017 (CSA)

Dynamite Hill M&I Resources				
Cut-off Au g/t	Mt	Au g/t	Au Oz	Au Moz
-	4.20	1.34	181,569	0.18
0.30	3.60	1.42	164,448	0.16
0.40	3.54	1.44	163,732	0.16
0.50	3.41	1.48	161,818	0.16
0.60	3.26	1.52	159,071	0.16
0.70	3.04	1.58	154,630	0.15
0.80	2.82	1.64	149,353	0.15
1.50	1.30	2.28	94,897	0.09
2.50	0.33	3.39	35,499	0.04

Table 14-49: Dynamite Hill Deposit – MRE reported at various Au g/t cut-offs, April 2014 (CJM)

Dynamite Hill M&I Resources					
Cut-off Au g/t	Tonnes	Mt	Au g/t	Au Oz	Au Moz
0.40	2,080,000	2.08	1.72	115,022	0.12
0.60	2,010,000	2.01	1.76	113,736	0.11
0.80	1,840,000	1.84	1.86	110,033	0.11
1.00	1,560,000	1.56	2.03	101,815	0.10

At a comparable 0.8 g/t Au cut-off, changes in the total Measured and Indicated Mineral Resources ("M&I") are an increase of 1.19 Mt, equivalent to a 65% increase in tonnage. Due to additional drilling, more confidence could be applied to the model and material previously classified as IMR has been upgraded to IMR. No Measured MR material was included in the January 2017 MRE.

At a 0.8 g/t cut-off Global M&I Au grade decreased from 1.86 g/t to 1.65 g/t. The decrease in grade can be attributed to both an increase in the modelled mineralisation volume, which includes more waste intervals, as well as a change in the estimation methodology. CJM previously used IK, with a 0.3 g/t Au cut-off to flag data as mineralised or not, to delineate the mineralised portion within the main litho-structural domains.

IMR material increased by 0.50 Mt, an increase of 97%. Additional IMR material is due to more confidence added to the model because of infill drilling, and re-modelling of the mineralisation volume.

14.9.10 Conclusions and Recommendations

14.9.10.1 Conclusions

CSA Global considers that data collection techniques are consistent with industry good practise and suitable for use in the preparation of a MRE to be reported in accordance with NI 43-101. Quality Control data supports the integrity of the analytical data which has been utilised.

A 3D block model representing the geology and mineralisation has been created by Asanko geologists using MicromineTM software. High-quality RC and DDH samples were used to interpolate grades into blocks using OK. The block model was validated visually and statistically.

The total drilling available for the geological model and MRE update was 185 holes for 21,572m. Since the previous MRE (CJM, April 2014), 56 additional RC holes for 5,176m were added to the database.

2,953 samples were flagged within the mineralised volume and composited down hole into 1m lengths. The resultant 3,005 composite samples were used in the estimate.

Additional In Situ Dry Bulk Density ("BD") analysis is required. 21 samples were available for review and the default BD's applied to the MRE are as follows: Oxide = 1.72 t/m^3, transitional = 2.24 t/m^3 and fresh = 2.64 t/m^3.

Following contact analysis, a single mineralisation domain was used for all geostatistical analysis. A variogram was modelled for Au on 1m top-cut composites.

Grade was estimated into parent blocks of 15m by 15m by 3m (X by Y by Z) using OK, controlled by Dynamic Anisotropy ("DA").

Grade estimates were validated against drill data. There is good correlation between the input composites and output model for the estimated Au grade. Generally, the model grade trends follow the pattern of the drill samples grades, with reasonable levels of smoothing of the higher and lower grades.

The Dynamite Hill MRE satisfies the requirements for Indicated and IMR categories as embodied in the NI 43-101 Canadian National Instrument for the reporting of Mineral Resources and Reserve.

The MRE indicates reasonable prospects for economic extraction. At a comparable 0.8 g/t Au cut-off, changes in the total Measured and Indicated Mineral Resources ("M&I") are an increase of 1.19 Mt, a 65% increase in tonnage. Due to additional drilling, more confidence could be applied to the model and material previously classified as Inferred Mineral Resource has been upgraded to IMR. No Measured MR material was included in the January 2017 MRE.

Global M&I Au grade decreased from 1.86 g/t to 1.65 g/t. The decrease in grade can be attributed to both an increase in the modelled mineralisation volume, which includes more waste intervals, as well as a change in the estimation methodology. CJM previously used IK, with a 0.3 g/t Au cut-off to flag data as mineralised, or not, to delineate the mineralised portion within the main litho-structural domains. The IK method will increase grade by minimising internal waste intervals.

IMR material increased by 0.50 Mt, an increase of 97%. Additional Inferred Mineral Resource material is due to more confidence added to the model as a result of infill drilling, and remodelling of the mineralisation volume.

14.9.10.2 Recommendations

CSA recommends the following actions are completed prior to completing MRE updates in the future:

- Additional BD data should be collected regularly from DDH core and/or open pit production and reviewed regularly to build up a useful bulk density database of values that can be used to determine the tonnage factors for the Dynamite Hill deposit. Methodology and measurements should be verified and standardised in the resource model
- The current level of understanding of the Au distribution and geological controls are sufficient for mine planning purposes. CSA recommends that instead of additional infill drilling to upgrade IMR to Measured MR's, grade control drilling should be sufficient to delineated blast and dig lines during open cast mining
- The resource is open down dip. CSA recommends additional drilling for resource delineation with depth to allow Inferred Mineral Resources to be considered for an IMR classification level. A drill spacing of about 25m Z (down dip) is recommended to allow the classification of Inferred Mineral Resources.
- Preliminary geotechnical assessment should be completed to develop a work programme that will enable ground support design.

14.10 Nkran Extension MRE (CJM)

Nkran Extension is a zone of mineralisation directly NE of the main Nkran pit locating along the Nkran shear structure. Mineralisation was initially detected during sterilisation drilling for the TDF. A strike length of approximately 900m was subsequently drilled by RC heel-toe on 40m line spacing to delineate zones that can be mined by open cast methods. Three zones were delineated that attracted whittle pits, but the site preparation and water plus infringement on the current tailings facility footprint ruled out two. The north-eastern pit (#3) is the subject of this MRE, and is scheduled to be drilled out in more detail Q1 and Q2 2017.

14.10.1 Weathering

The weathering profile was also used to code composites. The profile was modelled from drill data and comprised weathered oxide, transition material and fresh units. In general, the weathering surfaces are broadly parallel to the topographical profile Figure 14-53, although weathering tends to be deeper within zones of mineralisation and tends to parallel the footwall to the mineralisation where the footwall approaches the surface.

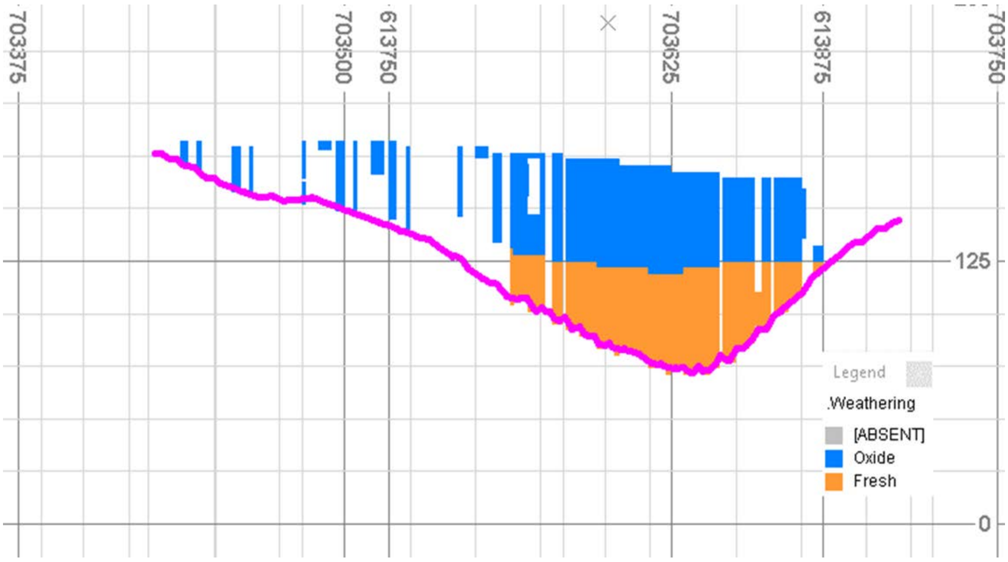


Figure 14-53: Nkran Extension Weathering profile and Pit shell

14.10.2 Domains

Based up upon the litho-structural work conducted, mineralisation was constrained to 3 broad / major domains all located within sandstone. (Figure 14-54). These were numbered 1000, 2000 and 3000. Figure 14-55 shows the search ellipse orientation overlaid on the Geological wireframes.

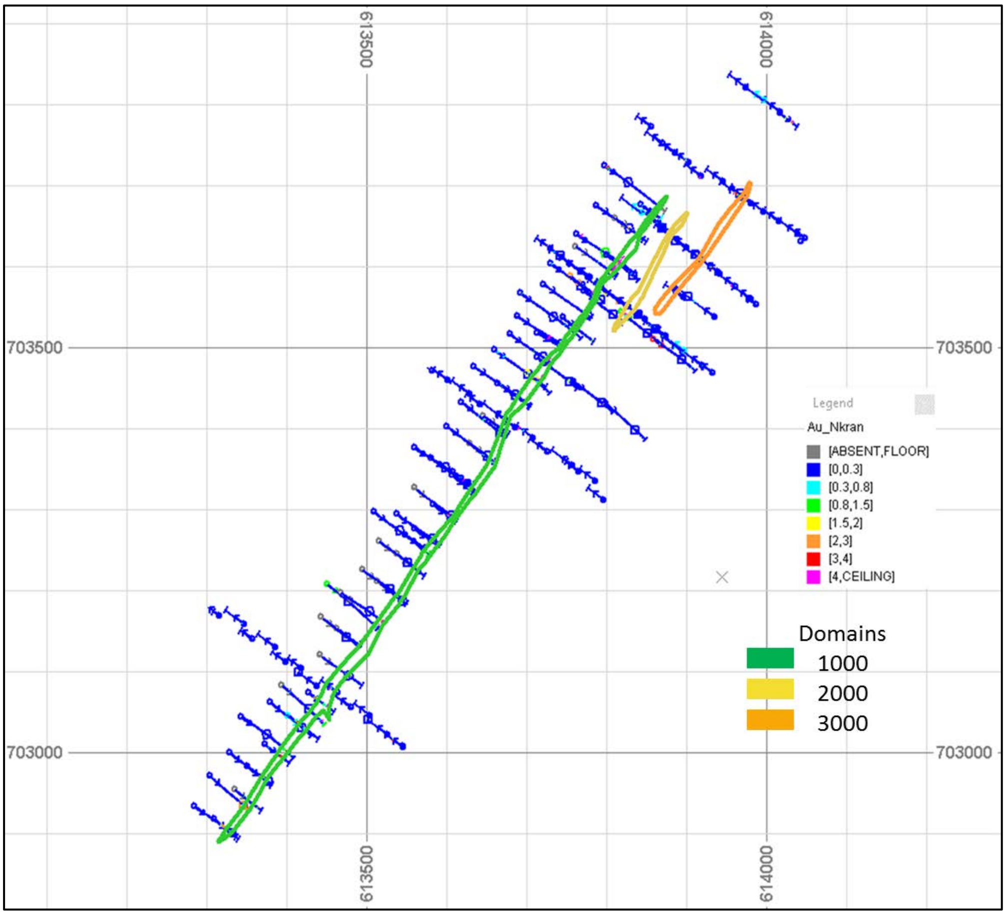


Figure 14-54: Plan view showing Drill hole layout and Geological Domains

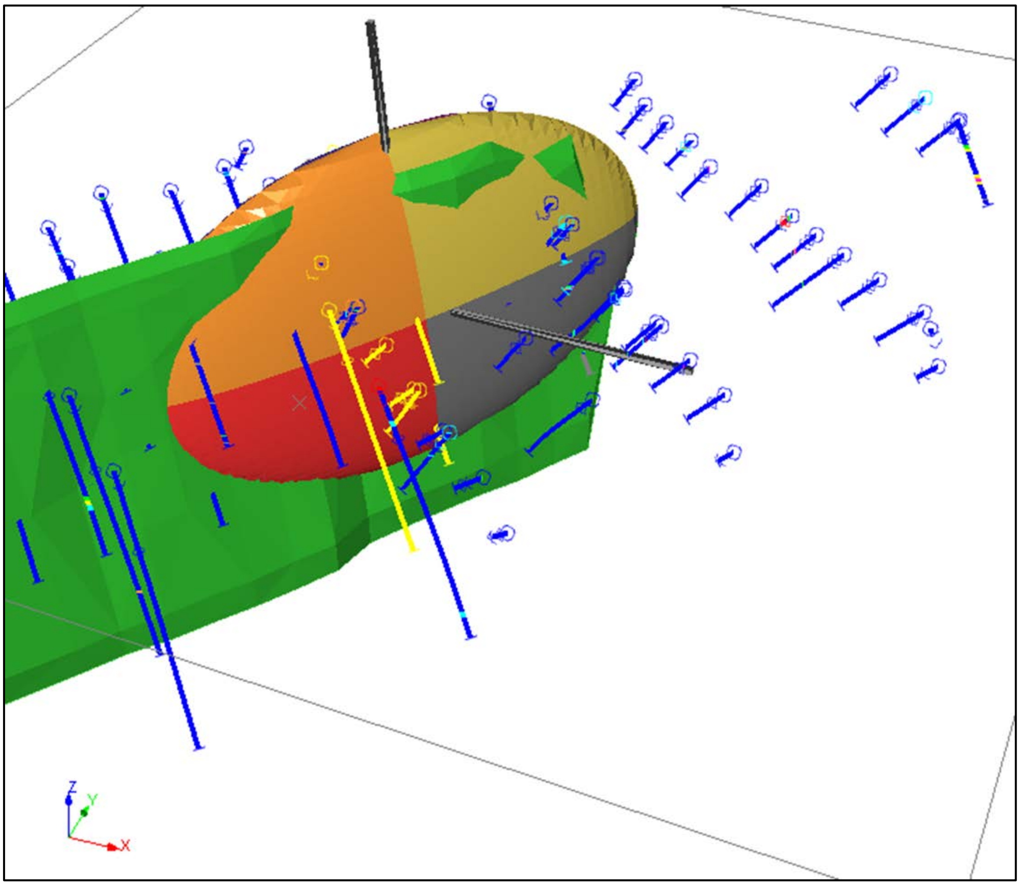


Figure 14-55: Nkran Extension Search Ellipse and Wireframes Oblique view

14.10.3 Nkran Extension Compositing

The de-surveyed drill holes were composited within DatamineTM on a 1m composite length. Figure 14-56 supports this approach and shows the drill hole sample lengths, where the majority of the sampling lengths are close to 1m.

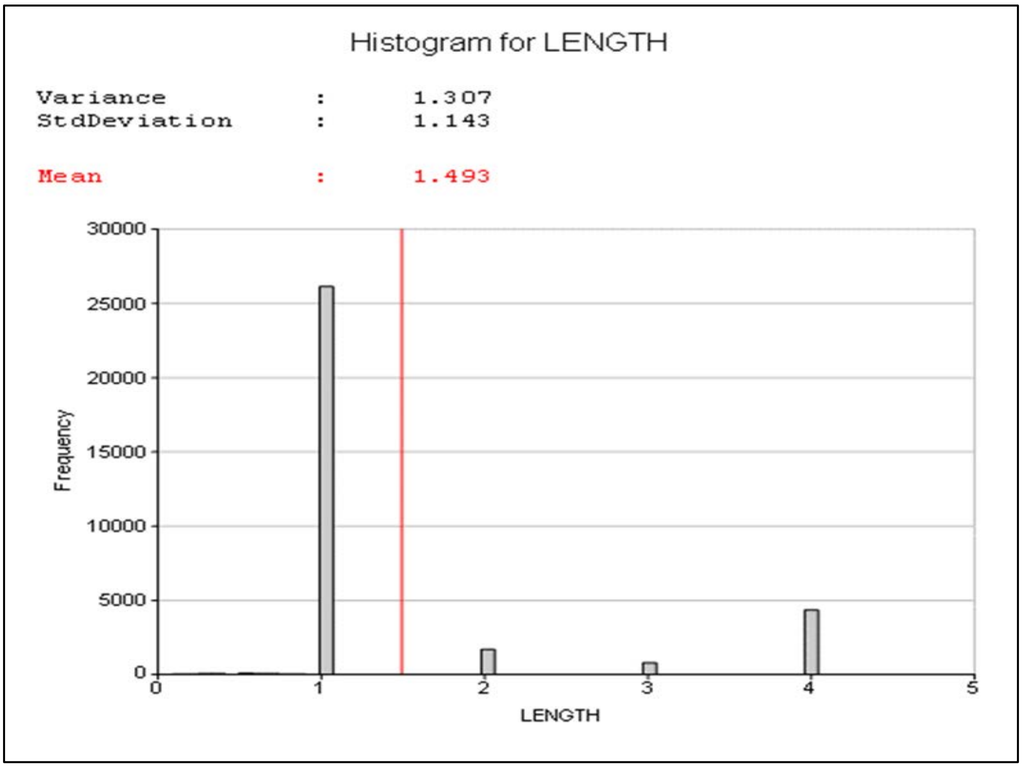


Figure 14-56: Histogram showing Sampling lengths for Nkran Extension

14.10.4 Nkran Extension Statistical Analysis

Statistical analyses are shown in Table 14-50.

Table 14-50: Nkran Extension Descriptive Statistics

Domain	Parameter	Num Records	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
1000	AU	659	0.01	55.00	1.36	16.76	4.09	3.01
2000	AU	91	0.01	8.66	0.67	1.63	1.28	1.90
3000	AU	69	0.01	5.67	0.67	1.43	1.20	1.80

14.10.5 Nkran Extension Outlier Analysis

Outlier analysis was undertaken using histograms and probability plots. The outliers were determined for both the variography and the kriging stages. For variography the outliers are cut at a specified value, but for kriging the outliers are capped at a specified value. Hence, all available data is used in the kriging of the data.

The histograms and probability plots indicate that the populations of gold grades are close to log normal, which is typical of many gold deposits. The co-efficients of variation ('CV') are moderately high, but typical for the Asankrangwa belt types of deposits.

Table 14-51: Nkran Extension- Kriging and Variogram Capping Values per Domain

Domain	Commodity	Capping g/t
D1000	AU	20
D2000	AU	15
D3000	AU	15

14.10.6 Nkran Extension Variography Parameters

Table 14-52: Nkran Extension Variography Parameters

Domain		Sill	Nugget%	Sill 1%	Range1m	Range2m	Range3m	Sill 2%	Range1m	Range2m	Range3m
D1000	AU	5.00	19.97	55.00	42	63	8	100	83	125	20
D2000	AU	5.10	21.57	55.90	42	63	8	100	83	125	20
D3000	AU	5.05	20.79	55.47	42	63	8	100	83	125	20

14.10.7 Nkran Extension Estimation Methodology

The Nkran Extension estimation was conducted on a block size of 5 x 5 x 3m (X, Y and Z) OK was completed using a discretisation of 5 x 5 x 3 (X & Y & Z). Table 14-50 and Table 14-51 show the search parameters used in the estimation.

Table 14-53: Nkran Extension Search Ranges and Angles

Domain		Sdist 1	Sdist 2	Sdist 3	Sangle 1	Sangle 2	Sangle 3	Saxis 1	Saxis 2	Saxis 3
D1000	AU	50	50	10	31	-80	5	3	2	3
D2000	AU	25	25	10	36	-84	5	3	2	3
D3000	AU	50	50	10	37	-88	5	3	2	3

Table 14-54: Nkran Extension Number of Samples in Search

Domain		Min num 1	Max num 1	Svolfac 2	Min num 2	Max num 2	Svolfac 3	Min num 3	Max num 3	Max Key
D1000	AU	4	20	2	4	40	5	1	20	5
D2000	AU	4	20	2	4	40	5	1	20	5
D3000	AU	4	20	2	4	40	5	1	20	5

Figure 14-57 shows a transverse section through the block model with the Au grades as estimated.

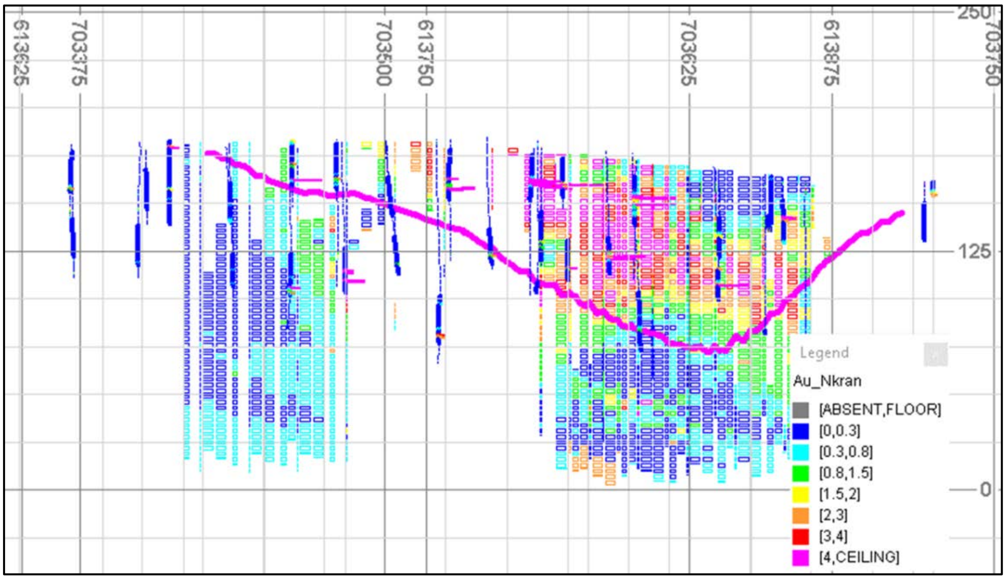


Figure 14-57: Nkran Extension- Block model transverse section

14.11 Nkran Extension Mineral Resource Statement (CJM)

The Nkran Extension resource was estimated for this project by CJM. This Resource is stated within a US$ 1,500/oz Au pitshell at a cut-off Au grade of 0.5 g/t. No Measured resources were detailed.

Table 14-55 shows the total MR sub-divided into Indicated and Inferred classes.

Table 14-55: Nkran Extension Mine Mineral Resources at 0.5g/t Au cut-off grade, as at December 2016

Nkran Extension 0.5 g./t cut-off constrained to $1500/oz Au Shell					
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Indicated	Oxide	0.13	2.80	11,357	0.011
	Transition	0	0	-	0
	Fresh	0.06	2.50	4,958	0.005
Total Indicated		0.19	2.70	16,316	0.016
Total measured and Indicated		0.19	2.70	16,316	0.016
Inferred	Oxide	0	0	-	0
	Transition	0	0	-	0
	Fresh	0.01	1.02	254	0.0003
Total Inferred		0.01	1.02	254	0.0003

Notes:

- Columns may not add up due to rounding.
- All figures are in metric tonnes
- The MR's are stated as in situ tonnes.
- Individual Densities was used per mineral zone.
- The tonnages and contents are stated as 100%, which means no attributable portions are stated in the Table.

Table 14-56 shows the total Mineral Resource sub-divided into Measured, Indicated and Inferred; at various Au cut-off grades.

Table 14-56: Nkran Extension MR's at Different Cut-off Grades, as December 2016

Resource Category	Cut-off Grade g/t	Tonnes Mt	Gold Grade g/t	Gold Ounces Moz
Indicated	0	0.37	1.50	0.018
	0.3	0.23	2.33	0.018
	0.4	0.22	2.48	0.017
	0.5	0.20	2.61	0.017
	0.6	0.19	2.72	0.017
	0.7	0.19	2.82	0.017
	0.8	0.18	2.91	0.017
	0.9	0.17	2.99	0.016
	1.0	0.17	3.06	0.016
	1.5	0.13	3.55	0.015
Inferred	0	0.03	0.78	0.001
	0.3	0.02	1.02	0.001
	0.4	0.02	1.07	0.001
	0.5	0.02	1.12	0.001
	0.6	0.01	1.20	0.001
	0.7	0.01	1.28	0.001
	0.8	0.01	1.36	0.000
	0.9	0.01	1.43	0.000
	1.0	0.01	1.51	0.000
	1.5	0.00	2.20	0.000

Figure 14-58 shows the Nkran Extension block resource classification. The Resource is also reported within a US$1,500/oz Au pit shell as depicted in Figure 14-59.

Previous Resources for the Nkran Extension were not reported within a pit shell.

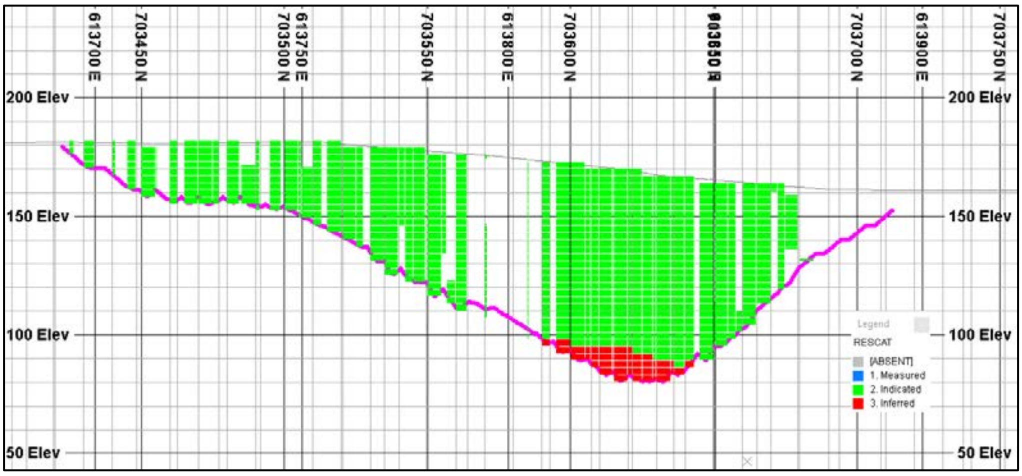


Figure 14-58: Nkran Extension Resource Classification Block model- Longitudinal section

14.11.1 Grade Tonnage Curve

The Nkran Extension pit #3 Grade – Tonnage Curve is shown below in Figure 14-58.

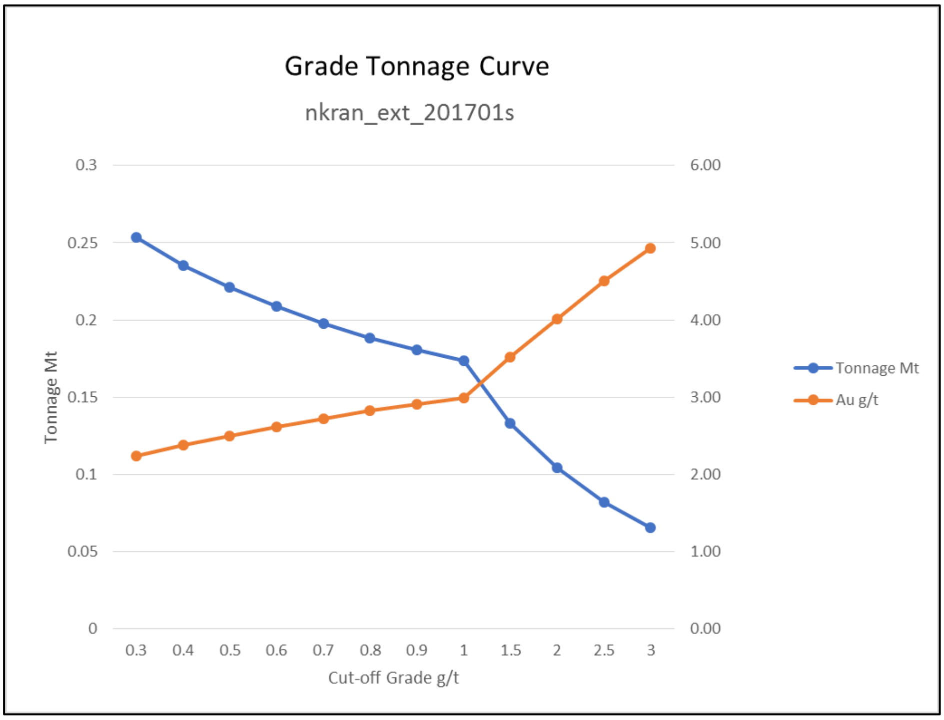


Figure 14-59: Nkran Extension Grade Tonnage Curve

14.12 Abore (CJM)

14.12.1 Weathering

The weathering profile also was used to code composites. The profile was modelled from drill data and comprises weathered oxide, transition material and fresh / unweathered units. Figure 14-60 In general, the weathering surfaces are broadly parallel the topographical profile, although weathering tends to be deeper within zones of mineralisation and tend to parallel the footwall to the mineralisation where the footwall approaches the surface.

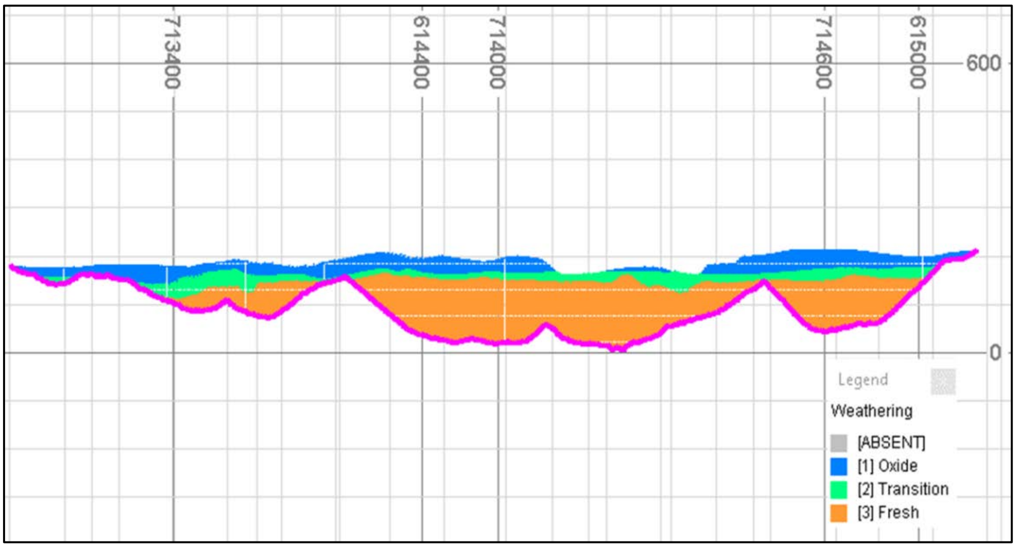


Figure 14-60: Abore Weathering Profile

14.12.2 Domains

The domains for the Abore Project were based on the latest litho-structural geological model completed during the July 2014 re-logging exercise (Figure 14-52). The main mineralisation is associated with the granitic intrusion. The mineralisation extends into the adjacent sedimentary rocks. These domains have not been changed for the 2017 report. The final domains (Table 14-57) used for estimation (2014) were:

- "GR" - the granite intrusive
- "SE" mineralisation in the sediments to the east of the granite
- "SW" mineralisation in the sediments to the west of the granite intrusive
- "SI" which is an internal small portion of sediment caught up in the granite intrusive
- "OX" which forms the mineralisation in the oxidised zone

Figure 14-61 shows the different domains used for estimation.

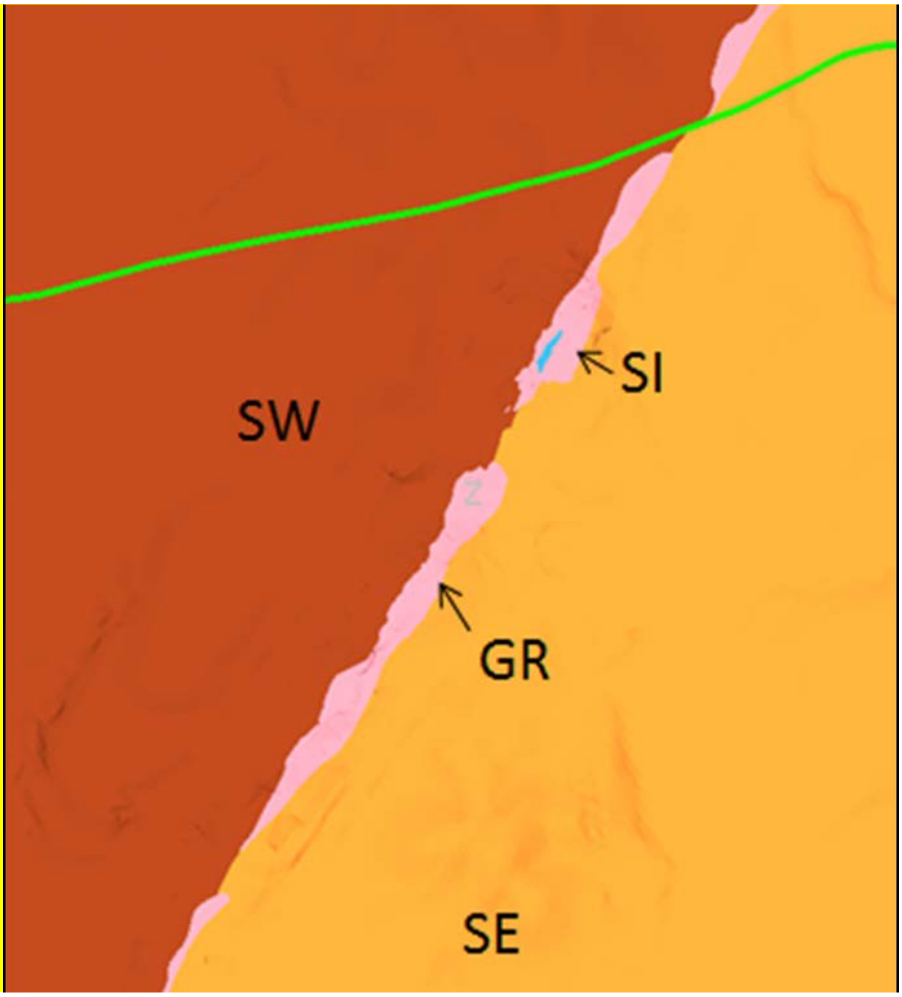


Figure 14-61: Abore Domains used for Resource Estimation

The mineralised portion within the main litho-structural domains was delineated using the IK method. Data flagging was based on a 0.3 g/t cut-off including up to 2m waste. This indicator flag was used to perform detailed spatial analysis on the mineralised orientations within each domain. Variogram analysis was done in all directions and the best continuity directions selected

A probability of 0.3 was selected to delineate the mineralisation in domain GR that strikes 030^o and dips 80^o northwest. For the estimation process, the domains were sub-divided into mineralised and waste portions. Figure 14-62 shows a section with the indicator estimation and orientation of the search ellipse. Figure 14-63 depicts an isometric view of the search ellipse along strike of domain GR.

Table 14-57: Domains used in Abore 2014 Estimation

Mineralised Zone	Material	Mineralised Zone	Type
GR	All	Granite	Ore
SE	All	Sandstone	Ore
SI	All	Sandstone	Ore
SW	All	Sandstone	Ore

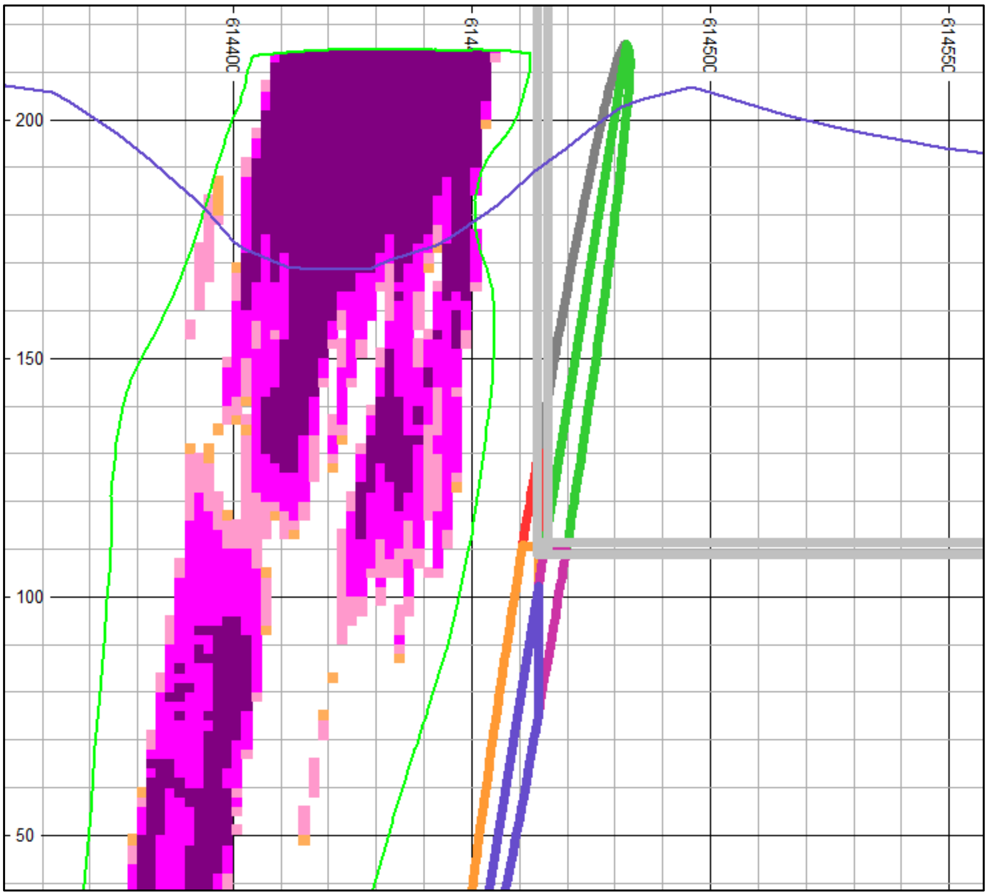


Figure 14-62: Abore Domains used for Resource Estimation

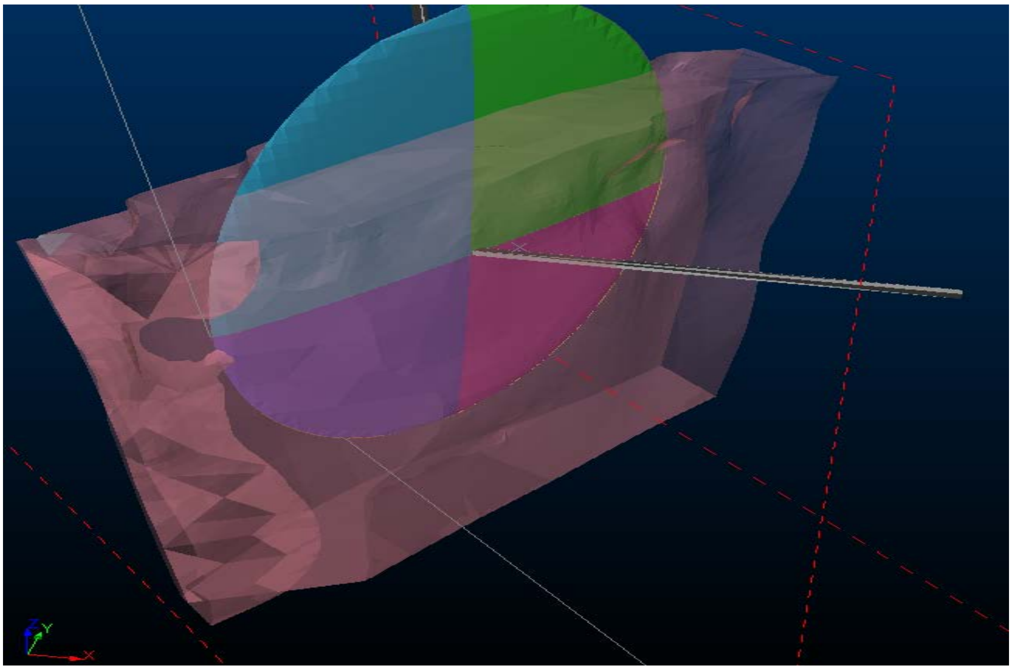


Figure 14-63: Abore Domains used for Resource Estimation

14.12.3 Abore Compositing

The de-surveyed drill holes were composited within DatamineTM on a 1m composite length. Figure 14-64 supports this approach and shows the drill hole sample lengths.

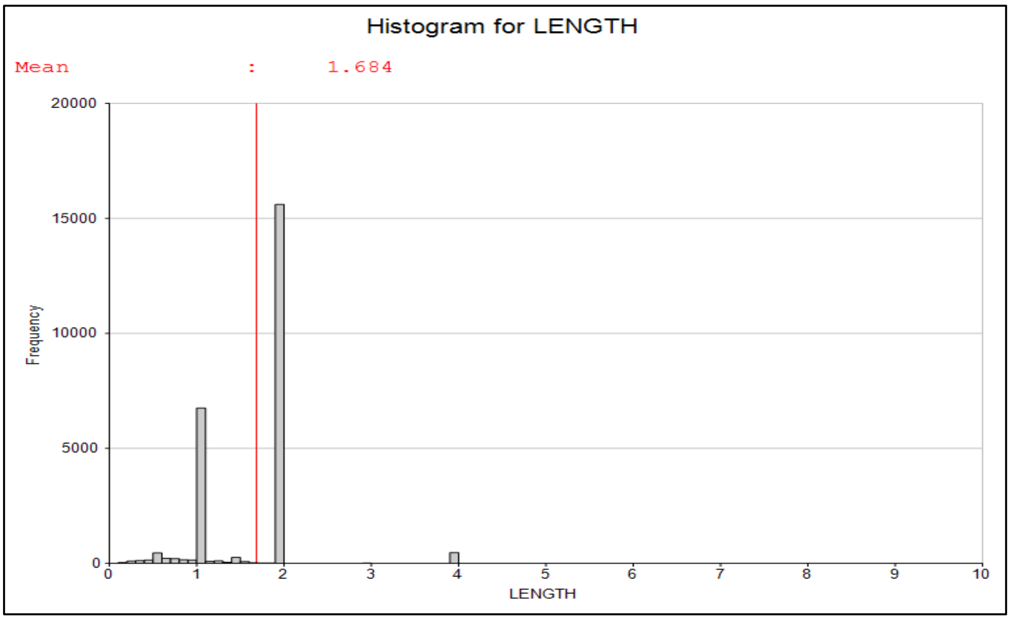


Figure 14-64: Abore - Histogram of Drill Hole Lengths

14.12.3.1 Abore Statistical Analysis

Descriptive statistics for Abore are shown for each domain in Table 14-58.

Table 14-58: Abore Descriptive Statistics for the Various Domains Generated

Domain	Parameter	Num Records	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
GR	AU	5 689	0.005	88.40	1.43	18.16	4.26	2.29
SI	AU	42	0.120	7.72	1.21	2.22	1.49	1.23
SE	AU	478	0.005	30.30	0.92	3.36	1.83	2.00
SW	AU	315	0.010	28.00	1.47	10.23	3.20	2.18
OX	AU	6 836	0.001	125.30	1.84	26.97	5.19	2.83

14.12.3.2 Abore Outlier Analysis

Outlier analysis was undertaken using histograms and probability plots. The outliers were determined for both the variography and the kriging stages. For variography the outliers are cut at a specified value, but for kriging the outliers are capped at a specified value. Hence, all available data is used in the kriging of the data, but a restricted data set is used for variography.

The histograms and probability plots indicate that the populations of gold grades are close to log normal, which is typical of many gold deposits. The CV is moderately high, but typical for these types of deposits. The kriging capping and the variogram top cut values applied are summarised in Table 14-59.

Table 14-59: Abore Kriging and Variogram Capping Values per Domain

Domain	Commodity	Capping g/t
GR	AU	40
SI	AU	999
SE	AU	10
SW	AU	10
OX	AU	60

14.12.3.3 Abore Variography Parameters

Point experimental variograms were generated and modelled for each domain.The parameters of the modelled variograms for the Abore Project are summarised in Table 14-60.

Table 14-60: Abore Variogram Parameters

Domain		Sill	Nugget %	Sill 1 %	Range1 m	Range2 m	Range3 m	Sill 2 %	Range1 m	Range2 m	Range3 m
GR	AU	5.37	51.05	67.09	39	25	4	100	109	71	4
SI	AU	2.27	34.16	45.68	19	25	4	100	35	66	4
SE	AU	3.37	32.06	69.47	13	11	4	100	52	30	4
SW	AU	1.31	30.38	53.16	11	22	4	100	30	50	4
OX	AU	11.49	51.05	67.09	39	25	4	100	109	71	4

14.12.3.4 Abore Estimation Methodology

The Abore Resource statement was re-evaluated in response to the May 2014 CIM Definition Standards. The Mineral Resource statement has been updated to reflect a $1500/oz Au pit-shell used to constrain the April 2014 Mineral Resource model, and a lower cut-off grade of Au 0.5g/t, based on the Nkran paylimit to date of 0.45 g/t Au.

The interpolations were conducted with a block size of 10 x 10 x 3m (X & Y and Z), ordinary kriging was undertaken with a discretisation of 5 x 5 x 3 (X & Y & Z). Table 14-61 and Table 14-62 show the search parameters used in the estimation.

Table 14-61: Abore Search Ranges and Angles

Domain	sdist1	sdist2	sdist3	sangle1	sangle2	sangle3	saxis1	saxis2	saxis3
OX	60	120	6	30	82	0	3	2	3
GR	60	120	6	30	82	0	3	2	3
SI	30	60	6	30	82	0	3	2	3
SE	30	60	5	30	82	0	3	2	3
SW	30	60	5	30	82	0	3	2	3

Table 14-62: Abore -Number of Samples in Search

Domain	Min num1	Max num1	svolfac2	Min num2	Max num2	svolfac3	Min num3	Max num3	Max key
OX	12	30	2	4	15	5	2	10	4
GR	12	30	2	4	15	5	2	10	4
SI	12	30	2	4	15	5	2	10	4
SE	12	30	2	4	15	5	2	10	4
SW	12	30	2	4	15	5	2	10	4

Figure 14-65 and Figure 14-66 show cross sections through the block model with the Au grades as estimated.

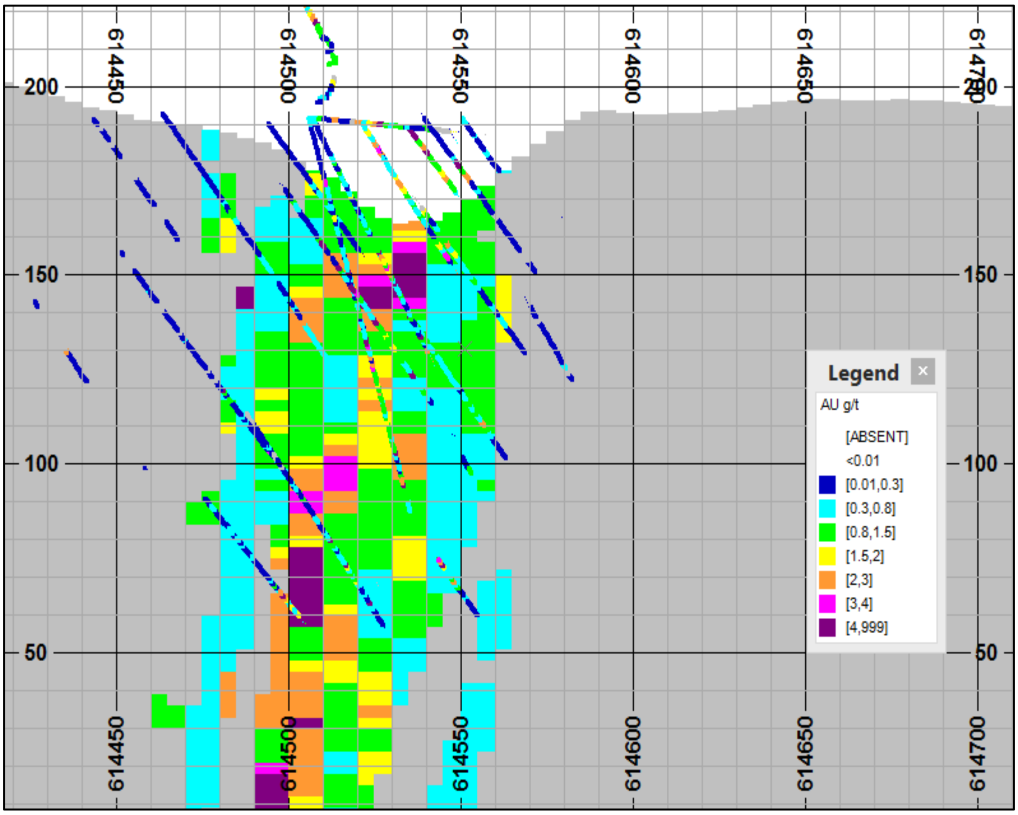


Figure 14-65: Abore Block Model Section N 713980

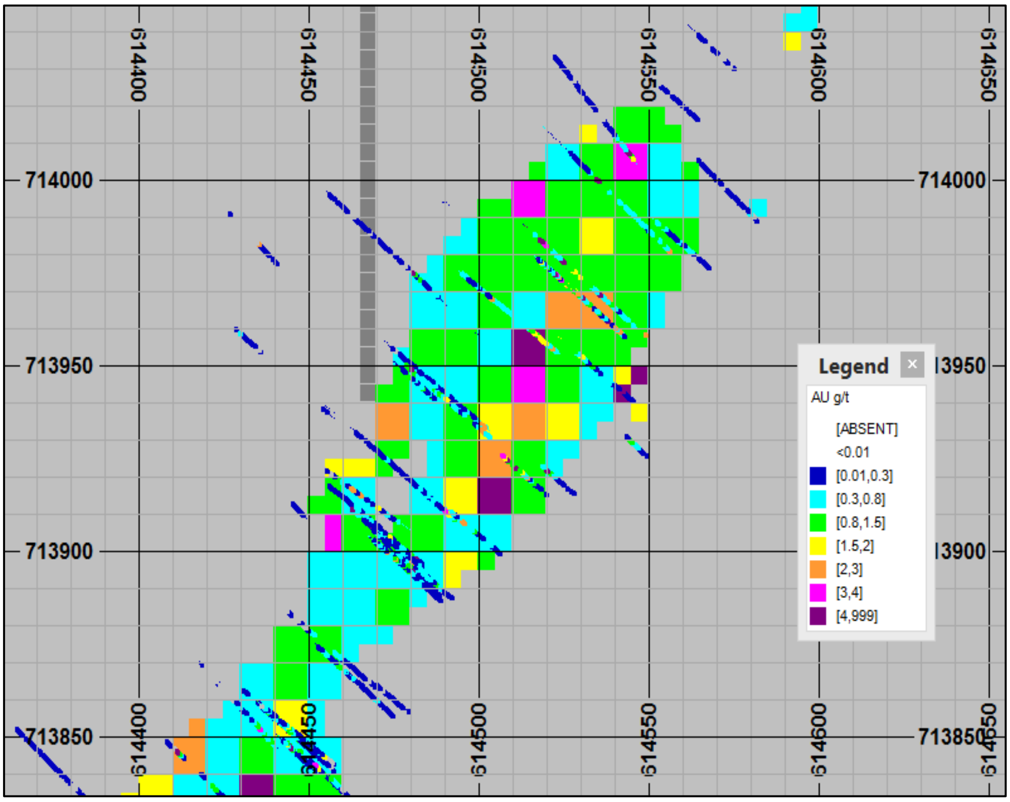


Figure 14-66: Abore Block Model Plan View 130m

14.12.4 Abore Resource Statement

The Abore Resource statement was re-evaluated as per the May 2014 CIM Definition Standards. A \$1500/oz Au pit-shell was used to constrain the April 2014 Mineral Resource model. The resource cut-off Au grade was also reduced to 0.5g/t for this estimation based on the Nkran paylimit to date of 0.45 g/t Au.

Table 14-63 shows the total MR sub-divided into Measured, Indicated and Inferred classes, as well as a breakdown into Oxide, Transition and Fresh zones.

Table 14-63: Abore Mine MR's at 0.5g/t Au cut-off grade as at April 2014, and within a US$1500 Au price shell

Abore	0.5 g/t cutoff constrained to $1500/oz Au shell				
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Measured	Oxide	0.47	1.33	19,984	0.020
	Transition	1.28	1.46	60,247	0.060
	Fresh	0.48	1.34	20,895	0.021
Total Measured		**2.23**	**1.41**	**101,127**	**0.101**
Indicated	Oxide	0.18	1.57	9,006	0.009
	Transition	0.49	1.69	26,648	0.027
	Fresh	2.42	1.43	111,136	0.111
Total Indicated		**3.09**	**1.48**	**146,790**	**0.147**
Total Measured and Indicated		**5.33**	**1.45**	**247,917**	**0.248**
Inferred	Oxide	0.09	1.90	5,667	0.006
	Transition	0.01	1.37	405	0.000
	Fresh	1.17	1.59	60,011	0.060
Total Inferred		**1.28**	**1.61**	**66,084**	**0.066**

Notes:

- Columns may not add up due to rounding.
- All figures are in metric tonnes.
- The MR's are stated as in situ tonnes.
- Individual Densities was used per mineral zone.
- The tonnages and contents are stated as 100%, which means no attributable portions are stated in the table.

Table 14-64: Abore Mine MR's at different cut-off grades, as at April 2014

Resource Category	Cut-off Grade g/t	Tonnes Mt	Gold Grade g/t	Gold Ounces Moz
Measured	0	2.41	1.35	0.104
	0.3	2.41	1.35	0.104
	0.4	2.39	1.35	0.104
	0.5	2.30	1.39	0.103
	0.6	2.10	1.47	0.099
	0.7	1.86	1.58	0.094
	0.8	1.61	1.71	0.088
	0.9	1.37	1.85	0.082
	1.0	1.18	2.00	0.076
	1.5	0.60	2.75	0.053
Indicated	0	4.97	1.28	0.205
	0.3	4.97	1.28	0.205
	0.4	4.91	1.29	0.204
	0.5	4.68	1.33	0.201
	0.6	4.25	1.41	0.193
	0.7	3.71	1.52	0.182
	0.8	3.19	1.65	0.169
	0.9	2.71	1.79	0.156
	1.0	2.29	1.94	0.143
	1.5	1.12	2.71	0.098
Inferred	0	6.03	1.32	0.255
	0.3	5.75	1.37	0.253
	0.4	5.68	1.38	0.252
	0.5	5.37	1.44	0.248
	0.6	4.78	1.55	0.237
	0.7	4.14	1.68	0.224
	0.8	3.57	1.83	0.210
	0.9	3.10	1.98	0.197
	1.0	2.70	2.13	0.185
	1.5	1.54	2.81	0.139

Figure 14-67 shows the Abore block model as classified. The Resource is also reported within a $1,500/oz Au price pit shell.

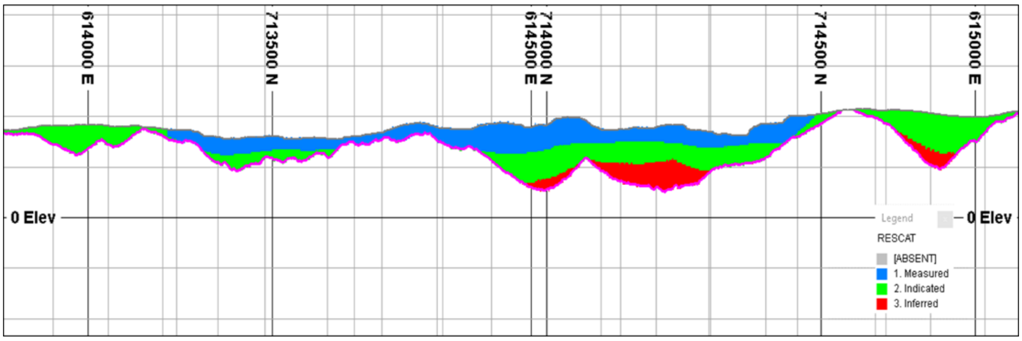


Figure 14-67: Abore Resource Classification Block model- Longitudinal S-N ransverse section

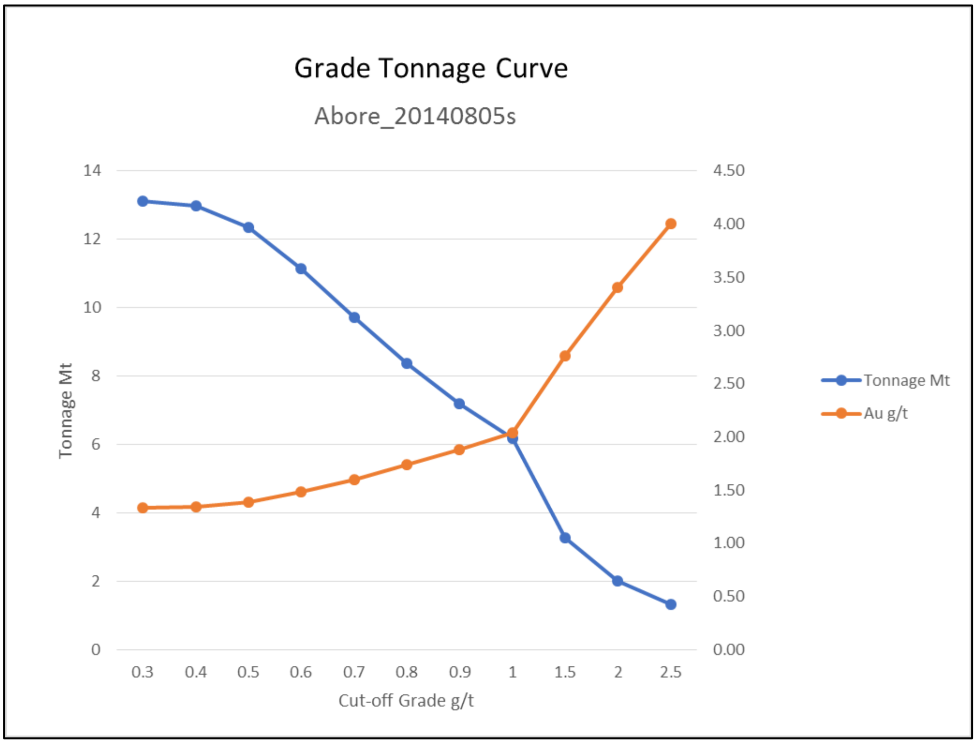


Figure 14-68: Abore Grade Tonnage Curve

14.13 Adubiaso (CJM)

14.13.1 Weathering

Composites were also coded by the weathering profile. The profile was modelled from drill data and comprises weathered oxide, transition and fresh units. In general, the weathering surfaces are broadly parallel to the topographical profile, although weathering tends to be deeper within zones of mineralisation and tend to parallel the foot wall to the mineralisation where the foot wall approaches the surface (Figure 14-69).

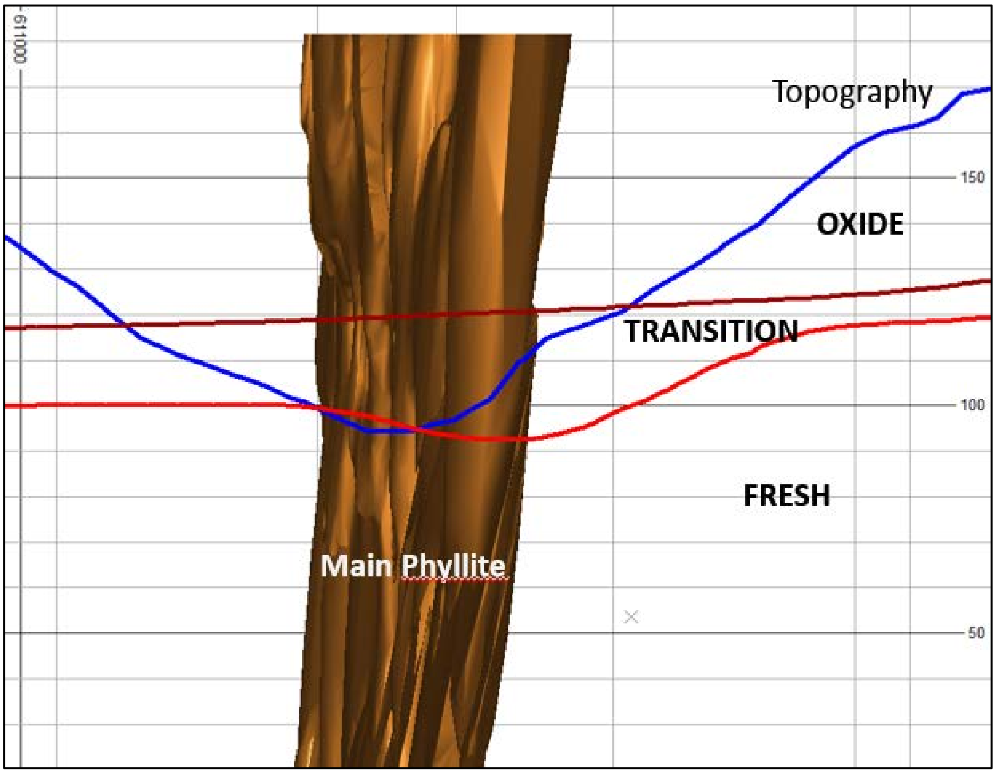


Figure 14-69: Adubiaso Weathering Profile

14.13.2 Domains

A probability of 0.3 was selected to delineate the mineralisation of the main phyllite zone mineralisation which is associated with quartz veins. The main zone strikes northeast 40° and dips 30° east (Figure 14-70). For the secondary domains (Figure 14-71) which strike northeast 040° and dips 85° east using a probability of 0.5. Figure 14-72 shows the variogram ellipse overlaid on the Geological wireframes for Domain “Q”.

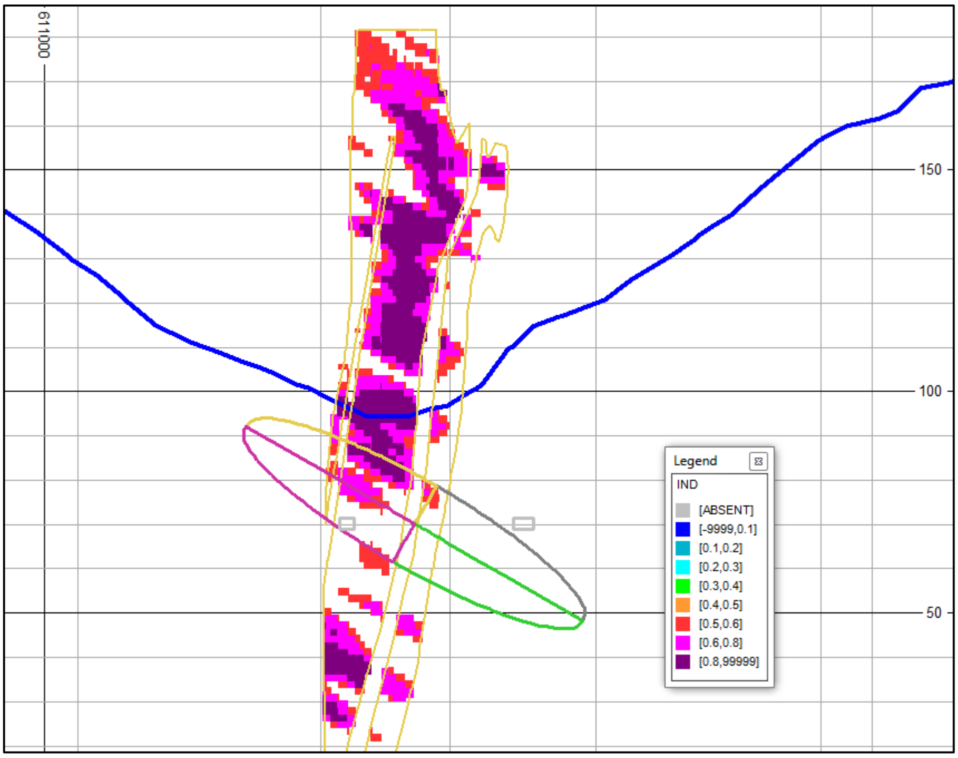


Figure 14-70: Adubiaso - Block Model Main Area, IK>0.3% and Search Ellipse

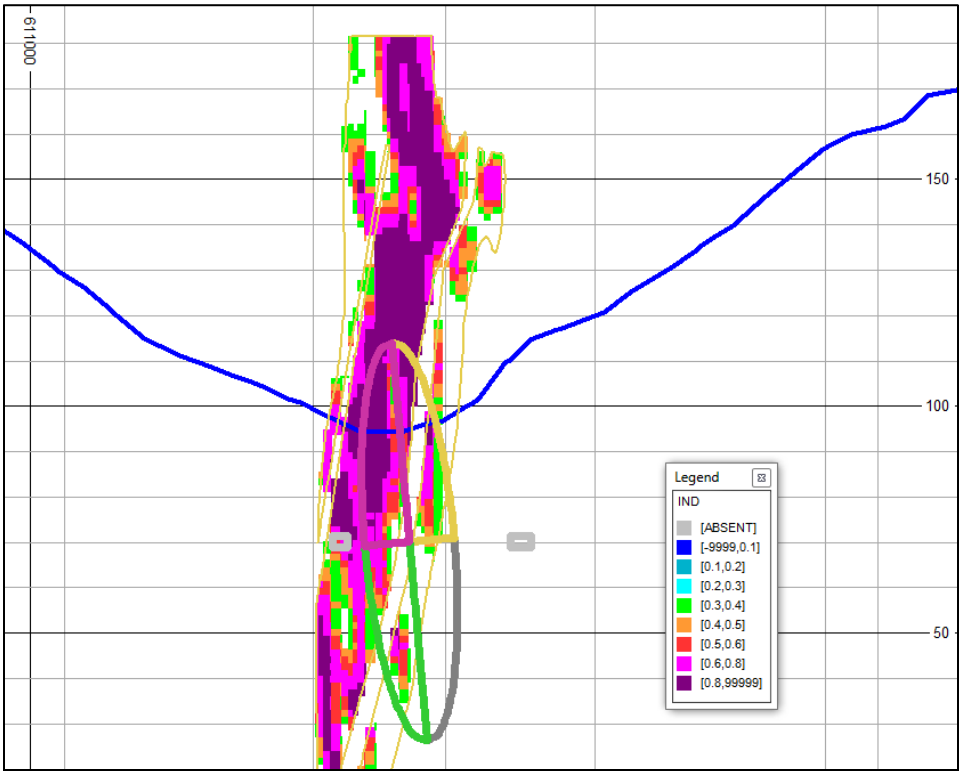


Figure 14-71: Adubiaso - Block Model Secondary Area, IK>0.5% and Search Ellipse

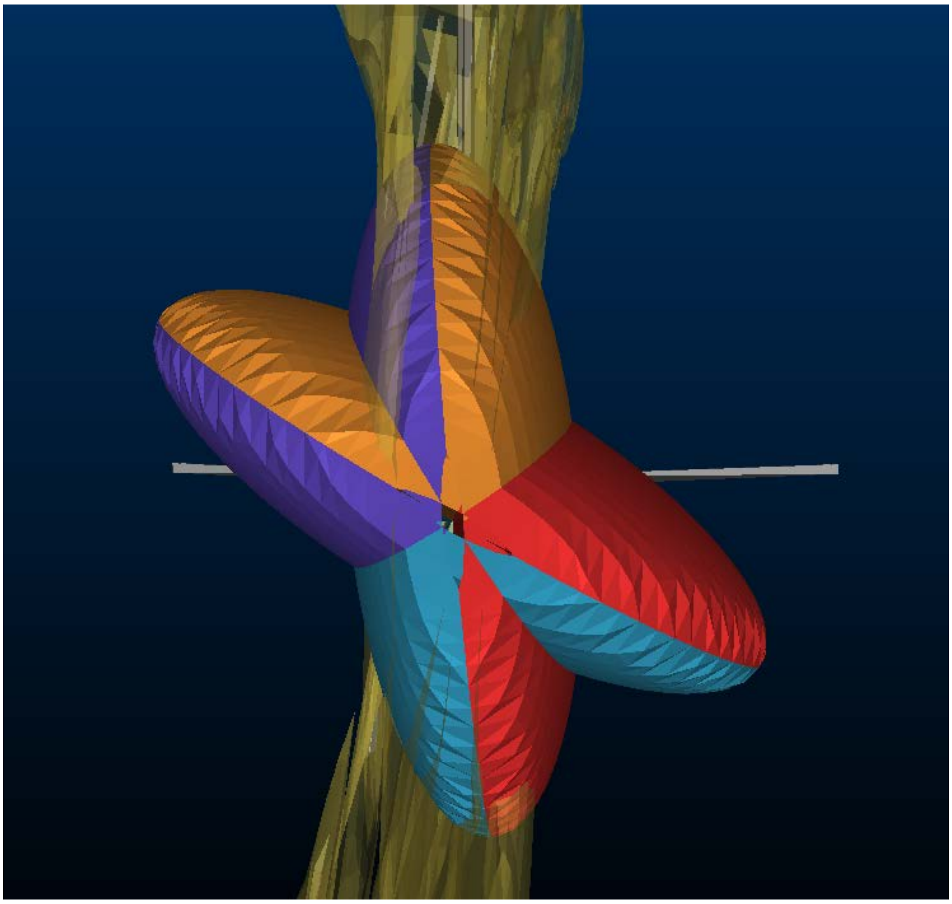


Figure 14-72: Adubiaso = Variogram Ellipse along Domain “Q

Figure 14-65 shows the domains used in the estimation.

Table 14-65: Adubiaso - Mineralised Domains used in the MRE

Mineralised Zone	Material	Mineralised Zone	Type
OQO	Oxide	Main	Ore
OQW	Oxide	Main	Waste
TQO	Transition	Main	Ore
TQW	Transition	Main	Waste
FQO	Fresh	Main	Ore
FQW	Fresh	Main	Waste
OSO	Oxide	Secondary	Ore
OSW	Oxide	Secondary	Waste
TSO	Transition	Secondary	Ore
TSW	Transition	Secondary	Waste
FSO	Fresh	Secondary	Ore
FSW	Fresh	Secondary	Waste

14.13.3 Adubiaso Compositing

The de-surveyed drill holes were composited within DatamineTM on a 1m composite length, the histograms support this approach Figure 14-73. A total of 27,458 composites were used in the statistical analysis and resource estimation. Twelve quartz vein zones, including internal waste, form part of the statistical analysis and resource estimation.

A composite analysis was performed for different composite sizes 1, 2 and 3m, the fresh quartz zone is the largest and the grades are very stable in all the composite sizes. Table 14-66 shows the grades are greater for the composite sizes of 2 and 3 due to outlier high grades zones.

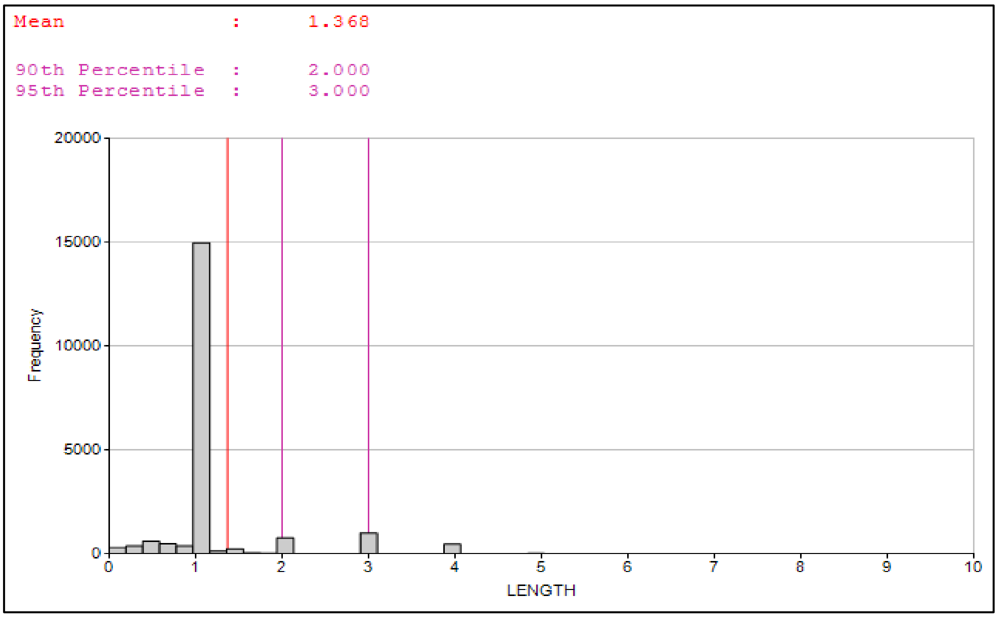


Figure 14-73: Adubiaso Histograms of the Sample Lengths for the Original

Table 14-66: Adubiaso Composite Analysis of Topcut data

Domain	Parameter	Composite 1m	Composite 2m	Composite 3m	1m vs 2m	1m vs 3m	2m vs 3m
OQO	AU	2.25	2.29	2.32	-1%	-3%	-1%
OQW	AU	0.08	0.09	0.10	-10%	-13%	-3%
OSO	AU	1.31	1.63	1.80	-20%	-27%	-9%
OSW	AU	0.16	0.17	0.17	-7%	-8%	-2%
TQO	AU	2.34	2.52	2.74	-7%	-15%	-8%
TQW	AU	0.08	0.08	0.08	-4%	-2%	1%
TSO	AU	1.28	1.49	1.58	-14%	-19%	-5%
TSW	AU	0.15	0.17	0.17	-13%	-11%	3%
FQO	AU	1.98	2.00	2.01	-1%	-2%	-1%
FQW	AU	0.10	0.12	0.14	-19%	-30%	-13%
FSO	AU	2.08	2.20	2.17	-5%	-4%	1%
FSW	AU	0.09	0.11	0.11	-22%	-18%	5%

14.13.4 Adubiaso Statistical Analysis

Following domain generation, descriptive statistics of the individual domains was undertaken shown in Table 14-67.

Table 14-67: Adubiaso Descriptive Statistics for the Various Domains Generated

Domain	Parameter	Num records	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
OQO	AU	1 544	0.001	80.000	2.29	47.14	6.87	2.99
OQW	AU	941	0.000	4.000	0.07	0.02	0.15	2.06
TQO	AU	634	0.001	50.000	2.43	34.52	5.88	2.42
TQW	AU	307	0.000	0.600	0.08	0.01	0.09	1.19
FQO	AU	592	0.001	30.000	1.94	17.73	4.21	2.17
FQW	AU	793	0.001	10.000	0.07	0.24	0.49	6.71
OSO	AU	201	0.009	15.000	1.35	8.30	2.88	2.13
OSW	AU	10 537	0.000	40.000	0.13	1.00	1.00	7.70
TSO	AU	256	0.008	20.000	1.38	9.99	3.16	2.30
TSW	AU	3 602	0.000	20.000	0.12	0.61	0.78	6.65
FSO	AU	913	0.008	50.000	2.14	29.96	5.47	2.55
FSW	AU	7 138	0.001	30.000	0.07	0.48	0.70	10.2

14.13.5 Adubiaso Outlier Analysis

Outlier analysis was undertaken using histograms and probability plots. The outliers were determined for both the variography and the kriging stages. For variography the outliers are cut at a specified value, but for kriging the outliers are capped at a specified value. Hence, all available data is used in the kriging of the data, but a restricted data set is used for variography.

The histograms and probability plots indicate that the populations of gold grades are close to log normal, which is typical of many gold deposits. The co-efficients of variation are moderately high, but typical for these types of deposits. The kriging capping and the variogram top-cut values applied are summarised in Table 14-68 and Table 14-69.

Table 14-68: Adubiaso Kriging Capping Values per Domain

Domain	Commodity	Capping g/t
FQO	AU	30
FQW	AU	10
FSO	AU	50
FSW	AU	30
OQO	AU	80
OQW	AU	4
OSO	AU	15
OSW	AU	40
TQO	AU	50
TQW	AU	0.6
TSO	AU	20
TSW	AU	20

Table 14-69: Adubiaso Variography Top-Cut Values per Domain

Domain	Commodity	Top Cut g/t
FQO	AU	20
FQW	AU	5
FSO	AU	40
FSW	AU	20
OQO	AU	40
OQW	AU	2
OSO	AU	10
OSW	AU	20
TQO	AU	40
TQW	AU	0.4
TSO	AU	10
TSW	AU	10

14.13.6 Adubiaso Variography Parameters

Point experimental variograms were generated and modelled for each domain. The parameters of the modelled variograms for the Adubiaso Project are summarised in Table 14-70.

Table 14-70: Adubiaso Variogram Parameters

Domain	Sill	Nugget %	Sill 1 %	Range1 m	Range2 m	Range3 m	Sill 2 %	Range1 m	Range2 m	Range3 m
OQO	16.034	43.16	73.58	4	13	8	100	10	30	15
OQW	0.008	25.00	75.00	11	15	8	100	25	40	15
TQO	17.38	40.13	40.50	14	12	3	100	25	40	10
TQW	0.006	33.33	33.33	13	19	9	100	25	30	15
FQO	7.395	29.40	74.67	20	14	10	100	55	35	20
FQW	0.033	24.24	33.33	31	5	8	100	60	30	30
OSO	2.359	31.33	52.69	13	20	5	100	45	50	10
OSW	0.475	33.47	68.42	7	20	5	100	25	50	10
TSO	2.078	29.31	47.26	10	19	5	100	25	50	10
TSW	0.262	32.44	44.66	9	20	5	100	30	80	10
FSO	14.025	20.00	55.00	13	20	5	100	30	40	10
FSW	0.175	24.57	73.71	19	14	5	100	85	25	10

14.13.7 Adubiaso Estimation Methodology

The interpolation was conducted with a block size of 10 x 10 x 6m (X and Y and Z), ordinary kriging was undertaken. For this exercise, ordinary kriging was used with a discretisation of 5 x 5 x 3 (X, Y and Z; Table 14-71).

Table 14-71 and Table 14-72 show the search parameters used in the estimation.

Table 14-71: Adubiaso Search Ranges and Angles

Domain	Commodity	sdist1	sdist2	sdist3	sangle1	sangle2	sangle3	saxis1	saxis2	saxis3
FQO	AU	55	35	20	40	-85	0	3	2	3
FQW	AU	60	30	30	40	-85	0	3	2	3
FSO	AU	30	40	10	40	-30	0	3	2	3
FSW	AU	85	25	10	40	-30	0	3	2	3
OQO	AU	10	30	15	40	-85	0	3	2	3
OQW	AU	25	40	15	40	-85	0	3	2	3
OSO	AU	45	50	10	40	-30	0	3	2	3
OSW	AU	25	50	10	40	-30	0	3	2	3
TQO	AU	25	40	10	40	-85	0	3	2	3
TQW	AU	25	30	15	40	-85	0	3	2	3
TSO	AU	25	50	10	40	-30	0	3	2	3
TSW	AU	30	80	10	40	-30	0	3	2	3

Table 14-72: Adubiaso - Number of Samples in Search

Domain	minnum1	maxnum1	svolfac2	minnum2	maxnum2	svolfac3	minnum3	maxnum3	maxkey
FQO	6	24	2	4	16	5	2	11	6
FQW	6	24	2	4	16	5	2	11	6
FSO	6	24	2	4	16	5	2	11	6
FSW	6	24	0	4	16	0	2	11	6
OQO	6	24	2	4	16	5	2	11	6
OQW	6	24	2	4	16	5	2	11	6
OSO	6	24	2	4	16	5	2	11	6
OSW	6	24	0	4	16	0	2	11	6
TQO	6	24	2	4	16	5	2	11	6
TQW	6	24	2	4	16	5	2	11	6
TSO	6	24	2	4	16	5	2	11	6
TSW	6	24	0	4	16	0	2	11	6

Figure 14-74 shows a cross section through the block model with the Au grades as estimated, and Figure 14-75 a plan view of the block model.

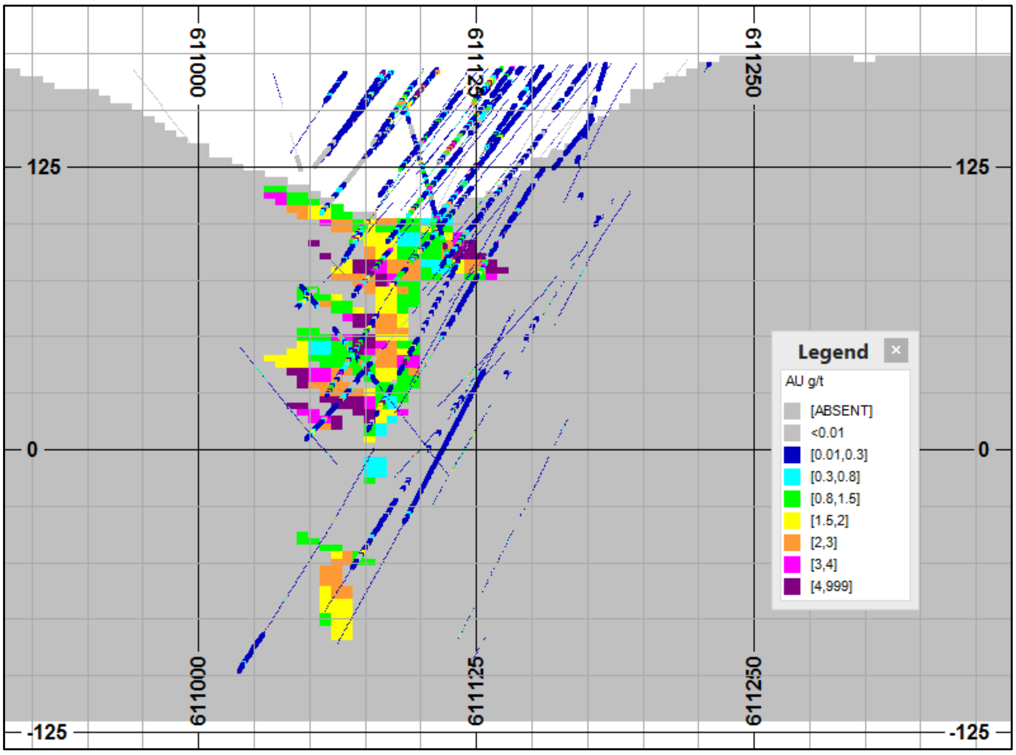


Figure 14-74: Adubiaso Block Model Section N 704400

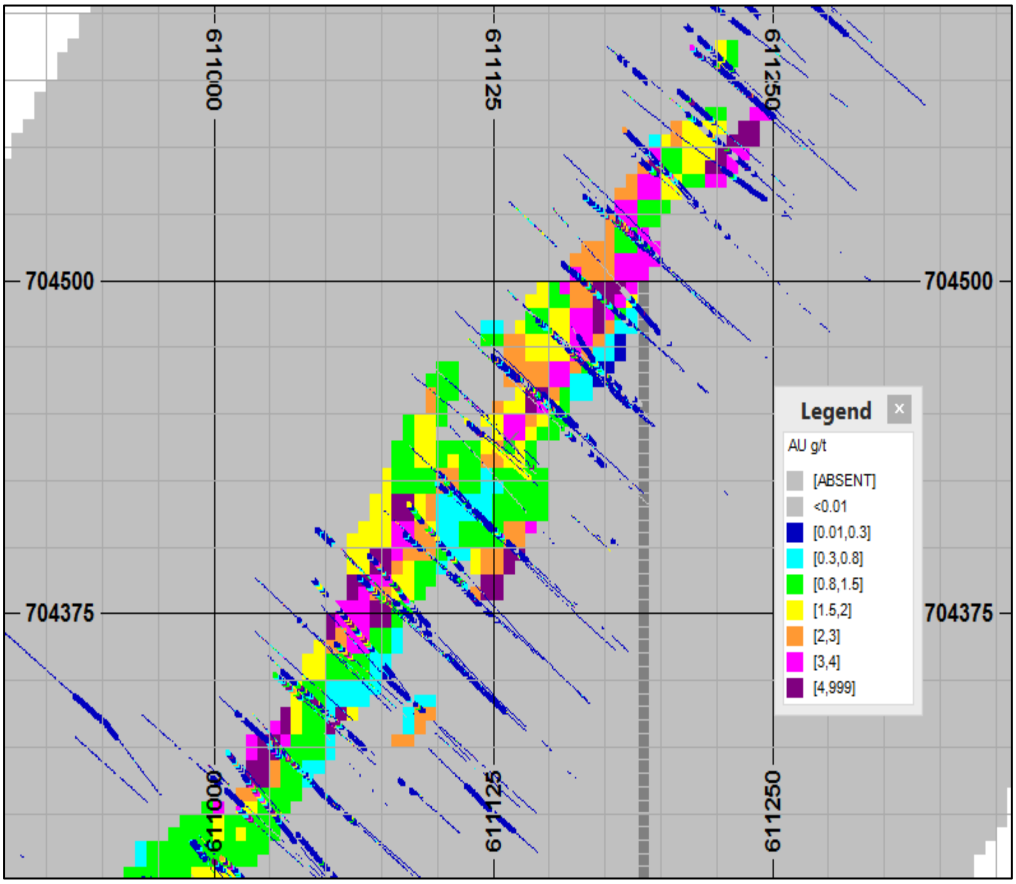


Figure 14-75: Adubiaso Block Model Plan View at 80m RL

14.13.8 Adubiaso Resource Statement

The Adubiaso Project was not re-estimated for this report. The Resource statement was updated using the original April 2014 estimation within a US$1,500/oz Au pit-shell. The cut-off Au grade was also reduced to 0.5g/t for this estimation.

Table 14-73 shows the total Mineral Resource sub-divided into Measured, Indicated and Inferred; as well as Oxide, Transition and Fresh zones.

Table 14-73: Adubiaso Mine MR's at 0.5g/t Au cut-off grade as at April 2014, within a US$1,500 pit shell

Adubiaso Main	0.5 g/t cutoff constrained to $1500/oz Au shell				
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Measured	Oxide	0.12	2.40	9,136	0.009
	Transition	0.41	2.14	27,871	0.028
	Fresh	0.30	2.63	25,384	0.025
Total Measured		**0.82**	**2.36**	**62,391**	**0.062**
Indicated	Oxide	0.00	0.80	96	0.000
	Transition	0.03	2.87	2,587	0.003
	Fresh	0.83	2.01	53,596	0.054
Total Indicated		**0.86**	**2.03**	56,279	**0.056**
Total Measured and Indicated		**1.68**	**2.19**	**118,670**	**0.119**
Inferred	Oxide	0.0001	1.46	4	0.000
	Transition	0.0023	3.75	276	0.000
	Fresh	0.0003	2.80	27	0.000
Total Inferred		**0.0027**	**3.57**	308	**0.000**

Notes:

- Columns may not add up due to rounding.
- All figures are in metric tonnes.
- The MR's are stated as in situ tonnes.
- Individual Densities was used per mineral zone.
- The tonnages and contents are stated as 100%, which means no attributable portions are stated in the table.

Table 14-74: Adubiaso Mine MR's at different cut-off grades as at April 2014

Resource Category	Cut-off Grade g/t	Tonnes Mt	Gold Grade g/t	Gold Ounces Moz
Measured	0	0.87	2.25	0.063
	0.3	0.87	2.27	0.063
	0.4	0.85	2.31	0.063
	0.5	0.83	2.35	0.063
	0.6	0.81	2.41	0.062
	0.7	0.77	2.50	0.062
	0.8	0.72	2.60	0.061
	0.9	0.69	2.70	0.059
	1.0	0.65	2.80	0.058
	1.5	0.46	3.45	0.051

Resource Category	Cut-off Grade g/t	Tonnes Mt	Gold Grade g/t	Gold Ounces Moz
Indicated	0	1.65	1.82	0.096
	0.3	1.64	1.83	0.096
	0.4	1.61	1.85	0.096
	0.5	1.57	1.89	0.095
	0.6	1.53	1.93	0.095
	0.7	1.47	1.98	0.093
	0.8	1.39	2.04	0.092
	0.9	1.31	2.12	0.089
	1.0	1.20	2.22	0.086
	1.5	0.75	2.81	0.068
Inferred	0	0.48	1.39	0.022
	0.3	0.47	1.41	0.021
	0.4	0.40	1.59	0.021
	0.5	0.30	1.98	0.019
	0.6	0.27	2.12	0.019
	0.7	0.26	2.18	0.018
	0.8	0.25	2.26	0.018
	0.9	0.24	2.32	0.018
	1.0	0.22	2.40	0.017
	1.5	0.14	3.07	0.014

14.13.9 Adubiaso Resource Classification

Figure 14-76 shows the Adubisaso block resource classification. The Resource is also reported within a US$ 1,500/oz Au pit shell.

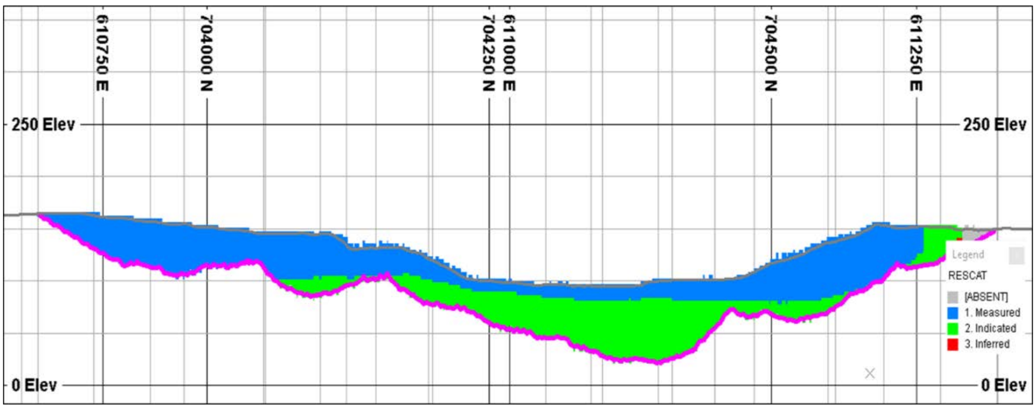


Figure 14-76: Adubiaso Resource Classification Block model- Cross section

The Grade – Tonnage Curve for Adubiaso Main is shown in Figure 14-77.

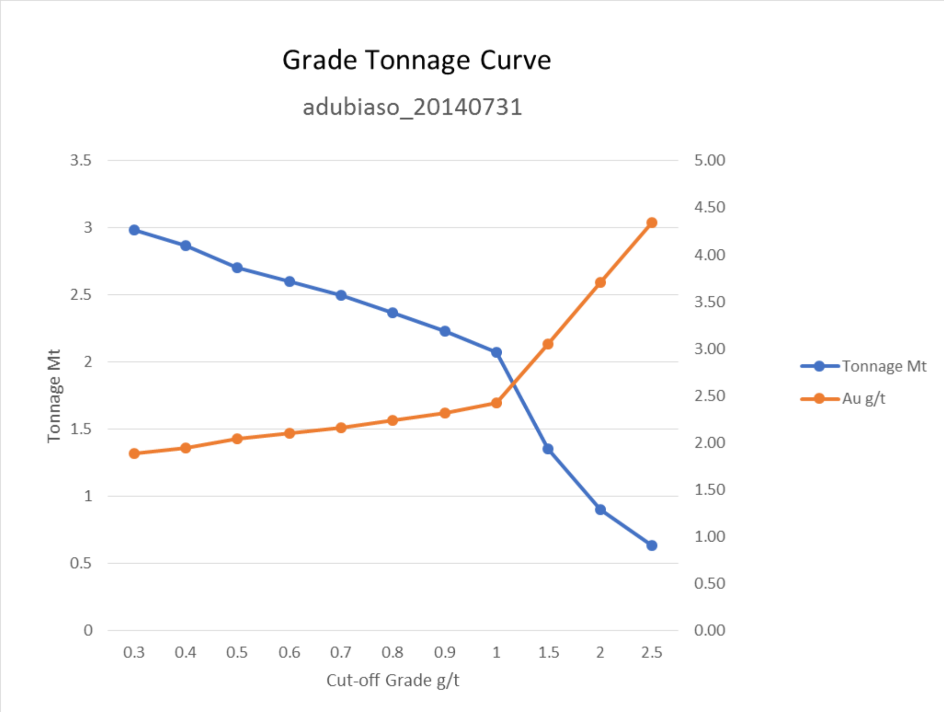


Figure 14-77: Adubiaso Grade Tonnage Curves

14.14 Adubiaso Extension (CJM)

Adubiaso Extension is the northerly strike extension of the Adubiaso Main satellite pit. The target was evaluated and advanced to being 20m x 20m infill drilled during 2016.

14.14.1 Weathering

The weathering profile for Adubiaso Extension was also used to code composites. The profile was modelled from drill data and comprised weathered oxide, transition material and fresh units. In general, the weathering surfaces are broadly parallel to the topographical profile, although weathering tends to be deeper within zones of mineralisation and tend to parallel the footwall to the mineralisation where the footwall approaches the surface Figure 14-78.

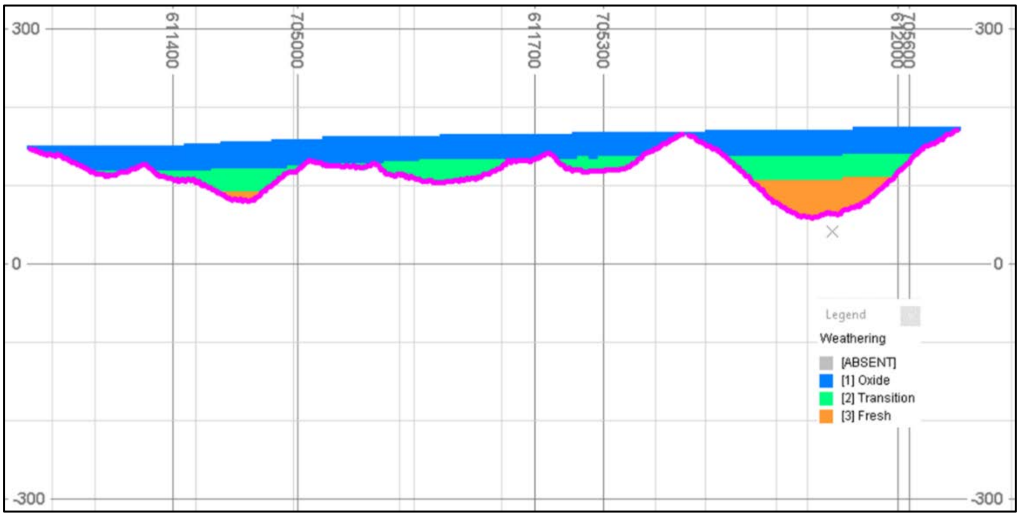


Figure 14-78: Adubiaso Extension Weathering profile within the 2017 pit shell

14.14.2 Domains

A single Domain GR for Indicator Kriging was used to delineate the zones of mineralisation. An indicator probability of greater than 0.4 and a minimum distance of 0.8m between estimation points was used as the selection criteria. This approach appears to provide reasonable delineation and is deemed a better approach than simply constructing grade shells; as they tend to overestimate the volume. Figure 14-79 shows the grades are not necessarily included in the estimation envelopes. Figure 14-80 shows the orientation of the search ellipse overlain on the Geological wireframes.

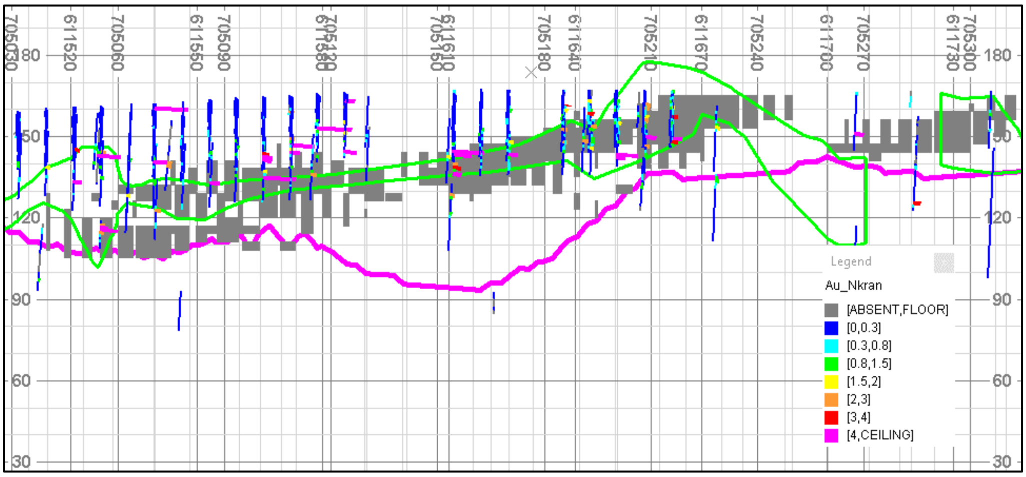


Figure 14-79: Adubiaso Extension Block model Longitudinal section

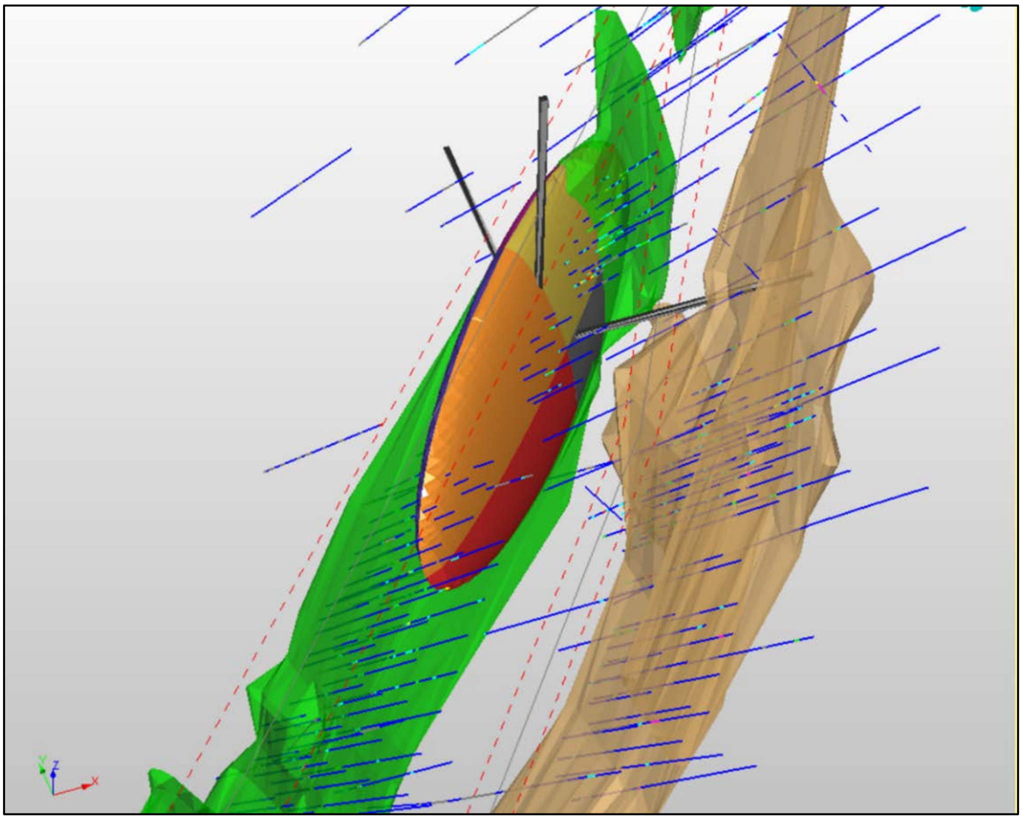


Figure 14-80: Adubiaso ExtensionSearch Ellipse and Wireframes Oblique view

14.14.3 Adubiaso Extension Compositing

The de-surveyed drill holes were composited within DatamineTM on a 1.5m composite length, the histograms in Figure 14-81 support this approach as it shows that the average sampling interval for the majority of the data is at 1m.

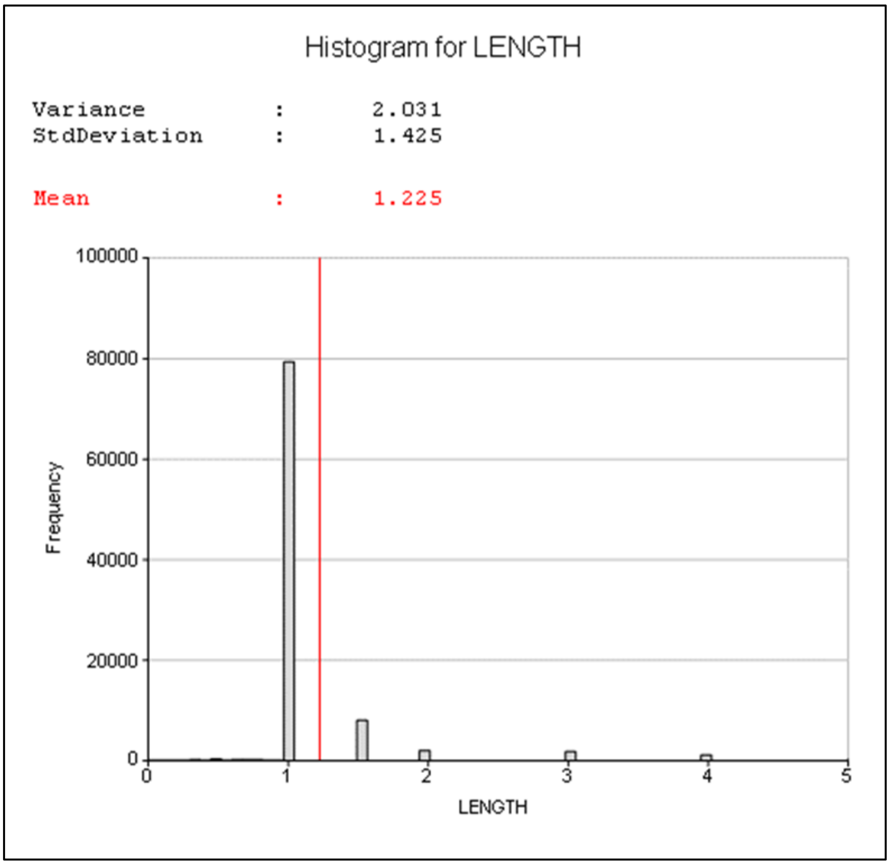


Figure 14-81: Adubiaso Extension - Histogram of Sample Lengths for Drill Holes

14.14.4 Adubiaso Extension Statistical Analysis

Following domain generation, descriptive statistics of the individual domains was undertaken shown in Table 14-75.

Table 14-75: Adubiaso Extension Descriptive Statistics for the Various Domains Generated

Domain	Parameter	Num records	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
Q	AU	1051	0.005	414.8	2.17	207.26	14.40	6.64
W	AU	14281	0	358	0.15	11.00	3.32	21.87

14.14.5 Adubiaso Extension Outlier Analysis

Outlier analysis was undertaken using histograms and probability plots. The outliers were determined for both the variography and the kriging stages. For variography the outliers were cut at a specified value, but for kriging the outliers were capped at a specified value. Hence, all available data was used in the kriging of the data, but a restricted data set was used for variography.

The histograms and probability plots indicate that the populations of gold grades are close to log normal, which is typical of many gold deposits. The co-efficient of variation is moderately high but typical for these types of deposits. The kriging capping and the variogram top-cut values applied are summarised in Table 14-76.

Table 14-76: Adubiaso Extension Kriging and Variogram Capping Values per Domain

Domain	Commodity	Capping g/t
Q	AU	50
W	AU	20

14.14.6 Adubiaso Extension Variogram Parameters

Point experimental variograms were generated and modelled for each domain. The parameters of the modelled variograms for the Adubiaso Extension Project are summarised in Table 14-77.

Table 14-77: Adubiaso Extension Variogram Parameters

Domain		Sill	Nugget %	Sill 1 %	Range1 m	Range2 m	Range3 m	Sill 2 %	Range1 m	Range2 m	Range3 m
Q	AU	16.18	23.70	55.21	14	20	5	100	40	85	10

14.14.7 Adubiaso Extension Estimation Methodology

The interpolations were conducted with a block size of 10 x 10 x 3m (X and Y and Z), ordinary kriging was undertaken with a discretisation of 5 x 5 x 3 (X and Y and Z).

Table 14-78 and Table 14-79 show the search parameters used in the estimation.

Table 14-78: Adubiaso Extension Search Ranges and Angles

Domain	Commodity	sdist1	sdist2	sdist3	sangle1	sangle2	sangle3	saxis1	saxis2	saxis3
Q	AU	40	80	10	40	-75	0	3	2	1

Table 14-79: Adubiaso Extension - Number Samples in Search

Domain	Min num 1	Max num 1	Svolfac 2	Min num 2	Max num 2	Svolfac 3	Min num 3	Max num 3	Max key
Q	6	24	2	4	16	5	4	11	5

Figure 14-82 shows a transverse section through the block model with the Au grades as estimated.

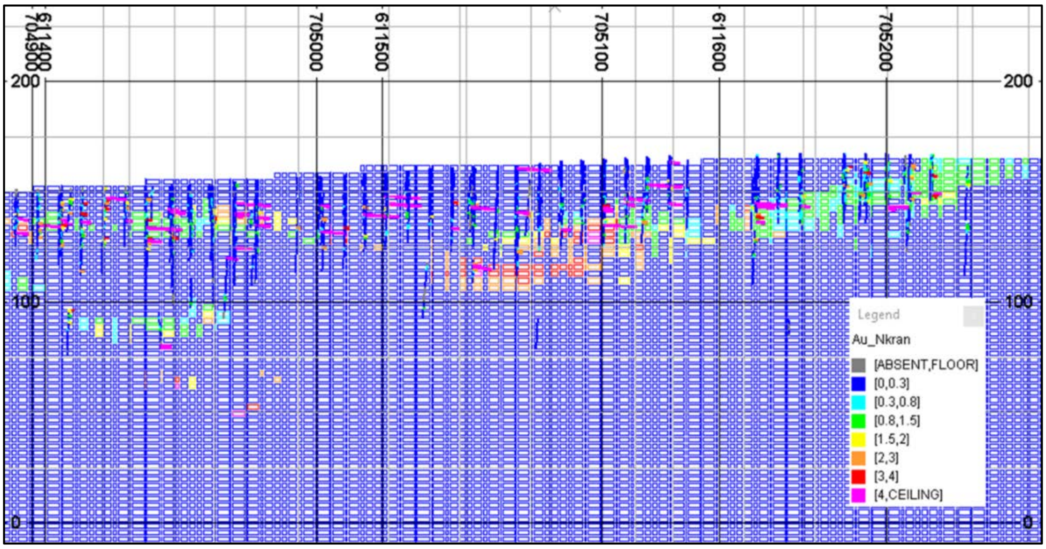


Figure 14-82: Adubiaso North Block Model Transverse Section

14.14.8 Adubiaso Extension Resource Statement

The Adubiaso Extension Project was estimated for this report. The Resource statement is constrained to be within a US$1,500/oz Au pit-shell. The cut-off Au grade is stated at 0.5 g/t for this estimation.

Figure 14-80 shows the total MR sub-divided into Measured, Indicated and Inferred; as well as Oxide, Transition and Fresh zones.

Table 14-80: Adubiaso Extension MR's at 0.5g/t Au cut-off grade as at December 2016

Adubiaso Extension	0.5 g/t cutoff constrained to $1500/oz Au shell				
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Measured	Oxide	0.12	1.98	7,369	0.007
	Transition	0.04	1.90	2,430	0.002
	Fresh	-	0.00	-	-
Total Measured		**0.16**	**1.96**	**9,800**	**0.010**
Indicated	Oxide	0.11	1.05	3,585	0.004
	Transition	0.09	1.19	3,299	0.003
	Fresh	0.05	3.04	4,943	0.005
Total Indicated		**0.26**	**1.71**	**11,827**	**0.014**
Total Measured and Indicated		**0.42**	**1.80**	**24,144**	**0.024**
Inferred	Oxide	0.01	2.74	521	0.001
	Transition	0.01	2.93	514	0.001
	Fresh	0.10	3.37	11,270	0.011
Total Inferred		**0.14**	**3.10**	**13,741**	**0.014**

Notes:

- Columns may not add up due to rounding
- All figures are in metric tonnes
- The Mineral Resources are stated as in situ tonnes
- Individual Densities was used per mineral zone
  - The tonnages and contents are stated as 100%, which means no attributable portions are stated in the table.

Table 14-81: Adubiaso Extension Mine MR's at different cut-off grades as at December 2016

Resource Category	Cut-off Grade Au g/t	Tonnes Mt	Grade Au g/t	Content Au Moz
Measured	0	2.19	0.15	0.011
	0.3	0.17	1.86	0.010
	0.4	0.16	1.89	0.010
	0.5	0.16	1.94	0.010
	0.6	0.15	2.06	0.010
	0.7	0.13	2.21	0.009
	0.8	0.12	2.39	0.009
	0.9	0.10	2.58	0.009
	1	0.09	2.77	0.008
	1.5	0.06	3.69	0.007
Indicated	0	8.56	0.07	0.019
	0.3	0.35	1.48	0.016
	0.4	0.34	1.50	0.016
	0.5	0.31	1.59	0.016
	0.6	0.28	1.70	0.015
	0.7	0.25	1.83	0.015
	0.8	0.23	1.96	0.014
	0.9	0.20	2.08	0.014
	1	0.18	2.25	0.013

Resource Category	Cut-off Grade Au g/t	Tonnes Mt	Grade Au g/t	Content Au Moz
	1.5	0.11	2.91	0.010
Inferred	0	5.94	0.11	0.021
	0.3	0.24	2.53	0.019
	0.4	0.24	2.54	0.019
	0.5	0.24	2.55	0.019
	0.6	0.23	2.60	0.019
	0.7	0.22	2.68	0.019
	0.8	0.21	2.76	0.019
	0.9	0.20	2.86	0.019
	1	0.19	2.97	0.018
	1.5	0.16	3.33	0.017

Figure 14-83 shows the Adubiaso North block resource classification. The Resource is also reported within a pit shell as depicted in Figure 14-83.

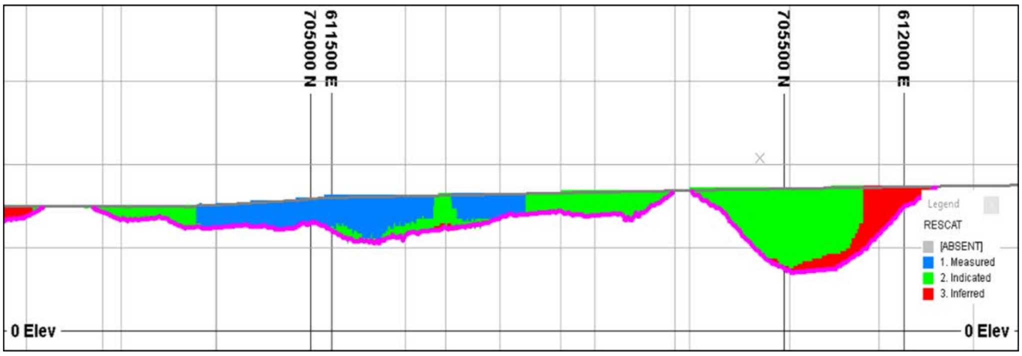


Figure 14-83: Adubiaso North Resource Classification Block model- Longitudinal section

Figure 14-84 shows the Adubiaso North grade-tonnage curve.

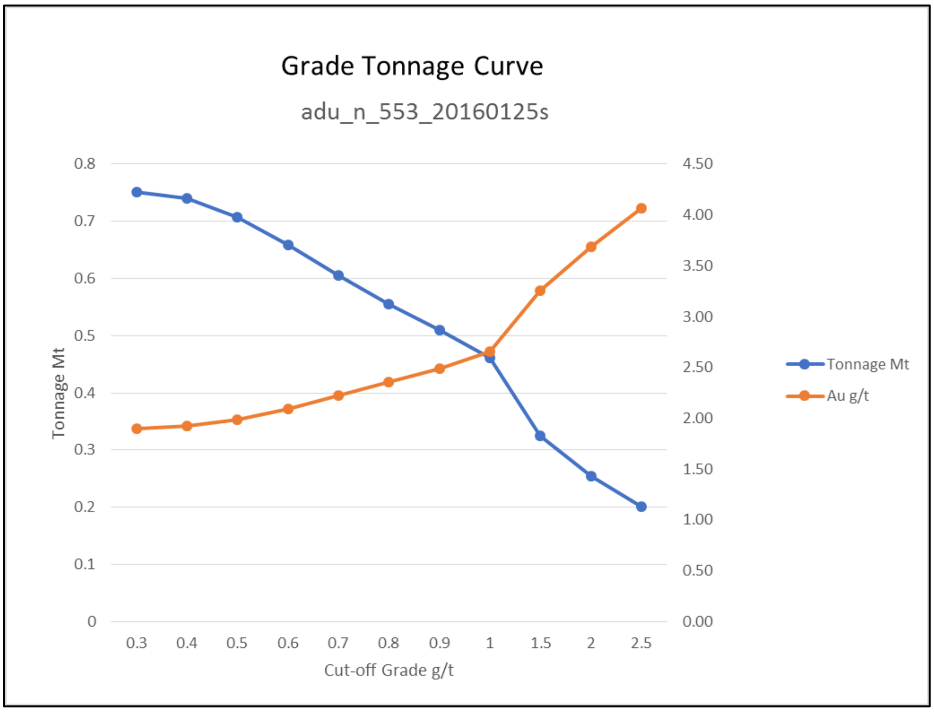


Figure 14-84: Adubiaso Extension Grade Tonnage Curve

14.15 Asuadai (CJM)

14.15.1 Weathering

Composites were also coded by the weathering profile. The profile was modelled from drill data and comprises weathered oxide, transition material and fresh units. In general, the weathering surfaces are broadly parallel with the topographical profile, although weathering tends to be deeper within zones of mineralisation and tend to parallel the foot wall to the mineralisation where the footwall approaches the surface. The main domains were sub-divided using the weathered surfaces which resulted into 12 distinct domains for estimation. Figure 14-85 shows a west east section of the weathered surfaces.

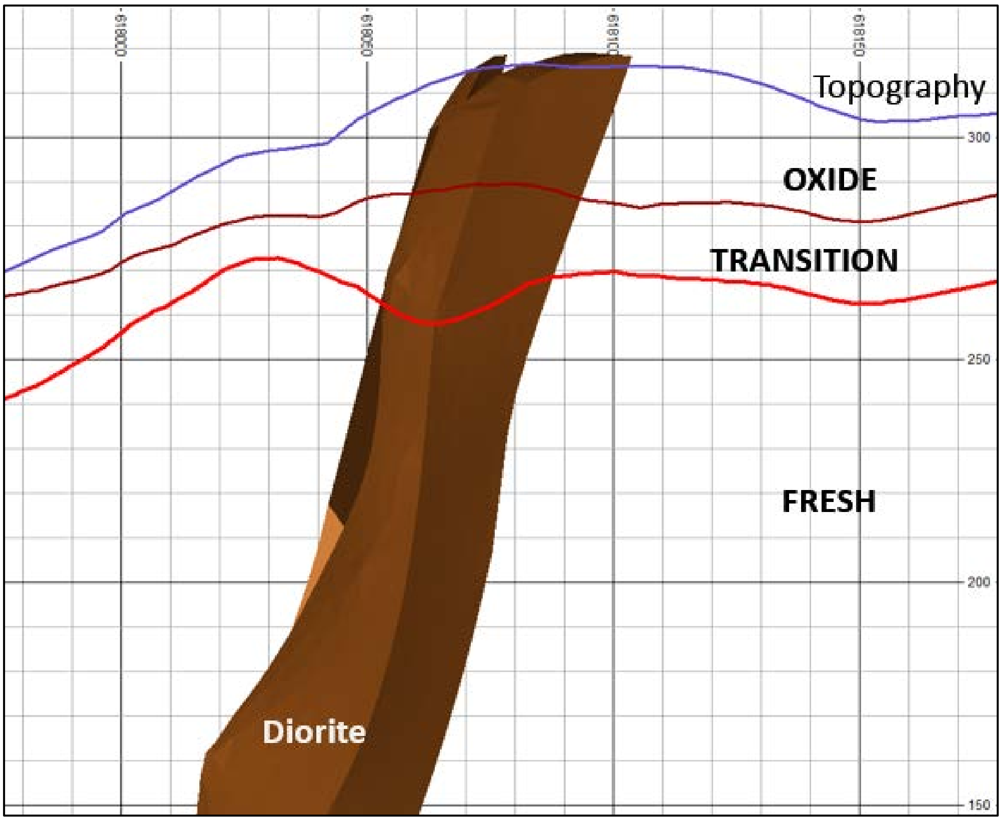


Figure 14-85: Asuadai West East View of Weathering Profile

14.15.2 Domains

The mineralised areas were defined by Optiro, and the granite zone is the main mineralised zone, whilst the other lithological zones were considered as one combined unit, called the Secondary Material outside the granite wireframes.

Figure 14-86 and Figure 14-87 show the indicator estimations and orientations of the search ellipses. All the materials outside the granite were considered as “Secondary Domain”. The strike of this domain is also north northeast 030^{o} and the dip is at 60^{o} east. Figure 14-88 depicts the search ellipse for domain GR.

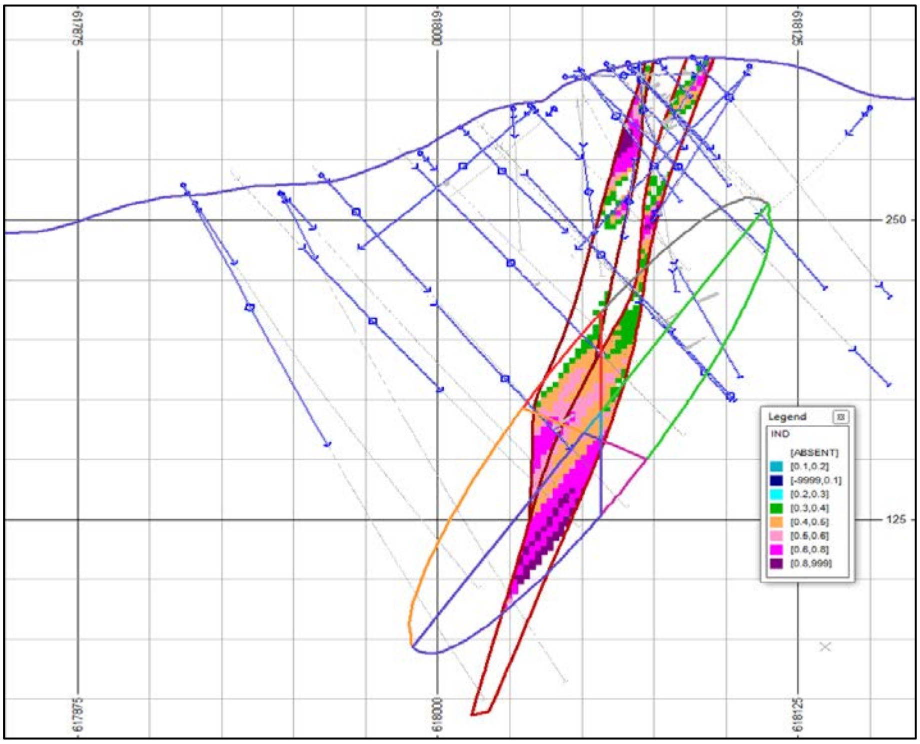


Figure 14-86: Asuadai - Block Model Granite Area, IK>0.3% and Search Ellipse

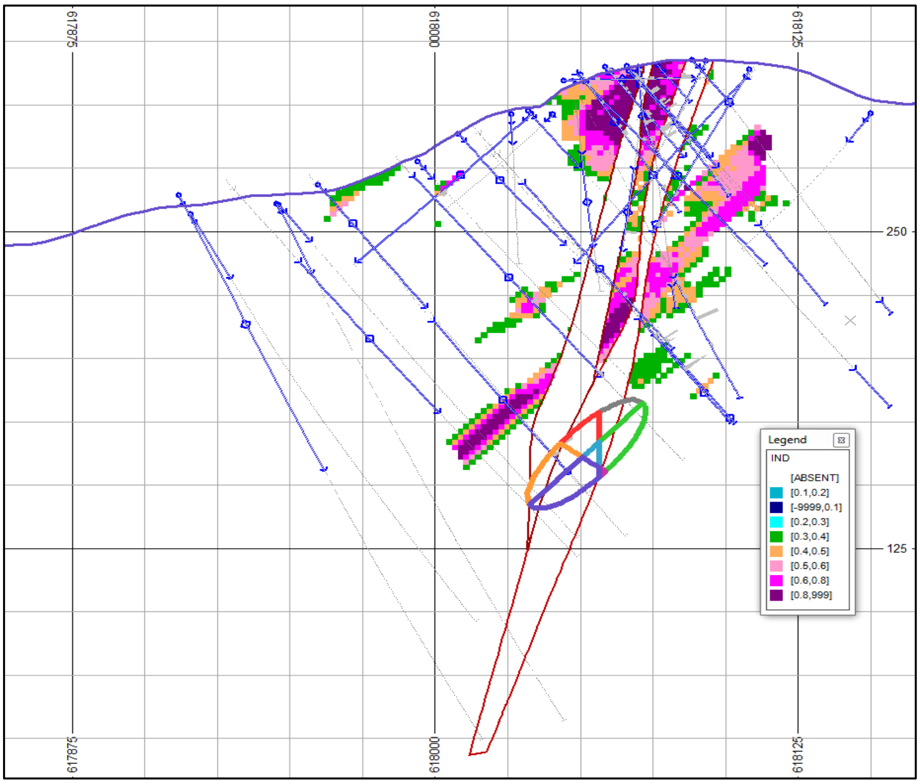


Figure 14-87: Asuadai - Block Model Secondary Area, IK>0.3% and Search Ellipse

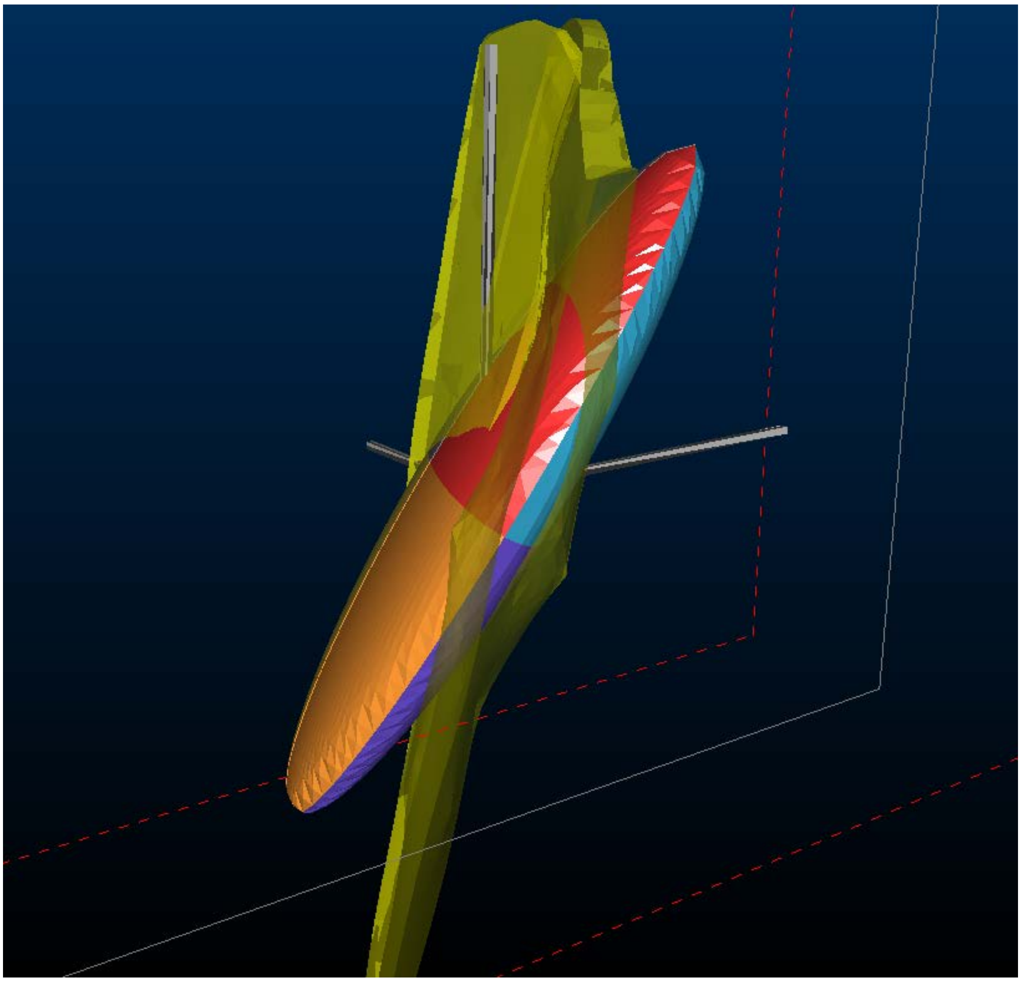


Figure 14-88: Asuadai - Variogram Ellipse for the Domain GR

For the purpose of the MRE twelve (12) mineralised domains were interpreted and were modelled using the limits of the oxide, transition and fresh mineral zones Table 14-82.

Table 14-82: Asuadai Mineralised Domains used in the MRE

Code	Material	Mineralised Zone	Type
ODO	Oxide	Granite	Ore
ODE	Oxide	Granite	External
TDO	Transition	Granite	Ore
TDE	Transition	Granite	External
FDO	Fresh	Granite	Ore
FDE	Fresh	Granite	External

Code	Material	Mineralised Zone	Type
OSO	Oxide	Secondary	Ore
OSE	Oxide	Secondary	External
TSO	Transition	Secondary	Ore
TSE	Transition	Secondary	External
FSO	Fresh	Secondary	Ore
FSE	Fresh	Secondary	External

14.15.3 Asuadai Compositing

The de-surveyed drill holes were composited within DatamineTM on a 1m composite length, supported by the histogram shown in Figure 14-89. A total of 9,381 composites were used in the statistical analysis and resource estimation. Twelve mineralized zones, including the waste, form part of the statistical analysis and MRE.

A composite analysis was performed for different composite sizes 1, 1.5 and 2m, the fresh quartz zone is the largest and the grades are very stable in all the composite sizes. Table 14-83 shows the grades are greater for the composite sizes of 1.5m due to outlier high grade zones.

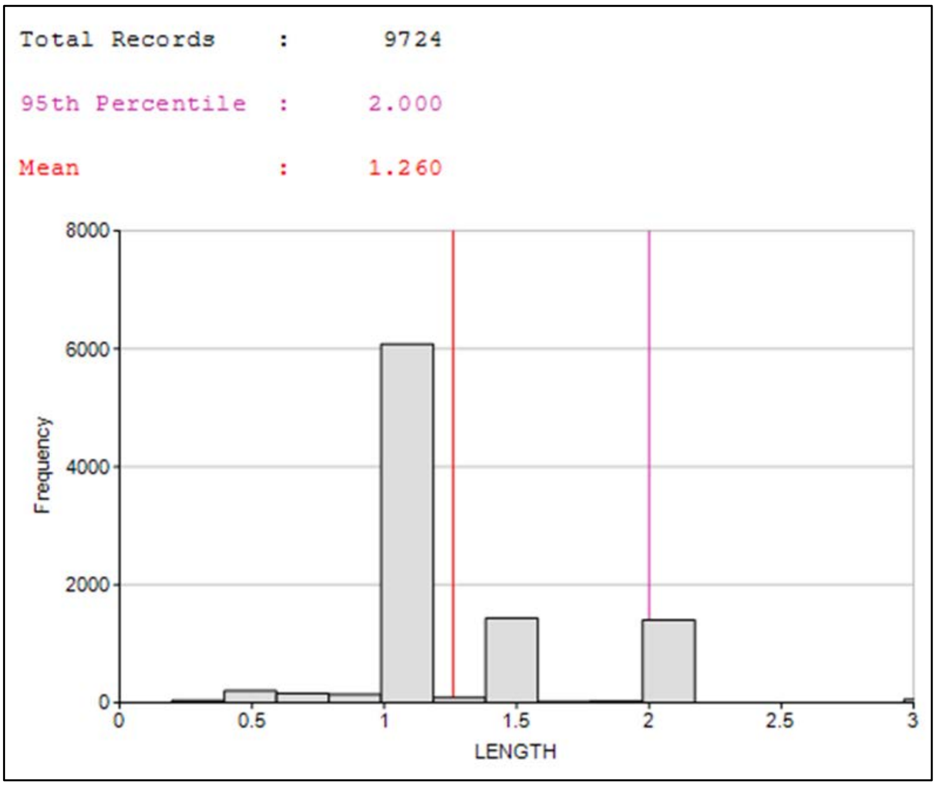


Figure 14-89: Asuadai Histograms of the Sample Lengths for the Original Drill Hole Data

Table 14-83: Asuadai Composite Analysis of Top cut data

Domain	Parameter	Composite 1m	Composite 1.5m	Composite 2m	1m vs 1.5m	1m vs 2m	1.5m vs 2m
ODO	AU	1.46	1.48	1.50	-1%	-3%	-1%
ODE	AU	0.17	0.16	0.17	-3%	1%	-3%
OSO	AU	1.30	1.30	1.28	1%	1%	1%
OSE	AU	0.15	0.16	0.15	3%	-5%	3%
TDO	AU	1.49	1.53	1.57	-2%	-6%	-2%
TDE	AU	0.18	0.19	0.19	-2%	-8%	-2%
TSO	AU	1.04	1.06	1.06	0%	-2%	0%
TSE	AU	0.09	0.09	0.09	-7%	-5%	-7%
FDO	AU	1.42	1.45	1.43	2%	0%	2%
FDE	AU	0.30	0.30	0.31	-4%	-6%	-4%
FSO	AU	1.22	1.21	1.20	0%	1%	0%

14.15.4 Asuadai Statistical Analysis

The following domain generation descriptive statistics of the individual domains were calculated Table 14-84

Table 14-84: Asuadai Descriptive Statistics for the Various Domains Generated

Domain	Parameter	Num records	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
ODO	AU	80	0.005	13.760	1.48	5.02	2.24	ODO
ODE	AU	80	0.005	1.930	0.16	0.08	0.29	ODE
OSO	AU	351	0.005	36.260	1.30	5.93	2.43	OSO
OSE	AU	1 269	0.025	77.000	0.16	4.90	2.21	OSE
TDO	AU	109	0.120	16.300	1.53	3.55	1.88	TDO
TDE	AU	106	0.005	3.700	0.19	0.25	0.50	TDE
TSO	AU	323	0.005	11.040	1.06	2.38	1.54	TSO
TSE	AU	1 796	0.002	9.030	0.09	0.15	0.39	TSE
FDO	AU	266	0.017	40.390	1.45	11.64	3.41	FDO
FDE	AU	209	0.002	18.710	0.30	1.91	1.38	FDE
FSO	AU	646	0.002	18.260	1.21	4.07	2.02	FSO
FSE	AU	4 146	0.002	11.770	0.09	0.16	0.40	FSE

14.15.5 Asuadai Outlier Analysis

Outlier analysis was undertaken using histograms and probability plots. The outliers were determined for both the variography and the kriging stages. For variography the outliers are cut at a specified value, but for kriging the outliers are capped at a specified value. Hence, all available data is used in the kriging of the data, but a restricted data set is used for variography.

The histograms and probability plots indicate that the populations of gold grades are close to log normal, which is typical of many gold deposits. The co-efficient of variation are moderately high but typical for these types of deposits. The kriging capping and the variogram top-cut values applied are summarised in Table 14-85 and Table 14-86 respectively.

Table 14-85: Asuadai Kriging Capping Values per Domain

Domain	Commodity	Capping g/t
FDE	AU	4
FDO	AU	15
FSE	AU	8
FSO	AU	10
ODE	AU	1
ODO	AU	8
OSE	AU	11
OSO	AU	10
TDE	AU	1
TDO	AU	6
TSE	AU	6
TSO	AU	10

Table 14-86: Asuadai Variography Top-Cut Values per Domain

Domain	Commodity	Top Cut g/t
FDE	AU	2
FDO	AU	7
FSE	AU	4
FSO	AU	8
ODE	AU	1
ODO	AU	4
OSE	AU	7
OSO	AU	7
TDE	AU	1
TDO	AU	6
TSE	AU	4
TSO	AU	6

14.15.6 Asuadai Variography Parameters

Point experimental variograms were generated and modelled for each domain.

The parameters of the modelled variograms for the Asuadai Project are summarised in Table 14-87.

Table 14-87: Asuadai Variogram Parameters

Domain		Sill	Nugget %	Sill 1%	Range1 m	Range2 m	Range3 m	Sill 2%	Range1 m	Range2 m	Range3 m
FDE	AU	0.086	31.40	38.37	6	5	8	100	80	14	14
FDO	AU	1.144	22.64	23.25	35	48	7	100	110	87	12
FSE	AU	0.046	26.09	67.39	5	11	3	100	85	27	19
FSO	AU	1.551	23.15	72.40	32	15	14	100	99	29	21
ODE	AU	0.027	33.33	40.74	5	8	11	100	23	28	20
ODO	AU	0.762	29.92	33.33	4	23	6	100	10	52	11
OSE	AU	0.085	35.29	50.59	4	8	5	100	26	19	23
OSO	AU	1.369	22.13	59.17	7	26	8	100	19	60	20
TDE	AU	0.008	25.00	37.50	20	14	21	100	31	30	31
TDO	AU	1.558	24.33	40.05	7	23	4	100	12	49	14
TSE	AU	0.052	36.54	36.54	10	12	11	100	61	48	24
TSO	AU	1.000	43.40	44.80	5	8	6	100	21	29	11

14.15.7 Asuadai Estimation Methodology

The interpolation was conducted with a block size of 10 x 10 x 3m (X & Y and Z), ordinary kriging was undertaken with a discretisation of 5 x 5 x 3 (X and Y and Z).

Table 14-88 and Table 14-89 show the search parameters used in the estimation.

Table 14-88: Asuadai Search Ranges and Angles

Domain		sdist1	sdist2	sdist3	sangle1	sangle2	sangle3	saxis1	saxis2	saxis3
FDE	AU	80	30	15	30	60	0	3	2	3
FDO	AU	110	80	15	30	60	0	3	2	3
FSE	AU	85	30	15	30	50	0	3	2	3
FSO	AU	100	30	20	30	50	0	3	2	3
ODE	AU	30	30	20	30	60	0	3	2	3
ODO	AU	10	50	10	30	60	0	3	2	3
OSE	AU	30	20	20	30	50	0	3	2	3
OSO	AU	20	60	20	30	50	0	3	2	3
TDE	AU	40	30	25	30	60	0	3	2	3
TDO	AU	20	50	15	30	60	0	3	2	3
TSE	AU	60	45	20	30	50	0	3	2	3
TSO	AU	30	30	10	30	50	0	3	2	3

Table 14-89: Asuadai -Number of Samples in Search

Domain		Min num1	Max num1	svolfac2	Min num2	Max num2	svolfac3	Min num3	Max num3	Max Key
FDE	AU	6	24	2	4	16	5	2	11	6
FDO	AU	6	24	2	4	16	5	2	11	6
FSE	AU	6	24	0	4	16	0	2	11	6
FSO	AU	6	24	2	4	16	5	2	11	6
ODE	AU	6	24	2	4	16	5	2	11	6
ODO	AU	6	24	2	4	16	5	2	11	6
OSE	AU	6	24	0	4	16	0	2	11	6
OSO	AU	6	24	2	4	16	5	2	11	6
TDE	AU	6	24	2	4	16	5	2	11	6
TDO	AU	6	24	2	4	16	5	2	11	6
TSE	AU	6	24	0	4	16	0	2	11	6
TSO	AU	6	24	2	4	16	5	2	11	6

A cross section and plan view of the Asuadai Block model is shown Figure 14-90 and Figure 14-91 respectively.

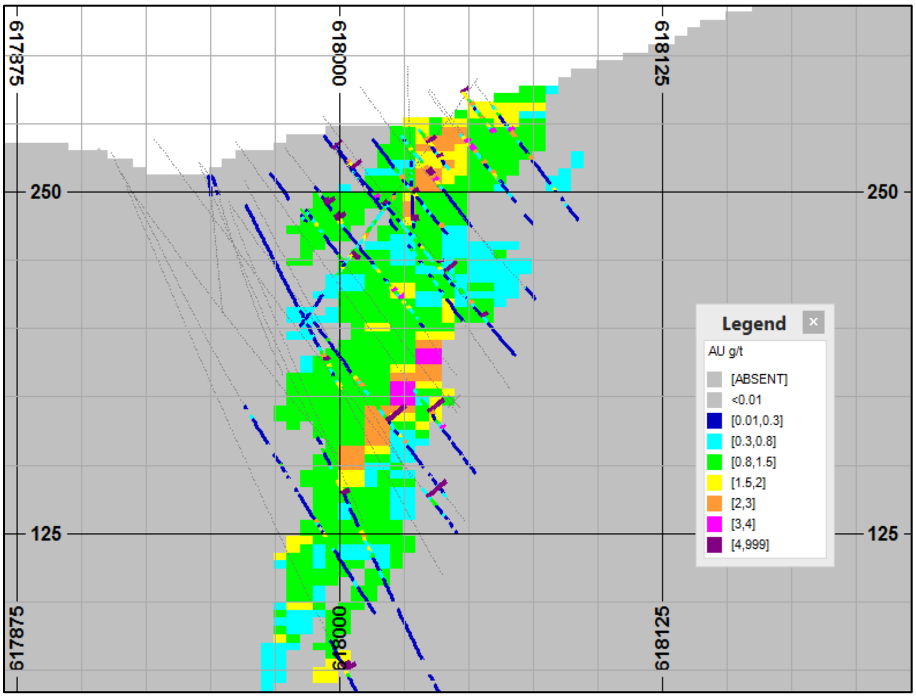


Figure 14-90: Asuadai Block Model Cross Section N 709200

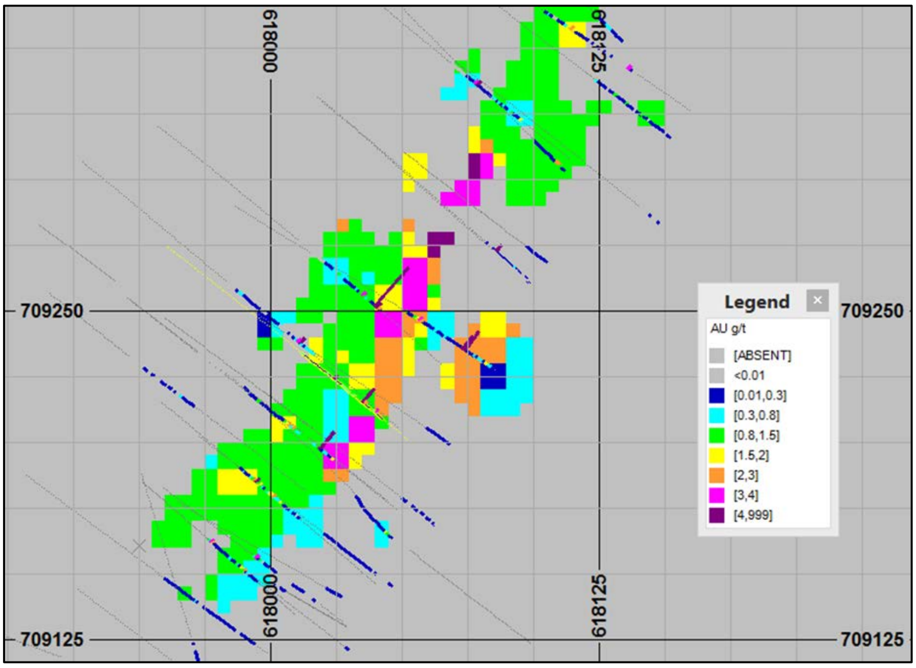


Figure 14-91: Asuadai Block Model Plan View 180m

14.15.8 Asuadai Resource Statement

The Asuadai Project was not re-estimated for this report. The Resource statement was updated using the original April 2014 estimation with a 0.5g/t Au cutoff grade within a US$1,500/oz Au pit-shell.

Table 14-90 shows the total MR sub-divided into Measured, Indicated and Inferred; as well as Oxide, Transition and Fresh zones.

Table 14-90: Asuadai Mine MR's at 0.5g/t Au cut-off grade, as at April 2014, within a US$1,500/oz pit shell

Asuadai	0.5 g/t cutoff constrained to $1500/oz Au shell				
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Indicated	Oxide	0.40	1.23	15,703	0.016
	Transition	0.51	1.10	18,046	0.018
	Fresh	0.97	1.27	39,852	0.040
Total Indicated		**1.88**	**1.22**	**73,600**	**0.074**
Total Measured and Indicated		**1.88**	**1.22**	**73,600**	**0.074**
Inferred	Oxide	0.00	0.69	88	0.000
	Transition	0.04	1.77	2,243	0.002
	Fresh	0.58	1.75	32,784	0.033
Total Inferred		**0.63**	**1.75**	**35,115**	**0.035**

Notes:

- Columns may not add up due to rounding .
- All figures are in metric tonnes.
- The MR's are stated as in situ tonnes. .
- The tonnages and contents are stated as 100%, which means no attributable portions are stated in the table.

The MR's for Asuadai are shown at different cutoff grades in Table 14-91.

Table 14-91: Asuadai Mine Mineral Resources at different cut-off grades, as at April 2014

Resource Category	Cut-off Grade Au g/t	Tonnes Mt	Grade Au g/t	ContentAu Moz
Indicated	0	2.03	1.18	0.077
	0.3	2.02	1.19	0.077
	0.4	2.00	1.20	0.077
	0.5	1.97	1.21	0.076
	0.6	1.88	1.24	0.075
	0.7	1.75	1.28	0.072
	0.8	1.57	1.34	0.068
	0.9	1.35	1.42	0.062
	1	1.14	1.51	0.055
	1.5	0.41	2.03	0.027
Inferred	0	0.94	1.58	0.048
	0.3	0.93	1.59	0.048
	0.4	0.93	1.60	0.048
	0.5	0.92	1.61	0.047
	0.6	0.89	1.64	0.047
	0.7	0.86	1.68	0.046
	0.8	0.81	1.74	0.045
	0.9	0.73	1.83	0.043
	1	0.65	1.94	0.041
	1.5	0.36	2.53	0.029

14.15.9 Resource Classification

Figure 14-92 shows the Asuadai block model resource classification. The Resource is also reported within a $1,500 / oz Au pit shell.

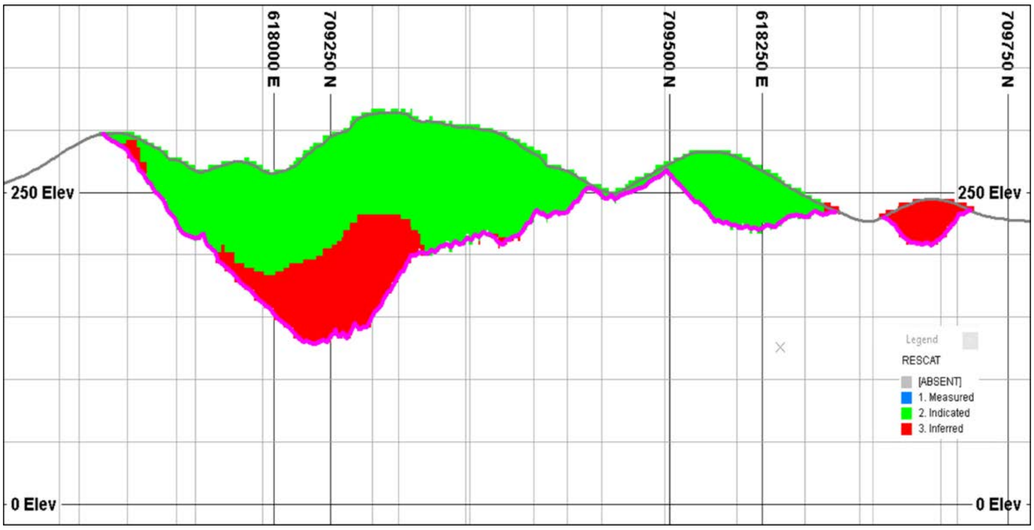


Figure 14-92: Asuadai Resource Classification Block model- Transverse section

The Asuadai grade-tonnage curve is shown in Figure 14-93.

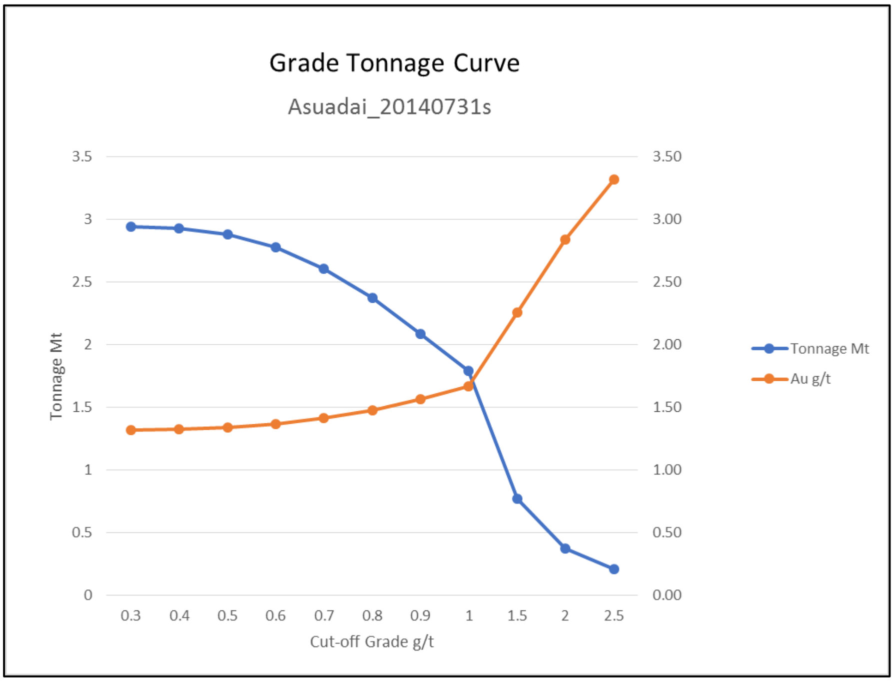


Figure 14-93: Asuadai Grade Tonnage Curve

14.16 Akwasiso (CSA/Asanko)

Akwasiso is a newly evaluated deposit located on the Nkran shear corridor approximately 4km north of the Nkran Pit.

14.16.1 Weathering

The weathering profile at Akwasiso was used to code composites. The profile was modelled from drill data and comprises weathered oxide and fresh / unweathered units Figure 14-94. In general, the weathering surfaces are broadly parallel to the topographical profile, although weathering tends to be deeper within zones of mineralisation and tend to parallel the footwall to the mineralisation where the footwall approaches the surface.

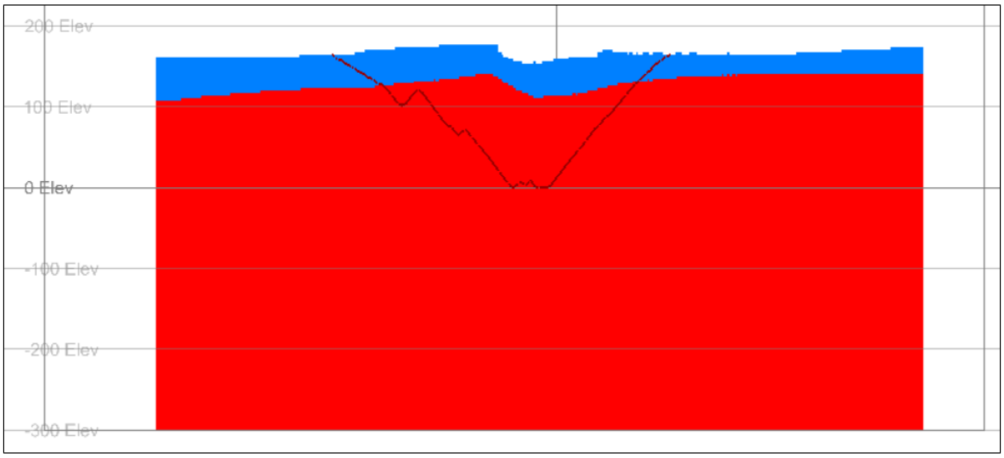


Figure 14-94: Akwasiso W-E cross section view of weathering profile

14.16.2 Domains

The domains for the Akwasiso Project are based on the latest litho-structural geological model completed during the Q1 2017 infill drilling program. The main mineralisation is associated with the granitic intrusion. The mineralisation extends into the adjacent sedimentary rocks.

The final domains used for estimation are listed in Table 14-92 and shown in Figure 14-95.

Table 14-92: Akwasiso domains used in the MRE

Litho Domain	Lithology	
0	Shale / Siltstones	shales_siltstonetr/pt.dm
1	Sandstone	Westernmost_Sandsttr/pt.dm
2	Sandstone	Western_Sandstonetr/pt.dm
3	Sandstone	Central_Sandstonetr/pt.dm
4	Sandstone	Eastern_Sandstonetr/pt.dm
5	Granite (skinny granite)	Skinny_Granitetr/pt.dm
6	Granite (main granite)	Granitetr/pt.dm

Litho Domain	Lithology	
Rock		
SI	Shales / Siltstones	Zone 0
SST	Sandstone	Zone 1, 2, 3 and 4
GR	Granite	Zone 5 and 6

The mineralised portion within the main litho-structural domains was delineated using the IK method. Data flagging was based on a 0.3 g/t cut-off including up to 2m waste. This indicator flag was used to perform detailed spatial analysis on the mineralised orientations within each domain. Variogram analysis was done in all directions and the best continuity directions selected. All of these estimation parameters were also related to the geological observations in the field and from drill holes.

A probability of 0.3 was selected to delineate the mineralisation in domain GR that strikes 030° and dips 80° northwest. For the estimation process, the domains are sub-divided into mineralised and waste portions. Figure 14-96 and Figure 14-97 show a cross section and isometric view of the domains. Figure 14-98 depicts an isometric view of the search ellipses along strike of the domains.

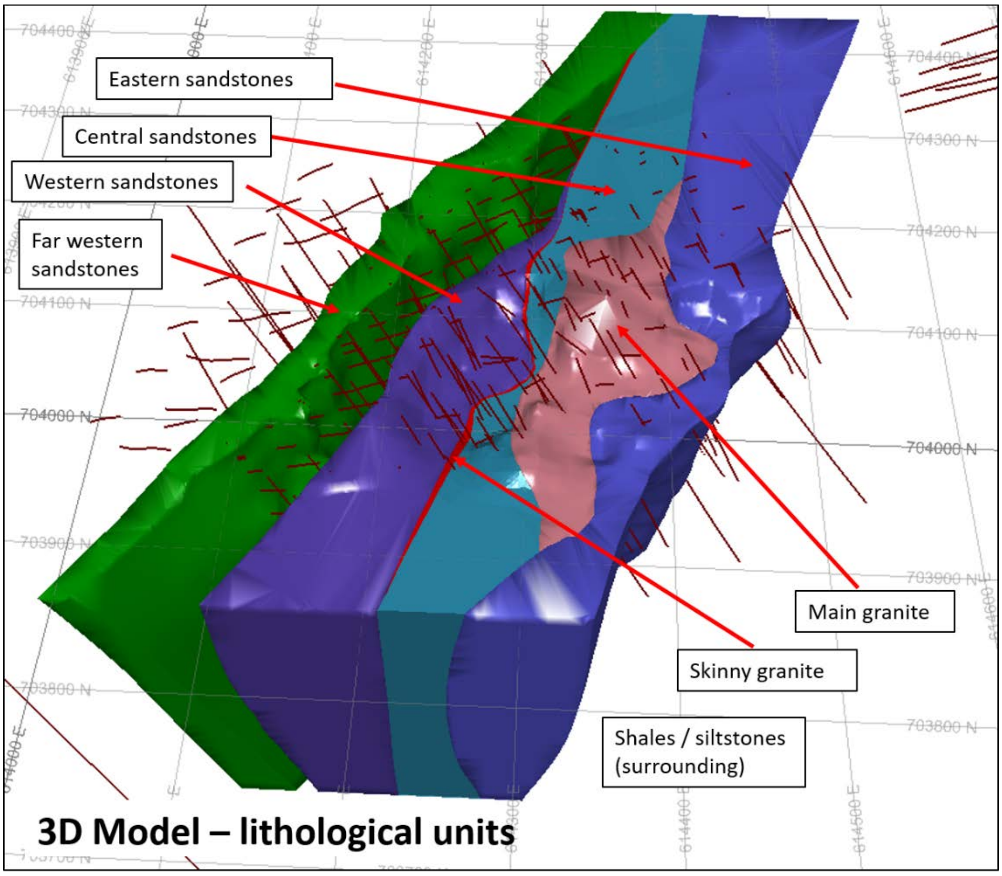


Figure 14-95: Akwasiso lithological domains

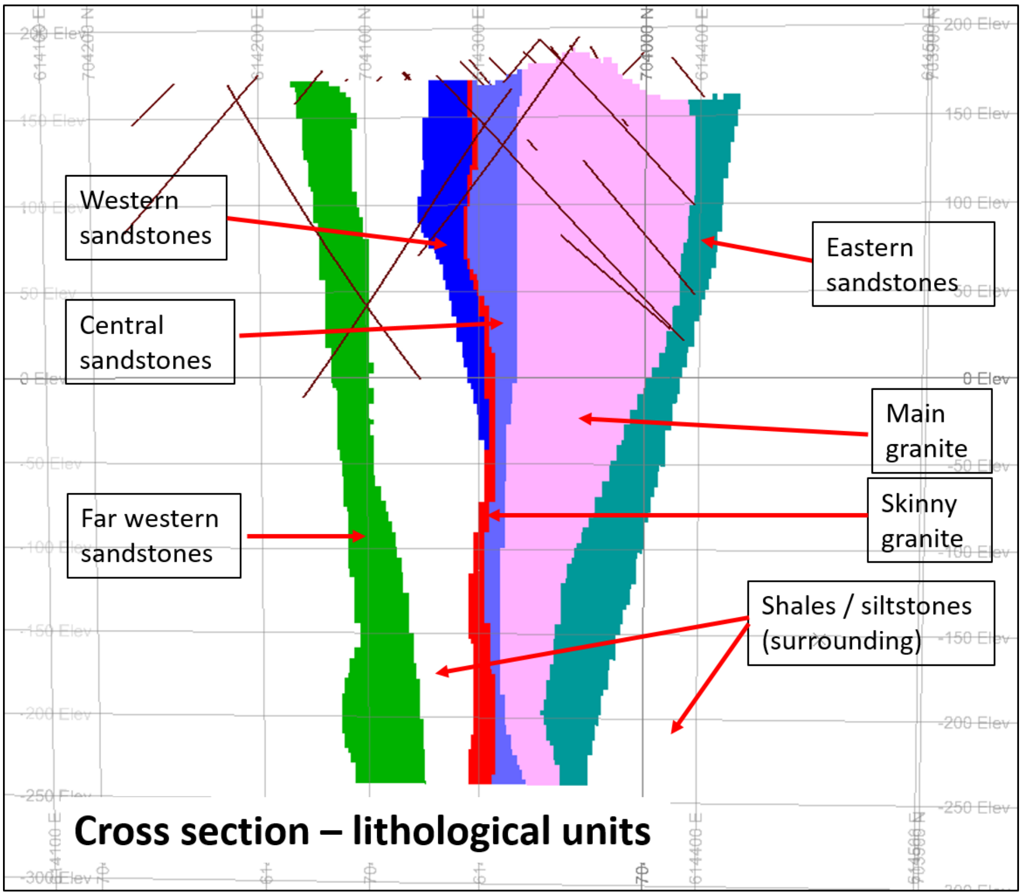


Figure 14-96: Akwasiso representative cross section of lithological units

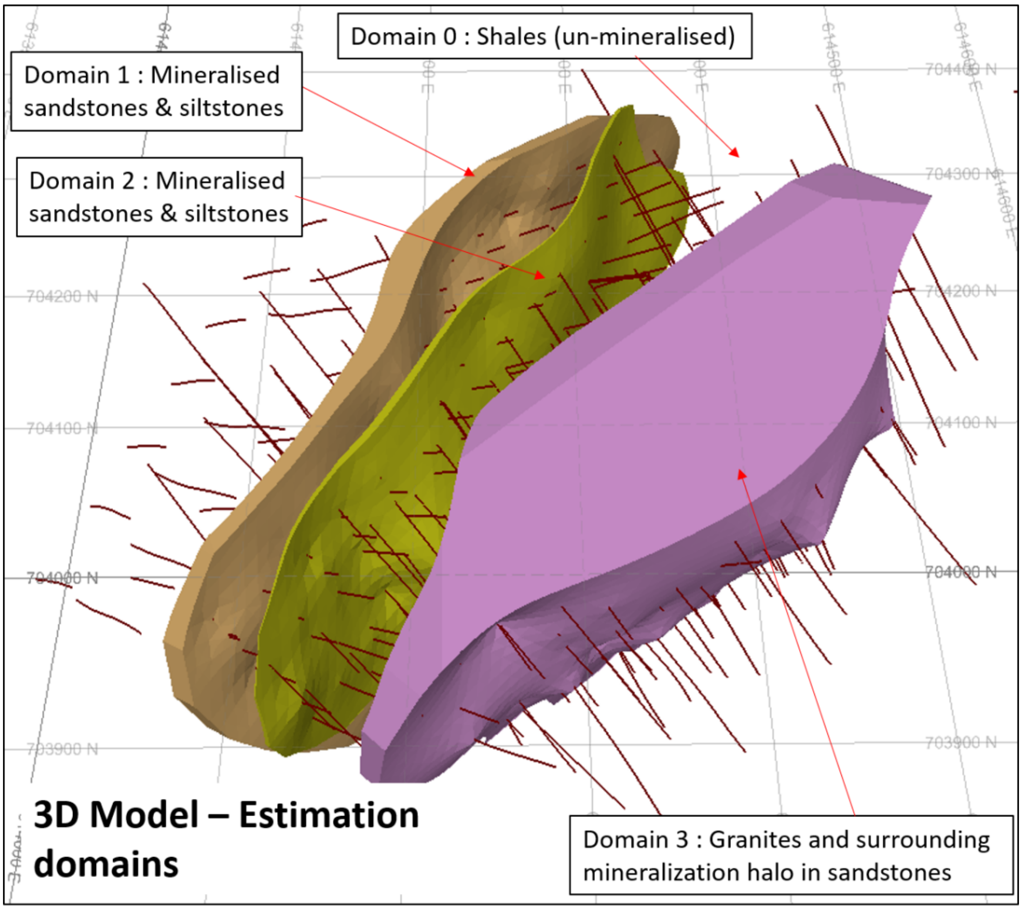


Figure 14-97: Akwasiso- Resource estimation domains

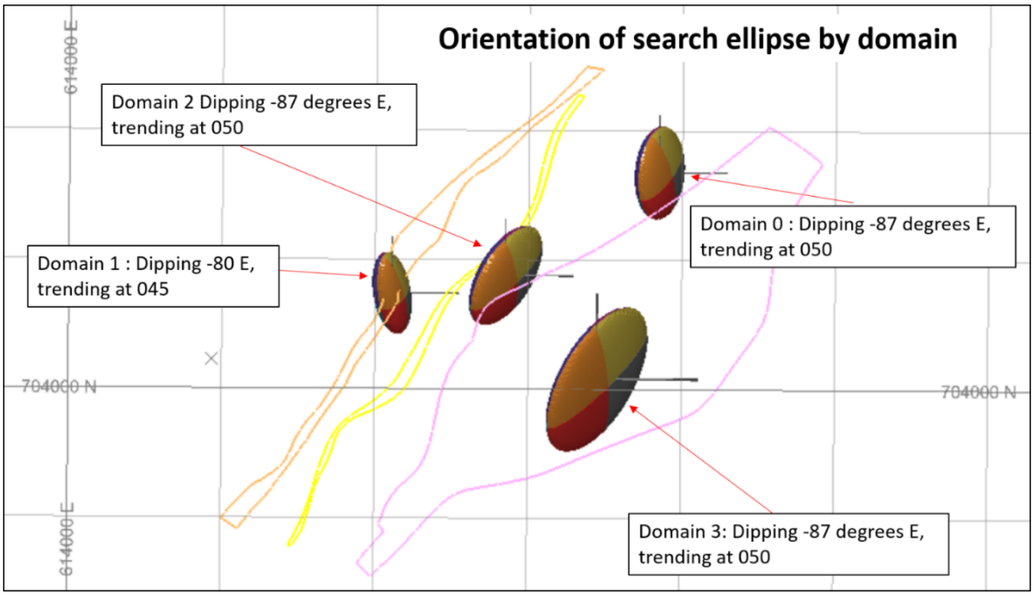


Figure 14-98: Akwasiso- search Ellipse along all domains

14.16.3 Akwasiso Compositing

The de-surveyed drill holes were composited within DatamineTM on a 1.5m composite length, supported by the histogram in Figure 14-99, which shows that 90% of the data is sampled at 1.5m intervals.

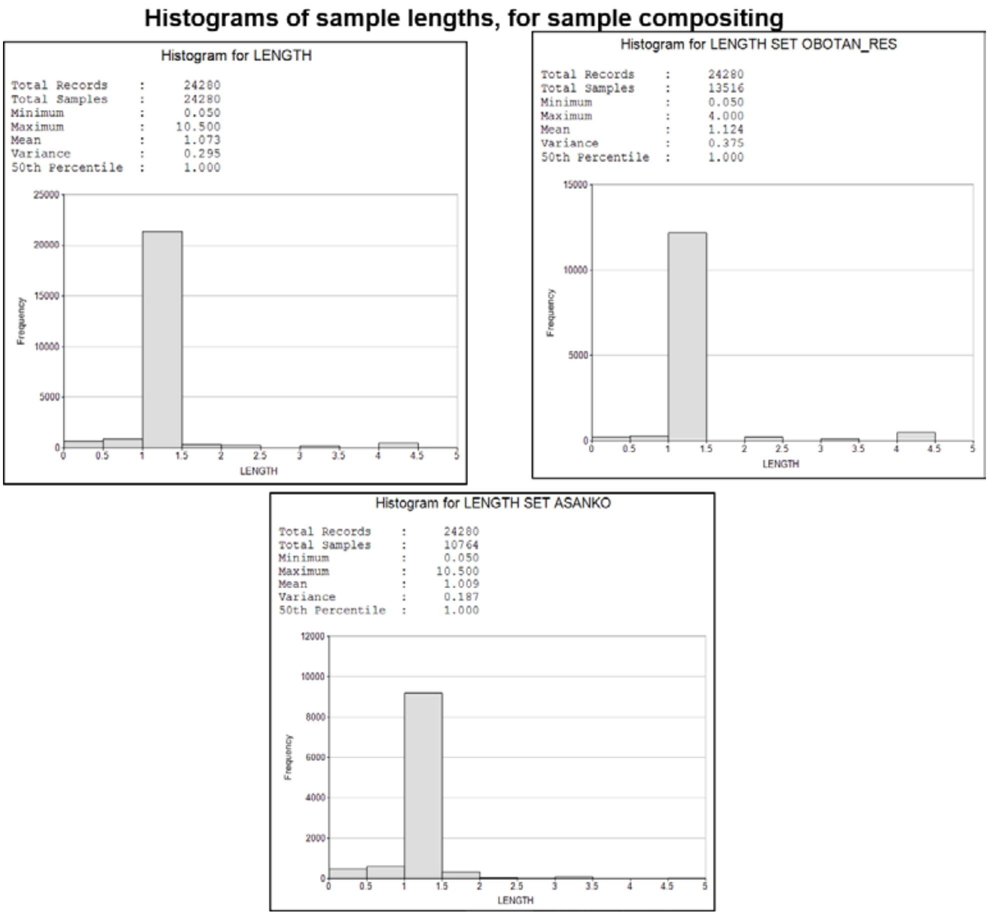


Figure 14-99: Akwasiso – Histogram of Drill Hole Sampling Lengths

14.16.4 Akwasiso Statistical Analysis

Following domain generation, descriptive statistics of the individual mineralized zones was undertaken Table 14-93 and Table 14-94 and Table 14-95

Zone Stats on 1m capped composites (declustered).

Table 14-93: Akwasiso Zones Descriptive Statistics (using capped composites)

Zone	Rock Type	Count	Min Au G/T	Max Au G/T	Mean Au G/T	Variance au G/T2	Standard Deviation Au G/T	Coefficient Of Variation
0	Shale	8,216	0.00	16.00	0.07	0.216	0.46	7.14
1	Sandstone	3,567	0.00	35.00	0.25	1.510	1.23	4.92
2	Sandstone	2,981	0.00	40.00	0.38	4.278	2.07	5.44
3	Sandstone	1,928	0.01	17.00	0.23	1.082	1.04	4.49
4	Sandstone	3,339	0.00	27.00	0.37	3.174	1.78	4.77
5	Granite	276	0.00	13.07	0.48	2.066	1.44	3.01
6	Granite	4,284	0.00	40.00	0.54	2.577	1.61	2.99

Domain Stats on 1m capped composites (declustered.

Table 14-94: Akwasiso Domains descriptive statistics (using capped composites)

LFDO Main	Rock type	Count	Min Au g/t	Max Au g/t	Mean Au g/t	Variance Au g/t2	Standard Deviation Au g/t	Coefficient of Variation
0	Shale	15,018	0.00	40.00	0.06	0.22	0.47	7.95
1	Sandstone	1,187	0.01	35.00	0.75	7.50	2.74	3.66
2	Sandstone	482	0.01	16.00	0.87	4.15	2.04	2.35
3	Sandstone/Granite	7,904	0.00	40.00	0.74	5.15	2.27	3.06

Final MRE MODEL: Grade Shell Domains.

Table 14-95: Akwasiso Resource Model Statistics by Domain

Zone	Rock type	Count	Min Au g/t	Max Au g/t	Mean Au g/t
0	Shale	206,621	0.00	9.16	0.07
1	Sandstone	22,636	0.02	13.24	1.41
2	Sandstone	12,198	0.01	9.26	0.92
3	SandstoneGranites	102,691	0.01	10.09	0.73

14.16.5 Akwasiso Outlier Analysis

Outlier analysis was undertaken using histograms and probability plots. The outliers were determined for both the variography and the kriging stages. For variography the outliers are cut at a specified value, but for kriging the outliers are capped at a specified value. Hence, all available data is used in the kriging of the data, but a restricted data set is used for variography.

The histograms and probability plots indicate that the populations of gold grades are close to log normal, which is typical of many gold deposits. The co-efficient of variation are moderately high but typical for these types of deposits. The kriging capping and the variogram top-cut values applied are summarised in Table 14-96.

Table 14-96: Top Cut analysis by Zone

Zone	Top Cut	Reduction in Mean Au	No of samples cut
Zone 0	16	7%	2
Zone 1	35	15%	4
Zone 2	40	21%	4
Zone 3	17	18%	4
Zone 4	27	10%	4
Zone 5	40	0%	0
Zone 6	40	2%	1

14.16.6 Akwasiso Variography Parameters

The mineralization follows a SW-NE trend, dipping at near vertically (80 - 87 degrees) to the SE. The gold is concentrated in sandstone and granitic units.

The direction of mineralization, based on the modelled grade envelopes, does not align with the trend of the lithological units. The concentration of the main mineralization (Domain 3) is offset and trends approximately 045 (NE-SW), whereas the lithology trends at 030. The variograms were modelled in the same direction as the search ellipsoids, see tables below.

Point experimental variograms were generated and modelled for each domain, on composite data with excluded outliers. The parameters of the modelled variograms for the Akwasiso Project are summarised in Table 14-97. Variograms were modelled on composite data with excluded outliers. Domain boundaries (Zone) were all hard with respect to the domain (i.e. grade envelope) boundaries and soft with respect to lithological contacts.

Table 14-97: Akwasiso Parameters of Modelled Variograms

Vrefnum	Vangle1	Vangle2	Vangle3	Vaxis1	Vaxis2	Vaxis3	Nugget
0	45	0	0	3	1	3	0.09
1	45	0	0	3	1	3	0.97
2	45	0	0	3	1	3	0.51
3	45	0	0	3	1	3	0.16
4	45	0	0	3	1	3	0.83
5	45	0	0	3	1	3	0.35
6	45	0	0	3	1	3	0.62

ST1	ST1 PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4
1	68.38	54.35	3.39	0.12	1	113.98	126.22	39.56	0.08
1	11.13	9.65	3.80	0.86	1	20.69	20.92	11.99	0.53
1	4.04	18.10	3.29	1.02	1	18.31	25.88	14.97	0.21
1	16.10	33.77	5.48	0.19	1	57.44	144.53	12.50	0.28
1	1.15	11.15	9.45	0.15	1	5.06	24.12	39.86	0.98
1	1.15	24.73	22.66	0.37	1	2.12	45.04	50.71	1.08
1	4.05	10.16	16.31	0.74	1	10.23	21.92	23.45	0.96

14.16.7 Akwasiso Estimation Methodology

For the MRE Ordinary Kriging was used as the gold grade interpolation method. The direction was taken from the Isoshell at a 0.5 g/t cut-off with and azimuth of 30 degrees and dip direction of -40 degrees in the “Y” direction. The block size used was 10m x 20m x 6m.

For the estimation and search parameters KNA was run for each domain based on the variograms. Slopes of regression and presence of negative weights in representative locations (well, moderately and poorly informed) were used as a guideline to choose the max number of samples. Searches were made larger than Variogram ranges, to there were sufficient samples in the first search pass. A second search pass using less samples was added to estimate fringe blocks. Discritization points of 5 x 5 x 6 points were used for all domains.

Table 14-98 and Table 14-99 show the search parameters used in the estimation.

Table 14-98: Akwasiso MRE Variography Search Parameters

Zone	S Method	Sdist1	Sdist2	Sdist3	Sangle1	Sangle2	Sangle3	Saxis1	Saxis2	Saxis3
0	1	40	25	10	30	0	-80	3	1	2
1	1	40	15	10	30	0	-80	3	1	2
2	1	40	35	10	40	0	-87	3	1	2
3	1	60	50	10	40	0	-87	3	1	2

Table 14-99: Akwasiso MRE Variography Search Parameters

Min Num1	Max Num1	Svolfac2	Min Num2	Max Num2	Svolfac2	Min Num2	Max Num2	Oct Meth
3	4							0
5	9	2	3	8	3	3	5	0
5	9	2	3	8	3	3	5	0
5	12	2	3	8	3	3	5	0

Figure 14-100 shows a plan view of the block model, and Figure 14-101 and Figure 14-102 and Figure 14-103 show cross sections through the block model with the Au grades as estimated.

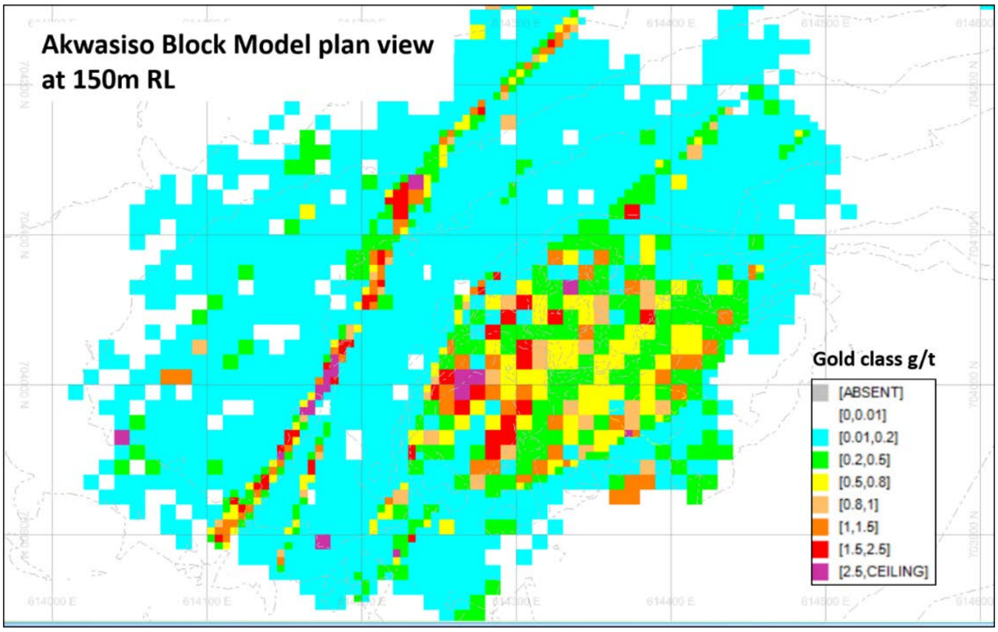


Figure 14-100: Akwasiso Block Model Plan View

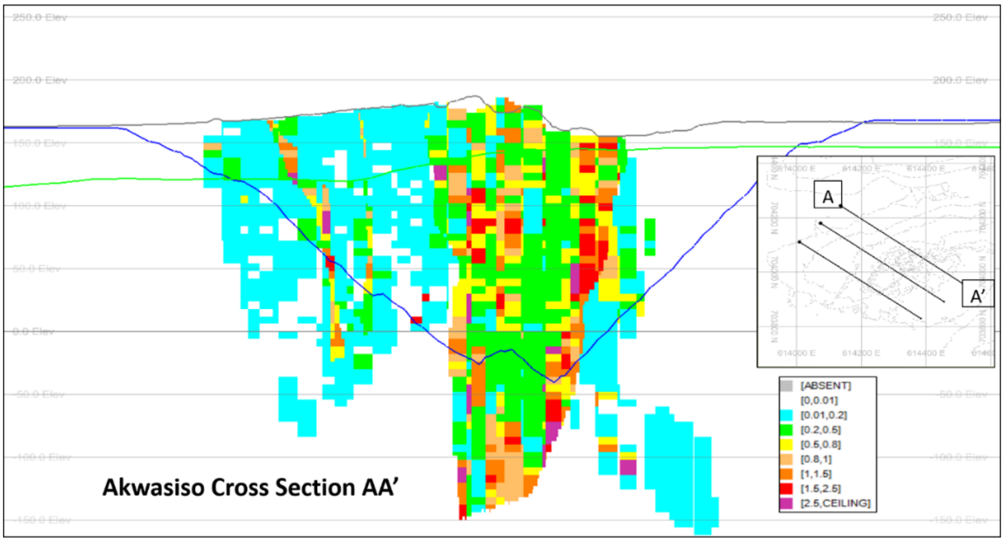


Figure 14-101: Akwasiso Block Model W-E Cross Section A-A'

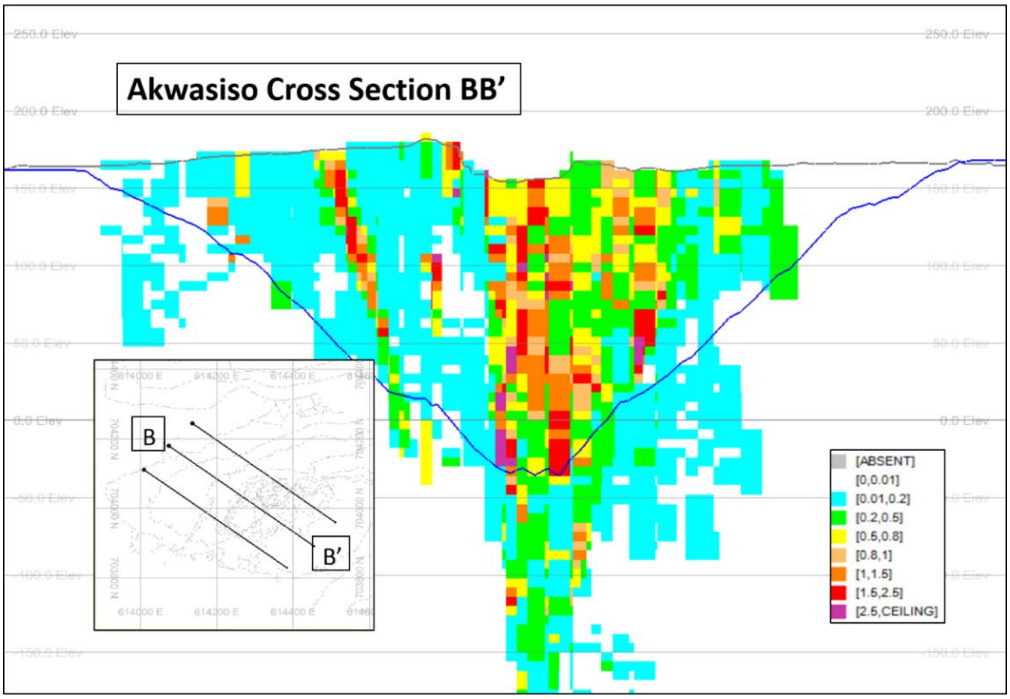


Figure 14-102: Akwasiso Block Model W-E Cross Section B-B’

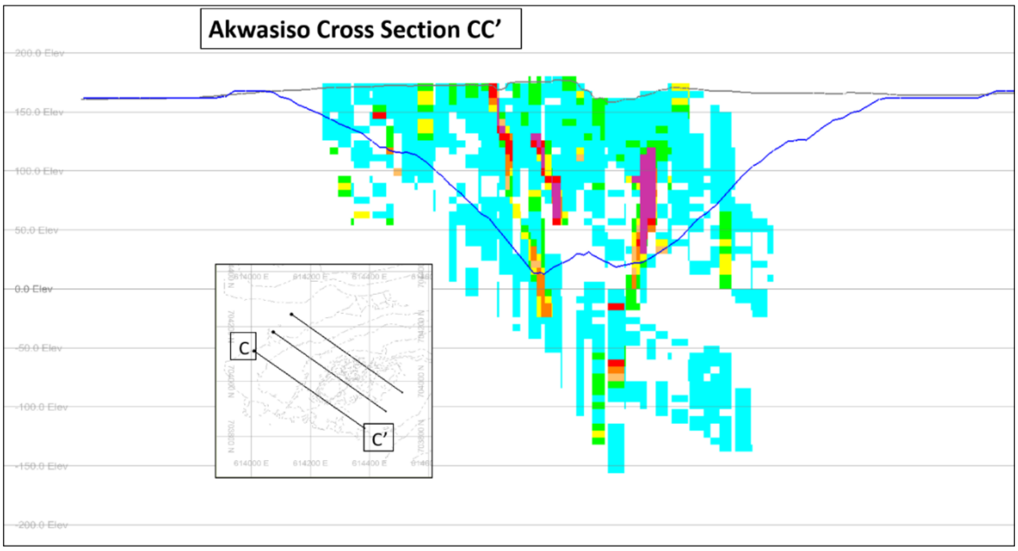


Figure 14-103: Akwasiso Block Model W-E Cross Section C-C’

14.16.8 Akwasiso Resource Statement

The Akwasiso deposit mineral resources have been re-estimated post a further phase of evaluation drilling during Q1 2017. The Resource statement was updated within a $1,500/oz Au pit-shell, at a Au cut-off grade of 0.5g/t Au.

Table 14-100 shows the total MR sub-divided into Measured, Indicated and Inferred; as well as Oxide, Transition and Fresh zones.

Table 14-100: Akwasiso MR's as at April 2017, 0.5 g/t cutoff within a $1,500/oz pit shell

Akwasiso	0.5 g/t cutoff constrained to $1500/oz Au shell				
Cut-off grade 0.5 g/t Au	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Measured	Oxide	0.00	-	-	-
	Transition	0.00	-	-	-
	Fresh	0.00	-	-	-
Total Measured		**0.00**	**-**	**-**	**-**
Indicated	Oxide	1.39	1.27	56,937	0.057
	Transition	0.00	-	-	-
	Fresh	4.93	1.56	248,244	0.248
Total Indicated		**6.33**	**1.50**	**305,181**	**0.305**
Total Measured and Indicated		**6.33**	**1.50**	**305,181**	**0.305**
Inferred	Oxide	0.06	0.97	1,784	0.002
	Transition	0.00	-	-	-
	Fresh	0.06	1.02	1,912	0.002
Total Inferred		**0.12**	**0.99**	**3,696**	**0.004**

Notes:

- Using a US$1,500/oz Au pit shell.
- Columns may not add up due to rounding.
- All figures are in metric tonnes.
- The MR's are stated as in situ tonnes.
- Individual Densities was used per mineral zone.
- The tonnages and contents are stated as 100%.

Table 14-101: Akwasiso MR's at Different Cut-off Grades

Akwasiso 25 April 2017; $1500/oz Au shell				
Resource Category	Cutoff Au g/t	Tonnes	Au g/t	Au KOz
Indicated	0.40	7,451,464	1.34	321.02
	0.50	6,327,673	1.50	305.16
	0.60	5,499,985	1.64	290.00
	0.70	4,789,283	1.79	275.62
	0.80	4,160,306	1.95	260.83
	1.50	2,083,387	2.80	187.55
	2.50	936,482	3.88	116.82
Inferred	0.40	176,014	0.81	4.58
	0.50	115,752	0.99	3.68
	0.60	100,485	1.06	3.42
	0.70	79,018	1.17	2.97
	0.80	68,633	1.24	2.74
	1.50	9,485	2.47	0.75
	2.50	3,252	3.79	0.40

14.16.9 Akwasiso Resource Classification

The classification of the Akwasiso block model at different cut-off grades is shown above in Table 14-101. A cross section of the Indicated and Inferred resource classes is shown in Figure 14-94.

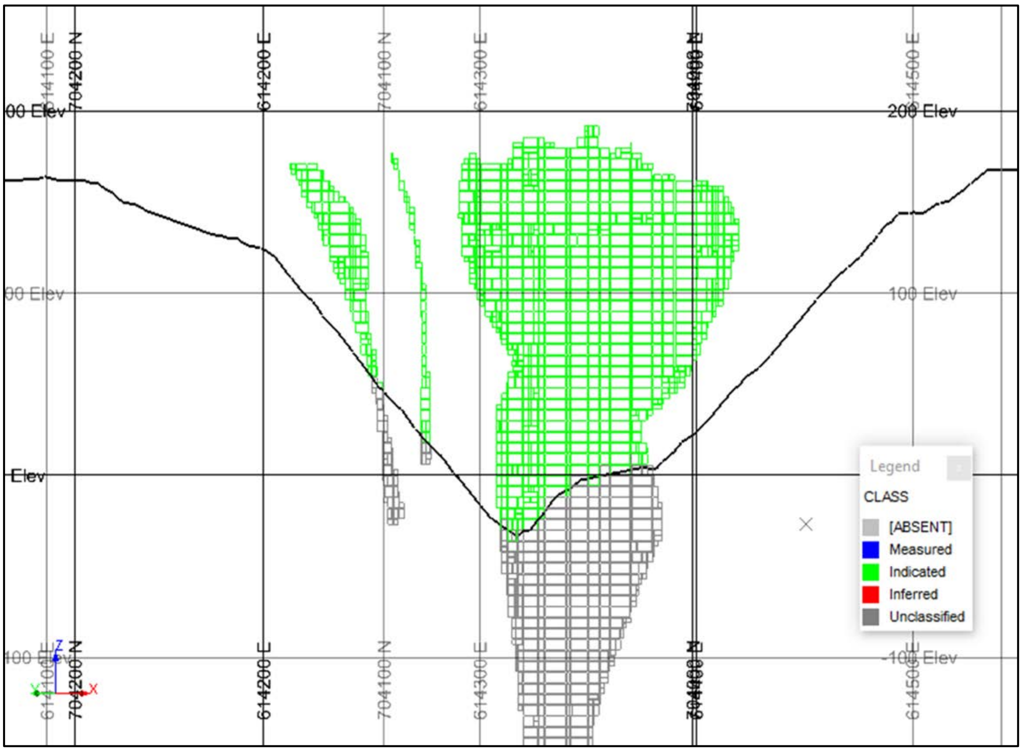


Figure 14-104: Akwasiso Resource Classification Block model- Cross Section

The grade – tonnage curve for the Akwasiso deposit is shown in Figure 14-105

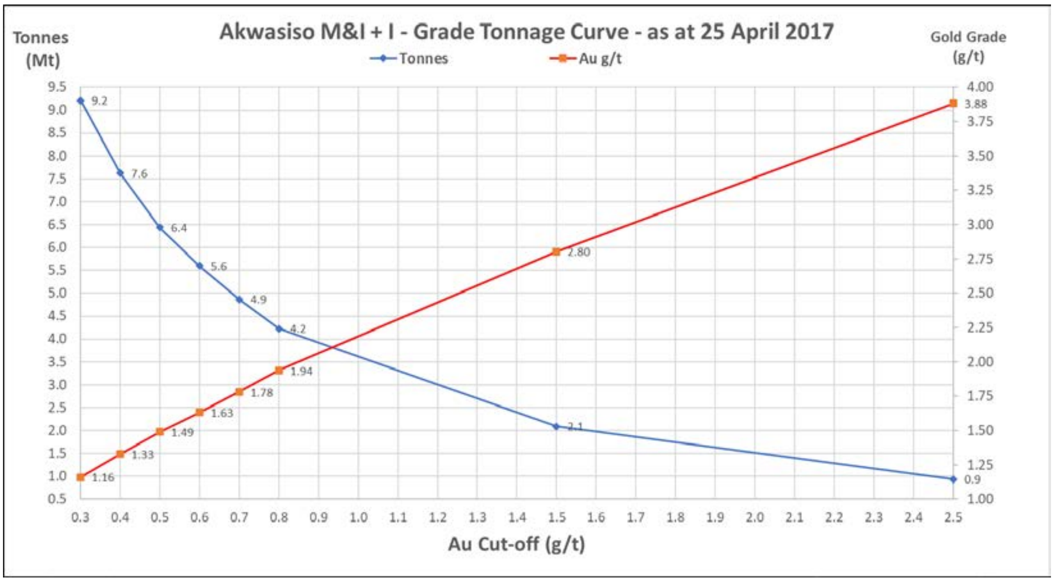


Figure 14-105: Akwasiso Grade Tonnage Curve

14.17 Esaase (CJM)

The Esaase MRE conducted by CJM in 2012 remains the basis of the estimation presented below, but has been audited by CSA Global in January 2017.

14.17.1 Weathering

Esaase composites were coded by the weathering profile. The profile was modelled from drill data and comprises strongly weathered saprolite, moderately weathered saprolite, transition material and fresh units. Esaase locates on a prominent topographic high, with one drainage cross cutting the ore body. In general, the weathering surfaces are broadly parallel to the topographical profile, although weathering tends to be deeper within zones of mineralisation and tend to parallel the foot wall to the mineralisation where the footwall approaches the surface. On some sections, the intermixing of the weathering types can be quite complicated. Figure 14-106 shows the distribution of the weathering types.

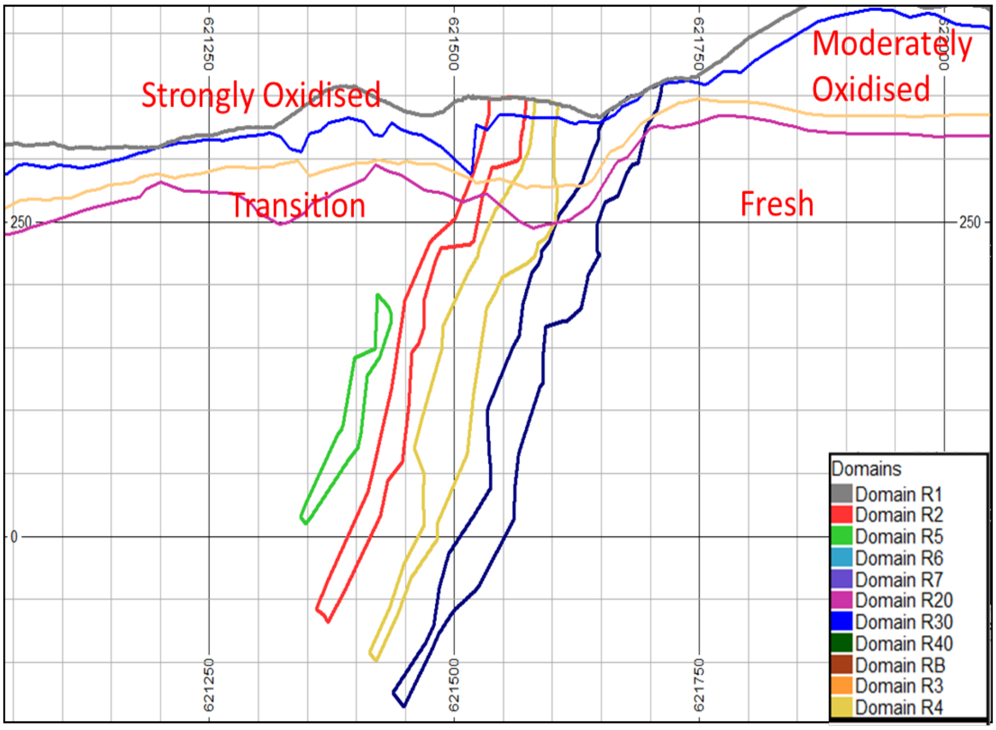


Figure 14-106: Sectional West to East View of Domains and Weathering Profile

Source: Esaase NI-43101, 2012

14.17.2 Domains

For the purpose of the MRE 11 main and 2 secondary mineralised domains were interpreted and modelled on an approximate lower cut-off grade of 0.3 g/t Au. The main mineralised domains are located within the previously broadly delineated mineralised zones, whereas the secondary mineralised domains are outside these main mineralised zones. The waste zone was assigned a default value of 0.005 g/t Au. The main domains are depicted in Figure 14-107.

To delineate the ore zones inside the previously delineated wireframes, IK was implemented using a cut-off grade of 0.3 g/t Au. A probability of 0.3 was selected as the best represented to delineate the mineralisation of the ore body.

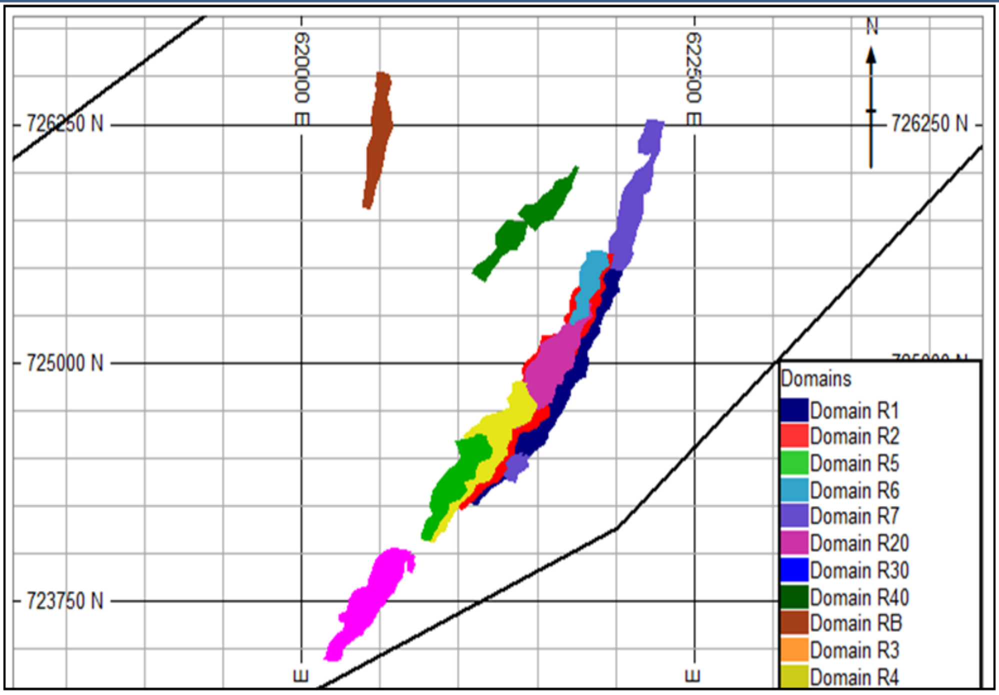


Figure 14-107: Geological Domains used for the MRE

Source: Esaase NI-43101, 2012

Figure 14-108 and Figure 14-109 show a plan view and East-West section through the block model showing the Indicator probabilities as estimated.

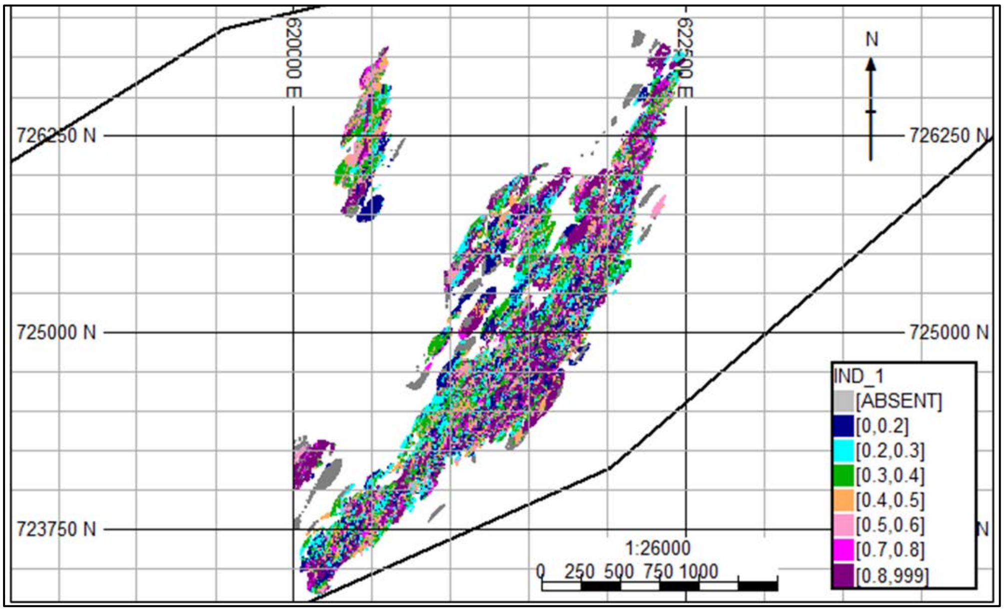


Figure 14-108: Plan View of Indicator Estimation

Source: Esaase NI-43101, 2012

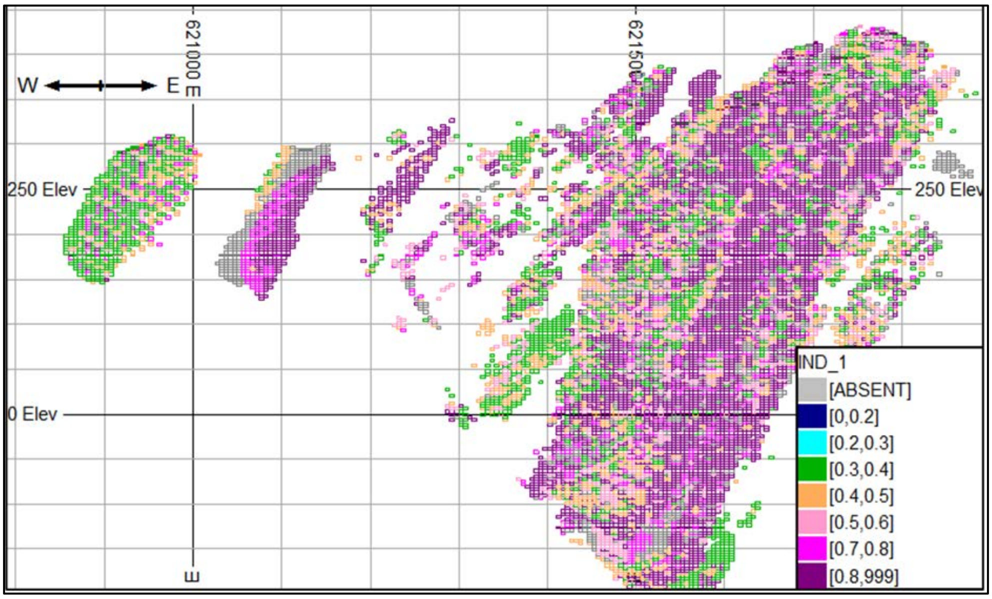


Figure 14-109: Esaase West – East Section View of Indicator Probabilities

Source: Esaase NI-43101, 2012

14.17.3 Esaase Compositing

The de-surveyed drill holes were composited within DatamineTM on a 1m composite length. A total of 233,503 composites were used in the statistical analysis and MRE Figure 14-110.

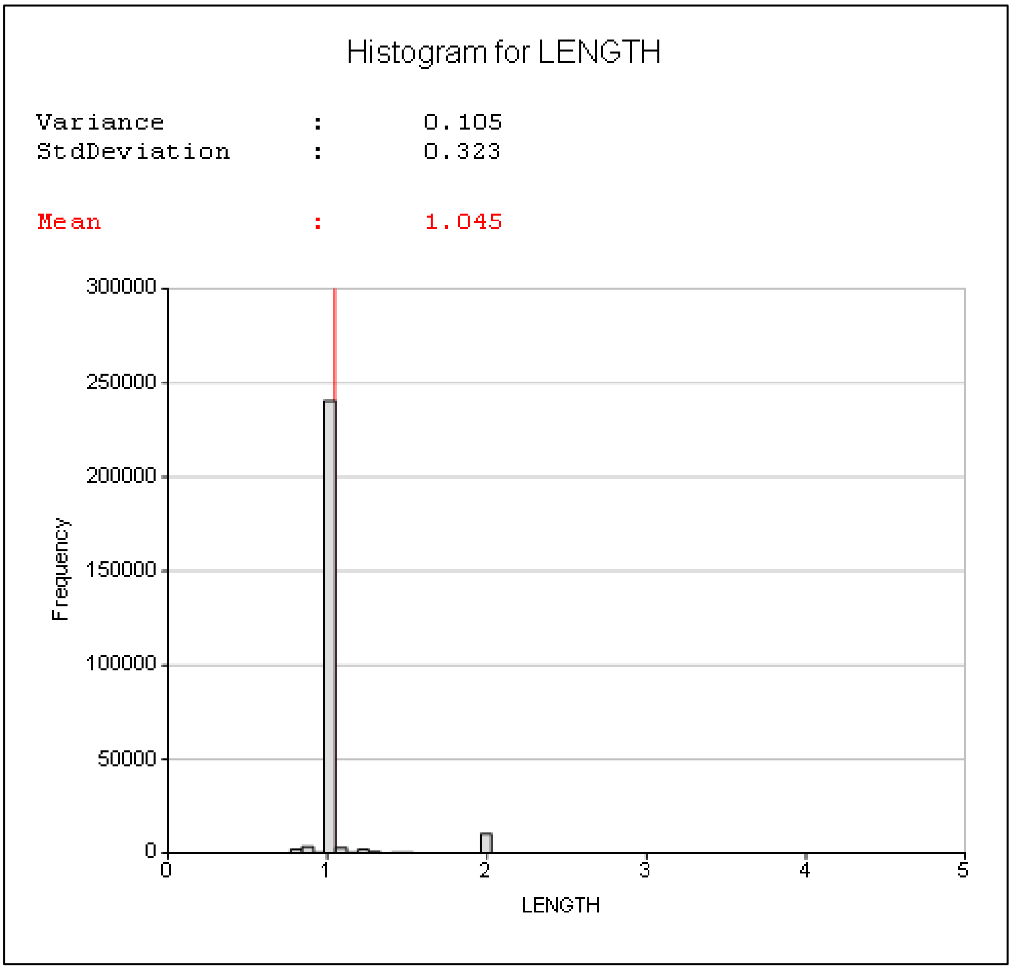


Figure 14-110: Esaase composited borehole length histogram

14.17.4 Esaase Statisitcal Analysis

Following domain generation, descriptive statistics of the individual mineralized zones was undertaken in Table 14-102.

Table 14-102: Esaase Descriptive Statistics for the Various Domains

Mineral Zone	Parameter	Domain	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
LR1	AU	1	0.001	20.000	1.11	3.708	1.926	1.735
LR2	AU	1	0.001	40.000	1.16	11.194	3.346	2.891
LR3	AU	1	0.001	10.000	1.11	4.729	2.175	1.961
LR4	AU	1	0.001	20.000	1.02	6.027	2.455	2.399
LR5	AU	1	0.001	3.000	0.57	0.387	0.622	1.095
LR6	AU	1	0.005	2.000	0.49	0.288	0.537	1.095
LR7	AU	1	0.001	1.500	0.45	0.201	0.449	0.998
LR20	AU	1	0.001	20.000	1.20	8.169	2.858	2.374
LR30	AU	1	0.001	5.000	0.77	1.610	1.269	1.639
LR40	AU	1	0.001	1.000	0.50	0.153	0.391	0.777
LRB	AU	1	0.040	3.000	0.94	0.490	0.700	0.748
OR1	AU	1	0.001	80.000	1.45	20.951	4.577	3.152
OR2	AU	1	0.005	30.000	1.01	5.950	2.439	2.420

Mineral Zone	Parameter	Domain	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
OR3	AU	1	0.005	40.000	1.68	17.496	4.183	2.490
OR4	AU	1	0.005	30.000	1.10	5.593	2.365	2.156
OR5	AU	1	0.001	10.000	0.78	1.652	1.285	1.612
OR6	AU	1	0.001	25.000	1.38	9.891	3.145	2.282
OR7	AU	1	0.026	2.000	0.47	0.230	0.480	1.021
OR3	AU	1	0.005	40.000	1.68	17.496	4.183	2.490
OR4	AU	1	0.005	30.000	1.10	5.593	2.365	2.156
OR5	AU	1	0.001	10.000	0.80	1.652	1.285	1.612
OR6	AU	1	0.001	25.000	1.38	9.891	3.145	2.282
OR7	AU	1	0.026	2.000	0.47	0.230	0.480	1.021
OR20	AU	1	0.001	40.000	1.15	9.252	3.042	2.639
OR30	AU	1	0.005	12.000	1.48	6.827	2.613	1.771
OR40	AU	1	0.010	15.000	1.19	5.929	2.435	2.046
ORB	AU	1	0.020	1.000	0.48	0.118	0.343	0.719
TR1	AU	1	0.001	80.000	1.68	31.967	5.654	3.366
TR2	AU	1	0.001	30.000	1.12	5.243	2.290	2.052
TR3	AU	1	0.005	69.000	2.00	41.453	6.438	3.216

Mineral Zone	Parameter	Domain	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
TR4	AU	1	0.005	42.000	0.85	7.310	2.704	3.176
TR5	AU	1	0.010	32.000	1.02	8.518	2.919	2.861
TR6	AU	1	0.005	112.000	4.17	322.942	17.971	4.307
TR7	AU	1	0.020	1.000	0.27	0.075	0.273	1.008
TR20	AU	1	0.005	40.000	1.19	8.383	2.895	2.434
TR30	AU	1	0.005	37.000	1.35	15.481	3.935	2.915
TR40	AU	1	0.005	15.000	1.49	9.856	3.139	2.103
TRB	AU	1	0.110	3.000	1.09	1.166	1.080	0.989
FR1	AU	1	0.001	80.000	1.26	12.459	3.530	2.795
FR2	AU	1	0.002	50.000	1.05	7.677	2.771	2.626
FR3	AU	1	0.005	60.000	1.22	11.320	3.364	2.746
FR4	AU	1	0.002	30.000	0.96	5.462	2.337	2.438
FR5	AU	1	0.005	15.000	0.82	2.811	1.677	2.038
FR6	AU	1	0.005	12.000	1.22	4.692	2.166	1.772
FR7	AU	1	0.005	15.000	0.94	2.739	1.655	1.757
FR20	AU	1	0.005	50.000	1.44	17.634	4.199	2.917
FR30	AU	1	0.005	20.000	1.25	7.223	2.688	2.143

Mineral Zone	Parameter	Domain	Minimum g/t	Maximum g/t	Average g/t	Variance	St Dev	CoV
FR40	AU	1	0.005	20.000	1.10	4.385	2.094	1.901
FRB	AU	1	0.005	4.000	0.76	0.693	0.832	1.096
W	AU	1	0.001	125.000	0.12	0.925	0.962	8.084
WE	AU	1	0.001	60.000	0.71	5.293	2.301	3.218

14.17.5 Esaase Outlier Analysis

Outlier analysis was undertaken using histograms and probability plots. The outliers were determined for both the variography and the kriging stages. For variography the outliers are cut at a specified value, but for kriging the outliers are capped at a specified value. Hence, all available data is used in the kriging of the data, but a restricted data set is used for variography.

The histograms and probability plots indicate that the populations of gold grades are close to log normal, which is typical of many gold deposits. The co-efficient of variation are moderately high but typical for these types of deposits. The kriging capping and the variogram top-cut values applied are summarised in Table 14-103 and Table 14-104, respectively.

Table 14-103: Esaase Kriging Capping Values per Domain

Mineral Zone	Domain	Commodity	Top Cut g/t	Mineral Zone	Domain	Commodity	Top Cut g/t
FR1	1	AU	60	OR20	1	AU	20
FR2	1	AU	30	OR3	1	AU	20
FR20	1	AU	30	OR30	1	AU	10
FR3	1	AU	40	OR4	1	AU	20
FR30	1	AU	15	OR40	1	AU	5
FR4	1	AU	20	OR5	1	AU	6
FR40	1	AU	10	OR6	1	AU	10
FR5	1	AU	10	OR7	1	AU	1
FR6	1	AU	8	ORB	1	AU	1
FR7	1	AU	10	TR1	1	AU	60
FRB	1	AU	3	TR2	1	AU	20
LR1	1	AU	15	TR20	1	AU	10
LR2	1	AU	20	TR3	1	AU	20
LR20	1	AU	15	TR30	1	AU	15
LR3	1	AU	4	TR4	1	AU	10
LR30	1	AU	3	TR40	1	AU	4
LR4	1	AU	10	TR5	1	AU	5
LR40	1	AU	1	TR6	1	AU	10
LR5	1	AU	2	TR7	1	AU	1
LR6	1	AU	2	TRB	1	AU	3
LR7	1	AU	1	W	1	AU	40
LRB	1	AU	3	WE	1	AU	40
OR1	1	AU	60	-	-	-	-
OR2	1	AU	15	-	-	-	-

Table 14-104: Esaase Variography Top Cut Values per Domain

Mineral Zone	Domain	Commodity	Top Cut g/t	Mineral Zone	Domain	Commodity	Top Cut g/t
FR1	1	AU	60	OR20	1	AU	20
FR2	1	AU	30	OR3	1	AU	20
FR20	1	AU	30	OR30	1	AU	10
FR3	1	AU	40	OR4	1	AU	20
FR30	1	AU	15	OR40	1	AU	5
FR4	1	AU	20	OR5	1	AU	6
FR40	1	AU	10	OR6	1	AU	10
FR5	1	AU	10	OR7	1	AU	1
FR6	1	AU	8	ORB	1	AU	1
FR7	1	AU	10	TR1	1	AU	60
FRB	1	AU	3	TR2	1	AU	20
LR1	1	AU	15	TR20	1	AU	10
LR2	1	AU	20	TR3	1	AU	20
LR20	1	AU	15	TR30	1	AU	15
LR3	1	AU	4	TR4	1	AU	10
LR30	1	AU	3	TR40	1	AU	4
LR4	1	AU	10	TR5	1	AU	5
LR40	1	AU	1	TR6	1	AU	10
LR5	1	AU	2	TR7	1	AU	1
LR6	1	AU	2	TRB	1	AU	3
LR7	1	AU	1	W	1	AU	40
LRB	1	AU	3	WE	1	AU	40
OR1	1	AU	60	-	-	-	-
OR2	1	AU	15	-	-	-	-

14.17.6 Esaase Variography Parameters

Point experimental variograms were generated and modelled for each domain. The parameters of the modelled variograms for the Keegan Esaase Project are summarised in Table 14-105.

Table 14-105: Esaase Variogram Model Parameter

Mineralized Zone	Sill	Nugget %	Sill 1 %	Range1 m	Range2 m	Range3 m	Sill 2%	Range1 m	Range2 m	Range3 m
LR1	2.18	40.50	87.12	38.97	64.95	9	100	50.08	67.78	9
LR2	2.71	34.38	85.88	37.50	62.50	9	100	38.54	64.37	9
LR3	0.47	30.32	76.88	31.47	42.88	9	100	69.61	75.00	9
LR4	2.67	31.74	55.00	38.81	46.94	9	100	63.10	92.17	9
LR5	2.60	32.76	55.00	53.58	71.46	9	100	75.00	78.28	9
LR6	1.72	37.95	55.00	38.95	111.44	9	100	84.64	95.53	9
LR7	0.04	32.75	55.00	13.21	13.21	9	100	9.15	9.15	9
LR20	1.94	37.23	87.83	62.22	65.19	9	100	101.86	143.56	9
LR30	0.46	31.79	86.02	38.82	187.96	9	100	77.64	139.49	9
LR40	0.06	31.26	99.90	36.06	36.06	9	100	43.05	43.05	9
LRB	0.91	44.66	76.96	28.12	65.27	9	100	70.28	83.55	9
OR1	2.22	40.07	79.17	56.76	68.05	9	100	29.01	77.22	9
OR2	1.78	49.50	87.01	38.98	55.41	9	100	41.25	49.51	9
OR3	1.81	44.58	85.65	23.94	47.78	9	100	39.54	49.25	9
OR4	1.82	35.89	81.58	38.97	55.51	9	100	20.73	91.76	9
OR5	1.82	39.67	55.00	31.13	52.41	9	100	67.77	83.73	9
OR6	2.38	30.27	55.00	69.62	73.15	9	100	94.03	69.04	9
OR7	0.08	32.73	76.56	26.99	36.99	9	100	41.48	45.10	9
OR20	2.16	38.14	55.00	39.96	47.06	9	100	30.88	58.69	9
OR30	4.24	32.45	76.88	85.10	93.27	9	100	81.92	107.57	9
OR40	1.97	45.05	55.00	62.53	62.53	9	100	87.68	87.68	9
ORB	1.55	61.67	89.54	5.60	5.60	9	100	6.40	6.40	9
TR1	2.14	39.97	89.26	38.99	64.99	9	100	52.47	129.98	9

Mineralized Zone	Sill	Nugget %	Sill 1 %	Range1 m	Range2 m	Range3 m	Sill 2%	Range1 m	Range2 m	Range3 m
TR2	2.17	37.29	76.84	47.60	43.13	9	100	32.08	79.36	9
TR3	1.99	50.79	76.14	50.12	83.53	9	100	63.42	98.29	9
TR4	1.45	61.67	93.88	45.00	51.36	9	100	90.00	150.00	9
TR5	1.60	48.16	84.38	95.12	124.81	9	100	200.13	84.98	9
TR6	2.72	59.23	59.82	29.53	48.65	9	100	97.21	162.01	9
TR7	1.43	43.74	55.00	6.49	10.81	9	100	12.97	15.91	9
TR20	1.92	45.86	90.81	42.76	84.55	9	100	17.96	63.30	9
TR30	2.87	30.79	55.00	76.94	131.79	9	100	105.85	145.87	9
TR40	1.85	45.71	55.00	77.01	101.30	9	100	154.02	193.97	9
TRB	0.42	56.33	87.04	360.62	360.62	9	100	295.36	295.36	9
FR1	2.22	45.80	85.94	54.87	9.49	9	100	159.44	130.50	9
FR2	2.22	40.91	74.91	71.03	64.96	9	100	89.32	53.33	9
FR3	6.17	32.86	80.50	59.43	31.85	9	100	86.50	76.89	9
FR4	1.98	46.12	55.00	39.07	39.27	9	100	63.22	83.62	9
FR5	1.80	31.13	75.59	59.33	64.93	9	100	126.80	129.88	9
FR6	2.59	34.36	91.89	26.19	42.20	9	100	34.19	29.43	9
FR7	2.29	38.94	75.92	29.58	37.66	9	100	25.13	69.79	9
FR20	2.49	34.43	75.40	53.17	27.52	9	100	90.32	21.92	9
FR30	3.19	49.15	55.00	42.19	66.03	9	100	61.03	79.27	9
FR40	2.56	39.02	84.88	77.00	52.54	9	100	61.47	77.70	9
FRB	1.83	30.75	75.25	51.35	52.82	9	100	131.75	126.16	9
W	2.20	43.90	97.45	78.04	126.72	9	100	107.54	125.24	9
WE	2.61	36.89	55.00	162.50	162.50	9	100	250.74	250.74	9

14.17.7 Esaase Estimation Methodology

Simple and ordinary kriging were undertaken. Simple kriging includes the global mean as a constituent of the kriging equation. Simple kriging is used primarily is areas which are not well defined spatially by data and the mean of the population is included as part of the estimate.

The global means for each domain and mineralized zone were determined through the analysis of the statistics of various regularised data set dimensions. Minxcon declustered the data and reviewed the means and average variances of each declustered data set in order to determine the most representative global mean for each domain. Table 14-106 summarises the global means determined per reef:

Table 14-106: Esaase Global Means

Mineral Zone	Domain	Commodity	Global Mean	Reef	Domain	Commodity	Global Mean
FR1	1	AU	1.26	OR2	1	AU	1.01
FR2	1	AU	1.06	OR20	1	AU	1.15
FR20	1	AU	1.44	OR3	1	AU	1.68
FR3	1	AU	1.23	OR30	1	AU	1.48
FR30	1	AU	1.25	OR4	1	AU	1.10
FR4	1	AU	0.96	OR40	1	AU	1.19
FR40	1	AU	1.10	OR5	1	AU	0.80
FR5	1	AU	0.82	OR6	1	AU	1.38
FR6	1	AU	1.22	OR7	1	AU	0.47
FR7	1	AU	0.94	ORB	1	AU	0.48

Mineral Zone	Domain	Commodity	Global Mean	Reef	Domain	Commodity	Global Mean
FRB	1	AU	0.76	TR1	1	AU	1.68
LR1	1	AU	1.11	TR2	1	AU	1.12
LR2	1	AU	1.16	TR20	1	AU	1.19
LR20	1	AU	1.20	TR3	1	AU	2.00
LR3	1	AU	1.11	TR30	1	AU	1.35
LR30	1	AU	0.77	TR4	1	AU	0.85
LR4	1	AU	1.02	TR40	1	AU	1.49
LR40	1	AU	0.50	TR5	1	AU	1.02
LR5	1	AU	0.57	TR6	1	AU	4.17
LR6	1	AU	0.49	TR7	1	AU	0.27
LR7	1	AU	0.45	TRB	1	AU	1.09
LRB	1	AU	0.94	W	1	AU	0.12
OR1	1	AU	1.45	WE	1	AU	0.68

Table 14-107 summarises the means and variances obtained for each declustered data set.

Table 14-107: Esaase Declustered Statistics per Reef

Mineralized Zone	Domain	Xsize	Ysize	Zsize	Num Blk	Av_AU	Var_AU
LR1	DOM0	80	80	250	20	0.91	0.19
LR2	DOM0	80	80	250	19	1.46	2.89
LR3	DOM0	80	80	250	4	0.67	0.39
LR4	DOM0	80	80	250	13	0.92	0.60
LR5	DOM0	80	80	250	9	0.57	0.08
LR6	DOM0	80	80	250	5	0.47	0.03
LR7	DOM0	80	80	250	1	0.45	-
LR20	DOM0	80	80	250	15	0.88	0.53
LR30	DOM0	80	80	250	6	0.72	0.15
LR40	DOM0	80	80	250	3	0.46	0.01
LRB	DOM0	80	80	250	4	0.90	0.11
OR1	DOM0	80	80	250	46	1.46	4.11
OR2	DOM0	80	80	250	44	1.02	1.11
OR3	DOM0	80	80	250	11	1.48	0.98
OR4	DOM0	80	80	250	26	0.86	0.14
OR5	DOM0	80	80	250	20	0.67	0.09
OR6	DOM0	80	80	250	10	0.89	0.68

Mineralized Zone	Domain	Xsize	Ysize	Zsize	Num Blk	Av_AU	Var_AU
OR7	DOM0	80	80	250	4	0.49	0.00
OR20	DOM0	80	80	250	27	1.11	1.08
OR30	DOM0	80	80	250	10	1.98	4.04
OR40	DOM0	80	80	250	7	1.53	0.94
ORB	DOM0	80	80	250	3	0.36	0.05
TR1	DOM0	80	80	250	47	1.46	1.11
TR2	DOM0	80	80	250	40	0.88	0.31
TR3	DOM0	80	80	250	8	2.00	0.71
TR4	DOM0	80	80	250	24	0.77	0.49
TR5	DOM0	80	80	250	12	1.50	2.84
TR6	DOM0	80	80	250	5	3.42	35.53
TR7	DOM0	80	80	250	1	0.27	-
TR20	DOM0	80	80	250	19	1.10	1.46
TR30	DOM0	80	80	250	11	1.55	2.48
TR40	DOM0	80	80	250	6	1.80	4.29
TRB	DOM0	80	80	250	2	0.70	0.35
FR1	DOM0	80	80	250	76	1.11	0.47

Mineralized Zone	Domain	Xsize	Ysize	Zsize	Num Blk	Av_AU	Var_AU
FR2	DOM0	80	80	250	58	1.08	0.66
FR3	DOM0	80	80	250	29	1.39	1.79
FR4	DOM0	80	80	250	29	0.91	1.26
FR5	DOM0	80	80	250	15	0.70	0.08
FR6	DOM0	80	80	250	5	1.01	0.24
FR7	DOM0	80	80	250	6	1.04	0.34
FR20	DOM0	80	80	250	28	1.22	0.71
FR30	DOM0	80	80	250	17	1.26	0.69
FR40	DOM0	80	80	250	18	1.10	0.44
FRB	DOM0	80	80	250	12	0.77	0.11
W	DOM0	80	80	250	1136	0.08	0.01
WE	DOM0	80	80	250	447	0.68	0.45

Minxcon reviewed the effect of the global means on the estimates for one of the ore bodies. On a global basis, there was minimal difference between the ordinary and simple kriged estimates, such that a material impact on the estimates by the global means was determined as insignificant. The zone reviewed contained a spatially well represented drill hole data set.

Figure 14-111 shows a transverse section through the block model with the Au grades as estimated, and Figure 14-112 the same in plan view.

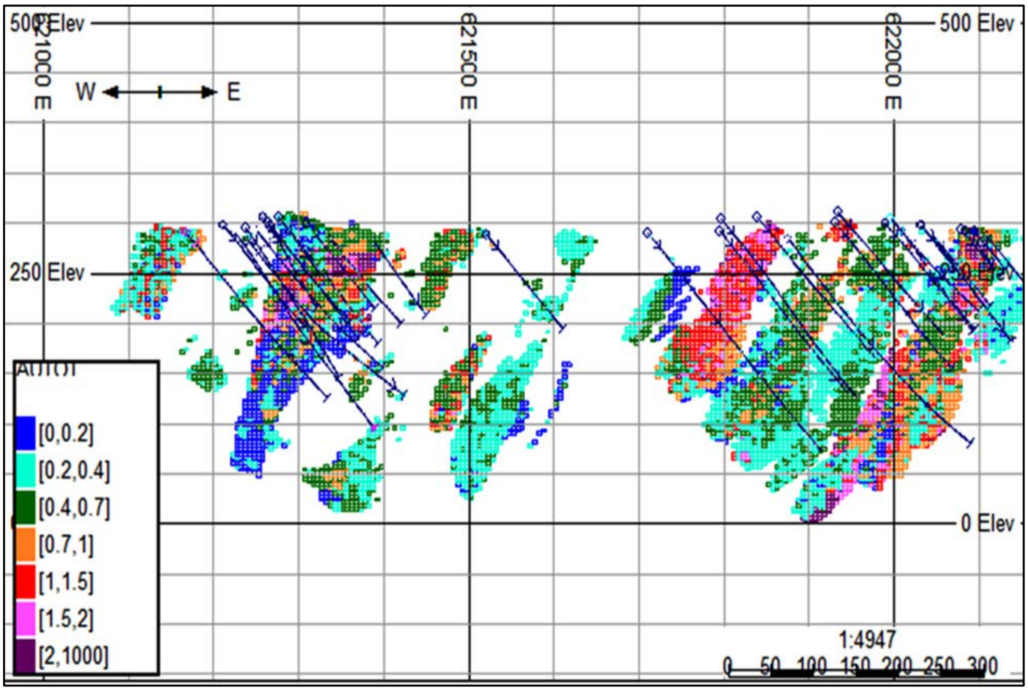


Figure 14-111: Esaase Block Model Transverse Section View

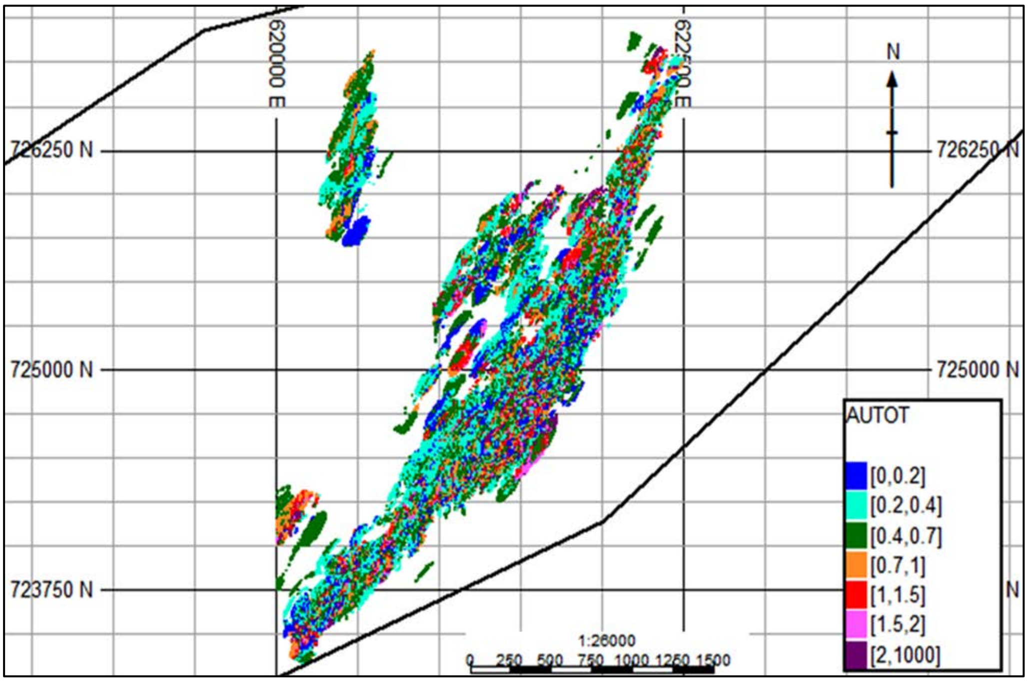


Figure 14-112: Esaase Block Model Plan View

14.17.8 Esaase Resource Statement

The Esaase Project was not re-estimated for this report. The Resource statement was updated using the original 2012 estimation within a new 2017 pit-shell. The cut-off Au grade was also reduced to 0.5g/t for this tabulation. For the 2012 estimation, no Transition or Oxide surfaces were used.

Table 14-108 shows the total Mineral Resource sub-divided into Measured, Indicated and Inferred; as well as Oxide, Transition and Fresh zones.

Table 14-108: Esaase MR's (Oct 2012) at 0.5g/t Au cut-off grade within a US$1500/oz Au Pit Shell

Esaase	0.5 g/t cutoff constrained to $1500/oz Au shell				
Resource Category	**Ore Type**	**Tonnes Mt**	**Grade Au g/t**	**Content Au oz**	**Content Au Moz**
Measured	Strong Oxide	3.50	1.21	136,444	0.136
	Oxide	8.21	1.34	353,975	0.354
	Transition	4.57	1.75	257,122	0.257
	Fresh	10.21	1.30	425,935	0.426
Total Measured		**26.49**	**1.38**	**1,173,475**	**1.173**
Indicated	Strong Oxide	1.81	1.36	79,280	0.079
	Oxide	6.11	1.36	266,737	0.267
	Transition	3.68	1.20	141,942	0.142
	Fresh	45.93	1.40	2,060,777	2.061
Total Indicated		**57.53**	**1.38**	2,548,736	**2.549**
Total Measured and Indicated		**84.02**	**1.38**	**3,722,211**	**3.722**
Inferred	Strong Oxide	0.04	1.22	1,497	0.001
	Oxide	0.05	0.83	1,315	0.001
	Transition	0.03	0.86	780	0.001
	Fresh	0.38	1.33	16,117	0.016
Total Inferred		**0.49**	**1.25**	19,708	**0.020**

Notes:

- Columns may not add up due to rounding.
- All figures are in metric tonnes.
- The Mineral Resources are stated as in situ tonnes.
- Individual Densities was used per mineral zone.
- The tonnages and contents are stated as 100%, which means no attributable portions are stated in the table.

14.17.9 Esaase Resource Classification

Figure 14-113 shows the Esaase block model resource classification in plan view, and Figure 14-114 in long section. The Resource is also reported at >0.5 g/t within a US$1,500/oz Au pit shell as depicted in Figure 14-113.

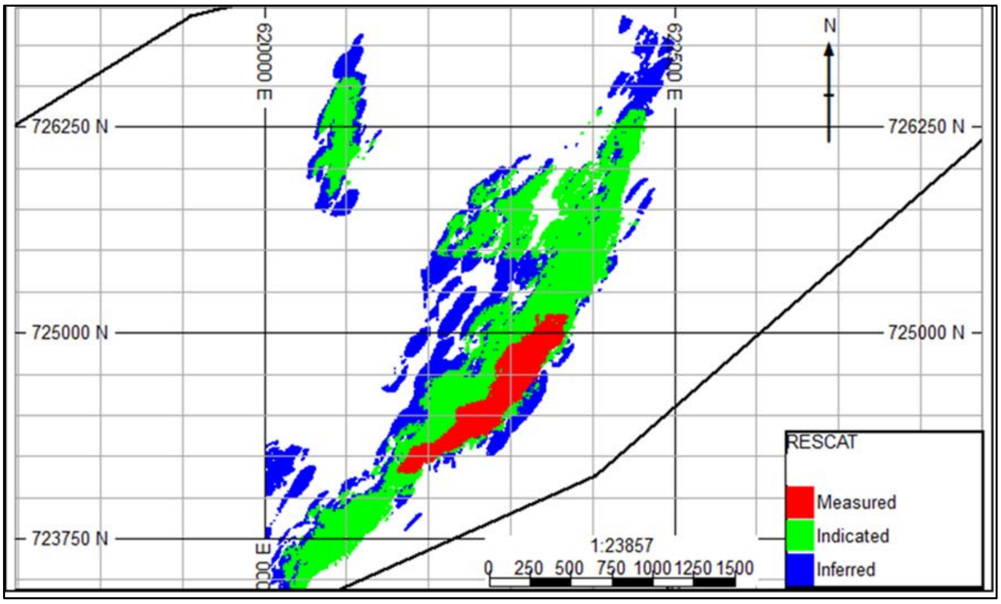


Figure 14-113: Esaase Resource Classification Block model- Plan View

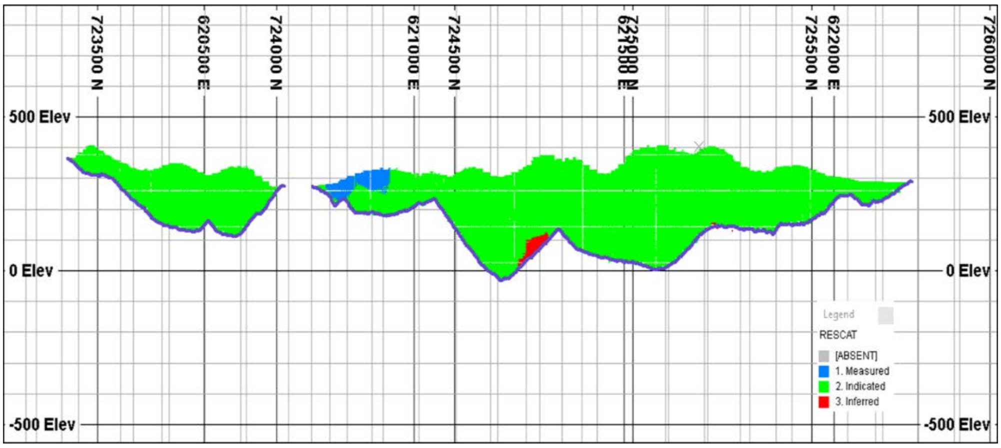


Figure 14-114: Esaase Resource Classification Block model Longitudinal Section

The Esaase grade-tonnage curve is shown in Figure 14-115.

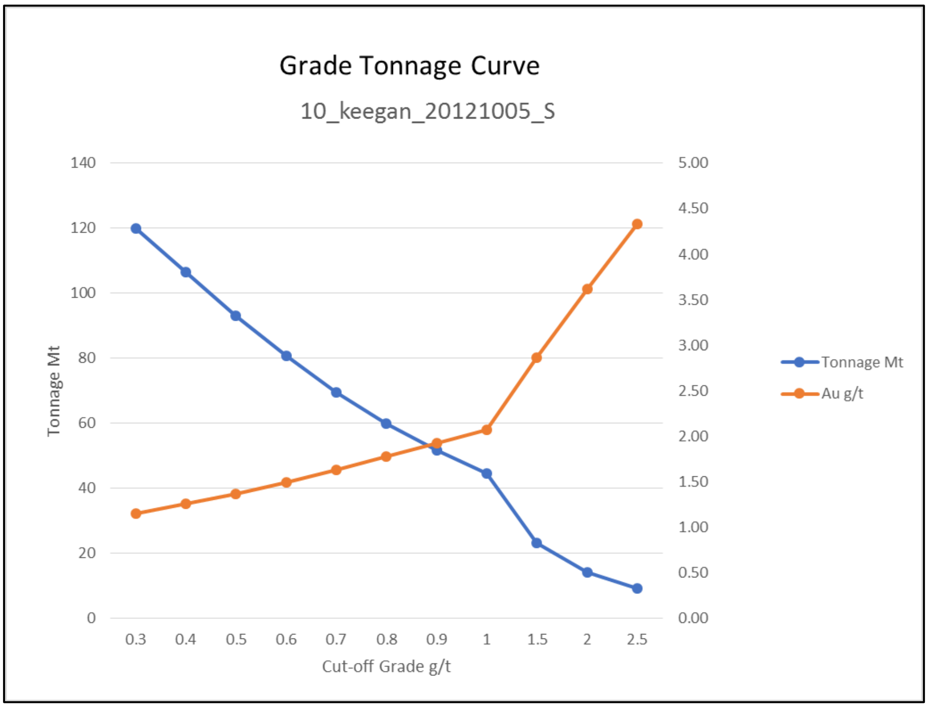


Figure 14-115: Esaase Grade Tonnage Curve

14.17.10 CSA Global Audit

CSA Global undertook a high-level review of the Esaase MRE for Asanko Gold, including review of input parameters and assumptions, MRE methodology, reasonableness and block model fatal flaw analysis. The following conclusions and recommendations were made:

- CSA considers the drilling and sampling procedures and techniques, as described in the NI43-101 (2012) guidance, as adequate for use in the MRE. Following validation, CSA considers the database adequate for use in the MRE
- CSA has reviewed the QAQC provided and notes that the duplicate assay precision (repeatability) is within acceptable limits. There are numerous failures in the blanks (possible contamination) and assay accuracy (CRM) which in part are due to apparent misidentified CRMs and blanks. Asanko has rectified the misidentified samples in the database
- CSA considers the modelled mineralisation wireframes reasonable for the MRE
- CSA considers the flagging and compositing reasonable for the MRE
- Visual checks of CSA desurveyed and composited data compares satisfactorily to the Minxcon estimation composite file. Domain statistical comparison confirmed the visual checks

- CSA considers the top cuts as applied by Minxcon for the kriging estimate as appropriate for the MRE
- CSA considers the default densities per oxidation zone for Transitional and Fresh material to be reasonable for the MRE. It is noted that the most oxidised material at Esaase is saprock, hence higher densities between 2.27 – 2.45 t/m³ have been applied (cf normal oxides ~ 1.80 t/m³)
- Review of the modelled variograms as presented in Appendix 2 of the NI-43-101 report appears reasonable
- Analysis of the Block Model showed that the metal within the drill holes very closely match the metal in the MRE. The block model looks reasonable with grades showing good continuity. Validation statistics are reasonable. Also the swath plots look reasonable, with smoothing as is expected
- CSA recommends Asanko review some of the criteria for the Measured – Indicated Resource interface, as well as the Inferred Resources estimation
- CSA is of the opinion that in some peripheral areas (these are isolated discrete zones) to the main zones of mineralisation, the amount of drilling supporting Indicated class material is marginal, and may not have enough drill hole data support, or continuity confidence for this level of classification
- CSA also observed that certain areas of Classified Measured Mineral Resources may warrant downgrading to Indicated Mineral Resources due to:
  - The high nugget nature of the deposit (high grade variability)
  - The current drill hole spacing, at best 20 x 20m may not be adequate to define required continuity confidence
  - Further density measurements in the more oxidised zones of the deposit would give more confidence to the 2.27-2.33 t/m³ density used for saprock in the MRE
  - The moderate confidence in the geological interpretation, (as noted by Minxcon) and the need for IK to define further estimation domains outside modelled mineralisation envelopes.

Asanko Gold has noted the review outcomes, and has initiated (as per for the Nkran deposit) an in-house geological and resource model review.

14.18 Mineral Resource Risk Analysis

The MRE tabled in this disclosure are all located in the Amansie west district of southwest Ghana. The main risks pertaining to these MRE are related to external factors including:

- Ghanaian socio-political issues which are not under the control of Asanko, and which include the stability of local communities, security of tenure and Government intervention on the right to mine, punitive environmental legislation, and increased taxes and duties

- A significant down turn in the gold price which could render the orebodies sub economic. Notably the current operations have a breakeven grade of around 0.45 g/t Au, compared to a Reserve grade of 1.93 g/t Au
- Asanko is a publicly listed company. Market pressures can impact on the ability of Asanko to conduct normal business due to short positions taken by speculative investment research companies or capital hedge funds
- Geotechnical failure of open pit sidewall/s, which would potentially delay profitable mining operations, and impact on the extraction of Resources
- The development of the Esaase resources and reserves assumes the resettlement of the Tetrem village, which in turn has been approved by the Environmental Protection Agency and Minerals Commission of Ghana as evidenced by the granting of the Environmental Permit referred to in section 5.4 of this technical report. In light of the collective nature of the ongoing negotiations with Tetrem representatives, the mature legal framework within Ghana, the advanced stage of the resettlement process under Ghanaian law, and the Company's recent experiences with resettlement efforts and associated costs during Phase 1, the resettlement of the Tetrem community is not considered to pose a material risk to the potential development of mineral resources and reserves.
-

14.19 Mineral Resource Estimate Summary

The updated global Asanko Gold Mine MR's include the Nkran, Adubiaso, Adubiaso Extension, Nkran Extension, Abore, Dynamite Hill, Asuadai, Akwasiso, Esaase Main, Esaase B and Esaase D deposits. The cut-off grade for all deposits was 0.5 g/t Au (based on Nkran actual break even operating costs April 2016 – May 2017 equivalent to 0.45 g/t Au). The > 0.5 g/t Au resources were constrained within US$1,500/oz Au pit shells, and are shown in Table 14-109 (Measured and Indicated) and Table 14-110 (Inferred).

Table 14-109: AGM Measured and Indicated Resources 25 April 2017 0.5 g/t Au Cut-off US$1,500/oz Au

AGM M & I 0.5 g/t Au cutoff within US$1,500/oz Au shell												
Deposit	Measured				Indicated				Total (M&I)			
	Mt	Au g/t	Au Oz	Au Moz	Mt	Au g/t	Au Oz	Au Moz	Mt	Au g/t	Au Oz	Au Moz
Esaase Main	26.49	1.38	1,173,475	1.17	57.53	1.38	2,548,736	2.55	84.02	1.38	3,722,211	3.72
Nkran Main	4.92	1.81	286,650	0.29	23.09	1.89	1,400,706	1.40	28.01	1.87	1,687,356	1.69
Abore	2.23	1.41	101,127	0.10	3.09	1.48	146,790	0.15	5.33	1.45	247,917	0.25
Dynamite Hill	-	0.00	0.00	0.00	0.33	3.39	35,499	0.04	0.33	3.39	35,499	0.04
Akwasiso	-	0.00	0.00	0.00	0.94	3.88	116,907	0.12	0.94	3.88	116,907	0.12
Adubiaso	0.82	2.36	62,391	0.06	0.86	2.03	56,279	0.06	1.68	2.19	118,670	0.12
Esaase D	0.83	1.11	29,819	0.03	1.16	1.42	52,922	0.05	2.00	1.29	82,741	0.08
Esaase B	0.75	1.01	24,291	0.02	1.90	0.78	47,369	0.05	2.65	0.84	71,660	0.07
Asuadai	-	0.00	0.00	0.00	1.88	1.22	73,600	0.07	1.88	1.22	73,600	0.07
Adubiaso Ext.	0.16	1.96	9,800	0.01	0.26	1.71	11,827	0.01	0.42	1.61	21,627	0.02
Nkran Ext.	-	0.00	0.00	0.00	0.19	2.70	16,316	0.02	0.19	2.70	16,316	0.02
Total	36.20	1.45	1,687,552	1.69	91.23	1.54	4,506,950	4.51	127.44	1.51	6,194,502	6.19

Table 14-110: AGM Inferred Mineral Resources April 2017

AGM Inferred Resources 25 Apr 2017 0.5 g/t Au cut-opff US$1,500/oz Au Shell				
Deposit	**Inferred**			
	Mt	**Au g/t**	**Au Oz**	**Au Moz**
Esaase Main	0.09	1.08	2,812	0.003
Nkran Main	0.07	3.37	7,450	0.007
Aboreq	1.28	1.61	66,084	0.066
Dynamite Hill	0.02	3.22	1,804	0.002
Akwasiso	0.00	3.79	396	0.000
Adubiaso	0.00	3.57	308	0.000
Esaase D	1.01	1.26	40,916	0.041
Esaase B	2.12	0.86	58,278	0.058
Asuadai	0.63	1.75	35,115	0.035
Adubiaso Ext	0.14	3.10	13,741	0.014
Nkran Ext	0.01	1.02	254	0.000
Total	**5.35**	**1.32**	**226,905**	**0.227**

Notes for Tables 14-109 and 14-110

- CJM - Esaase was estimated in October 2012, Abore, Adubiaso, Asuadai estimated in April 2014
- CSA Global Nkran and Dynamite Hill were re-estimated in January 2017, Akwasiso April 2017.
- The resource cut-off grade used for all deposits was 0.5 g/t Au within a Whittle PitShell at US$ 1,500/oz Au.
- Columns may not add up due to rounding.
- All figures are in metric tonnes.
- The MR's are stated as in situ tonnes.
- Individual Densities were used per mineral zone.
- The tonnages and contents are stated as 100%, which means no attributable portions are stated in the table.
- Conversion from grams to ounces – 31.1035

15 MINERAL RESERVE ESTIMATE

The Mineral Reserve Estimate ("MRev") has been classified and reported in accordance with the Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral projects' of June 2011 (the Instrument), updated in 2015 and the classifications adopted by the CIM Council in November 2011.

The effective date of the MRev is 31st December 2016.

The Mineral Reserves were derived from the Mineral Resource block models that are presented in Section 14. The Mineral Reserves are the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining losses and the addition of waste dilution. The Mineral Reserves form the basis of the mine plan presented in Section 16.

15.1 Obotan Mineral Reserves

The process to develop the MRev for the AGM Phase 1 was as follows, and as reported in the November 2014 Definitive Project Plan ("DPP") Study:

- The open pit optimisation has been undertaken on the Measured and Indicated resources only
- The grades and tonnes of the mineral resource model have been modified by a mining recovery and a mining dilution based on ore body geometry and mining methodology. The Mining Model contains undiluted ore tonnes and ore grade. Fixed dilution and mining recovery percentages of 5% end 95% respectively, were subsequently applied in the optimisations
- The Datamine NPVS® suite of optimisation software was used to perform the pit optimisations. Datamine NPVS® is an accepted industry optimisation tool that uses the 3D Lerchs-Grossmann algorithm to determine the economic pit limits based on input of mining and processing costs and revenue per block
- The source of the parameters are summarised below, along with the source of the information:
    - A base gold price of US$1,300/oz Au (PMI). A government royalty of 5% and a Refining Cost of $4.00/oz Au were then applied. This resulted in a Net Gold Price of ~$1,231/oz Au
    - Pit slopes inter-ramp angles ranging from 28.5o to 58.2o. Resulting overall pit slopes account for access ramps where applicable
    - Gold recovery ranging from 94.0% to 94.5% dependent on ore body and material weathering
    - Processing throughput of 3.6 Mtpa
    - Mining costs based on actual costs achieved with current existing mining contractor
    - Processing costs based on actual costs achieved
- The Deswik® suite of design software was used to perform the detailed pit designs

- Pit shells were selected for each deposit that were then used as the basis for ultimate pit designs
- A cut-off grade of 0.5 g/t Au for Oxide material and 0.7 g/t Au for Fresh and Transitional material was used in the optimisations

Table 15-1 summarises the MRev based on the work detailed above, undertaken as part of the AGM 2017 DFS.

Table 15-1: Obotan Gold Project MRev (Source: DRA 2017)

Deposit	Classification	Tonnage (Mt)	Grade (g/t Au)	Moz Au
Nkran	Proven	4.40	1.85	0.26
	Probable	18.37	1.93	1.14
	Total	22.77	1.91	1.40
Nkran Ext	Proven	0.11	2.47	0.01
	Probable	0.08	1.91	0.00
	Total	0.19	2.24	0.01
Abore	Proven	1.59	1.44	0.07
	Probable	1.60	1.53	0.08
	Total	3.18	1.48	0.15
Adubiaso	Proven	1.04	2.00	0.07
	Probable	1.04	1.82	0.07
	Total	2.09	2.08	0.14
Adubiaso Ext	Proven	0.12	1.66	0.01
	Probable	0.09	1.34	0.00
	Total	0.21	1.53	0.01
Dynamite Hill	Proven	-	-	-
	Probable	2.84	1.49	0.14
	Total	2.84	1.49	0.14
Akwasiso	Proven	-	-	-
	Probable	4.95	1.51	0.24

Deposit	Classification	Tonnage (Mt)	Grade (g/t Au)	Moz Au
	Total	4.95	1.51	0.24
Asuadai	Proven	-	-	-
	Probable	1.30	1.09	0.05
	Total	1.30	1.09	0.05
Total Obotan Reserve Note 8	Proven	7.26	1.79	0.42
	Probable	30.47	1.76	1.72
	Total	37.74	1.76	2.14

Note: All tonnes quoted are dry tonnes. Differences in the addition of deposit tonnes to the total displayed is due to rounding.

1. Mineral Reserves are defined within a mine design guided by Lerchs-Grossman ("LG") pit shells.
2. The LG shell generation was performed on Measured and Indicated materials only.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal.
4. Tonnage and grade measurements are in metric units.
5. Minimum economic cut-off grade for the Nkran Fresh material 0.7g/t Au. All other Obotan pits use an economic cut-off grade of 0.5g/t Au and 0.7g/t Au for Oxide and Fresh material respectively.
6. No Inferred, deposit, or mineralised waste contributes value to the pit optimisation.
7. No Inferred, deposit, or mineralised waste is included in the Mineral Reserve.
8. Reserve excludes Obotan surface stockpiles (as at 31st December 2016) of 1.95 Mt @ 1.22 g/t Au.

The estimate of Mineral Reserves for the Obotan ore sources are not at this stage materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issue. Furthermore, the estimate of Mineral Reserves is not materially affected by any known mining, metallurgical, infrastructure, or other relevant factor.

15.2 Esaase Mineral Reserves

The process to develop the P10M MRev for the AGM is detailed in the Section 16 of this report:

- The Obotan MRev and mine schedule was maintained from the Obotan production plan (December 2016)
- The open pit optimisation for the Esaase resource has been undertaken on the Measured and Indicated resources only
- The grades and tonnes of the Esaase mineral resource model have been modified by a mining recovery of 92.6% to allow for cross hauling and ore

loss. The mining dilution is based on ore body geometry and mining methodology. A static 8% dilution at 0 g/t Au was used to allow for waste rock inclusion into the ore blast blocks

- The Whittle® suite of optimisation software was used to perform the pit optimisations. Whittle® is an accepted industry optimisation tool that uses the 3D Lerchs-Grossmann algorithm to determine the economic pit limits based on input of mining and processing costs and revenue per block
- The source of the parameters are summarised below, along with the source of the information:
  - A base gold price of US$1,300/oz Au. A government royalty of 5%, Nett Smelter Royalty 0.5% and a Refining Cost of $4.00/oz Au were then applied. This resulted in a Net Gold Price of approximately $1,224.50/oz Au
  - Geotechnical recommendations to maintain pit slope stability was used in the optimisation. The Overall Slope Angles ("OSA") used in the pit optimisation process were 50.0° NW / 56.3° SE, 46.6° NW / 52.8° SE and 40.4° NW / 40.4° SE for Fresh, Transitional and Oxide material respectively
  - Gold recovery formulae based on test work dependant on the type of material and destination of the ore. Two processing paths were used in the optimisation (CIL P5M and CIL P10M processing circuits)
  - CIL P5M processing plant requiring a throughput of 1.67 Mtpa to 2.0 Mtpa and CIL P10M processing plant requiring a throughput of 5.0 Mtpa
  - Mining costs are based on the current mining contractor operating at Obotan. A cost estimate following a Request for Quotation ("RFQ") based on the Esaase production schedule was received from the mining contactor. The base date for the mining costs is Q2 2017. Reference mining cost of US$2.36/t and a fixed monthly management fee of US$ 1.35 million / month
  - The CIL P5M and CIL P10M processing costs are detailed in Section 7
  - A minimum economic cut-off grade of 0.6 g/t, based on project specific projected revenue and cost was applied as a minimum input process grade
  - An additional 10Mt at 0.55g/t of very low grade reservable ore will be stockpiled separate to waste
- The Datamine® suite of design software was used to perform the detailed pit designs
- The MRev for the Esaase deposit was evaluated against the Esaase pit design and is within 5% from the optimal pit shell generated by the Whittle ® open pit optimisation software

Table 15-2 summarises the MRev for the Esaase deposit based on the open pit optimisation and detailed design work.

Table 15-2: MRev for the Esaase Deposit (Source: DRA 2017)

Deposit	Classification	Tonnage (Mt)	Grade (g/t Au)	Moz Au
Esaase (Main Pit)	Proven	21.51	1.44	1.00
	Probable	41.05	1.47	1.94
	Total Main Pit	62.57	1.46	2.94
Esaase (B Zone)Note9	Proven	0.10	0.83	-
	Probable	0.00	0.92	-
	Total B Zone	0.10	0.83	-
Esaase (D Zone)	Proven	0.20	1.05	0.01
	Probable	0.40	1.70	0.02
	Total D Zone	0.60	1.56	0.03
Total Esaase ReserveNote10	Proven	21.81	1.44	1.01
	Probable	41.45	1.47	1.96
	Total Esaase	63.27	1.46	2.97

Notes:

1. Mineral Reserves are defined within a mine design guided by LG pit shells.
2. The LG shell generation was performed on Measured and Indicated materials only.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal.
4. Tonnage and grade measurements are in metric units.
5. Minimum economic cut-off grade for the Esaase deposit is 0.6g/t Au.
6. No Inferred, deposit, or mineralised waste contributes value to the pit optimisation.
7. No Inferred, deposit, or mineralised waste is included in the Mineral Reserve.
8. Proven and Probable Mineral Reserves are modified to include ore-loss (7.4%) and dilution (8%).
9. Esaase B Zone values for contained metal are negligible.
10. Mineral Reserve excludes ~10Mt at 0.55g/t Au of very low grade material in the measured and indicated categories contained within the Esaase main pit design.

The estimate of Mineral Reserves for the Esaase deposit are not at this stage materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issue. Furthermore, the estimate of Mineral Reserves is not materially affected by any known mining, metallurgical, infrastructure, or other relevant factor.

15.3 AGM – P10M Mineral Reserves (Esaase and Obotan Combined)

Table 15-3 summarises the MRev for the combined P10M based on the work detailed above.

Table 15-3: MRev for AGM P10M (Source: DRA 2017)

Deposit	Classification	Tonnage (Mt)	Grade (g/t Au)	Moz Au
Nkran	Proven	4.40	1.85	0.26
	Probable	18.37	1.93	1.14
	Total	22.77	1.91	1.40
Nkran Ext	Proven	0.11	2.47	0.01
	Probable	0.08	1.91	0.00
	Total	0.19	2.24	0.01
Abore	Proven	1.59	1.44	0.07
	Probable	1.60	1.53	0.08
	Total	3.18	1.48	0.15
Adubiaso	Proven	1.04	2.00	0.07
	Probable	1.04	1.82	0.07
	Total	2.09	2.08	0.14
Adubiaso Ext	Proven	0.12	1.66	0.01
	Probable	0.09	1.34	0.00
	Total	0.21	1.53	0.01
Dynamite Hill	Proven	-	-	-
	Probable	2.84	1.49	0.14
	Total	2.84	1.49	0.14
Akwasiso	Proven	-	-	-
	Probable	4.95	1.51	0.24
	Total	4.95	1.51	0.24

Deposit	Classification	Tonnage (Mt)	Grade (g/t Au)	Moz Au
Asuadai	Proven	-	-	-
	Probable	1.30	1.09	0.05
	Total	1.30	1.09	0.05
Total Obotan Reserve Note 9	**Proven**	**7.26**	**1.79**	**0.42**
	Probable	30.47	1.76	1.72
	Total	37.74	1.76	2.14
Esaase (Main Pit)Note 10	Proven	21.51	1.44	1.00
	Probable	41.05	1.47	1.94
	Total Main Pit	62.57	1.46	2.94
Esaase (B Zone)	Proven	0.10	0.83	-
	Probable	0.00	0.92	-
	Total B Zone	0.10	0.83	-
Esaase (D Zone)	Proven	0.20	1.05	0.01
	Probable	0.40	1.70	0.02
	Total D Zone	0.60	1.56	0.03
Total AGM Reserve	**Proven**	**29.08**	**1.52**	**1.42**
	Probable	**71.92**	**1.59**	**3.68**
	Total	**101.00**	**1.57**	**5.11**

Notes:

1. Mineral Reserves are defined within a mine design guided by Lerchs-Grossman ("LG") pit shells.
2. The LG shell generation was performed on Measured and Indicated materials only
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal
4. Tonnage and grade measurements are in metric units
5. Reserves for each pit are based on detailed pit designed informed by US$ 1,300/oz pit shells

6. Minimum economic cut-off grade for Esaase deposits is 0.6g/t Au and Nkran Fresh 0.7g/t Au. All other pits use an economic cut-off grade of 0.5g/t Au and 0.7g/t Au for Oxide and Fresh material respectively
7. No Inferred, deposit, or mineralised waste contributes value to the pit optimisation
8. No Inferred, deposit, or mineralised waste is included in the Mineral Reserve
9. Proven and Probable Mineral Reserves are modified to include ore-loss and dilution
10. Reserve excludes Obotan surface stockpiles (as at 31st December 2016) of 1.95 Mt @ 1.22g/t Au
11. Mineral Reserve excludes ~10Mt at 0.55g/t Au of very low grade material in the measured and indicated categories contained within the Esaase main pit design

The estimate of MRev's for P10M are not at this stage materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issue. The costs to resettle the Tetrem village have been considered in the estimates of reserves. Given the advanced stage of the resettlement process under Ghanaian law, including the receipt of the Environmental Permit and the collective nature of the ongoing negotiations, and experience with resettlement and the associated costs during Phase 1, the resettlement process of the Tetrem village is not considered to pose a material risk to the potential development of the mineral reserves. Furthermore, the estimate of MRev not materially affected by any known mining, metallurgical, infrastructure, or other relevant factor.

15.4 DRA Comments

DRA is confident that sufficient geological work has been undertaken and sufficient geological understanding gained, to enable the construction of an ore body model suitable for the derivation of Mineral Resource and Mineral Reserves. DRA considers that both the modelling and the grade interpolation have been carried out in an unbiased manner and that the resulting grade and tonnage estimates should be reliable within the context of the classification applied. In addition, DRA is not aware of any additional metallurgical, infrastructural, environmental, legal, title, taxation, socio-economic, or marketing issues that would impact on the MRE and MRev as presented.

16 MINING METHODS

16.1 Introduction

Asanko Gold commissioned DRA to complete the DFS to a ±10% level of accuracy for the AGM.

Following the merger with PMI in early 2014, Asanko Gold combined its Esaase Gold Project with PMI's Obotan Gold Project to form the AGM. Asanko Gold is intending to develop the AGM in phases, the first phase is in production. The second phase of expansion will be completed as two discreet projects, P5M and P10M.

This document will provide mining information and data for P5M and P10M. P5M is the upgrade of the existing 3 Mtpa CIL processing facility, located at Obotan, to be capable of processing 5 Mtpa. Ore tonnages will be supplied from the Obotan deposits initially until the Esaase overland conveyor, which is scheduled for completion in January 2019, is completed. The processing facility will be fed with 3 Mtpa from Obotan and 2 Mtpa from Esaase at that time.

P10M is an increase of the processing capacity to 10 Mtpa, targeting approximately 450,000 oz Au per year. This increase will be achieved by constructing and integrating an additional 5 Mtpa CIL processing facility into the 5 Mtpa facility at the Obotan site. It is planned that the new processing facility will begin wet commissioning in April 2020, ramping-up to full capacity over three months.

Esaase will supply 7 Mtpa and Obotan 3 Mtpa if the full 10 Mtpa processing facility is at full capacity.

16.2 Mining Strategy

The overall strategy of the Project is divided into 2 discreet phases.

16.2.1 P5M (Obotan Production and Esaase Start-up)

P5M consists of the following:

- Mining of the Nkran pit at a rate of between 2.4 Mt to 3.6 Mt of ore per annum
- The targeted plant feed for the first 6 months of 2017 is 300 ktpm, supplied from the currently operating Nkran pit
- Production from the satellite deposits will assist with the plant feed requirements from Q3 2017 onwards when the target changes to 333 ktpm (equivalent to 4.0 Mtpa)
- The plant will be upgraded by the end of 2017, thereafter the plant feed target will change to 417 ktpm (equivalent to 5.0 Mtpa) from January 2018 to April 2020
- The 5.0 Mtpa for 2018 will be produced from the Nkran pit and other satellite pits at Obotan

- The overland conveyor is planned for completion at the end of December 2018, when the Esaase deposit will start contributing 2.0 Mtpa from January 2019 of the 5.0 Mtpa. The 3.0 Mtpa plant feed requirement will be contributed from Obotan
- Esaase to produce 2 Mtpa from Q1 2019 until the plant is upgraded to 10 Mtpa in April 2020
- Obotan to produce 3 Mtpa from Q1 2019 until the plant is upgraded to 10 Mtpa in April 2020

16.2.2 P10M (Obotan 3 Mtpa and Esaase 7 Mtpa Ore Production)

P10M will include the following:-

- Total ore production from Obotan of 3.0 Mtpa from April 2020 until end of Obotan mine life
- Total ore production from Esaase of 7.0 Mtpa from April 2020 until end of Obotan mine life, thereafter producing 8.4 Mtpa (constrained by the maximum capacity of the overland conveyor belt)

16.3 Obotan Project Mine Design

16.3.1 Geology and Geological Resource

For complete geological information, refer Section 14 of this report.

16.3.2 Nkran Resource Characteristics

The resource model [nk_md0117ik.mreUTM.dm] forms the basis of the Nkran pit optimisations and MR. Pit optimisations were carried out on Measured and Indicated Resources only.

The Nkran block model average grade is approximately 1.8 – 2.0 g/t Au, it is classified as a medium grade open pit deposit. The ore body is not tabular in shape, but massive. Hence the development of a 3D resource block model for best spatial representation of the deposit.

The Nkran ore body strike direction is in a northeast to southwest direction. The mineralised lenses dip steeply (at approximately 70^{o}) in the northwest direction.

Figure 16-1, Figure 16-2 and Figure 16-3 illustrate the Gold Grade distribution, Rock type orientation and Resource classification respectively in relation to the Nkran Pit.

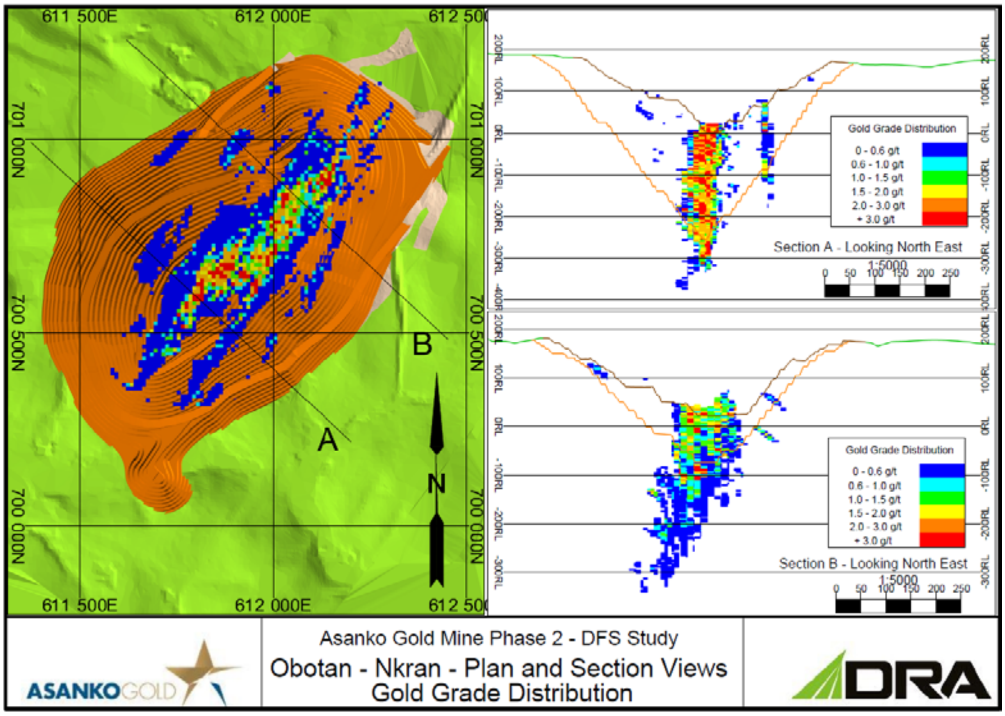


Figure 16-1: Plan and Section Views of the Gold Grade Distribution (Source: DRA 2017)

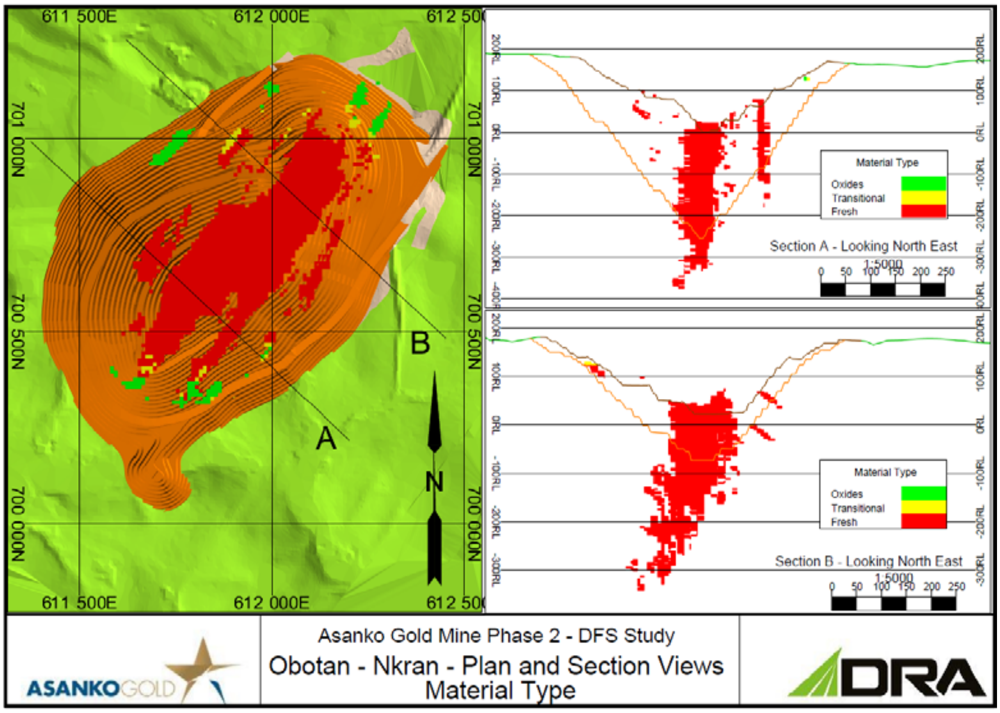


Figure 16-2: Plan and Section Views of the Material Types (Source: DRA 2017)

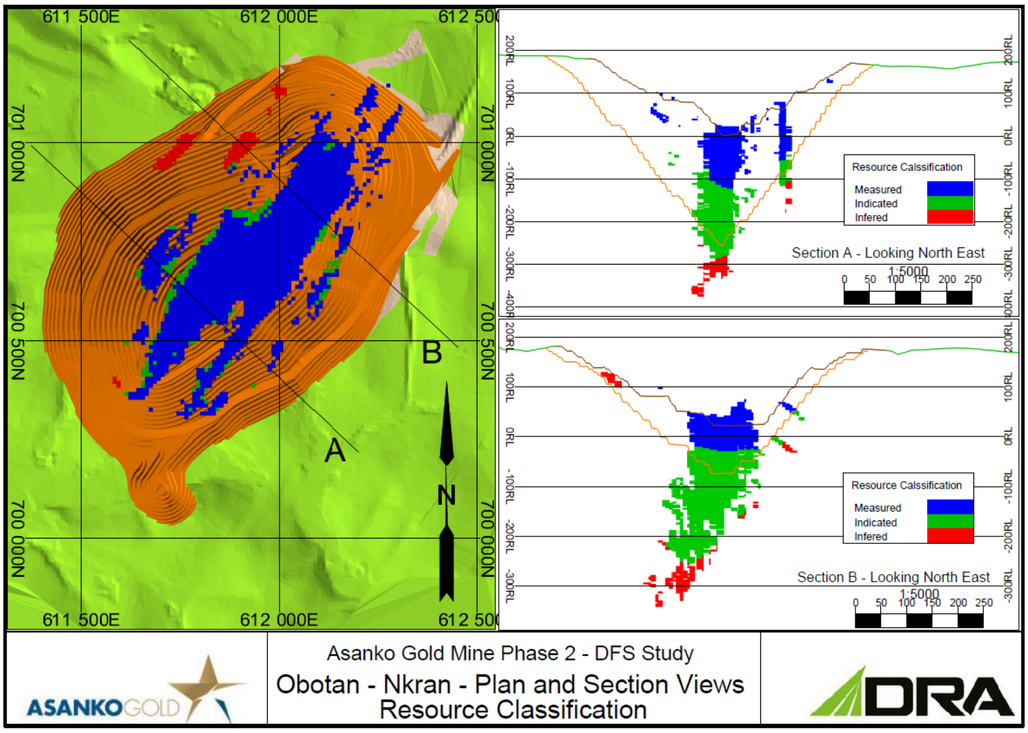


Figure 16-3: Plan and Section Views of the Resource Classification (Source: DRA 2017)

16.3.3 Open Pit Optimisation

The Obotan ore sources consist of several resources, namely:

- Nkran Main Pit and Nkran Extension (satellite pit)
- Adubiaso and Adubasio Extension (satellite pits)
- Abore (satellite pit)
- Dynamite Hill (satellite pit)
- Akwasiso (satellite pit)
- Asuadai (satellite pit)

The sequence of extraction of these pits require Nkran Main Pit to deliver a maximum of 3 Mtpa. During the construction and ramp-up phase of the Esaase open pit, whereby the current CIL plant is upgraded from 3 Mtpa to 5 Mtpa, the additional ore tonnage requirements will be sources from the satellite pits. The remainder of the satellite pits tonnages will form a “filler” towards the tail end of the Nkran Main Pit production.

The objective of open pit optimisation is to determine an open pit shape (shell) that provides the highest value for a deposit. Analysis of the pit shells generated in the optimisation process, lead to the selection of a single shape to serve as a guide for a practical, ultimate pit design.

The final pit design defines the Mineral Reserve estimate and subsequently LoM schedules, from which associated cash flows can be developed. The pit optimisation process is the critical first step in the development of any mineral extraction project. This process defines the scale of the Project as a whole.

In addition to defining the ultimate size of the open pit, the pit optimisation process also indicates potential areas for interim mining stages. These intermediate mining stages ensure the pit is developed in a practical and incremental manner.

The key parameters utilised are summarised in Table 16-1. NPV Scheduler is considered one of the pre-eminent mining software programmes for open pit optimisation and was selected for the Nkran, Dynamite Hill and Akwasiso deposits because they contribute the largest portion of the Obotan life of mine plan. The other satellite pits were derived using the Deswik® optimisation tools as they are smaller pits which only contribute a minor portion of the total life of Obotan. NPV Scheduler utilises the Lerchs-Grossman algorithm, which generates an optimal shape for an open pit in three dimensions and so does the Deswik® optimisation tool.

To generate this optimal shape, a 3D block model is utilised, therefore accounting for the spatial distribution of ore and associated waste rock. It utilises a large amount of input data, either from the block model, or from input directly programmed into the software. This includes, but is not limited to the following:-

- The type and quantity of material, as well as associated grade, (in this case gold) of every block
- The overall slope angle of any pit wall based on material type, geotechnical regions and the strike direction of the wall
- The mining cost, mining ore loss and mining dilution for any given block
- The cost of processing a block, the cost of "selling" the recovered commodity and the revenue generated by the commodity
- Mill throughput rates and mining rates over time
- Discount rate

The following sections detail the parameters used for the optimisation and analysis of the optimisation results.

16.3.3.1 Optimisation Input Parameters

The input parameters are the foundation of the optimisation process. The input parameters included all input parameters for the whole value chain. This includes parameters from in situ geology to the saleable product which includes mining, processing, and selling costs. The physical inputs include the production rates and geotechnical parameters.

Table 16-1: Obotan Pit Optimisation Input Parameters (Source: DRA 2017)

Financial Parameters	Unit	Nkran	Dynamite Hill	Akwasiso	Adubiaso	Nkran Extension	Abore	Asuadai
Commodity Price	$/oz	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Royalties	%	5%	5%	5%	5%	5%	5%	5%
Refining + Selling Cost	$/oz	4.00	4.00	4.00	4.00	4.00	4.00	4.00
Nett Commodity Price	$/g	1,231	1,231	1,231	1,231	1,231	1,231	1,231
Overall Slope Angle								
Oxide	Degrees	32	32	32	32	32	32	32
Transition	Degrees	46	46	46	46	46	46	46
Fresh	Degrees	51.5/54	51.5	51.5	51.5	51.5	51.5	51.5
Mining Cost Parameters								
Reference Level Elevation	RL	1190	275	187	175	187	181	319
Waste Mining Cost								
Oxide	$/t	1.2313	1.2313	1.2313	1.2313	1.2313	1.2313	1.2313
Transition	$/t	2.0583	2.0583	2.0583	2.0583	2.0583	2.0583	2.0583

Financial Parameters	Unit	Nkran	Dynamite Hill	Akwasiso	Adubiaso	Nkran Extension	Abore	Asuadai
Fresh	$/t	3.3390	3.3390	3.3390	3.3390	3.3390	3.3390	3.3390
Ore Mining Cost								
Oxide	$/t	1.2313	1.2313	1.2313	1.2313	1.2313	1.2313	1.2313
Transition	$/t	2.2026	2.2026	2.2026	2.2026	2.2026	2.2026	2.2026
Fresh	$/t	3.4008	3.4008	3.4008	3.4008	3.4008	3.4008	3.4008
Processing Cost Base								
Oxide	$/t milled	6.5	6.5	6.5	6.5	6.5	6.5	6.5
Transition	$/t milled	11.7	11.7	11.7	11.7	11.7	11.7	11.7
Fresh	$/t milled	11.7	11.7	11.7	11.7	11.7	11.7	11.7
GA's	$/t milled	5.04	5.04	5.04	5.04	5.04	5.04	5.04
Process Recovery								
Oxide	%	94.5%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%
Transition	%	94.5%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%
Fresh	%	94.5%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%
Other Cost Parameters								
RoM Stockpile Re-handle	$/t milled	0.91	0.91	0.91	0.91	0.91	0.91	0.91
Hauling Cost	$/t ore	-	1.65	1.07	1.07	0.89	3.03	3.19
MCAF	$/Vertical m	0.0061	0.0062	0.0064	0.0063	0.0071	0.0056	0.0063
Dewatering	$/t ore	0.19	0.19	0.19	0.19	0.19	0.19	0.19

Financial Parameters	Unit	Nkran	Dynamite Hill	Akwasiso	Adubiaso	Nkran Extension	Abore	Asuadai
GC Drilling	$/t ore	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Rehabilitation	$/t ore	0.30	0.30	0.30	0.30	0.30	0.30	0.30
Difference in Waste and Ore Mining Cost								
Oxide	$/t	-	-	-	-	-	-	-
Transition	$/t	0.1443	0.1443	0.1443	0.1443	0.1443	0.1443	0.1443
Fresh	$/t	0.0619	0.0619	0.0619	0.0619	0.0619	0.0619	0.0619
Modifying Factors								
Dilution	%	5%	5%	5%	5%	5%	5%	5%
Ore Loss	%	5%	5%	5%	5%	5%	5%	5%

16.3.3.2 Slope Design Angles

In July 2014 Mining One ("M1") was contracted by Asanko Gold to:

- Review the available FS from SRK
- Assess the potential to increase the overall slope design angles
- Identify areas that require further work to advance the study to feasibility level

Asanko Gold provided access to the following:

- Geological reports
- PMI Mining reports
- Site visit
- SRK Geotechnical reports
- Borehole geotechnical logs and photographs
- The outcome of the investigation includes the following:
  - Geotechnical recommendations for the upper saprolite zone (laterites and oxides)
  - Geotechnical recommendations for the lower saprolite zone (transition zone and saprocks)
  - Geotechnical recommendations for the hard rock zone (fresh rock)
  - The pit slope design parameters were provided by Mining One Consulting for the Nkran pit. The slopes were provided based on the weathering codes within the block model (i.e. Oxide / Transition / Fresh)
  - Two sets of parameters were provided for Nkran. Walls in the north and south were deemed to require a similar geotechnical approach, whereas the east and west wall geotechnical parameters are the same

Table 16-2: Nkran Slope Design Parameters (Source: M1 2014)

Nkran						
Parameter	Oxide		Transition		Fresh	
	E&W	N&S	E&W	N&S	E&W	N&S
Flitch Height (m)	3	3	3	3	3	3
Number of Flitches	2	2	4	4	6	6
Bench Height (m)	6	6	12	12	18	18
Batter Angle °	60	60	70	70	70	80
Berm Width (m)	6	6	7	7	7	8
Inter-Ramp Angle (IRA)	32.4	32.4	47.8	47.8	53.0	58.2
Stack Height (m)	40	40	60	60	108	108

The slope design parameters for all the satellite pits (Akwasiso, Dynamite Hill, Adubiaso, Adubiaso Extension, Abore, Asuadai and Nkran Extension) were interrelated to the Nkran slopes produced by M1 Consultants and the least aggressive slope angle for Fresh was used for the satellite pits.

Table 16-3: Satellite Pits Slope Design Parameters (source: M1 2014)

Satellite Pits						
Parameter	Oxide		Transition		Fresh	
	East	West	East	West	East	West
Flitch Height (m)	3	3	3	3	3	3
Number of Flitches	2	2	4	4	6	6
Bench Height (m)	6	6	12	12	18	18
Batter Angle °	60	60	70	70	70	70
Berm Width (m)	6	6	7	7	7	7
Inter-Ramp Angle (IRA)	32.4	32.4	47.8	47.8	53.0	53.0
Stack Height (m)	40	40	60	60	108	108

16.3.3.3 Geohydrology – Reference M1 2014

16.3.3.4 Mining Costs

Mining costs are based on actual contractor prices and costs associated with the actual cost currently being experienced at the Nkran pit. The cost estimates comprise of the following elements:

- Mobilisation, establishment and demobilisation
- Provision and maintenance of all equipment necessary to carry out the works

- Short term planning of the works
- Clearing and grubbing, topsoil stripping and stockpiling
- Drill and blast of all relevant ore and waste material, including pre-splitting
- Excavate and load of all materials
- Hauling, dumping and stockpiling of all materials to the designated destinations
- Construction, or reconstruction and maintenance of all necessary haul roads
- Grade Control drilling and sampling from dedicated drill rigs as per BOQ
- Re-handle of ROM stockpile to the ROM tip as per the plant feed schedule
- The provision and control of surface drainage
- The management and removal of all water within the open pit area and associated surface activities, including removal of storm water and ground water
- Provision of all pit and dump lighting facilities if required
- Profiling of final dumps and other disturbed areas as directed by the superintendent
- Grade Control drilling
- Carry out secondary breaking of ore as required
- Provision and management of all personnel for the mining activities
- Provision of safety, environmental and quality assurance plans
- Attend meetings and report progress of the works

16.3.3.5 Processing Recoveries and Costs

Processing costs and recoveries for each metallurgical domain were received from DRA and modelled in 3D for each of the deposits. Refer to Section 13.

16.3.3.6 Financial Parameters

The gold price and discount rate used in the optimisations are summarised in the Table 16-4 below.

Table 16-4: Optimisation Financial Parameters (source: Asanko Gold)

Parameter	Unit	Value
Discount Rate	%	5
Base Price	$US/oz Au	1,300
Government Royalty	%	5.0
Refining Cost	$US/oz Au	4.00
Nett Selling Price	$US/oz Au	1,231

16.3.3.7 Bench Height Selection

The geological resource estimation was conducted on a 5m x 5m x 3m (X, Y and Z respectively) block model size. So in order to achieve mining selectivity and dilution reduction, the decision was made to select mining equipment to mine the ore zone in 3m flitches, the waste zones are to be mined in 6m heights.

16.3.3.8 Ore Dilution and Ore Loss

For the purpose of the optimisations, a grade dilution of 5.0% was applied to allow for any waste inclusion in the blasted ore and a mining recovery of 95.0% was applied to allow for any cross-haulage of ore to the waste rock dumps. The mining dilution and mining recovery factors were applied in the optimisation software.

16.3.3.9 Optimisation Results

The initial pit optimisation runs were focused on determining the best cut-off grade for the Project. Pit optimisations where completed for a range of different cut-off grades between 0.5 g/t and 0.85 g/t, these where linked with a financial evaluation to determine highest NPV value.

The result indicated that a 0.7 g/t cut-off grade produce the highest value NPV The 0.7g/t cut-off grade was then used for further pit optimisation analysis.

Nkran Shell Selection

Table 16-5 details the pit optimisation results for Nkran. Pit 90 in the table indicated the highest NPV value, but after analysing all the results, it was evident that for a minor decrease in NPV, an $80/oz saving on the AISC was likely and therefore Pit 85 was selected as the pit for detailed design purposes.

Table 16-5: Nkran Optimisation Results (source: Asanko Gold 2017)

Pit	Total Tonnes	Ore Tonnes	Waste Tonnes	Strippin g Ratio	Reserves	NPV	Feed Grade	Cost (AISC)
#	Mt	Mt	Mt	tw:to	Koz	$m	g/t	$/oz
Pit 85	161.39	23.48	137.91	5.87	1 461.26	555.684	1.94	760.40
Pit 86	163.24	23.60	139.64	5.92	1 468.58	555.921	1.94	806.64
Pit 87	185.62	25.13	160.49	6.39	1 557.63	557.199	1.93	830.39
Pit 88	187.94	25.32	162.62	6.42	1 567.95	557.587	1.93	832.78
Pit 89	191.58	25.72	165.86	6.45	1 587.01	557.686	1.92	837.68
Pit 90	194.62	25.95	168.67	6.50	1 600.57	557.767	1.92	841.09

Figure 16-4 illustrates the results generated in NPVS and the highest NPV and selected Pits. Also evident from the graph is that selecting Pit 85 instead of Pit 90 had very little impact on NPV, but a much lower AISC, likely because of a lower stripping ratio.

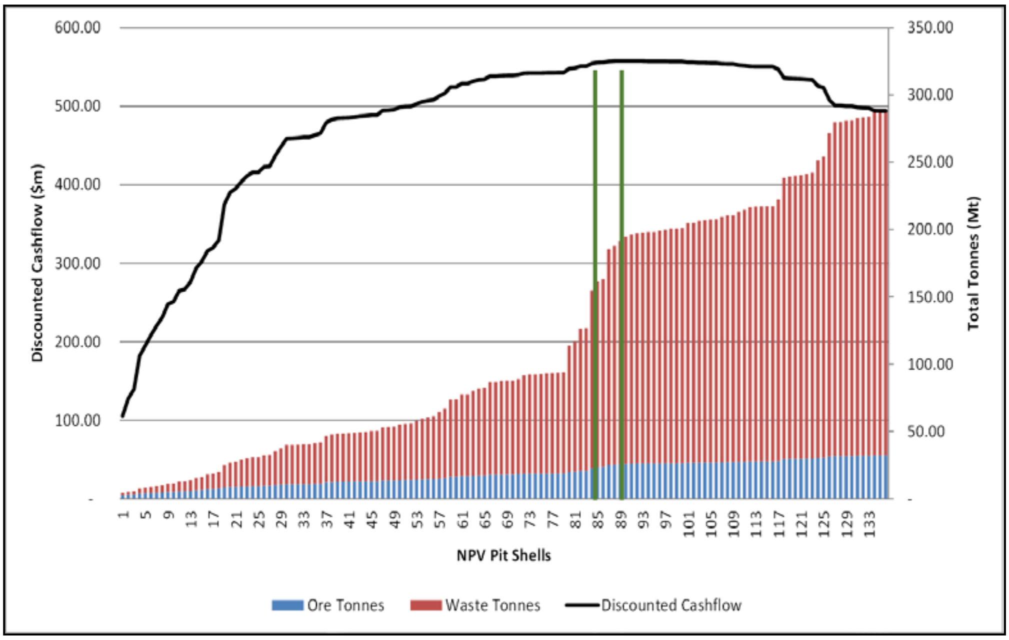


Figure 16-4: Nkran NPVS® Analysis (source: Asanko Gold 2017)

Akwasiso Shell Selection

Table 16-6 details the results from the Akwasiso pit optimisation run. The maximum NPV pit was selected for Akwasiso because of very little change in the results around the ultimate pit (Pit 80).

Table 16-6: Akwasiso Shell Selection (source: Asanko Gold 2017)

Pit	Total Tonnes	Ore Tonnes	Waste Tonnes	Stripping Ratio	Reserves	Discounted Cash flow	Feed Grade	Cost (AISC)
#	Mt	Mt	Mt	tw:to	Koz	$m	g/t	$/oz
Pit 78	34.34	4.62	29.72	6.44	240.71	86.043	1.62	857.32
Pit 79	35.10	4.67	30.43	6.52	243.48	86.071	1.62	862.17
Pit 80	35.28	4.68	30.60	6.53	244.15	86.072	1.62	863.36
Pit 81	35.38	4.69	30.69	6.55	244.52	86.071	1.62	864.03
Pit 82	36.11	4.71	31.40	6.66	246.56	86.040	1.63	867.83

Figure 16-5 illustrates minimal changes from the ultimate pit.

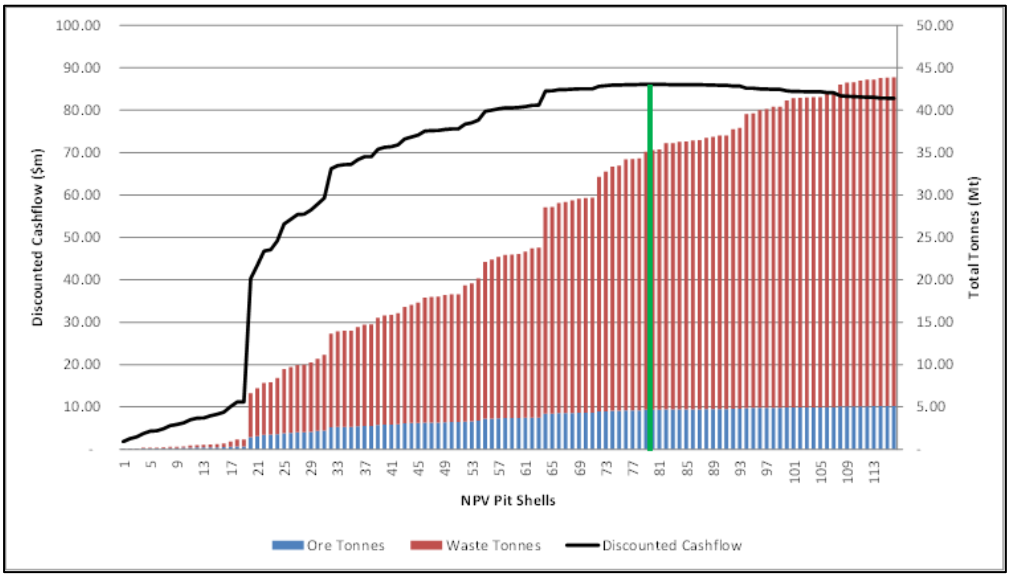


Figure 16-5: Akwasiso NPVS Analysis (source: Asanko Gold 2017)

Dynamite Hill Shell Selection

Table 16-7 details the results from the Dynamite Hill pit optimisation runs. The maximum NPV pit was also selected for Dynamite Hill because of the insignificant changes evident between the ultimate pit (Pit 56) and the adjacent pits.

Table 16-7: Dynamite Hill Shell Selection (source: Asanko Gold 2017)

Pit	Total Tonnes	Ore Tonnes	Waste Tonnes	Stripping Ratio	Reserves	Discounted Cash flow	Feed Grade	Cost (AISC)
#	Mt	Mt	Mt	tw:to	Koz	$m	g/t	$/oz
Pit 54	17.76	2.79	16.05	5.76	153.76	57.105	1.72	731.97
Pit 55	17.79	2.79	16.07	5.76	153.87	57.111	1.72	732.24
Pit 56	17.85	2.80	16.12	5.76	154.23	57.112	1.71	733.23
Pit 57	18.25	2.84	16.47	5.79	156.15	57.096	1.71	738.60
Pit 58	18.30	2.85	16.52	5.80	156.31	57.094	1.71	739.08

Figure 16-6 below illustrates minimal changes from the ultimate pit.

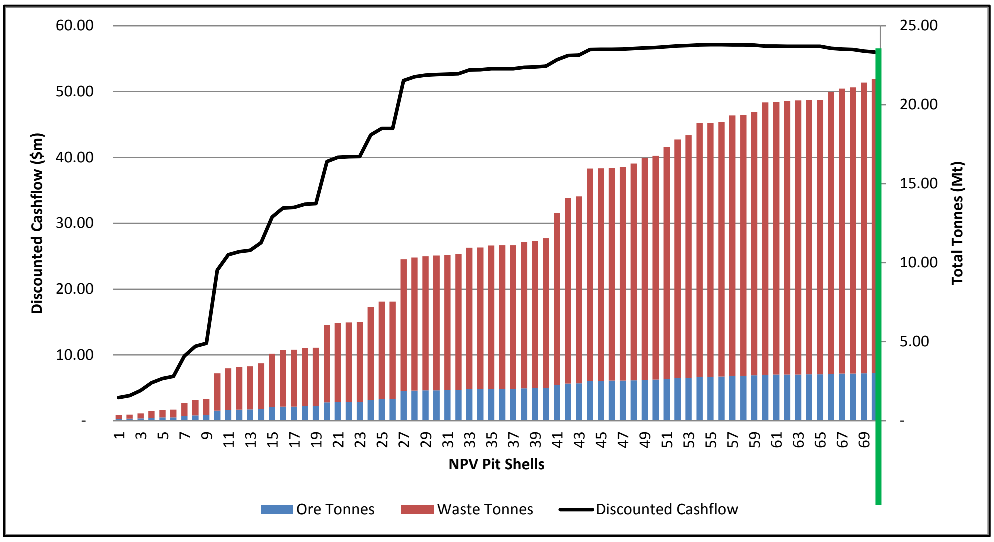


Figure 16-6: Dynamite Hill NPVS Analysis (source: Asanko Gold 2017)

Adubiaso, Adubiaso Extension, Abore, Asuadai and Nkran Extension Shell Selection

The Deswik® software was used to produces the optimal pit shells for Adubiaso, Adubiaso Extension, Abore, Asuadai and Nkran Extension. Deswik® also uses the Lerchs-Grossman algorithm to produce optimal pit shells based on input parameters. The ultimate pit shells were selected for all these pits.

Pit Optimisation Summary

Table 16-8 is a summary illustrating the content of each of the shells selected for the different deposits of Obotan. These shells where used to guide the detailed designs of all the pits.

Table 16-8: Results of the optimal pit shells selected for Obotan Deposits (source: Asanko Gold 2017)

Deposit	Optimisation Method	Total Tonnes (Mt)	Ore Tonnes (Mt)	Waste Tonnes (Mt)	Stripping Ratio (tw:to)	Ounces (Moz)	Feed Grade (g/t)
Nkran	NPVS®	161.39	23.48	137.91	5.87	1.46	1.94
Akwasiso	NPVS®	35.28	4.68	30.60	6.53	0.24	1.62
Dynamite Hill	NPVS®	17.85	2.80	16.12	5.76	0.15	1.71
Adubiaso	Deswik®	15.58	2.34	13.24	5.65	0.15	1.95
Adubiaso Ext	Deswik®	3.84	0.23	3.62	15.92	0.01	1.87
Nkran Ext	Deswik®	2.70	0.21	2.50	12.17	0.02	2.30
Abore	Deswik®	19.32	3.57	15.76	4.42	0.18	1.54
Asuadai	Deswik®	5.28	1.26	4.03	3.21	0.05	1.16
Total		261.25	38.56	223.76	5.80	2.26	1.82

16.3.4 Obotan Mine Design

The purpose of this section of the Report is to list the various design inputs that were used in the designs of all the Obotan 2017 DFS pits. These design criteria were used in correlation with Deswik® software to complete the final schedules and designs for the Obotan pits.

The first phase of the mine planning cycle required that an economic open pit optimisation be conducted and this was completed in March 2017 for the Obotan pits.

The next phase in the cycle was conducted in the general mine planning programme Deswik® and involved the design of the final and interim stage pits.

The objective of the pit design process was to transform the final optimal shells generated in NPV Scheduler and Deswik® into a practical workable designed pit, which involved the inclusion of practical operational design parameters such as ramp layouts, benches, slope angles, pushbacks and berm configurations.

These practical pit designs formed a critical input for the scheduling and reserving processes. All criteria used in these designs are discussed in this document. The geotechnical information has been derived from using inputs from internal discussions, supplier’s recommendations, equipment strategy philosophy and outside consultant recommendations such as the M1 Consultants for geotechnical design criteria.

16.3.4.1 Geotechnical Considerations

The slope design parameters used for designing the Obotan pits are summarised in Table 16-9 and Table 16-10 below. These parameters were provided by M1 and were also used in the open pit optimisation process. No hydrological parameters were considered in the designing of the pits.

Table 16-9: Nkran Geotechnical Parameters (source: M1 2014)

	East and West Walls			North and South Walls		
	Fresh	**Rl / Rs**	**Ru**	**Fresh**	**Rl / Rs**	**Ru**
Bench Height	18.0	12.0	6	18.0	12.0	6
Berm width	7.0	7.0	6	8.0	7.0	6
Batter angle	70.0	70.0	60.0	80.0	70.0	60.0
Step out	13.6	11.4	9.5	11.2	11.4	9.5
Batter width	6.6	4.4	3.5	3.2	4.4	3.5
Ramp	22.0	22.0	22.0	22.0	22.0	22.0
Inter Ramp Angle	53.0	47.8	32.4	58.2	47.8	32.4

Table 16-10: All Other Obotan Pits Geotechnical Parameters (source: M1 2014)

	West Wall with Ramp			East Wall		
	Fresh	**Rl / Rs**	**Ru**	**Fresh**	**Rl / Rs**	**Ru**
Bench Height	18.0	12.0	6	18.0	12.0	6
Berm width	7.0	7.0	6	7.0	7.0	6
Batter angle	70.0	70.0	60.0	70.0	70.0	60.0
Step out	13.6	11.4	9.5	13.6	11.4	9.5
Batter width	6.6	4.4	3.5	6.6	4.4	3.5
Ramp	22.0	22.0	22.0	22.0	22.0	22.0
Inter Ramp Angle	53.0	47.8	32.4	53.0	47.8	32.4

16.3.4.2 Technical Design Parameters

Production Rate Selection

The mining strategy requires that additional ore tonnage must be produced to build sufficient capacity on the ore stockpiles to ensure the treatment strategy can be maintained at the required grade profile. As a result, the Obotan pits will be mining different tonnage profiles as required to fit the overall Project strategy and plant feed targets.

All design parameters were based on a delivered plant feed rate of between 300 ktpm and 417 ktpm. The reason for this is that the current plant available for the Project has a capacity of 300 ktpm and after the plant upgrade (January 2018) will have a capacity of 417 ktpm. The ore tonnes required will be produced from a combination of the Obotan pits.

Pit Access

Access to the pits for mining activities will be through ramps, starting at surface and declining at a 10% slope gradient to intersect the floor of the pits. In some instances, like at pit bottom, the gradient might be steeper but only for short distances.

Wherever possible the ramp exits were located at the closest possible distance to the waste dumps to minimise ex-pit haulage. The criteria for pit and waste dump ramp designs were based on the width and turning circle of a CAT 777.

Within the pit designs, a minimum distance of 50m is required between the pit edge and final dump toe, which is considered acceptable. However, this is only a minimum separation distance, as the 150% revenue optimisation shell perimeter associated with improved commodity prices was used to demarcate surface area to be left free of infrastructure that might sterilise remaining underground resource. Hence, pit rim to Waste Rock Dump (“WRD”) toe distances will exceed the minimum of 50m in most cases.

In general, a minimum mining width of 25m is maintained, although tight conditions in and around the existing pits sometimes result in pinch points where narrower widths are inevitable. However, these occurrences are only over limited lateral and vertical distances.

The ramp design parameters used for the pit design process were based on industry norm and the experience with the current road width being used at Nkran, which has been adequate. In some instances, a single, or double lane ramp will be designed depending on the available space, practicality and the size of the pit.

These dimensions can be substantiated by using the rule of thumb for determining ramp lane dimensions. Guidelines specify that the vehicle width should be multiplied by a factor of 2.5 to 3 for two-lane traffic and 1.5 to 2 for single-lane traffic in order to determine the effective operating width of the ramp and incorporating the road infrastructure like the safety berm and drainage channel. The designs for the ramps are detailed in Table 16-11 below.

Table 16-11: CAT 777 Road Width Design (source: DRA 2017)

Description	Unit	Value	Comment
Equipment Width	m	5.5	Cat 777
Effective Operating Width for Two Way Traffic	m	16.5	3 x Equipment Width
Effective Operating Width for One Way Traffic	m	11	2 x Equipment Width
Safety Berm	m	3.6	Depending on Truck Wheel Diameter
Design Width Calculation	m	16.26	12 + 3.6 + 1.4 (Drain)
Practical Design Width for One Way Traffic	m	16	
Design Width Calculation	m	21.76	18 + 3.6 + 1.4 (Drain)
Practical Design Width for Two Way Traffic	m	22.5	

16.3.4.3 Pit Designs

Nkran Pit Design

The Nkran pit was designed in 4 mining stages. These mining stages ensure the pit is developed in a practical and incremental manner, while achieving your production targets.

Nkran final pit design is illustrated below in Figure 16-7.

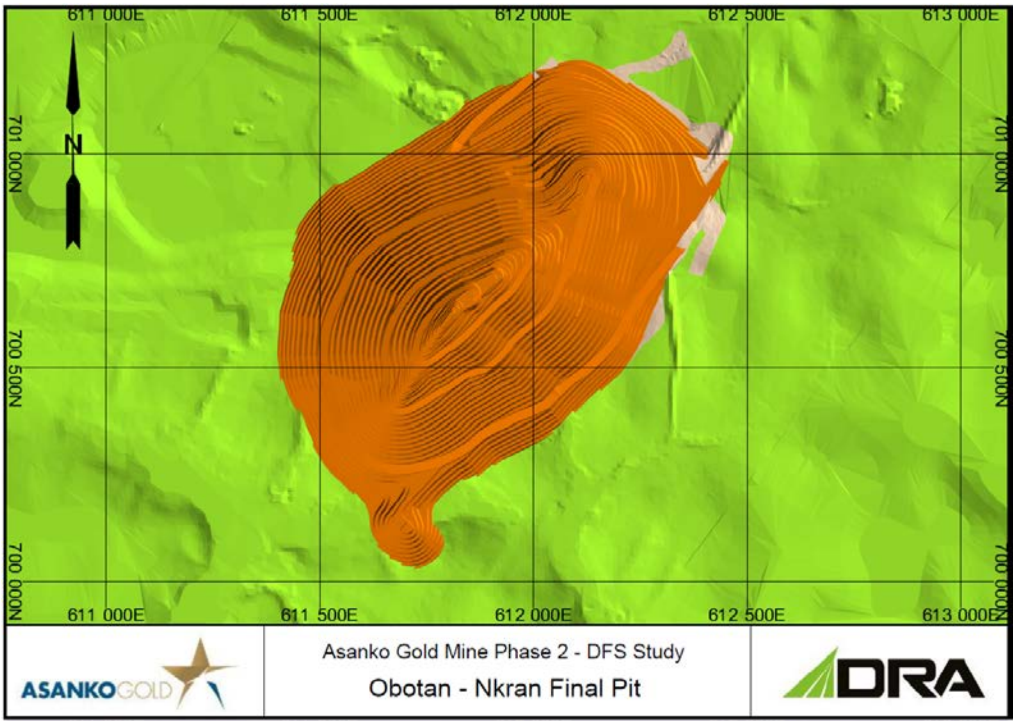


Figure 16-7: Nkran Final Pit Design (source: Asanko Gold 2017)

Adubiaso Pit Design

Adubiaso and Adubiaso Extension final pit designs are illustrated below in Figure 16-8 and Figure 16-9 respectively.

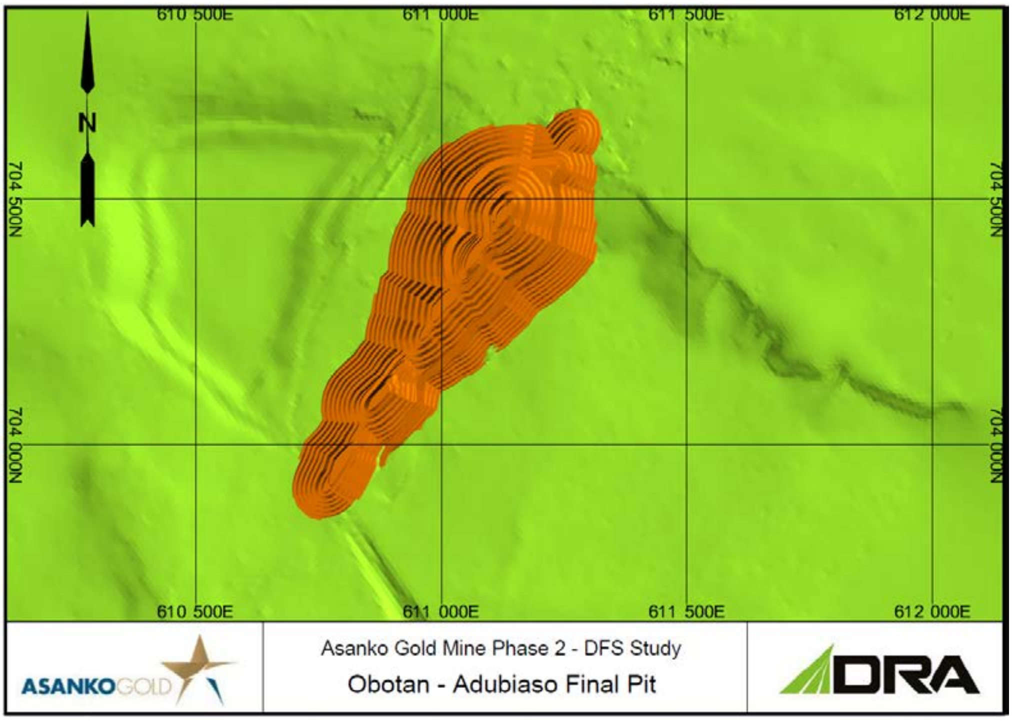


Figure 16-8: Adubiaso Pit Design (source: Asanko Gold 2017)

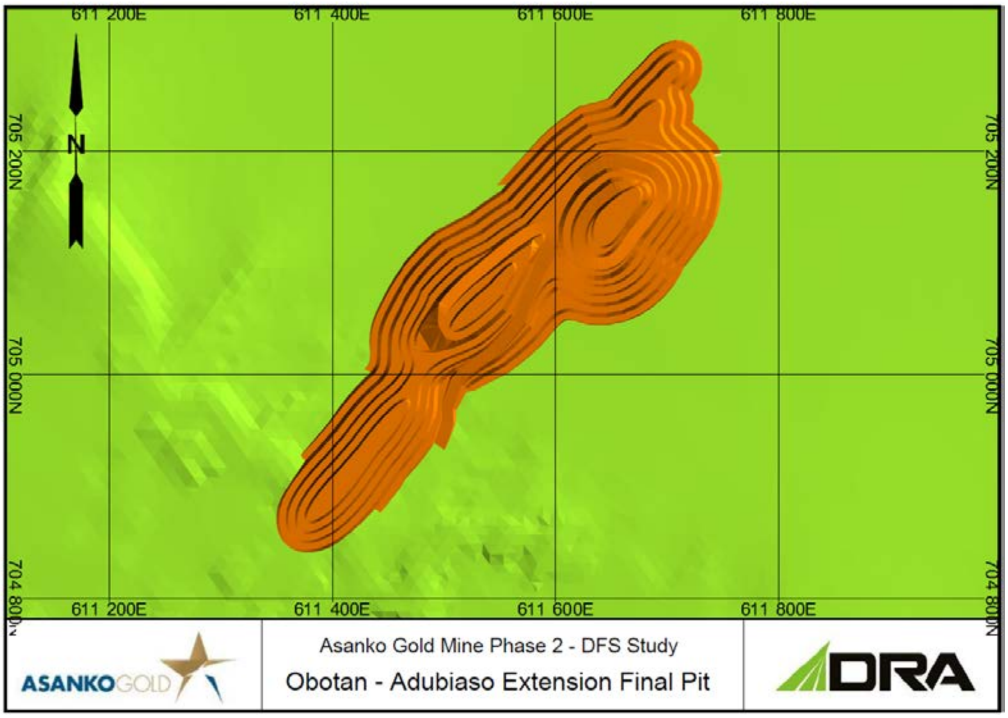


Figure 16-9: Adubiaso Extension Pit Design (source: Asanko Gold 2017)

Abore Pit Design

The Abore deposit makes up 3 separate pits. Abore Main, Abore North and Abore South. The designs are illustrated below in Figure 16-10.

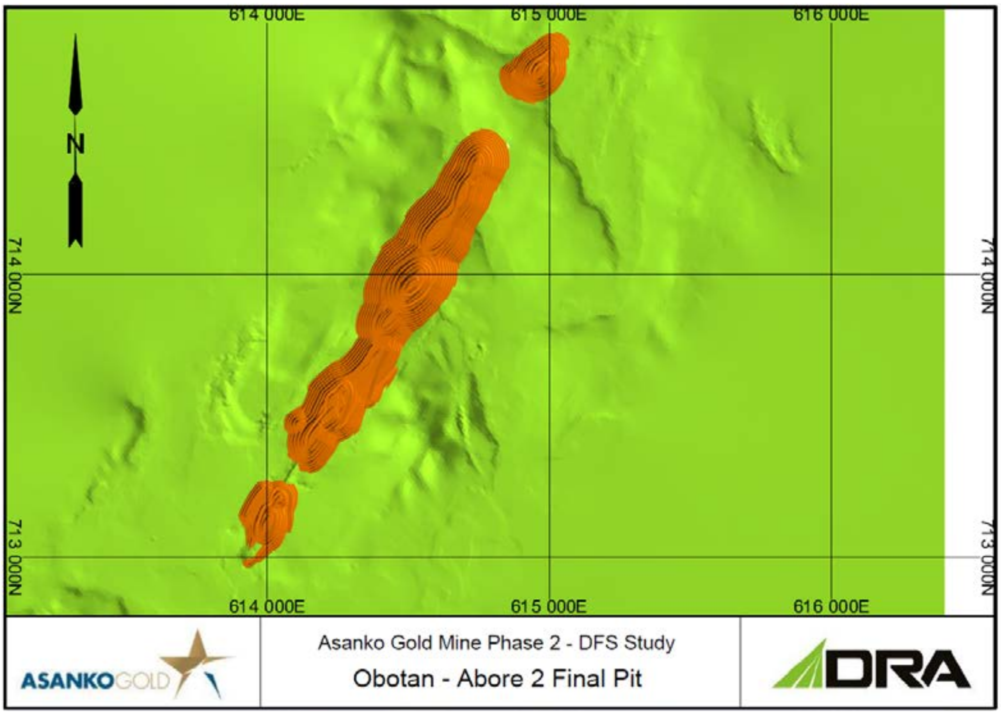


Figure 16-10: Abore Pit Design (source: Asanko Gold 2017)

Dynamite Hill Pit Design

The Dynamite Hill pit will be mined in two stages, the development of the two stages are illustrated below in Figure 16-11.

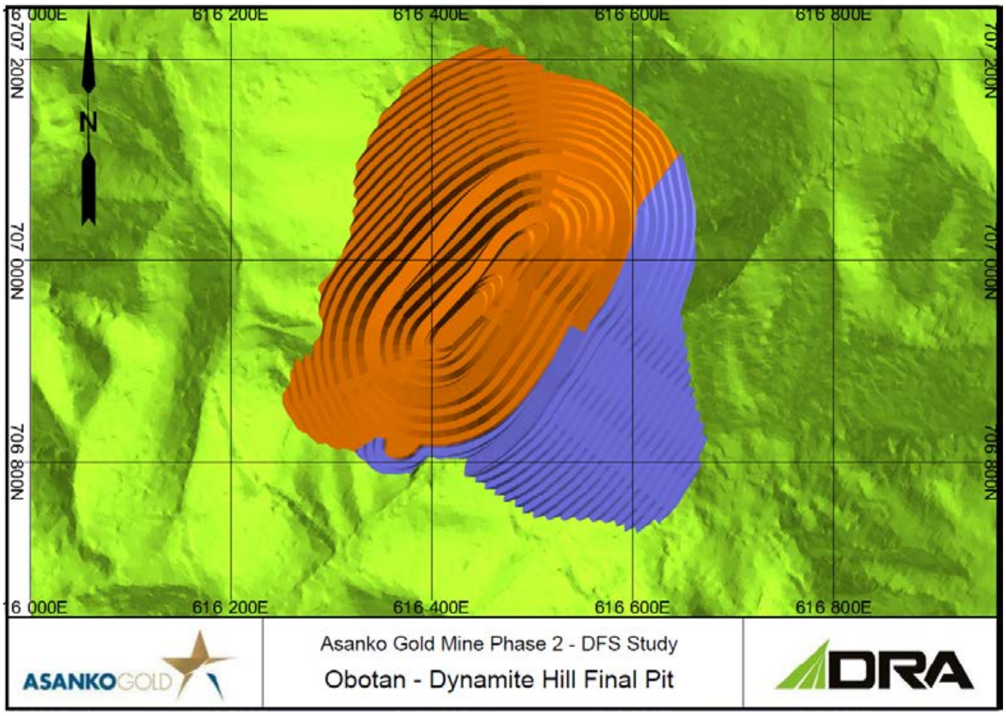


Figure 16-11: Dynamite Hill Final Pit Design (source: Asanko Gold 2017)

Nkran Extension Pit Design

Nkran Extension final pit design is illustrated below in Figure 16-12.

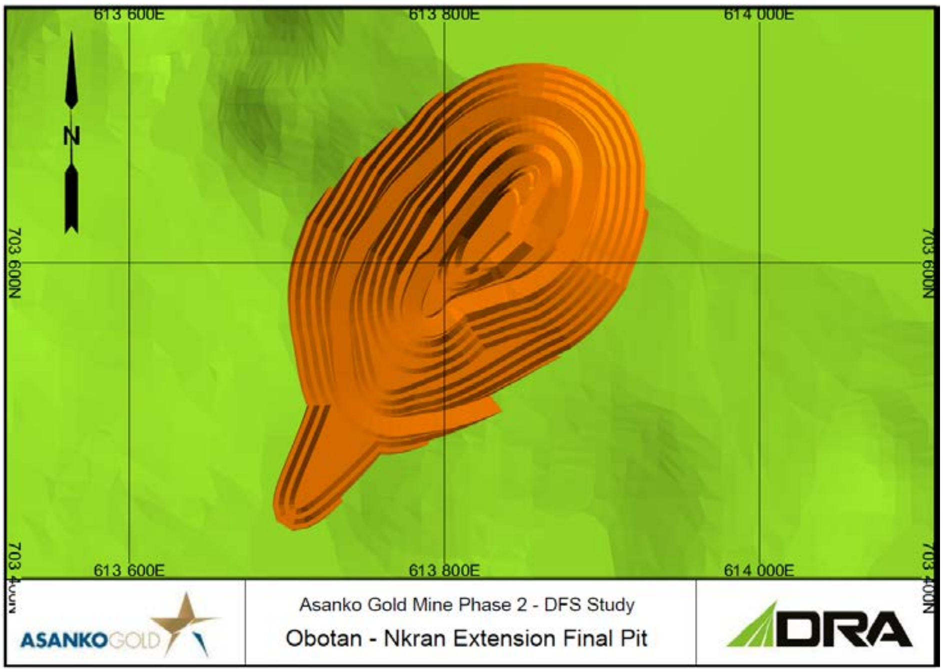


Figure 16-12: Nkran Extension Pit Design (source: Asanko Gold 2017)

Asuadai Pit Design

Asuadai final pit designs are illustrated below in Figure 16-13.

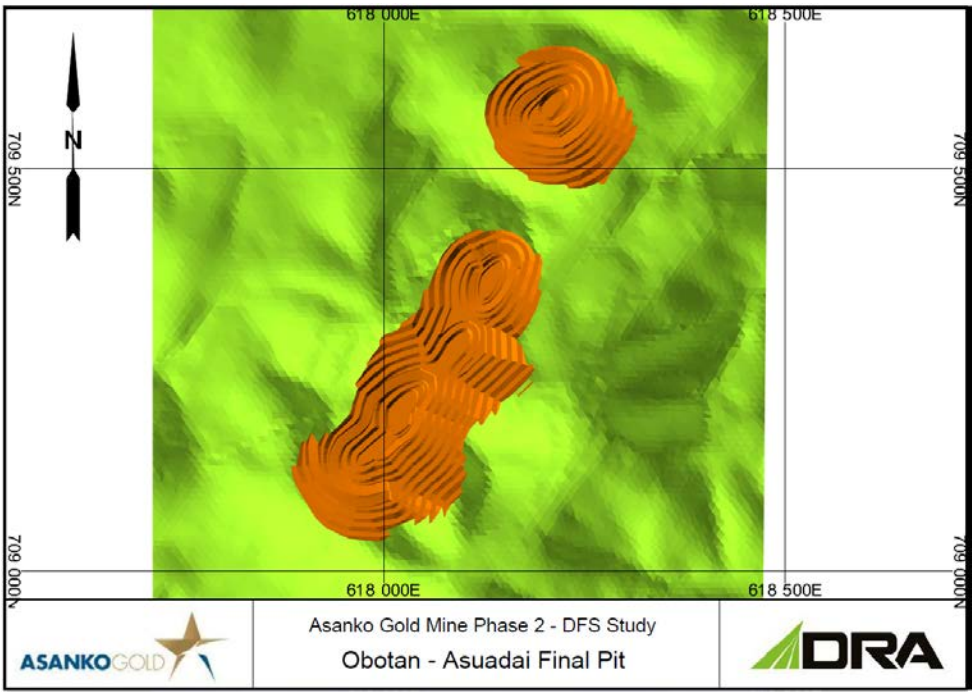


Figure 16-13: Asuadai Pit Design (source: Asanko Gold 2017)

Akwasiso Pit Design

The Akwasiso pit will be mined in 3 stages, the first stage is targeting the oxides ore of the pit, the second and third stages are practical stages to optimise value.

The Akwasiso final pit design is illustrated below in Figure 16-14.

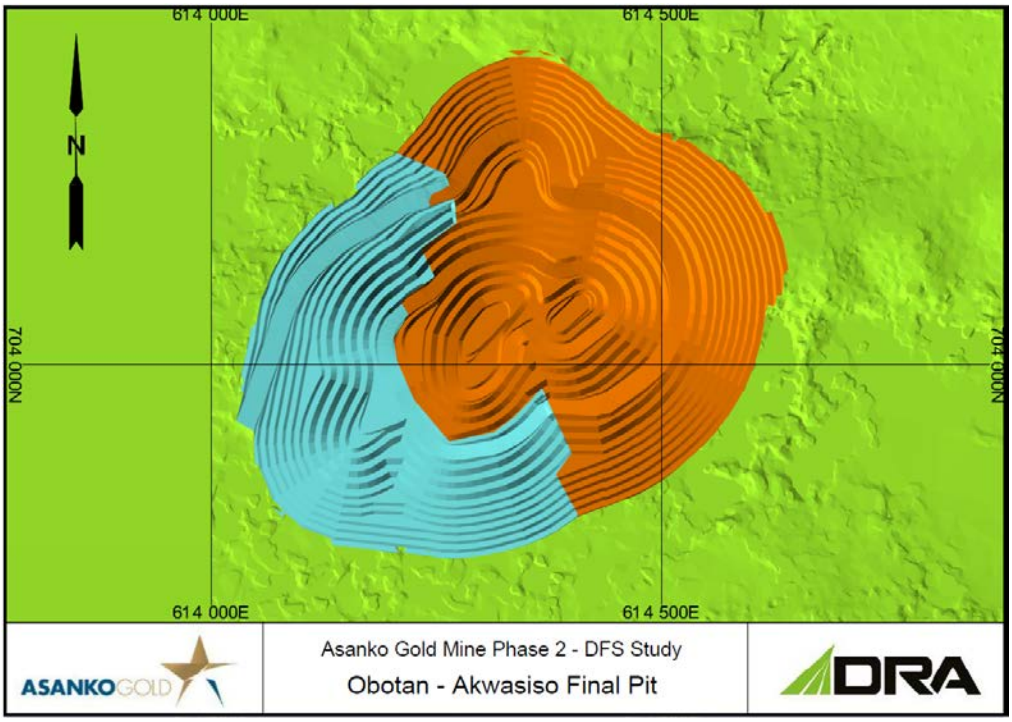


Figure 16-14: Akwasiso Final Pit (source: Asanko Gold 2017)

16.3.4.4 Optimal Pit vs Detailed Pit Design

In order to determine whether the final pit designs are sufficiently similar in shape, size and position, to the original optimised pits, the potential ore and waste contents within the design are measured, and compared to the relevant pits selected for design. A comparison between the selected pits and designed pits is outlined below.

Table 16-12: Optimal Pits vs. Pit Design Comparison (source: Asanko Gold 2017)

Description	Units	Pit Shells	Design	Difference
Ore Tonnes	Tonnes	38,563,658	38,230,554	-0.9%
Waste Tonnes	Tonnes	223,761,221	229,838,292	2.7%
Ounces	Oz's	2,259,188	2,162,574	-4.3%
Grade	g/t Au	1.82	1.76	-3.4%
Stripping Ratio	Wt : Ot	5.80	6.01	

Note [1] Includes very low grade ore (~10Mt at 0.55 g/t Au) and mineralised waste (<0.5 g/t Au)

Variances between the pit shells and the final pit design will invariably occur due to the application of design factors such as the ramp design parameters, as well as the detailed slope design. In the design, actual batter angles and berm widths are used, as opposed to the overall slope angle, (inclusive of a ramp system) as indicated in the pit optimisation results.

16.3.4.5 Mine Production - Obotan

See Section 16.5 for the Obotan production schedules graphs. Table 16-13 below details a summary of the Obotan pits.

Table 16-13: Obotan Production Pit Summary (source: DRA 2017)

Pit	Total Tonnes (t)	Ore Tonnes (t)	Waste Tonnes (t)	Stripping Ratio	Ounces (oz)	Feed Grade (g/t)
Nkran	162 306 585	22 771 779	139 534 806	6.13	1 400 624	1.91
Akwasiso	36 072 064	4 948 844	31 123 220	6.29	240 628	1.51
Dynamite Hill	19 470 767	2 842 807	16 627 960	5.85	136 492	1.49
Adubiaso	18 208 695	2 281 026	15 927 669	6.98	139 388	1.90
Adubiaso Extension	2 620 290	211 643	2 408 647	11.38	10 401	1.53
Nkran Extension	2 381 701	192 699	2 189 001	11.36	13 848	2.24
Abore	20 545 249	3 184 877	17 360 372	5.45	151 942	1.48
Asuadai	6 463 495	1 304 190	5 159 305	3.96	45 616	1.09

16.3.4.6 Fleet Requirements

Asanko Gold will make use of contractors to mine all the deposits and therefore all equipment requirements will be determined by the contractor which will be invited to tender. Asanko Gold are in the fortunate position to have PW mining currently on site with mining rates to compare all future work and tender prices to. A tender process for each of the pits will be followed whereby the contractor will specify the equipment requirements and Asanko Gold will adjudicate, with the help of external companies, the contractor and through this process the capacity and capability will be determined.

16.4 Esaase Project Mine Design

16.4.1 Geology and Geological Resource

For complete geological information, refer Section 14 of this report.

16.4.2 Esaase Resource Characteristics

The resource model [10_keegan_20121005r.dm] forms the basis of the pit optimisations and Mineral Reserves. Pit optimisations were carried out on Measured and Indicated Resources only.

The Esaase block model average grade is approximately 1.4g/t Au, it is classified as a low to medium grade deposit. The ore body is not tabular in shape, but massive. Hence the development of a 3D resource block model for best spatial representation of the deposit.

The Esaase ore body strike direction is in a northeast to southwest direction. The mineralised lenses dip steeply (at approximately 70^{o}) in the northwest direction. Figure 16-15, Figure 16-16 and Figure 16-17 illustrate the Gold Grade distribution, Rock type orientation and Resource classification respectively in relation to the Esaase Pit.

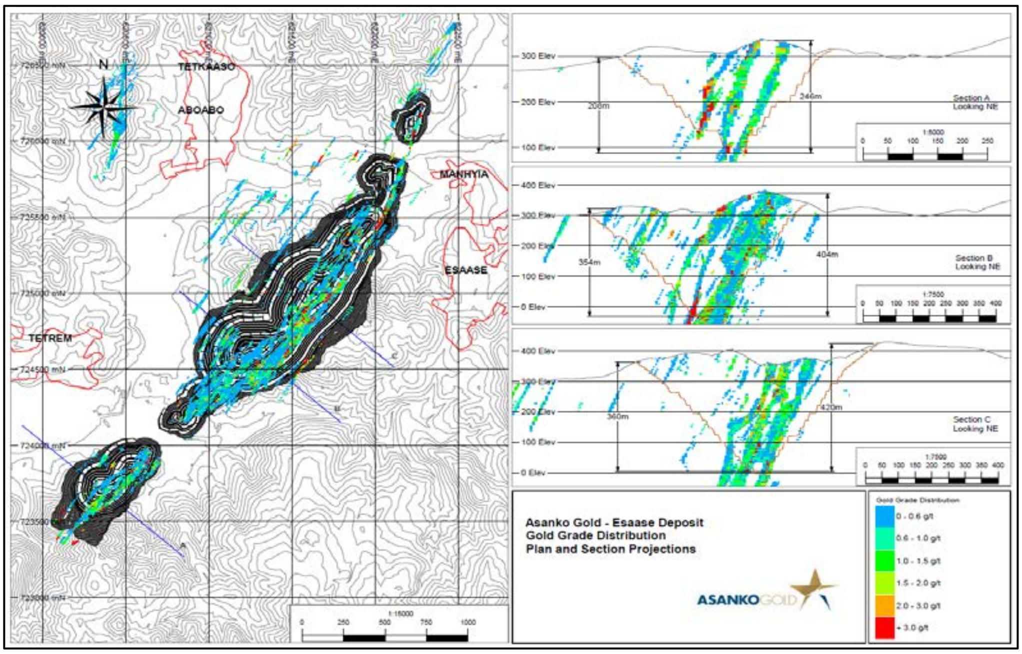


Figure 16-15: Plan and Section Views of the Gold Grade Distribution (source: DRA 2017)

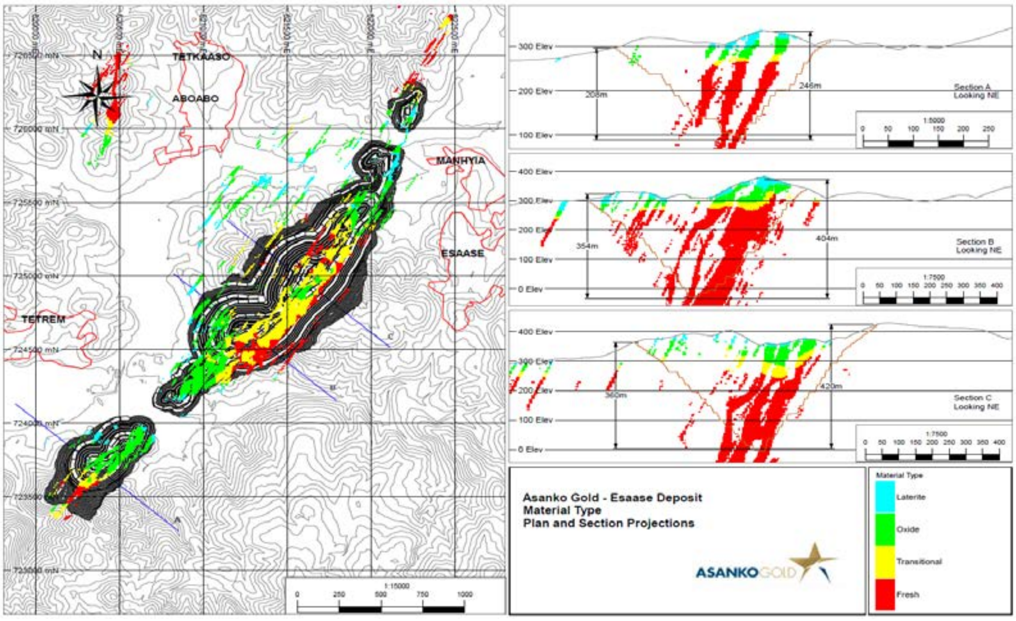


Figure 16-16: Plan and Section Views of the Material Types (source: DRA 2017)

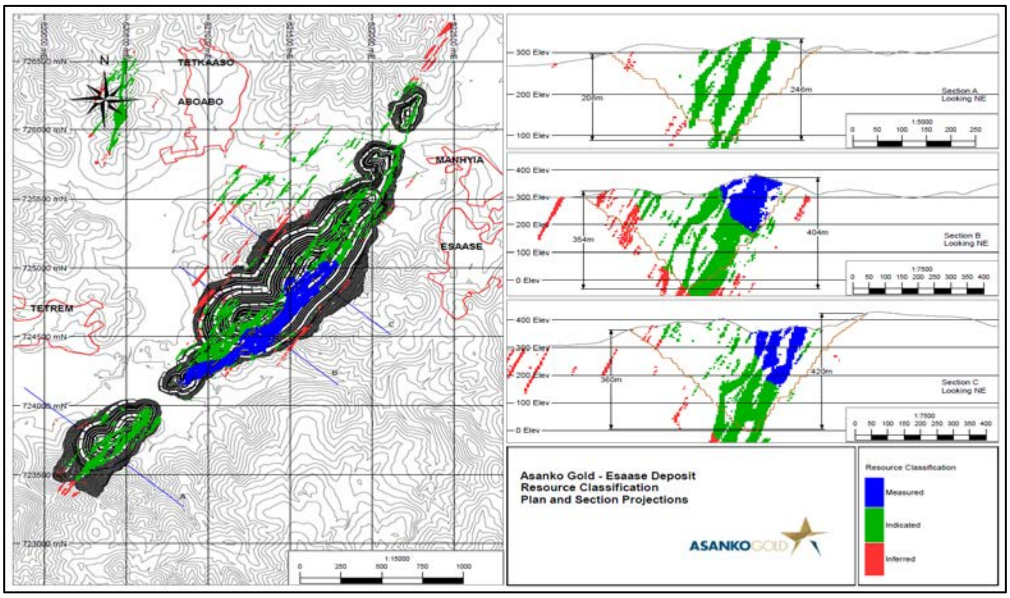


Figure 16-17: Plan and Section Views of the Resource Classification (source: DRA 2017)

16.4.3 Open Pit Optimisation

16.4.3.1 Whittle Four-X® Optimisation Methodology

Pit optimisations were conducted using Whittle Four-X® pit optimisation software. The software utilises the Lerchs-Grossmann algorithm to generate a pit shell that provides the maximum operating margin, or cash flow (before capital, taxes or discounting), based on a resource model and a set of economic and technical input parameters.

Pit optimisation economic parameters include unit mining costs, processing costs, general and administrative costs, and unit revenue estimates. Pit optimisation technical parameters include pit footprint constraint, estimates of mining dilution, mining loss, process recovery, and pit overall slope angles. Pit overall slope angles are derived from geotechnical criteria adjusted for the expected haulage ramp layout.

In accordance with the guidelines of National Instrument 43-101 Standards of Disclosure for Mineral Projects and the CIM's Definition Standards for MRE and MRev, only those ore blocks classified in the Measured and Indicated categories are allowed to drive the pit optimisation for a feasibility level study. Inferred resource blocks, regardless of grade and recovery, bear no economic value and are treated as waste.

A series of nested pit shells was generated by varying, or factoring unit revenue estimates, referred to as revenue factor ("RF"). Nested pit shells are utilised for incremental and present value analysis, and to guide phase pit and ultimate pit design. The cash flows are exclusive of any capital expenditure, or project start-up costs and should be used for pit optimisation comparison purposes only. No project Net Present Value ("NPV") can be derived, or assumed from these cash flows.

No open pit/underground mine cross-over optimisation was conducted on the Esaase resource as part of the feasibility study

16.4.3.2 Mining Dilution and Recovery

For the purpose of the optimisations, a grade dilution of 8.0% was applied to allow for any waste inclusion in the blasted ore and a mining recovery of 92.4% was applied to allow for any cross-haulage of ore to the waste rock dumps. The mining dilution and mining recovery factors were applied in the block model prior to importation into the optimisation software.

Waste block size in the X-Y-Z plane was also reduced to 10m x 10m x 6m to allow for greater accuracy of the slope calculations in the pit optimisation software. Ore block size (x, y and z dimensions respectively) were retained at 5m x 5m x 3m ore blocks.

16.4.3.3 Overall Pit Slopes

The overall pit wall slope angles adopted for the pit optimisation were based on the slope parameters as described in Section 16.4.4.2. The geotechnical parameters used in the Whittle® optimisation are summarised Table 16-14 below.

Table 16-14: Geotechnical Slope Parameters used in the Optimisation (source: M1 2016)

Geo-Technical Slopes	Units[1]	Esaase – 2017 DFS	
		SE Wall	NW Wall
Weathered Zone	°OSA	40.4° SE	40.4° NW
Transitional Zone	°OSA	52.8° SE	46.6° NW
Fresh Zone	°OSA	56.3° SE	50.0° NW

Note [1] OSA (Overall Slope Angle)

16.4.3.4 Geohydrology reference M1 2016

16.4.3.5 Base Date, Currency and Money Terms

The base date for all costs used in the Whittle® optimisation was based on Quarter 2, 2017. Unless otherwise specifically stated, all monetary values presented in this document are in United States Dollars.

16.4.3.6 Mining Cost Parameters

The mining cost parameters were based on a quotation from a mining contractor currently executing the Obotan mining production plan. The estimate of the mining costs was based on the current mining contractor tendered values for the Esaase deposit. The detail of the mining cost parameters is shown in the (Table 16-15) below.

Table 16-15: Mining Cost Parameters (source: Asanko Gold)

Mining Costs (Esaase)	Units	Value
Reference Elevation	m AMSL	280
Reference Drill and Blast	$/t	0.85
Oxide - Waste	$/t	0.06
Trans - Waste	$/t	0.83
Fresh - Waste	$/t	0.74
Oxide - Ore	$/t	0.08
Trans - Ore	$/t	0.98
Fresh - Ore	$/t	1.06
Pre-split Costs	$/t	0.11
Reference Load and Haul	$/t	1.16
Oxide - Waste	$/t	1.38
Trans - Waste	$/t	1.30
Fresh - Waste	$/t	1.16
Oxide - Ore	$/t	1.16
Trans - Ore	$/t	1.10
Fresh - Ore	$/t	0.98
Dewatering Cost [1]	$/t	-
Pit Wall Support	$/t	0.03
Ancillary and Support Equipment [2]	$/t	-
Grade Control Cost	$/$t_{ore}$	0.17
Ore Re-handle Cost (33%)	$/$t_{ore}$	0.46
Rehabilitation Costs [3]	$/t	-
Ore Incremental Cost - Oxide	$/$t_{ore}$	0.03
Ore Incremental Cost - Trans	$/$t_{ore}$	0.18
Ore Incremental Cost – Fresh	$/$t_{ore}$	0.37
Extra Over - Hauling	$/bcm/100m	0.049
Mining Cost Adjustment Factors		
Ore MCAF – (Positive Elevations)	$/t/Vm	0.0077
Ore MCAF – (Negative Elevations – Oxides)	$/t/Vm	0.0066
Ore MCAF – (Negative Elevations – Transitional)	$/t/Vm	0.0062

Mining Costs (Esaase)	Units	Value
Ore MCAF – (Negatived Elevations - Fresh / Competent)	$/t/Vm	0.0055
Waste MCAF – (Positive Elevations)	$/t/Vm	0.0064
Waste MCAF – (Negative Elevations – Oxides)	$/t/Vm	0.0061
Waste MCAF – (Negative Elevations – Transitional)	$/t/Vm	0.0057
Waste MCAF – (Negatived Elevations - Fresh / Competent)	$/t/Vm	0.0051
Reference Mining Cost	$/t	2.05
Reference Cost (including 15% Mark-up)[4]	$/t	2.36
Monthly Management Fee (including 15% Mark-up)	$/mth	1,347,394

Note [1] Dewatering Cost included in Monthly Management Fee

Note [2] Ancillary Equipment Costs are included in Unit Rate Costs for each activity

Note [3] Rehabilitation Costs included in Mine Closure Costs

Note [4] Reference cost refers to cost of mining Fresh rock type at 280 mamsl. The Esaase LoM average mining cost is US$ 3.19 / tonne (Total Tonnes Mined) and US$ 20.95 / tonne (Ore Tonnes Mined)

16.4.3.7 Mining Cost Adjustment Factors

The mining cost adjustment factors was calculated based on the rock type, relative elevation and the distinction between ore and waste.

The increase in mining cost to adjust for vertical hauling distances, was 0.066 US$/t/Vm, 0.062 US$/t/Vm and 0.055 US$/t/Vm, (benches below reference elevation) for Oxide, Transitional and Fresh economic ore material respectively, and 0.077 US$/t per vertical metre for benches above reference elevation (reference elevation 280 mamsl). The increase in mining cost to adjust for vertical hauling distances, was 0.061 US$/t/Vm, 0.057 US$/t/Vm and 0.051 US$/t/Vm (benches below reference elevation) for Oxide, Transitional and Fresh waste material respectively, and 0.064 US$/t per vertical metre for benches above reference elevation (reference elevation 280 mamsl).

Horizontal hauling distance costs were adjusted using US$0.049/bcm/100m supplied by the mining contractor for all hauling distance in excess of 500m from pit haul road exit. summarises the overhaul distances applied to the different areas of the Esaase pit (zero indicates the pit exit is within 500m of the destination). Table 16-16.

Table 16-16: Horizontal overhaul distances applied in Pit Optimisation (source: DRA 2017)

Pushback	Ore (km)[1]	Waste (km)[2]
Cut 1	0.8	0.3
Cut 2	1.0	0 (0.5)[3]

Pushback	Ore (km)[1]	Waste (km)[2]
Cut 3	0.2	1.3
Cut 4	0.8	0.3
Cut 5	1.1	2.3
Cut 6	1.7	2.9 (0.3)[4]

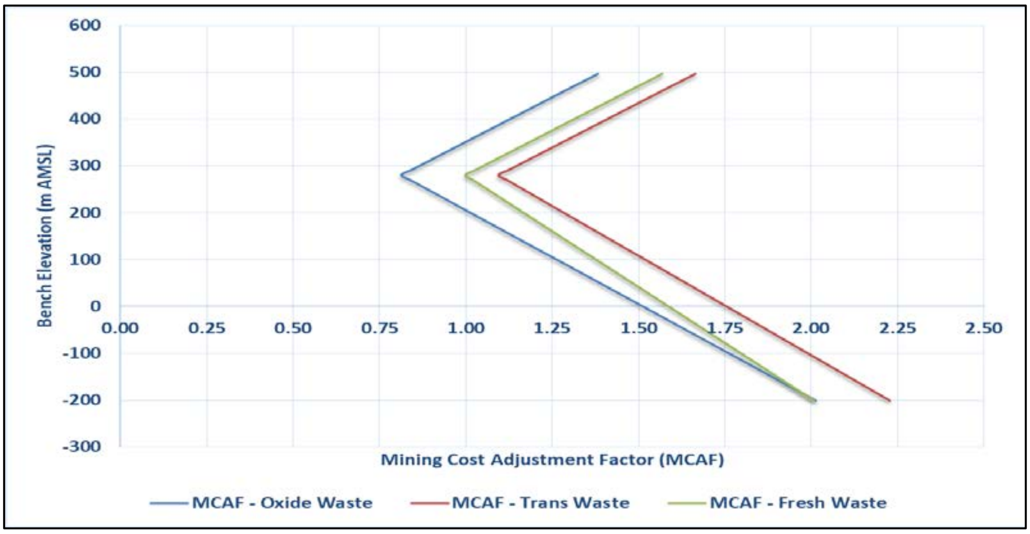
Note [1] Destination ROM Pad

Note [2] Ramp entrance to dumping destination

Note [3] Close proximity dumping for first 3 years until Tetrem village is relocated

Note [4] Backfilling of Cut 5 has been planned in the material movement schedule

Figure 16-18 below illustrates the vertical mining cost adjustment factors used in the pit optimisation.



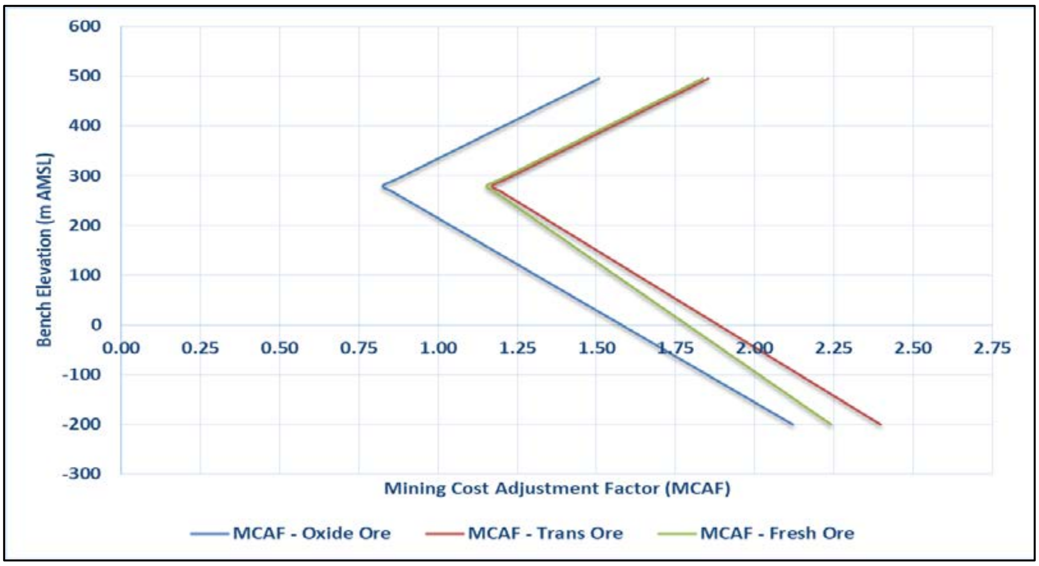


Figure 16-18: Vertical Mining Cost Adjustment Factor for Ore and Waste Materials (source: DRA 2017)

16.4.3.8 Processing Costs

The processing costs are detailed in Section 13. A summary of the processing costs used in the pit optimisation is presented in Table 16-17.

Table 16-17: Processing Costs used in Pit Optimisation (source: DRA 2017)

Process Costs	Units	CIL P5M Plant	CIL P10M Plant
Labour	$/t	0.97	0.14
Laboratory	$/t	0.03	0.03
Consumables	$/t	3.35	3.50
Power	$/t	5.56	5.46
Maintenance	$/t	0.65	0.65
Ore Re-handling	$/t	0.32	0.73
Tailings Disposal and Management	$/t	0.01	0.00
Fixed Cost Portion	$/month	589,875	963,417
Total Processing Costs	$/t	10.89	10.51

16.4.3.9 Processing Recoveries

The processing recoveries are detailed in Section 13. Figure 16-19 below illustrates the metallurgical recovery curves used in the pit optimisation for the Esaase material.

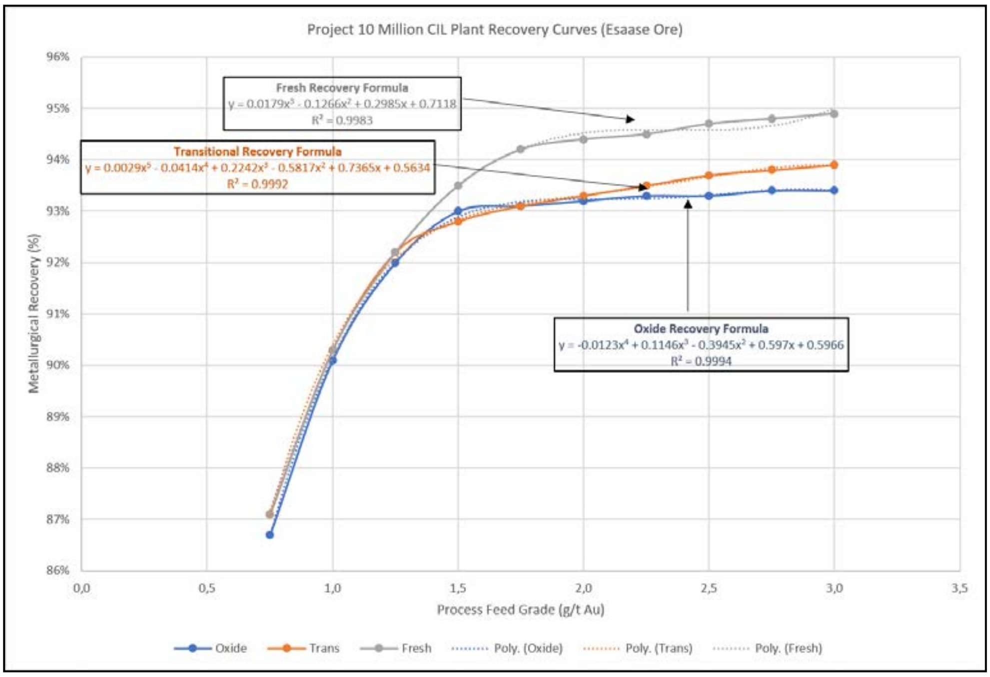


Figure 16-19: Metallurgical Recovery Curves per Rock type (source: DRA 2017

The metallurgical recovery performance of the Esaase ore material in relation to plant feed grade in the CIL processing circuit was formularised by fitting polynomial trend lines to the recovery dataset obtained from the more complex metallurgical recovery models (detailed in Section 13). The dataset used to determine the recovery formula used in the pit optimisation is detailed in Table 16-18 below.

Table 16-18: Metallurgical Recovery Formula for Esaase Ore used in Pit Optimisation (source: DRA 2017)

Process Feed Grade (g/t Au)	Oxide	Trans	Fresh
0.75	86.70%	87.10%	87.10%
1.00	90.10%	90.30%	90.30%
1.25	92.00%	92.20%	92.20%
1.50	93.00%	92.80%	93.50%
1.75	93.10%	93.10%	94.20%
2.00	93.20%	93.30%	94.40%
2.25	93.30%	93.50%	94.50%
2.50	93.30%	93.70%	94.70%
2.75	93.40%	93.80%	94.80%
3.00	93.40%	93.90%	94.90%

Due to the performance of the polynomial formula at higher grades, constant recoveries were applied to higher feed grade as outlined in Table 16-19 below.

Table 16-19: Metallurgical Higher Grade Cut-offs (source: DRA 2017)

	Oxide	Trans	Fresh
Trigger Feed Grade (g/t Au)	2.5	2.5	2.75
Fixed Recovery Applied (%)	93.4	93.8	94.8

16.4.3.10 Processing Method

The Esaase deposit was optimised using ore selection by cash flow method. The following processing parameters were selected in the optimisations:

- Minimum Process Grade ....: 0.6 g/t Au
- Ore Selection Method ....: Cash flow
- Multiple Process Paths ....: Yes

16.4.3.11 On-mine General and Administrative Charge

The on-mine charges consisted of the G&A charges. The synergies of operating the two sites resulting in a saving attributable to the Esaase deposit. The G&A charge is:

- G&A Charge (Obotan) .... : US$ 1,950,000 / month
- G&A Charge (Esaase) : US$ 355,798 / month

16.4.3.12 Off-mine Charges

The off-mine charges consisted of the refining charge.

- Refining Charge: US$ 4.00/oz Au

16.4.3.13 Revenue Parameters

The base case revenue parameters used for the Whittle® optimisation were provided as follows:

- Gold price ....: US$ 1,300/oz Au

16.4.3.14 Discount Rate

The discount rate used in the Whittle® optimisations.

- Discount Rate: 5% (per annum)

16.4.3.15 Royalty Rate

The royalty rate used in the Whittle® optimisations as per Ghanaian Mineral Royalty Legislation:

- Royalty Rate: 5.5% (of gross sales)

16.4.3.16 Constraints

Constraints in the Whittle® optimisation typically consist of mining, processing and element selling limits. The following constraints / limits were used in the various Whittle® optimisations (Table 16-20).

Table 16-20: Esaase Pit Optimisation Constraints / Targets (source: DRA 2017)

Constraint / Target	Units	2019	2020	2021 - 2027	Post 2027
Total Mined Tonnage	Mtpa	12	36	54	54
Ore Tonnage	Mtpa	2.5	6 / 7 Note 3	6 / 7 Note 3	55
Processing Throughput Note 1	Mtpa	2.0	7.0	7.0	8.4 Note 2
Element Limit	Koz / yr	Nil	Nil	Nil	Nil

Note 1 Esaase ore contribution only

Note 2 Overland conveyor belt design limit

Note 3 2020 - 2023 years required only 6 Mtpa of ore to be moved dues to stockpile inventory and management

Two areas were identified surrounding the Esaase deposit that required the optimisation to be constrained due to various surface constraints, namely:

- A low lying area between the South pit and the Southern extension of the main pit. Previous exploration work has proved difficult with low core recovery due to the fact that the ground has been previously worked resulting in “swamp-like” conditions and poor rock conditions
- An area between the North satellite pit and the Northern extension of the main pit. Haul road and community road infrastructure placement required the isolation of this area from the open pit optimisation

Both areas were excluded from the pit optimisation by the application of “heavy-blocks” or excessive mining cost adjustment factors.

16.4.3.17 Optimisation Method

The Milwa® Balanced Algorithm was used in all optimisations. The Milwa® Balanced Algorithm seeks to optimise both the discounted pit value whilst maintaining processing throughputs, as well as adjuring the mining rate, as far as possible to the targeted mining rate / limit.

16.4.3.18 Overall Project Schedule Assumptions (including Obotan)

The Obotan mining schedule is as per the updated mine planning (January 2017) presented for the annual Mineral Reserve update and described in Section 6.2. The Obotan MRev was updated as at 31st December 2016. The Obotan mine production schedule commenced on 1st January 2016. The Obotan CIL plant upgrade is planned to be fully commissioned in Q1 2018, achieving the final nameplate throughput of 5.0 Mtpa. Esaase mining is planned to start Q1 2019 (clear and grub, pre-stripping and contractor site and services establishment are planned six months ahead of feeding ore via overland conveyor belt) contributing approximately 2.0 Mtpa Oxides and Transitional material for a period of 15 months following which the additional 5 Mtpa processing facility will begin ramping up.

The new CIL processing facility (P10M) with a nameplate capacity of 5.0 Mtpa is planned to start wet commissioning in April 2020 ramping up to nameplate capacity by July 2020. The Esaase ore source will provide the 6 Mtpa to 7 Mtpa blended Oxide, Transitional and Fresh material via overland conveyor belt to the P10M CIL processing complex.

16.4.3.19 Optimisation Results

The selected option from the AGM 2017 DFS study is inclusive of Measured and Indicated material only, the purpose of this optimisation is to provide a key mechanism in the process of converting resource to reserve.

The pit-shell 37 (revenue factor 0.924 pit-shell) was chosen as the estimated peak discounted pit value.

Pit-shell 37 formed the basis of the mine design, production schedule and financial analysis.

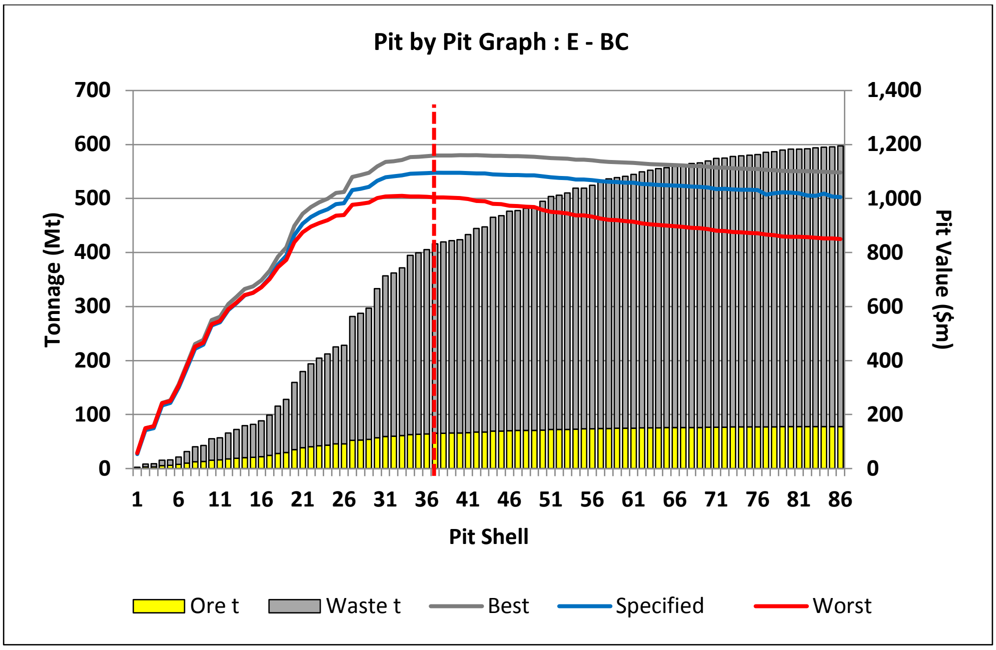


Figure 16-20: Whittle Pit-by-Pit Graph for Measured, Indicated Resources only (source: Asanko Gold 2017)

The above pit by pit analysis graph (Figure 16-20) shows an optimal pit value at pit-shell 37. Pit shell 29 through to pit shell 70 demonstrate that the indicated discounted pit value changes within ± 5% of the peak pit value indicating the Esaase pit value appears relatively robust for a range of ultimate final shells (297 Mt to 570 Mt [equivalent to a US$1,950/oz optimisation price]). This indicates a low risk to final wall commitment as a positive move in gold price would not significantly influence pit size. A summary of the statistics associated with the selected pit (pit-shell 37) is listed below.

Table 16-21: Results for M&I inclusive Whittle Optimal Scenario (source: Asanko Gold / DRA 2017)

Parameter	Units	Optimal Pit-shell	RF 1 Pit-shell
Ore Tonnes Mined	Mt	65.1	67.5
Waste Tonnes Mined	Mt	351.2	377.0
Total Tonnes Mined	Mt	416.3	444.5
Life Of Mine	Years	~11	~11
Overall Stripping Ratio	Wt : Ot	5.4	5.6
Au Metal Mined	Moz Au	3.07	3.17
Average Au Grade	g/t Au	1.47	1.46

The following pit size sensitivities were conducted to determine the robustness of the Esaase pit to changes in economic and technical conditions:

- 2017 Expansion DFS Base Case – Gold Price $1,300/oz
- Gold Price $1,000/oz
- Gold Price $1,100/oz
- Gold Price $1,200/oz
- Gold Price $1,400/oz
- Gold Price $1,500/oz
- Mining Cost -15%
- Mining Cost +15%
- Applied Processing Cost -15%
- Applied Processing Cost +15%
- Met Recovery -2.5%
- Met Recovery +2.5%
- Pit Slopes -2.5%
- Pit Slopes +2.5%
- DFS 2017 Base Case – Including Inferred Material

A summary of the pit size sensitivity analysis completed in the pit optimisation is presented in Table 16-22 and illustrated in Figure 16-21.

Table 16-22: Pit Size Sensitivities for the Esaase Deposit (source: DRA 2017)

Mining Costs	Disc $m	Ore (kt)	Waste (kt)	Gold Grade (g/t)	Gold to Plant (Au koz's)	LoM (Qtrs)	Strip Ratio (Wt:Ot)
DFS 2017 Base Case – Gold Price $1,300/oz	1,096	65,107	351,214	1.47	3,069	46.0	5.4
Gold Price $1,000/oz	524	45,036	186,515	1.50	2,176	32.0	4.1
Gold Price $1,100/oz	695	53,963	243,375	1.47	2,548	38.0	4.5
Gold Price $1,200/oz	894	63,608	335,748	1.47	3,001	45.0	5.3
Gold Price $1,400/oz	1,299	66,148	360,639	1.46	3,110	48.0	5.5
Gold Price $1,500/oz	1,507	69,836	401,967	1.45	3,266	51.0	5.8
Mining Cost -15%	1,238	69,101	395,725	1.46	3,241	48.0	5.7
Mining Cost +15%	963	60,649	305,349	1.47	2,862	43.0	5.0
Processing Cost -15%	1,165	65,847	357,100	1.46	3,096	47.0	5.4
Processing Cost +15%	1,014	65,098	351,692	1.47	3,069	45.0	5.4
Met Recovery -2.5%	1,025	65,484	354,261	1.46	3,083	45.0	5.4
Met Recovery +2.5%	1,166	65,770	356,724	1.46	3,094	46.0	5.4
Pit Slopes -2.5%	1,045	63,922	360,927	1.46	3,003	46.0	5.6
Pit Slopes +2.5%	1,149	66,274	337,431	1.47	3,127	45.0	5.1
Base Case – Incl. Inferred	1,154	68,927	384,263	1.49	3,291	49.0	5.6

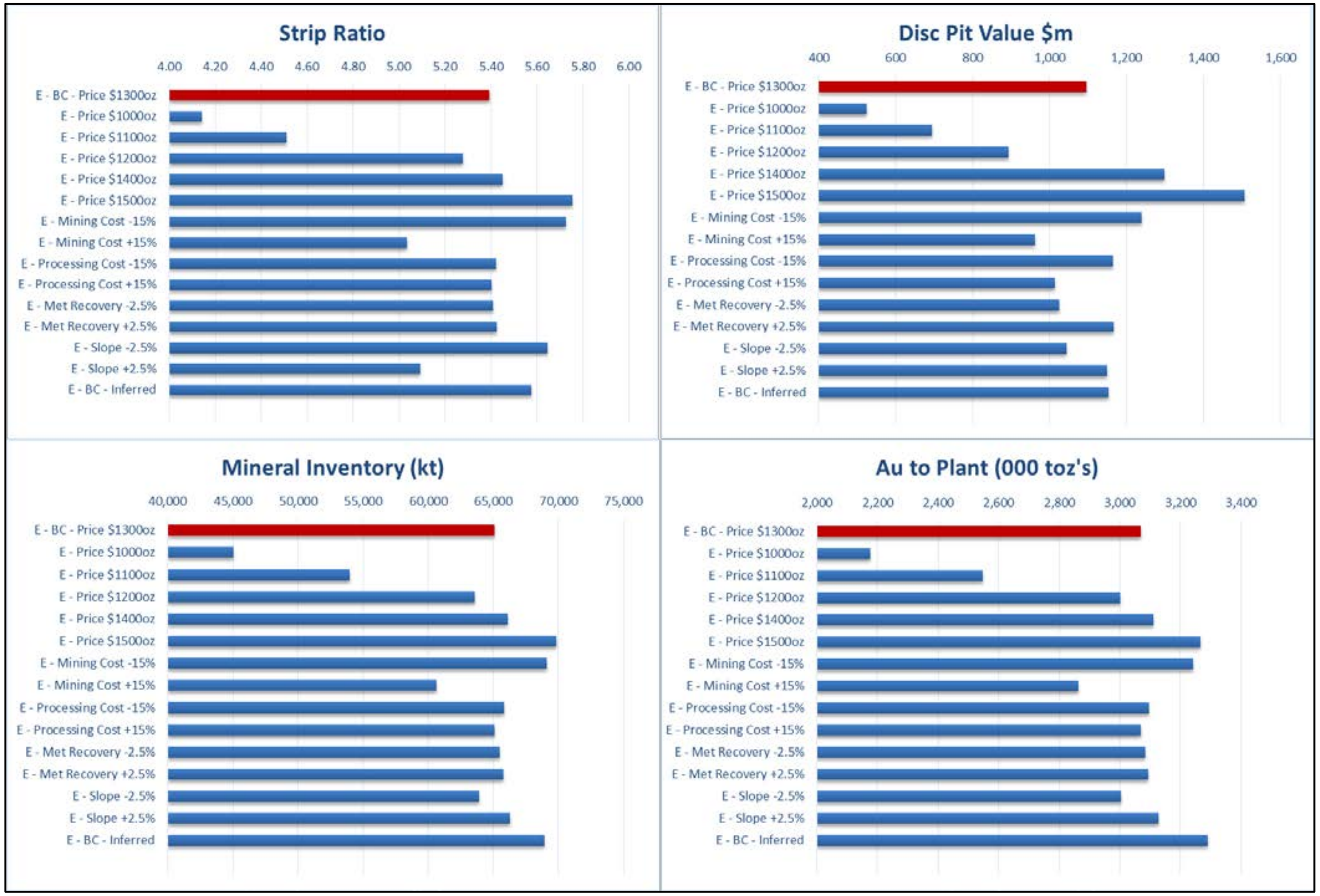


Figure 16-21: Pit Size Sensitivity Analysis (source: DRA 2017)

16.4.4 Esaase Mine Design

16.4.4.1 Pit Design Parameters

Pit design parameters are in keeping with established mining practice and are described in the. following sections. The Esaase pit design is based on Measured and Indicated material only.

16.4.4.2 Geotechnical Considerations

In March 2016 M1 was contracted by Asanko Gold to review the Geotechnical Recommendation Report at FS level completed by M1 in November 2013 and to align the bench height and configuration to the Nkran design.

The alignment would allow for similar equipment and Grade Control procedures to be implemented from the operational Nkran open pit. M1 proposed new slope configurations for the NW and SE Esaase pit slopes based on this requirement and utilising past data from studies completed by Mining One previously.

The slope recommendations were provided based on the weathering codes within the block model (i.e. Oxide / Transition / Fresh). Two main sets of slope parameters were provided for the Esaase deposit. One slope set for the North Western wall and another for the South Eastern wall. The design parameters are detailed in (Table 16-23) below.

Table 16-23: Esaase Slope Design Parameters (source: M1 2016)

Esaase Mining One Slope Recommendations						
Parameter	Oxide		Transition		Fresh	
	NW Wall	SE Wall	NW Wall	SE Wall	NW Wall	SE Wall
Flitch Height (m)	6	6	6	6	6	6
Number of Flitches	4	4	4	4	4	4
Bench Height (m)	6	6	12	12	24	24
Batter Angle °	80	80	70	80	70	80
Berm Width (m)	6	6	7	7	8.5	9
Inter-Ramp Angle (IRA)	40.4	40.4	46.6	52.8	50.0	56.3
Geotechnical Berm	-	-	-	-	20	20
Stack Height (m)	24	24	24	24	96	96

The pit slope recommendations outlined in the table above were modified to provide for practical design and operational requirements based on equipment selection, Grade Control and blast design. Slope angles were not modified and the detailed design configuration was approved by M1 geotechnical consultants. The modified configuration considered the mining of 24m benches in 6m flitches and the capability of the drilling rig to achieve sub-vertical holes.

Additional pit wall support has been included in the mining costs that allow for 10% of exposed wall area to be drilled and cable anchored and the installation of drapes and barriers above each main in-pit haul road.

16.4.4.3 Haul Road Design

The ramp width of an in-pit haul road should be at least 3.5 to 4 times the width of the selected haul truck. From the equipment selection process and contractor submissions, the CAT 777D (or equivalent) is considered suitable for the expected mining volumes.

The pit exits for the various mining areas are positioned as a trade-off between proximity to waste dumps, ROM Pad, surface infrastructure, terrain at exit and minimal hauling distances.

Table 16-24: Pit Design Parameters (source: DRA 2017)

Pit Design		Parameter
Haul Road Design	Width – Dual lane – Single lane	25 m 17 m
	Gradient Minimum radius of turning circle Switchback Gradient Cross-fall Gradient	10% 27 m 0% 2%
Working Widths	Minimum pit base width Minimum cutback width	30 m 50 m

16.4.4.4 Optimal Pit vs Detailed Pit Design

In order to determine whether the final pit design is sufficiently similar in shape, size and position, to the original optimised pit, the potential ore and waste contents within the design are measured, and compared to the relevant Whittle® optimisation results. As Whittle® employs a range of modifying factors in calculating the final output results, a similar process is followed when estimating the comparative figures from the pit design evaluation. A summary of this calculation and subsequent comparison is outlined below.

Table 16-25: Whittle Selected Pit vs. Pit Design Comparison (source: DRA 2017)

	Ore (Mt)	Waste (Mt)[Note 1]	Total (Mt)	Strip Ratio	Ore Grade (g/t Au)	Metal (Moz Au)
Whittle® Selected Pit (Pit 37)	65.1	351.2	416.3	5.4	1.47	3.07
Detailed Design	62.6	348.6	411.2	5.6	1.46	2.94
Variance % (Whittle vs Design)	-3.9%	-0.7%	-1.2%	3.3%	-0.3%	-4.2%

Note [1] Includes very low grade reservable ore (~10Mt at 0.55 g/t Au) and mineralised waste (<0.5 g/t Au)

Variances between the Whittle® pit and the final pit design will invariably occur due to the application of design factors such as the ramp design parameters, as well as the detailed slope design. In the design, actual batter angles and berm widths are used, as opposed to the overall slope angle, (inclusive of a ramp system) as indicated in the pit optimisation results.

16.4.5 Proposed Mining Operation

16.4.5.1 Mining Method

Mining by conventional open pit methods of drill and blast followed by load and haul will be employed. Drilling and blasting will be performed on 24m benches. Loading of the blasted material will be performed on four 6m flitches.

The envisaged scale of mining at the Esaase project is medium in size with a peak total material movement of 52 Mtpa. The annual mill feed requirement is approximately 2 Mtpa (2019) ramping up to approximately 6 Mtpa (year 2020 – 2023) and finally 7 Mtpa to 8 Mtpa (2024 onward) until end of LoM.

16.4.5.2 Load and Haul Equipment

The mining fleet will consist of CAT 6030 (or equivalent) hydraulic excavators (300 tonne class) with bucket capacity of 17m^3 in backhoe configuration and 16.5m^3 in face shovel configuration, and CAT 6015, (or equivalent) hydraulic excavator (100 tonne class) with bucket capacity of 6m^3 in backhoe configuration. CAT 777D off highway rigid frame dump trucks with ~94t capacity will form the primary hauling fleet.

The primary mining fleet of trucks and excavators will be supported by standard open-cut drilling and auxiliary equipment.

Waste material will be hauled to the two allocated waste rock dump positions to the south-west and south of the pit. Backfilling of depleted pit areas will commence in year 8 (2026) of the mine production schedule.

16.4.5.3 Drill and Blast

Rock fragmentation will be accomplished through drilling and blasting. Geotechnical test work has confirmed that the weathered material will not require blasting, and will be excavated using “free dig” equipment, or ripped with a CAT D9 dozer, (or equivalent). An allowance of 15% of the weathered material will be drilled and blasted. Exploration drilling has indicated an oxidised zone with an average depth of 24m to 40m below surface.

All transitional as well as Fresh competent material will be drilled and blasted.

The drill and blast activity will be supported with Sandvik DP1500, (or equivalent) drill rigs capable of drilling 102 mm to 152 mm vertical and inclined holes. The rigs will be supported by a stemming TLB and explosive delivery vehicle and several special purpose LDV’s carrying personnel and explosive accessories.

Table 16-26 summarises the estimated blast design for the Esaase ore.

Table 16-26: Ore Blast Design (source: DRA 2017)

Ore Blasting Parameters	Units	Oxide (15%)	Transitional	Fresh/Competent
Bench Height	(m)	6	6	6
Hole Diameter	(mm)	127	127	127
Powder Factor	(kg/bcm)	0.5	0.6	0.8
Spacing	(m)	5.0	4.5	4.0
Burden	(m)	5.6	4.8	3.5
Stem	(m)	2.5	2.5	2.5
Sub-Drill	(m)	1.87	1.60	1.17
Explosive per hole	(kg)	77.6	73.7	67.4
P80 Particle Size	mm	553	579	514

Table 16-27 summarises the estimated blast design for the Esaase waste.

Table 16-27: Waste Blast Design (source: DRA 2017)

Waste Blasting Parameters	Units	Oxide (15%)	Transitional	Fresh/Competent
Bench Height	(m)	6	6	6
Hole Diameter	(mm)	127	127	127
Powder Factor	(kg/bcm)	0.4	0.5	0.6
Spacing	(m)	6.2	5.0	4.5
Burden	(m)	6.1	5.0	4.5
Stem	(m)	2.5	2.5	2.5
Sub-Drill	(m)	2.03	1.67	1.50
Explosive per hole	(kg)	80.0	74.7	72.3
P80 Particle Size	mm	676	638	663

The pit configuration bench height and waste material type anticipated at the project suit drill rigs capable of drilling drill holes with a diameter of 127 mm. Drill burden, spacing and sub-drill design will be functions of the varying material types of the deposit.

An emulsion based product with water resistant characteristics and a higher velocity of detonation is recommended to achieve a better fragmentation.

The blast pattern is dictated by the powder factor required to ensure appropriate fragmentation and heave. The selection of the powder factor is based on the Unconfined Compressive Strength ("UCS") measurement results obtained from the preliminary excavation characterisation work. For weathered material the UCS range is between 8 MPa and 12 MPa, which suggests a very weak rock. For Fresh material the UCS range is between 28 MPa and 80 MPa, which suggests a weak to moderately strong rock.

As part of the geotechnical optimisation of the pit, it was recommended that pre- split blasting will be required for the complete final wall position. The pre-split cost has been included in the operating cost. The pre-split holes will be drilled at a spacing of 1.2m (6m deep) with a hole diameter of 102 mm and a 15m buffer trim shot will blasted in conjunction with the pre-split final wall. The pre-split blasting will achieve two goals, reduction of ground vibration for compliance to Ghana regulations regarding surrounding villages, and protection of the high wall condition.

16.4.5.4 Pit Support Equipment

Pit support equipment for the Esaase operation will consists of dozers, graders, fuel bowsers, water bowsers, hydraulic hammer, TLB and wheel loaders. The function of this equipment will be support the primary mining equipment by the maintenance of pit floor and haul road conditions, provide clean-up around the excavators to prevent excessive tire damage, secondary breakage of oversize rocks and to water-down road surfaces to supress dust.

16.4.5.5 Rehandle Equipment

Stockpiles will be required at Esaase to manage the feed requirements to the primary crusher, secondary crusher, overland conveyor belt and finally process front-end ore handling systems. The configuration and capacity of the ore handling system will allow for direct blended ore feed, or alternatively switch-over between Oxide / Transitional and Fresh material as and when required.

The proposed management will consist of two buffer stockpiles (Oxide/Transitional and Fresh) located in close proximity to the primary crusher. It is estimated that 70% of the plant feed tonnage will be able to direct tipped from a pit source and the remaining 30% will require re-handle with a front end loader.

A longer term strategic stockpile located approximately 750m from the ROM Pad will be maintained to balance the mining schedule with the plant feed schedule.

Re-handle equipment for the operation will consists of CAT 992, (or equivalent) front end loaders and CAT 777D, (or equivalent) haul trucks when re-handling from the strategic stockpile.

16.4.5.6 Ancillary Equipment

Ancillary equipment for the operation will consists of service trucks, tyre handlers, mobile crane, water pumps, lighting plants, TLB, LDV's and wheel loaders. The function of this equipment will be support the pit equipment and maintenance workshops.

16.4.5.7 Mobile Primary Crusher

During the site establishment, clear and grub, construction and for the first 12 to 15 months of operation, during which the ROM Pad, primary and secondary crusher installations are being completed, the Oxide and Transitional ore mined from the pit will require crushing for transport via the commissioned overland conveyor belt. A mobile primary crushing plant will be used for the primary sizing onto the overland conveyor belt. Once the main primary crushing and secondary crushing facilities have been commissioned the mobile crusher will no longer be required.

16.4.5.8 Mining Equipment Summary

Mine equipment is summarised in Table 16-28 below. The expansion equipment in the table below is the result of the increased mining rate starting in Q2 2020.

Table 16-28: Mining Equipment Summary (source: DRA 2017)

Area	2019 (10 Mtpa)	2020 (35 Mtpa)	Steady State (50 Mtpa)
Pit Load and Haul (Primary Fleet)			
CAT 6030	1	4	5
CAT 777D	7	21	36
CAT 6015	1	1	1
Pit Load and Haul (Pit Support Fleet)			
CAT D9 Dozer	2	4	6
CAT 770 35 kl Water Bowser	1	2	2
CAT 16H Motor Grader	1	2	2
CAT 992 FEL	1	2	2
Drilling Fleet			
Sandvik 1500 Pantera	4	8	12
TLB (Stemming)	1	2	2
Support LDV (Supervision and Accessories)	1	2	2
Stockpile Ore to Crusher			
CAT 992 FEL	1	1	1
CAT 777D	0	0	0
Waste Dump Profiling			
CAT D9 Dozer	0	1	2
Ancillary Fleet			
30t Excavator with Hammer	1	1	1
35 kl Fuel Bowser (ADT)	1	2	2
Service Truck	1	1	1
Lighting Plant	3	5	6
Water Pumps	1	2	3
Compactor	0	0	0

Area	2019 (10 Mtpa)	2020 (35 Mtpa)	Steady State (50 Mtpa)
Mobile Site Crane	1	1	1
Tyre Handler (Workshop)	1	1	1
Wheel Loader (General Site Assistant)	1	1	1
TLB 428F - Grubbing (General Site Work)	1	1	1
LDV	4	5	6
Total Equipment	36	70	96

16.4.5.9 Grade Control

Grade Control drilling and sampling at the Esaase deposit will be required to determine whether material in a given area is above, or below the cut-off grade. The ore material is not anticipated to be visually controlled in the active mining face. Grade Control definition drilling and sampling will be required to delineate the ore zone prior to the planning of the ore blast blocks.

Grade Control drilling will be conducted by a dedicated reverse circulation drill that will drill a 5m by 5m pattern to a depth of 18m. Sampling will be conducted at a 1m intervals using composited drill chippings along the full length of the hole. It is estimated that 18 holes and 320 samples per day will be assayed at peak ore production of 7 Mtpa. This will define zones above cut-off and the extra drilling depths will define three 6m flitches of Grade Control in a single pass of the drill. Marginally sub economic areas of mineralisation will also be drilled and sampled in order to identify any pockets of potential ore above cut-off grade.

The technical services team will be responsible for collecting the assay data from definition drilling and interpreting the results to define the economic ore zones. They will communicate the ore zones to the mine planning engineer and drill and blast supervisor for inclusion into the short term mine plan. Finally, they will indicate those zones in the field and provide direction to the to the mine operations crew during excavation.

16.4.5.10 Pit Dewatering and Drainage

In-pit water management will primarily consist of run-off control and sumps. The dewatering infrastructure and equipment is sized to handle ground water inflows and precipitation. The pit dewatering plan is based on diverting as much surface water as possible away from the open pits, then collecting the water that does report to the open pits, using ditches and sumps before pumping it the Mine Water Pond. There will be intermediate sumps on the pits walls as well as on the surface between the pit and the Mine Water Pond.

As the LoM pit will be operating at depths greater than 200m below crest, specialist high lift pumps will be required. Pontoon mounted pumps will be used to draw from sumps. This will ensure the pumps are not submerged as sump water levels rise rapidly in response to a rainfall event. Pumping infrastructure will advance with the active mining as it advances deeper.

The key operational requirements will be to:

- Minimise water flows into the pit using perimeter bunds, drains and fill, where practicable
- Provide pit pumping capacity for foreseeable extreme events
- Maintain pit wall drainage
- Provide permanent and temporary sumps capable of handling the expected water inflows
- Install settling ponds for the removal of solids prior to discharge off-site

16.4.5.11 Open Pit Work Roster

The mining operations are scheduled to work 365 days in a year, less unscheduled delays such as high rainfall events which may cause mining operations to be temporarily suspended. Table 16-29 outlines the proposed work roster for the Esaase operation.

Table 16-29: Asanko Gold Roster System (source: DRA 2017)

Expats	6 weeks on site, 2 weeks off	6-2
Admin Staff	5 day week (45 hours)	5-2
Shift Workers	12 hr shift roster, 1 swing shift	2 x 12
Engineering	11 day fortnight (i.e. 5 x 10 hr days per week, 5 hr every alternate Saturday and Sunday)	11 df

A contract mining approach will be adopted.

The estimated incremental Asanko Gold staff required to ensure efficient and sustainable service provision to the Esaase mining operation is outlined in Table 16-30 below.

Table 16-30: Asanko Labour Numbers for P10M (source: Asanko Gold / DRA 2017)

Labour	P5M	P10M (Incremental)	P10M (Total)
Management	1	0	1
Supervision	8	0	8
Mining	5	0	5
Engineering	10	0	10
Administration	2	0	2
Technical Services	0	0	0
Process	22	43	65
Security	22	0	22
Other	0	0	0
Total Workforce	**70**	**43**	**113**

The estimated mining contractor staff required to ensure delivery of the Esaase production mining plan is outlined in Table 16-31 below.

Table 16-31: Estimated Mining Contractor Labour Requirements (source: Asanko Gold / DRA 2017)

Labour	2019 (10 Mtpa)	2020 (35 Mtpa)	Steady State (50 Mtpa)
Management	3	3	3
Supervision	6	16	16
Operators	108	219	306
Maintenance	29	137	137
Administration	9	15	15
Technical Services	1	2	2
Others	-	-	-
Total Workforce	**156**	**392**	**479**
Total (Expatriates)	**6**	**13**	**13**

16.4.5.12 Human Settlement Considerations

A significant challenge in terms of the proposed development of the Esaase and the associated overland conveyor is the vicinity of human settlements in the form of the Tetrem, Esaase and Aboabo villages. Both the size (number of inhabitants) and proximity of the settlements to mining operations will constrain mining operations. The surrounding villages are home to approximately 25,000 inhabitants, although only a small proportion of inhabitants will be directly impacted and require resettlement.

Some significant considerations around settlement proximity and relocation are:

- As the planned final pit pushback perimeter (pit rim) extends over the 500m blasting radius line[1] of both the Esaase, Manhyia and Tetrem villages, the stated intention is to relocate portions[2] of these settlements in order to optimise the mining reserve. Preliminary financial investigations have confirmed that the value of the underlying material justifies the initial relocation expense
- In order to optimise the project NPV, the intention is to commence mining operations before the relocation process is complete. Therefore, while the relocation is in progress, production will be constrained to the areas inside the constrained respective blasting radii.
- As the targeted WRD positions also overlap with the perimeter of the Tetrem village, waste rock will be hauled to an alternative WRD[3] until the village is completely relocated

Note [1] Measured horizontally from Pit Rim Daylight. 500m blast radius not projectile arc distance equivalent.

Note [2] Total relocation is planned for the Tetrem village settlement, amounting to approximately 250 structures.

Note [3] WRD 1a & 1b lies outside the Tetrem 500m blast radii. WRD 2a is located a minimum 300m from Tetrem village.

Please refer to section 18.11 on resettlement for more information on the proposed relocation of the impacted villages.

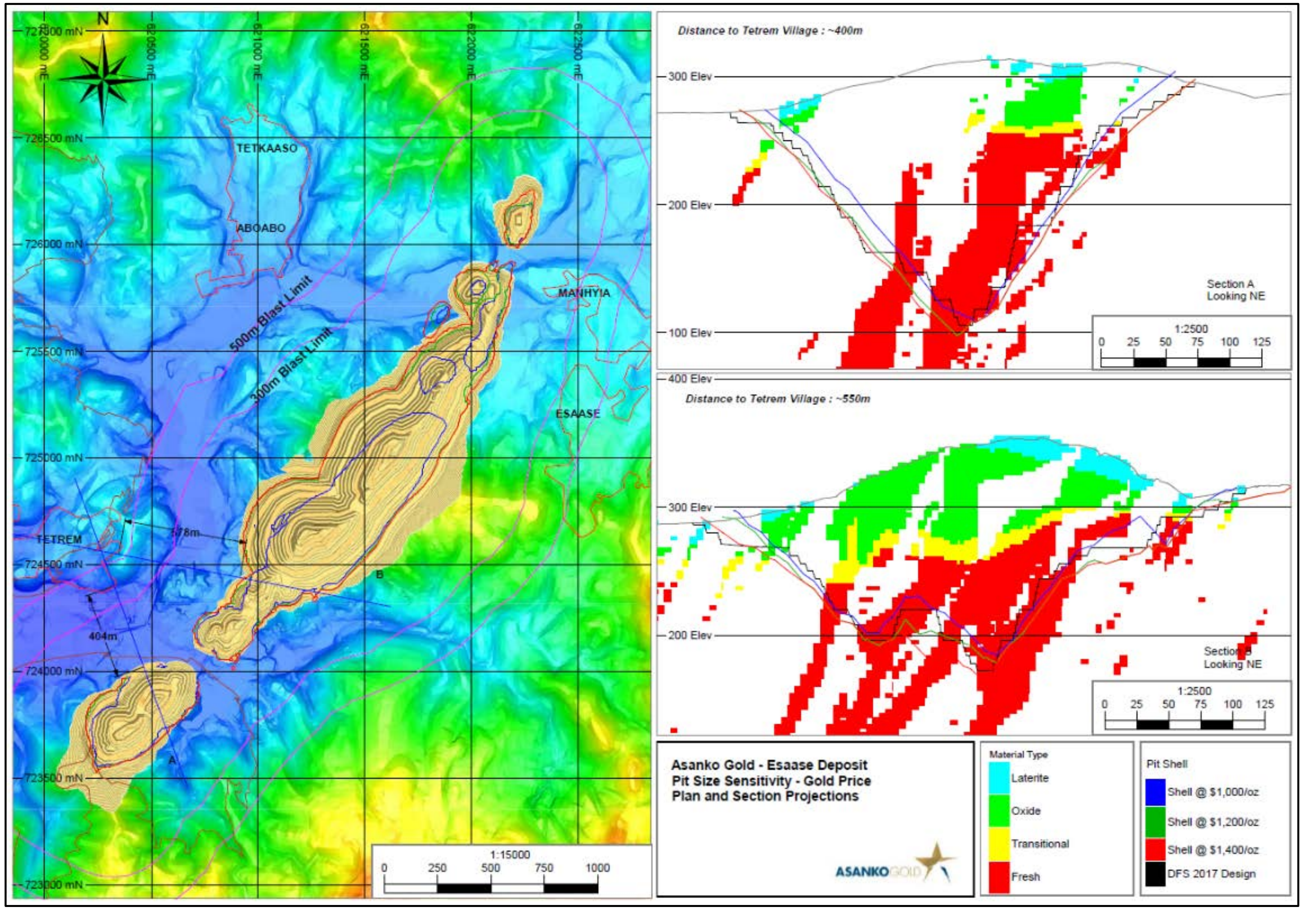


Figure 16-22: Blast Radii in relation to Tetrem Settlement (source: DRA 2017)

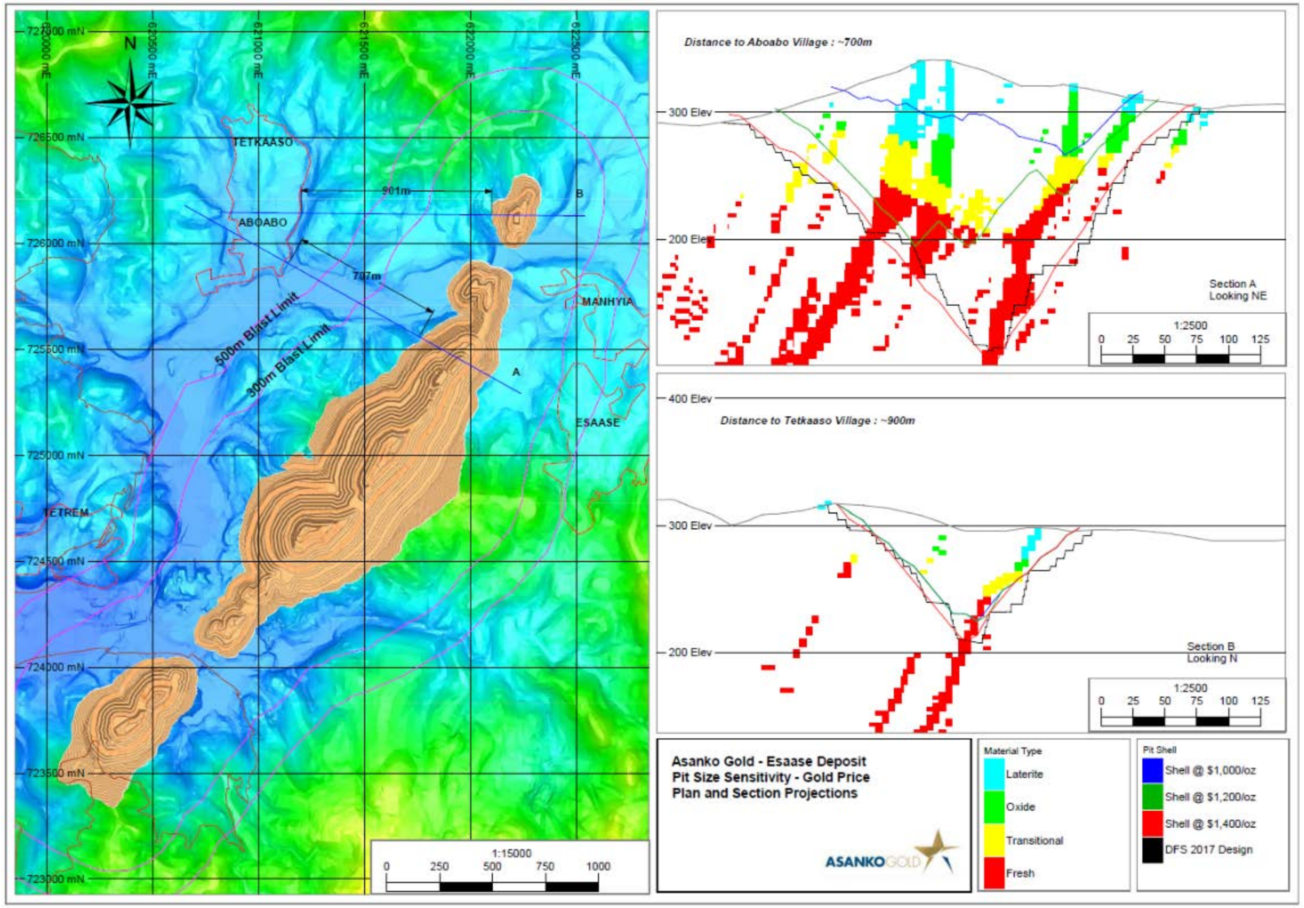


Figure 16-23: Blast Radii in relation to Aboabo / Tetkaaso Settlement (source: DRA 2017)

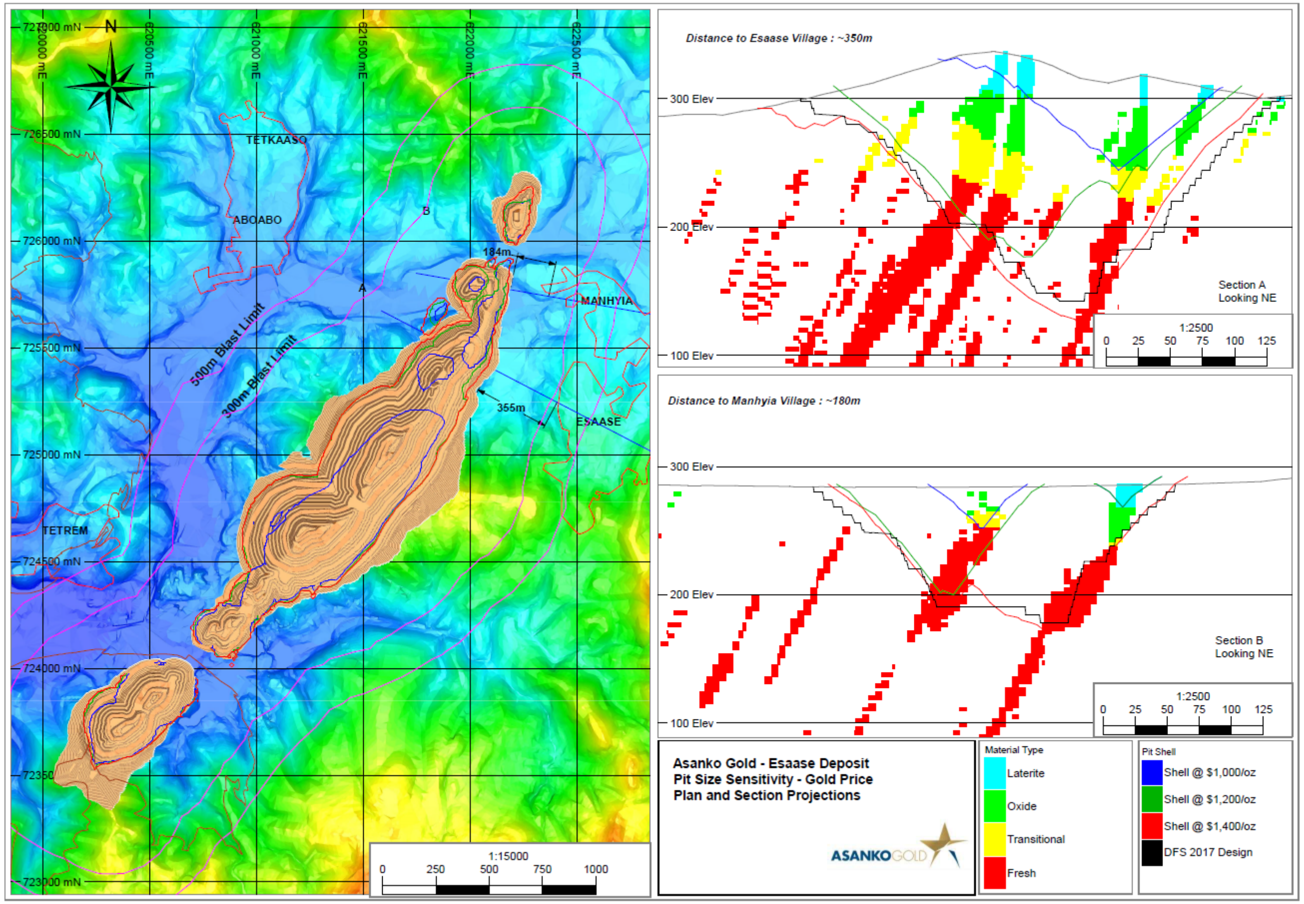


Figure 16-24: Blast Radii in relation to Esaase / Manhyia Settlement (source: DRA 2017)

16.4.5.13 Waste Rock Dumps

WRD Positions and Layout

The waste rock dumps associated with mining operations will be constructed to meet the requirements of the Ghanaian Mining Regulations and AKOBEN guidelines. They will initially to be constructed with the natural rill angle of approximately 37° degrees, which is the angle of repose of the dumped material. This is then to be contoured progressively to an overall slope angle of 18.5° (1:3) to allow for slope stability and re-vegetation. The waste dump will be progressed by tipping from a higher level against a windrow and progressively pushing the waste out with a dozer. Figure 16-25 illustrates the WRD design criteria used to design the Esaase waste rock dumps.

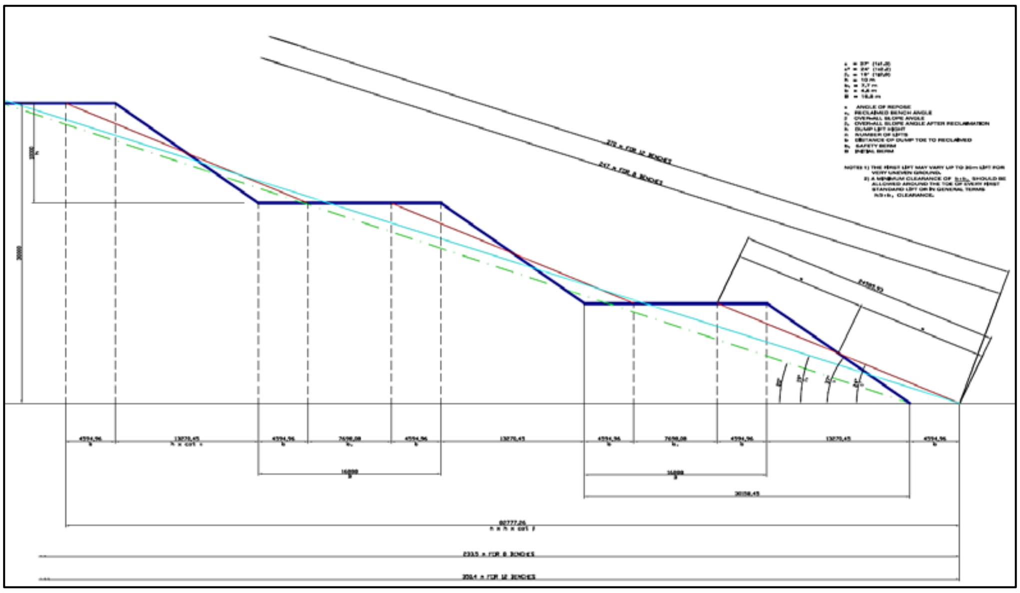


Figure 16-25: Esaase WRD Slope Design Criteria (source: DRA 2017)

Waste dumps will be progressively rehabilitated with topsoil, where possible. Surfaces of dumps will be contoured to minimise batter scour and ripped at 1.5m centres to a depth of 400 mm, where practicable. All such rehabilitation work will be carried out progressively. Seepage and shallow ground water flow along the perimeter of the mine residue deposits should be controlled with suitable toe drains.

Selected waste rock will also be used for the construction of the ROM pad, TSF walls and other infrastructure items during the site construction phase and for further TSF wall lifts during the LoM.

Good mining practice dictates that mine sites be rehabilitated to a sustainable state after the mining operations reach completion. Hence, all areas impacted by the project will be stripped of topsoil before commencement of construction. This topsoil will be stockpiled for future rehabilitation work at the end of the mine production life.

There are a number of factors that need to be considered in evaluating the best option for a waste storage area. They include:

- Storage capacity
- Stored material
- Visual impact
- Haulage distance costs
- Site drainage
- Site access and preparation

As result of the mountainous topography surrounding the Esaase project area, the WRD layouts are have been planned against the slope of mountains[1] on the South and South-West of the planned pit position. Neither of the WRD's extend higher than the peaks of their respective neighbouring topographic structures.

Note [1] 100% Valley Fill Method (unless partial between adjacent topographic peaks).

In planning the WRD positions, special cognisance was taken of existing natural water courses. The proposed dump footprints were selected carefully not to create dams, or otherwise constrict water flow in the naturally undulating environment.

As illustrated in Figure 16-26 two main areas have been allocated for Esaase waste rock storage:

- WRD 1 – backfilled south pit and final south WRD
- WRD 2 – located directly south-west of the main pit
- Selective backfilling of north pit and northern extension of the main pit
- Optional selective backfilling of south extension of the main pit

Figure 16-27 illustrates the interim waste rock dump positions (indicated in magenta) that will be used during the first two years of mining operations in whilst the Tetrem resettlement village is being constructed. The pit backfilling (indicated in red) will commence from year 8 (2026) of production once the South pit has been completely depleted and will continue until the end of mining operations.

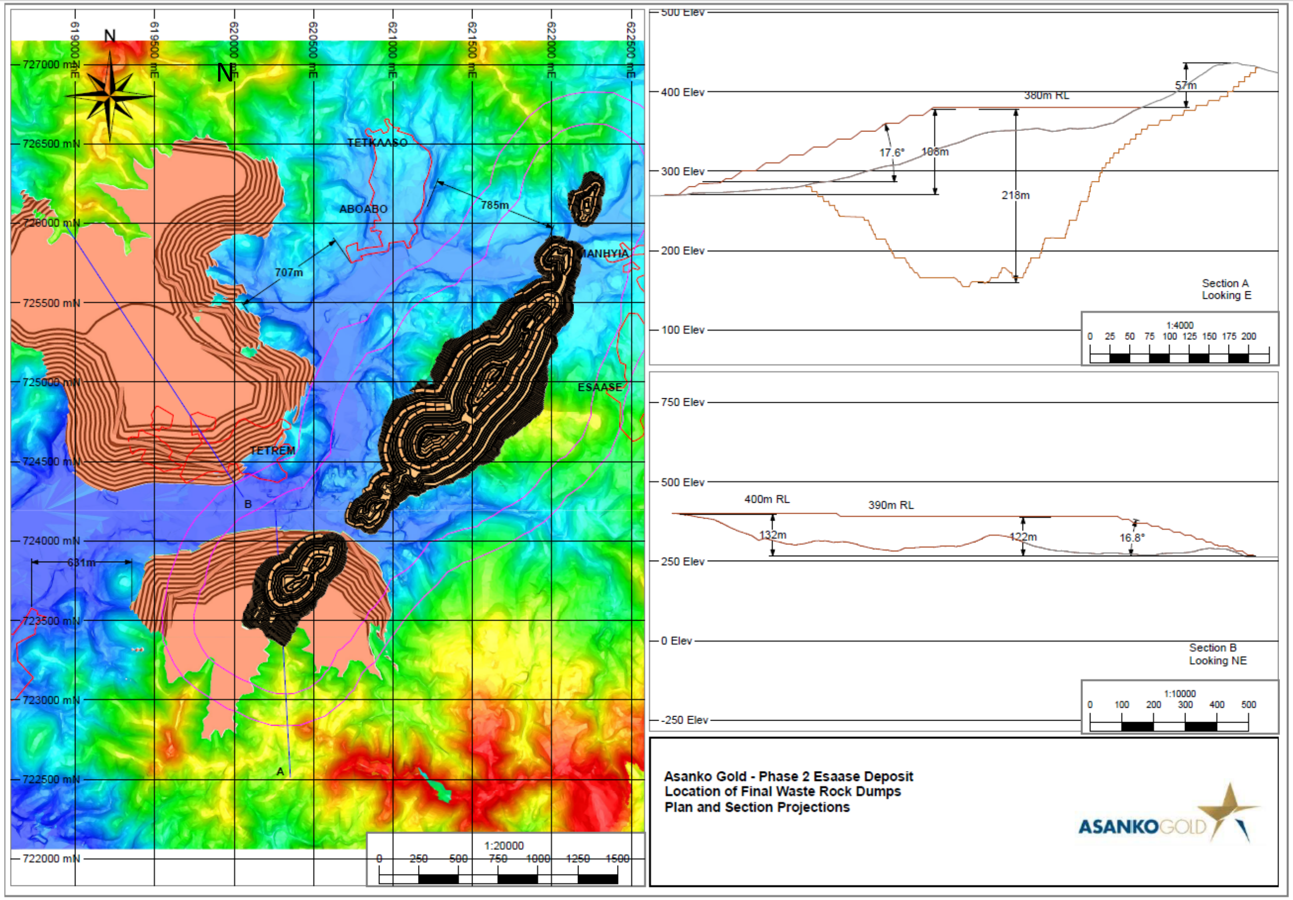


Figure 16-26: Final Esaase Pit and WRD Layout (source: DRA 2017)

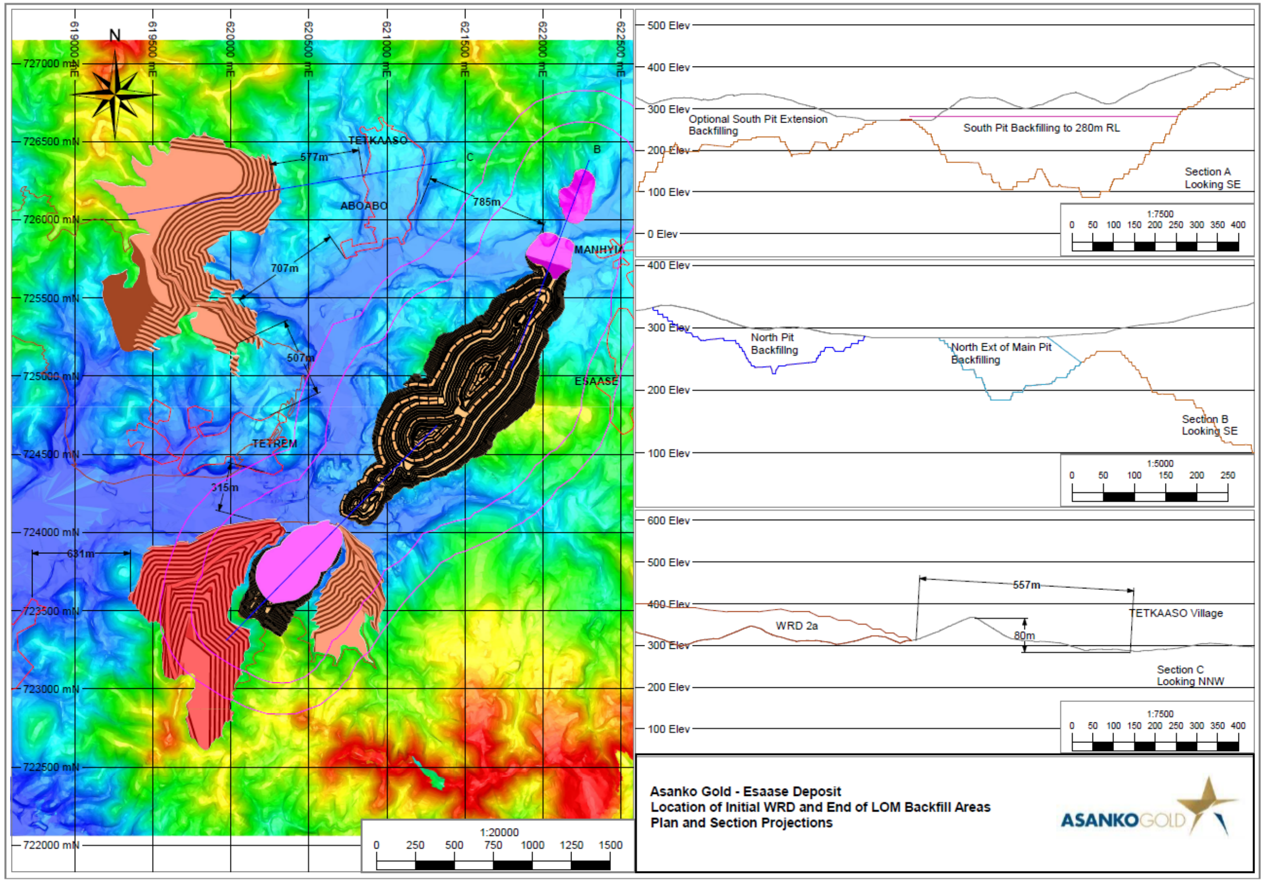


Figure 16-27: Initial WRD Positions and End of LoM Backfilling Areas (source: DRA 2017)

16.4.5.14 Storage Capacity

For the purpose of optimising the waste hauling costs, the two main Esaase WRDs (WRD1 and WRD2) were split into three stages each.

After running various optimisations, WRD1 (located to the South) was designed to a maximum RL of 380m and WRD2 (located to the South-West) to a maximum 400m RL.

The partial backfilling of the Northern extension of the main pit and the full backfilling of the North pit assumes that the original ground RL will be restored.

16.4.5.15 Waste Storage Schedule and Constraints

In planning the waste haulage routes and waste hauling schedule, the first priority is to minimise early waste haulage costs as a means of further optimising the project NPV. The intention is to fill WRD1a and WRD1b to capacity during the resettlement of the Tetrem village as it falls within the footprint of WRD2c. Waste storage at WRD2c is unavoidably delayed until the relocation of the entire Tetrem village is complete. Once the village has been completely relocated, waste hauling to WRD2c site will be allowed to commence.

In the case of WRD1c, waste movement logistics are further complicated by the position of WRD1c over the south zone of the main pit. Hence, the south zone will only be backfilled once it has been mined out completely. WRD positions have been prioritised on the basis of haulage distances and time constraints associated with target area availability.

16.4.5.16 Esaase Arsenic Rich Waste Rock Management

To minimise the leaching of arsenic from the WRDs, it will be essential that high arsenic waste, (which is defined as waste rock containing more than 400 mg/kg) and especially high arsenic Fresh waste, is not unduly exposed to atmospheric conditions on active dump heads. Instead, such material will be identified prior to mining and selectively handled in a manner that allows covering by low arsenic material and/or typical oxide waste in a timely manner. This will involve segregation and selective placement of high arsenic rock within small cells that become encapsulated within the core of the dump and entirely surrounded by a much larger mass of low arsenic or typical oxide material. It is estimated that around 7% of total waste rock will have an arsenic content that is greater than 400 mg/kg.

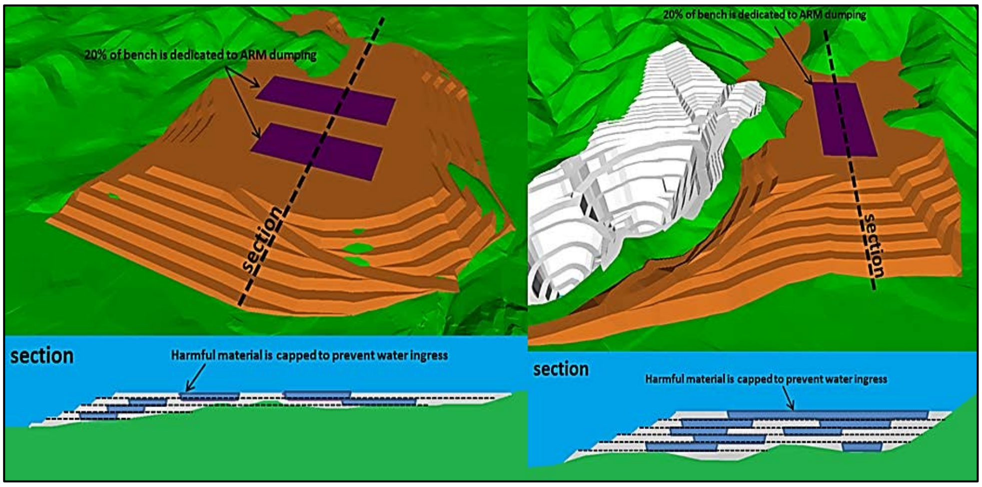


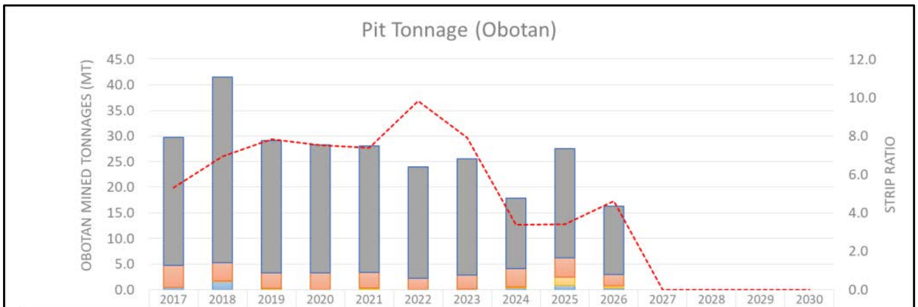
Figure 16-28: Management of Arsenic Rich Waste in WRD's (source: DRA 2017)

The cells for containing high arsenic material will be constructed in a manner that results in a low permeability within the cell and the overlying oxide cover. This will ensure a high level of control on water flux through high arsenic material. Furthermore, as most arsenic rich material will naturally occur in Fresh rock as arsenopyrite, a high degree of compaction will also limit diffusive movement of oxygen into cells and thereby limit the potential for oxidation of arsenopyrite (as well as associated pyrite). The low permeability cells will be constructed by paddock dumping of oxide waste (i.e. dumped as a single layer on a level surface), as this will allow the rock to be levelled and compacted in small lifts.

16.5 P10M Production Schedule Summary

16.5.1 Mine Production Schedules

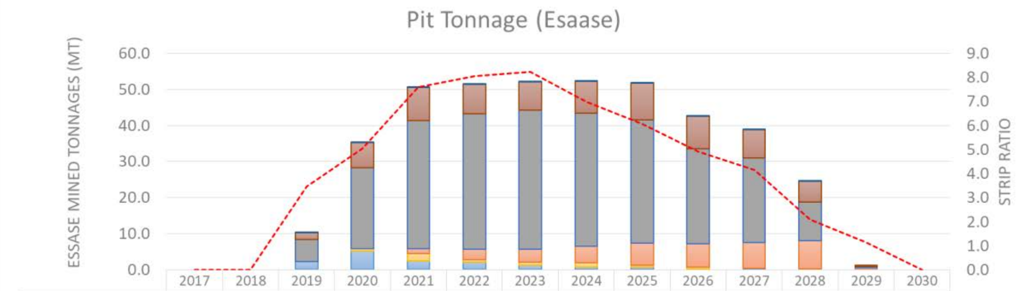
The Obotan mine production schedule as per updated mine planning (31st December 2016). Figure 16-29 below shows the updated Obotan mine schedule.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Waste - Obotan	25.1	36.3	25.8	24.9	24.7	21.8	22.7	13.8	21.3	13.4	0.0	0.0	0.0	0.0
Fresh - Obotan	4.3	3.5	3.1	3.3	3.0	2.2	2.8	3.6	3.9	2.2	0.0	0.0	0.0	0.0
Trans - Obotan	0.0	0.1	0.1	0.0	0.1	0.0	0.1	0.2	1.5	0.4	0.0	0.0	0.0	0.0
Oxide - Obotan	0.4	1.6	0.1	0.0	0.2	0.0	0.0	0.4	0.9	0.3	0.0	0.0	0.0	0.0
Total Ex-pit	29.8	41.6	29.1	28.3	28.0	24.0	25.5	17.9	27.6	16.3	0.0	0.0	0.0	0.0
Strip Ratio	5.3	6.9	7.8	7.5	7.4	9.8	7.9	3.4	3.4	4.6	0.0	0.0	0.0	0.0

Figure 16-29: Obotan Mine Schedule (source: Asanko Gold / DRA 2017)

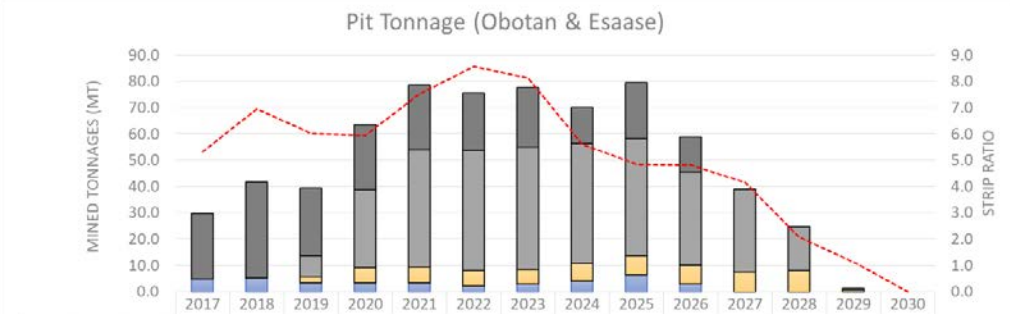
The Esaase mine production schedule is illustrated in Figure 16-30 below.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Inferred - Esaase	0.0	0.0	0.1	0.0	0.1	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0
VLG - Esaase	0.0	0.0	1.9	6.9	9.3	8.3	7.8	8.7	10.3	9.0	7.9	5.9	0.3	0.0
Waste - Esaase	0.0	0.0	6.1	22.5	35.4	37.5	38.6	36.9	34.2	26.4	23.5	10.7	0.3	0.0
Fresh - Esaase	0.0	0.0	0.0	0.0	1.4	3.0	3.5	4.6	6.0	6.5	7.1	7.8	0.6	0.0
Trans - Esaase	0.0	0.0	0.0	0.6	2.1	0.6	0.9	1.2	0.6	0.4	0.1	0.1	0.0	0.0
Oxide - Esaase	0.0	0.0	2.3	5.1	2.4	2.1	1.3	0.8	0.7	0.2	0.3	0.1	0.0	0.0
Total Ex-pit	0.0	0.0	10.4	35.2	50.6	51.5	52.1	52.2	51.8	42.6	38.9	24.6	1.3	0.0
Strip Ratio	0.0	0.0	3.5	5.1	7.6	8.1	8.2	7.0	6.1	4.9	4.2	2.1	1.1	0.0

Figure 16-30: Esaase Production Schedule (source: Asanko Gold / DRA 2017)

(Figure 16-31) below shows the combined Obotan and Esaase Mine production schedule.

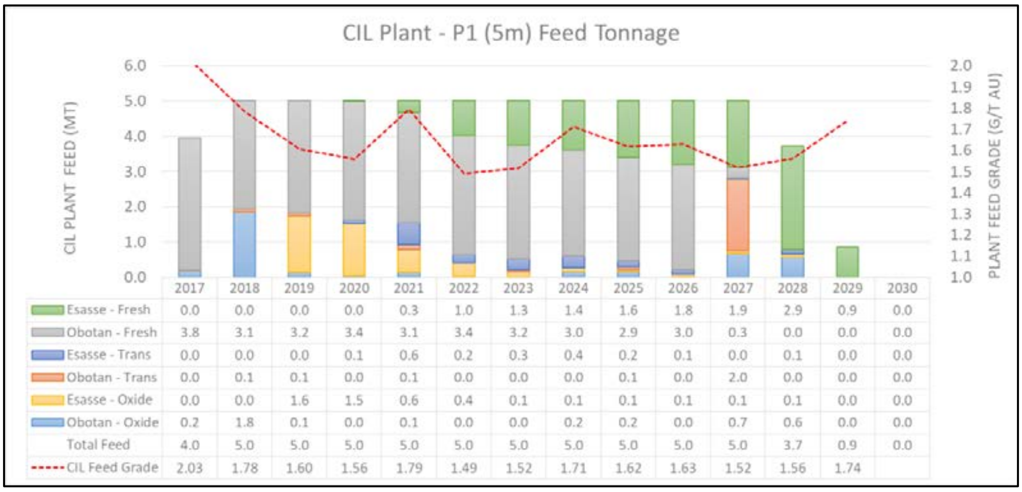


	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Obotan (Waste)	25.1	36.3	25.8	24.9	24.7	21.8	22.7	13.8	21.3	13.4	0.0	0.0	0.0	0.0
Essase (Waste)	0.0	0.0	8.0	29.4	44.7	45.8	46.4	45.7	44.5	35.4	31.3	16.6	0.7	0.0
Essase (Ore)	0.0	0.0	2.3	5.8	5.9	5.7	5.7	6.5	7.3	7.2	7.5	8.0	0.6	0.0
Obotan (Ore)	4.7	5.2	3.3	3.3	3.3	2.2	2.9	4.1	6.3	2.9	0.0	0.0	0.0	0.0
Total Ex-pit	29.8	41.6	39.5	63.5	78.6	75.5	77.6	70.2	79.4	58.9	38.9	24.6	1.3	0.0
Strip Ratio	5.3	6.9	6.0	5.9	7.5	8.5	8.1	5.6	4.8	4.8	4.2	2.1	1.1	0.0

Figure 16-31: Total Mine Production Schedule (source: DRA 2017)

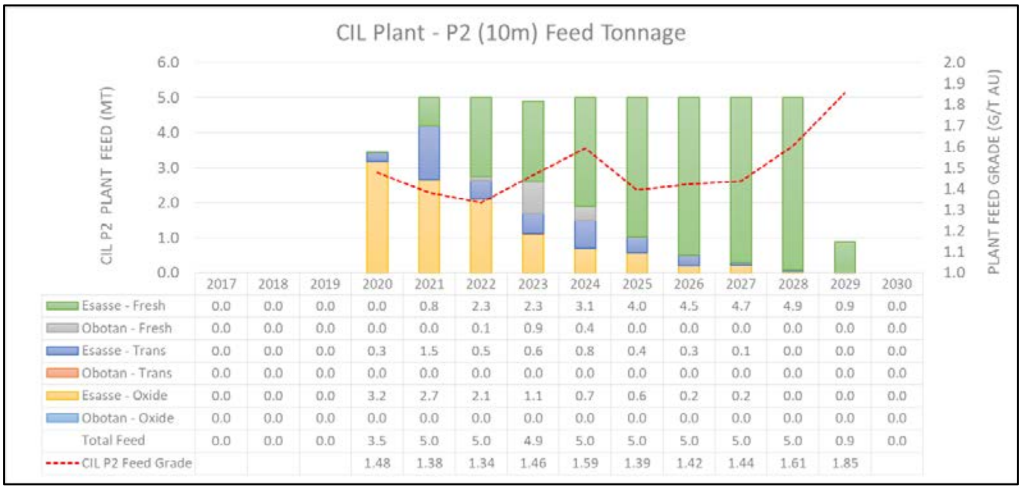
16.5.2 Plant Feed Schedules

The plant feed schedule is developed from the expected commissioning dates for the two plants and the aligned mine production schedules. Cognisance of stockpiling and re-handling is taken into account. Figure 16-32 and Figure 16-33 below shows the P5M CIL and P10M CIL processing plant feed tonnage schedule respectively.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	0.0	0.0	0.0	0.0	0.3	1.0	1.3	1.4	1.6	1.8	1.9	2.9	0.9	0.0
Obotan - Fresh	3.8	3.1	3.2	3.4	3.1	3.4	3.2	3.0	2.9	3.0	0.3	0.0	0.0	0.0
Esasse - Trans	0.0	0.0	0.0	0.1	0.6	0.2	0.3	0.4	0.2	0.1	0.0	0.1	0.0	0.0
Obotan - Trans	0.0	0.1	0.1	0.0	0.1	0.0	0.0	0.0	0.1	0.0	2.0	0.0	0.0	0.0
Esasse - Oxide	0.0	0.0	1.6	1.5	0.6	0.4	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0
Obotan - Oxide	0.2	1.8	0.1	0.0	0.1	0.0	0.0	0.2	0.2	0.0	0.7	0.6	0.0	0.0
Total Feed	4.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	3.7	0.9	0.0
CIL Feed Grade	2.03	1.78	1.60	1.56	1.79	1.49	1.52	1.71	1.62	1.63	1.52	1.56	1.74	

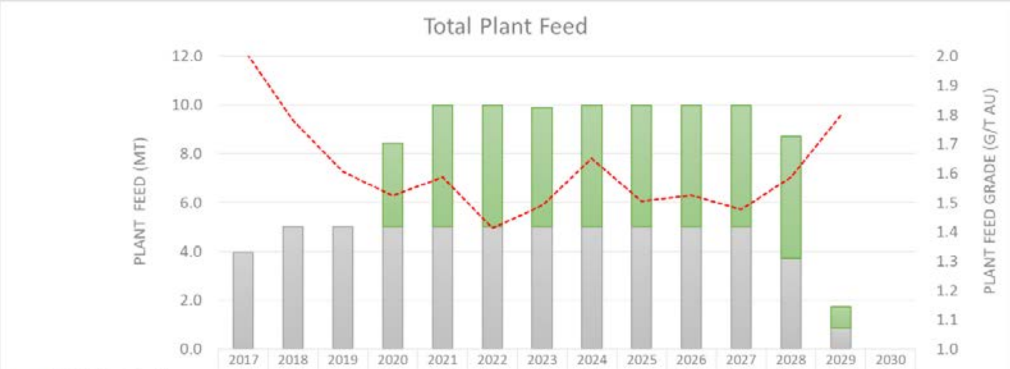
Figure 16-32: CIL P5M - Plant Feed Schedule (source: DRA 2017)



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	0.0	0.0	0.0	0.0	0.8	2.3	2.3	3.1	4.0	4.5	4.7	4.9	0.9	0.0
Obotan - Fresh	0.0	0.0	0.0	0.0	0.0	0.1	0.9	0.4	0.0	0.0	0.0	0.0	0.0	0.0
Esasse - Trans	0.0	0.0	0.0	0.3	1.5	0.5	0.6	0.8	0.4	0.3	0.1	0.0	0.0	0.0
Obotan - Trans	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Esasse - Oxide	0.0	0.0	0.0	3.2	2.7	2.1	1.1	0.7	0.6	0.2	0.2	0.0	0.0	0.0
Obotan - Oxide	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Feed	0.0	0.0	0.0	3.5	5.0	5.0	4.9	5.0	5.0	5.0	5.0	5.0	0.9	0.0
CIL P2 Feed Grade				1.48	1.38	1.34	1.46	1.59	1.39	1.42	1.44	1.61	1.85	

Figure 16-33: CIL P10M - Plant Feed Schedule (source: DRA 2017)

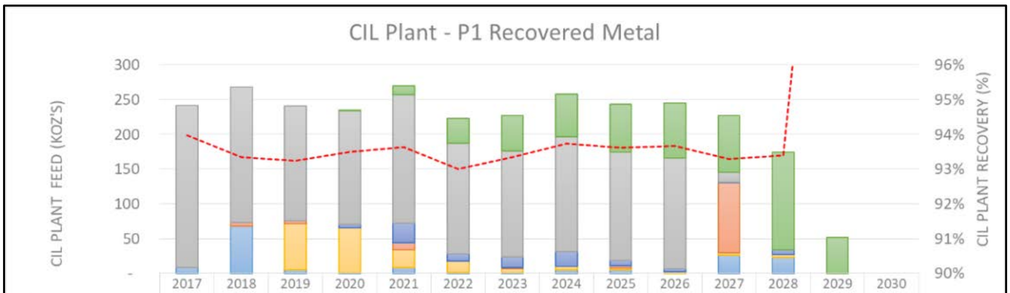
Figure 16-34 below shows the combined P10M CIL processing plant feed schedule.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
CIL P2 Plant Feed Tonnage	0.0	0.0	0.0	3.5	5.0	5.0	4.9	5.0	5.0	5.0	5.0	5.0	0.9	0.0
CIL P1 Plant Feed Tonnage	4.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	3.7	0.9	0.0
Total Feed	4.0	5.0	5.0	8.5	10.0	10.0	9.9	10.0	10.0	10.0	10.0	8.7	1.7	0.0
Average Feed Grade	2.03	1.78	1.60	1.52	1.59	1.41	1.49	1.65	1.51	1.52	1.48	1.59	1.80	

Figure 16-34: Combined Plant Feed Schedule (source: DRA 2017)

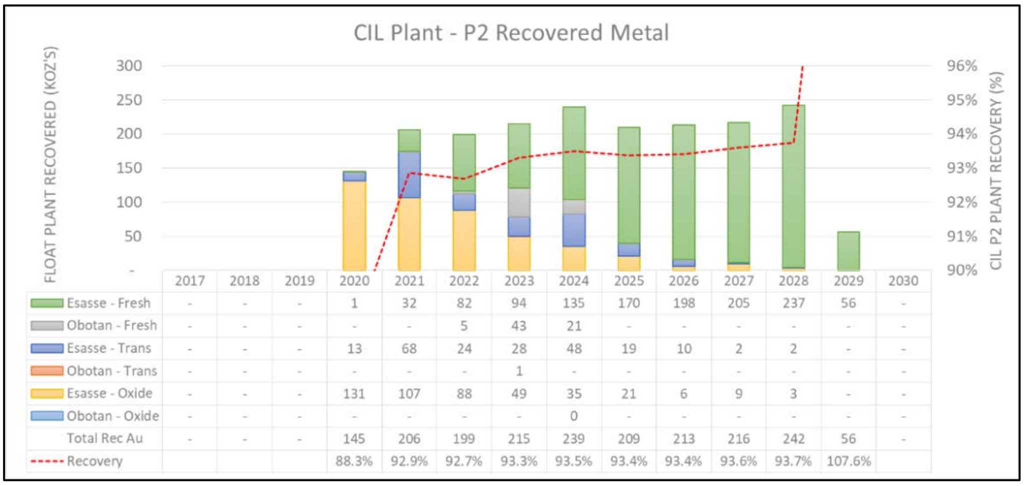
Figure 16-35 and Figure 16-36 below shows the P5M CIL and P10M CIL processing plant recovered metal respectively.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	-	-	-	0	13	36	52	61	69	79	82	141	52	-
Obotan - Fresh	234	195	165	163	185	159	152	166	156	160	15	-	-	-
Esasse - Trans	-	-	-	6	28	10	15	22	8	4	1	7	-	-
Obotan - Trans	0	5	4	0	11	-	2	-	3	-	100	-	-	-
Esasse - Oxide	-	-	66	64	26	16	6	5	2	2	4	4	-	-
Obotan - Oxide	8	68	4	1	8	1	0	5	5	-	26	22	-	-
Total Rec Au	242	268	241	234	270	223	227	258	243	245	228	174	52	-
Recovery	94.0%	93.3%	93.2%	93.5%	93.6%	93.0%	93.3%	93.7%	93.6%	93.7%	93.3%	93.4%	108.4%	

Figure 16-35: CIL P5M Plant Recoverable Gold (source: DRA 2017)

Note : Recovery in final year accounts for release of final gold lockup in the processing circuit.

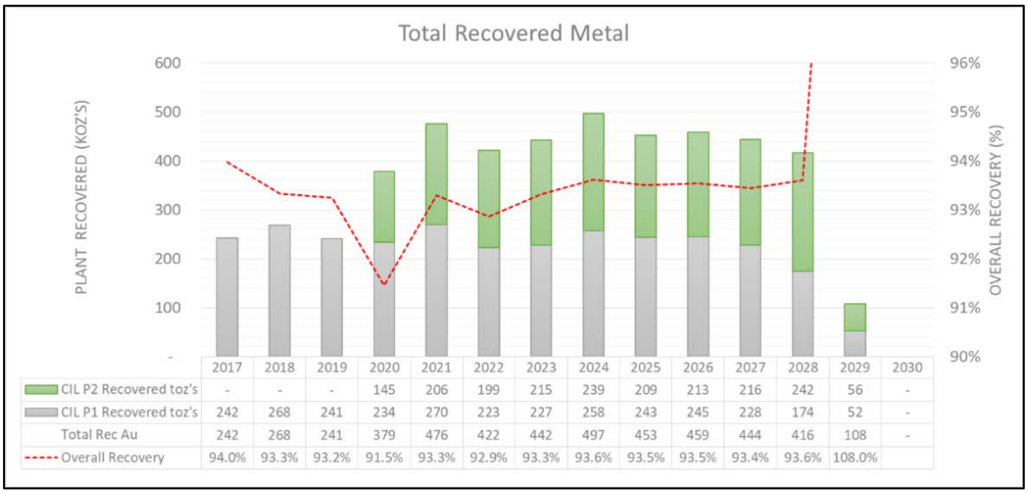


	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Esasse - Fresh	-	-	-	1	32	82	94	135	170	198	205	237	56	-
Obotan - Fresh	-	-	-	-	-	5	43	21	-	-	-	-	-	-
Esasse - Trans	-	-	-	13	68	24	28	48	19	10	2	2	-	-
Obotan - Trans	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Esasse - Oxide	-	-	-	131	107	88	49	35	21	6	9	3	-	-
Obotan - Oxide	-	-	-	-	-	-	-	0	-	-	-	-	-	-
Total Rec Au	-	-	-	145	206	199	215	239	209	213	216	242	56	-
Recovery				88.3%	92.9%	92.7%	93.3%	93.5%	93.4%	93.4%	93.6%	93.7%	107.6%	

Figure 16-36: CIL P10M Plant Recoverable Gold (source: DRA 2017)

Note : Recovery in final year accounts for release of final gold lockup in the processing circuit.

Figure 16-37 below shows the combined P10M CIL processing recovered metal.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
CIL P2 Recovered toz's	-	-	-	145	206	199	215	239	209	213	216	242	56	-
CIL P1 Recovered toz's	242	268	241	234	270	223	227	258	243	245	228	174	52	-
Total Rec Au	242	268	241	379	476	422	442	497	453	459	444	416	108	-
Overall Recovery	94.0%	93.3%	93.2%	91.5%	93.3%	92.9%	93.3%	93.6%	93.5%	93.5%	93.4%	93.6%	108.0%	

Figure 16-37: Total Recoverable Gold (source: DRA 2017)

Note : Recovery in final year accounts for release of final gold lockup in the processing circuit.

16.6 Mine Operating Costs

Mine operating costs are detailed in Section 21 of the NI 43-101 Technical Document.

16.7 Mine Capital Costs

Mine capital costs are detailed in Section 21 of the NI 43-101 Technical Document.

17 RECOVERY METHODS

17.1 AGM Phase 1

The AGM Phase 1 processing plant was commissioned during Q1 2016 and is currently operating at a throughput of 3.6 Mtpa and achieving recoveries in excess of 94%.

The AGM Phase 1 processing plant design is based on a typical single stage crushing, SAG and ball milling circuit followed by a CIL plant. The flow sheet includes a single stage jaw crusher that can either feed onto a live stockpile, or directly into an open circuit SAG, (complete with pebble crusher) and ball milling unit in closed circuit with classification cyclones. A gravity recovery circuit is utilised to treat a portion of the cyclone underflow stream to recover coarse free gold from the re-circulating load.

The milled product, (cyclone overflow) gravitates to a pre-leach thickener, via a trash removal screen. Thickener underflow is pumped directly to a pre-oxidation stage followed by a seven stage CIL circuit. Leached gold adsorbs onto activated carbon, which flows counter-currently to the gold-bearing slurry. Loaded carbon is directed to the 5t elution circuit while tailings gravitates to the cyanide destruction circuit.

Provision was made in the design for the detoxification of cyanide in the CIL tailings by means of the SO_2/Air process, during which the WAD cyanide concentration is reduced in a single tank by means of SMBS and air. The detoxified tailings product gravitates to the CIL tailings disposal tank via a sampling system from where it is pumped to the TSF.

Absorbed gold is eluted from the activated carbon by means of a heated solution of sodium cyanide and caustic soda via the split AARL procedure. Barren carbon from the batch elution process is directed to the carbon regeneration circuit, while the PLS is routed to the electro winning circuit.

After washing the gold sludge from the electrowinning cathodes, the sludge is decanted and treated in a drying oven after which it is mixed with fluxes and loaded into an induction smelting furnace. After smelting the gold bullion bars are cleaned, labelled, assayed and prepared for shipping.

The Phase 1 CIL plant further incorporates water treatment, reagent preparation, oxygen generation and supply, compressed air and water services.

This process flow sheet is well known in industry, and is relatively low risk as it has historically been proven a successful processing route for Obotan region ores during Resolute's operations of 1998 to 2002.

17.1.1 Process Design Criteria

The key process design criteria listed in Table 17-1 form the basis of the detailed PDC and MEL for the Phase 1 CIL plant.

Table 17-1: AGM Phase 1 Key Process Design Criteria (source: DRA 2017)

Parameter	Units	Value
Plant Capacity	t/a	3,600,000
LoM Average Head Grade	Au g/t	2.05
Sulphides	*Au g/t*	*2.06*
Oxides	*Au g/t*	*1.88*
LoM Average Ore Split		
Sulphides	*%*	*94.7*
Oxides	*%*	*5.3*
LoM Average Overall Recovery *	%	94.5
Sulphides	*%*	*94.6*
Oxides	*%*	*92.0*
Gravity Recovery	%	54.1
Sulphides	*%*	*55.0*
Oxides	*%*	*37.0*
Crushing Plant Running Time	hpa	5 125
Crushing Plant Feed Rate	tph	700**
Milling & CIL Plant Running Time	hpa	7 996
Milling & CIL Plant Feed Rate	tph	450
ROM Feed Size (F_{100})	mm	800
Leach Feed Size (F_{80})	µm	106
Leach Residence Time	hr	20.0
Leach Slurry Feed Density	% w/w	45.0
Number of Pre-Oxidation Tanks	#	1
Number of CIL Tanks	#	7
LoM Average CIL Cyanide	kg/t	0.42
LoM Average Lime Consumption	kg/t	0.64
Elution Circuit Type		Split AARL
Elution Circuit Size	t	5
Frequency of Elution	batches/day	2
Cyanide Destruction Process		SO_2 / Air Process

Note

- * Based on current operating and test work data and includes calculated discount factors to account for CIL solution gold losses and fine carbon gold losses as follows:

  Concentrate CIL carbon fines losses at 40g carbon per ton milled at a grade of 50 g/t Au.

  Solution gold losses based on 45% solids in the CIL tails stream and 0.01 g/l Au in solution.
- ** Mobile crusher unit procured and employed for additional capacity.

17.1.2 Plant Design

A simplified process flow diagram for the AGM Phase 1 CIL plant is provided in Figure 17-1.

Refer to the NI 43-101 Technical Report "Asanko Gold Mine – Phase 2 PFS" issued in June 2015 for more details on the Phase 1 CIL circuit plant design and building provided.

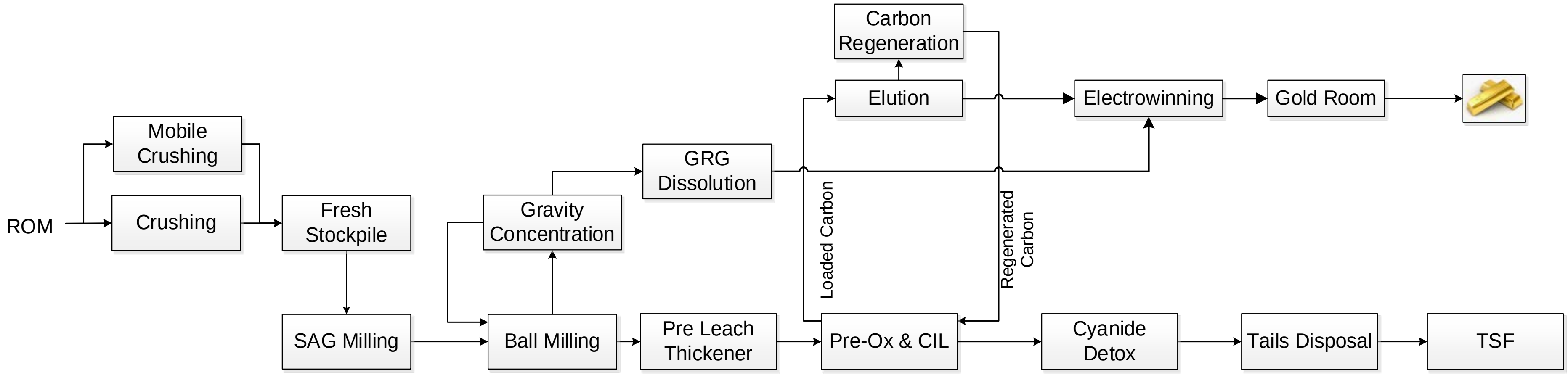


Figure 17-1: AGM Phase 1 Block Flow Diagram (source: DRA 2014)

17.2 AGM DFS

17.2.1 Introduction

The AGM DFS includes the upgrading of the existing Phase 1 CIL plant from the current throughput of 3.6 Mtpa to 5 Mtpa. A number of equipment upgrades (referred to as P5M) are required to the existing circuit in order to process 5 Mtpa. For the first year of the expansion, the increased throughput will be achieved by supplementing the current 3 Mtpa of Nkran Fresh material with 2 Mtpa of Oxide material obtained from satellite pits. After year 1 of P5M, the 3 Mtpa of Nkran Fresh material will be supplemented by 2 Mtpa of Esaase Oxide material.

The second stage (referred to as P10M) of the AGM DFS includes the addition of a second 5 Mtpa processing facility, to take the total processing throughput of the AGM to 10 Mtpa.

Previously P10M included the addition of a flotation plant consisting of a ROM handling and two-stage crushing circuit located at the Esaase mining site, followed by an overland conveying circuit to transport the crushed material to the AGM Phase 1 processing site where the gravity recovery, milling, flotation, concentrate regrind and concentrate CIL circuits would be located.

Test work conducted as part of this DFS investigated the opportunity to combine the P10M concentrate CIL circuit with the P5M CIL circuit in an attempt to reduce operating and capital costs. The test work however reported poor loading kinetics of the leached gold onto the activated carbon when blending the Nkran Sulphide gravity tailings material with the reground Esaase flotation concentrate material in a common CIL circuit. The poor adsorption kinetics was attributed to the poisoning of the activated carbon by the flotation reagents. Refer to Section 13 for more details on the test work conducted. The poor gold loading kinetics due to carbon poisoning resulted in the circuits remaining separate with dedicated elution and carbon regeneration circuits (as per the PFS flowsheet as reported in the NI 43-101 Technical Report “Asanko Gold Mine – Phase 2 PFS”, issued June 2015) in order not to jeopardise the performance of the existing Phase 1 CIL circuit – which is currently achieving recoveries in excess of 94%. Further, due to the slow adsorption of the leached gold onto the activated carbon, additional adsorption capacity would be required in the flotation concentrate CIL circuit. The design would have to allow for a short carbon residence time in the concentrate CIL circuit resulting in more frequent elution cycles, and thus increasing operating and capital costs for this processing option. The risk of high soluble gold losses was also present.

Previous evaluations conducted on the processing options for the Esaase material, (flotation vs whole ore leach), indicated that the flotation processing route achieved similar recoveries when compared to the whole ore leach circuit, whilst the operating cost for the flotation route was lower compared to a whole ore leach circuit. Subsequent to these evaluations, cyanide consumption optimisation test work have been conducted on Nkran material, which resulted in substantial operating cost reductions on the whole ore leach process.

Based on the above, a decision was taken to re-evaluate the opportunity of a whole ore leach circuit for the Esaase material. Further test work was conducted at ALS in Perth during the second half of 2016 to assess the gold recovery of the Esaase Sulphide material when processed in a whole ore leach circuit similar to the existing Phase 1 CIL circuit, whilst incorporating the learnings from the cyanide optimisation test work conducted on the Nkran material. The test work indicated that the Esaase Sulphide material resulted in similar recoveries compared to the Nkran Sulphide material. Refer to Section 13 for more details. A re-evaluation of a flotation processing route and whole ore leach processing route (incorporating these 2016 test work results) indicated that the whole ore leach processing route is a viable option to consider for the processing or the Esaase Sulphide material.

In addition to the above, the following benefits of a CIL circuit for AGM is noted:

- Experience gained from operating the existing Phase 1 CIL plant
- Familiarity of the Ghanaian workforce with CIL compared to flotation
- Savings on reagents and holding costs, reduced insurance and operating spares holdings

17.2.2 AGM DFS Flowsheet Summary

17.2.2.1 P5M - Phase 1 CIL Plant Upgrades

P5M is to be executed in two parts. The first part of P5M (metallurgical upgrade) includes the upgrading of the existing CIL circuit to treat a maximum of 5 Mtpa by means of the following modifications:

- Modifications to the secondary mill classification cyclone cluster internals, and upgrading of the existing cyclone overflow pipeline
- Expansion of the gravity recovery circuit by the addition of a third gravity gold concentrator and auxiliary equipment, with space provision for a fourth unit in future
- Addition of a second ILR (intensive leach reactor) and auxiliary equipment
- Upgrading of the current pre-leach thickener underflow pump motors, and upgrading of the thickener underflow pipeline
- Addition of a dedicated mill circuit water system, which will receive thickener overflow product for re-use as mill circuit dilution water by a dedicated pumping system
- Upgrading of the CIL intertank screens to cater for increased throughput
- Upgrading of the tailings disposal pump system by increasing motor ratings and the addition of a third pump train and second tailings pipeline
- Upgrading of the electrowinning circuit by the addition of a third electrowinning cell and rectifier, upgrading of the eluate electrowinning feed pumps and motors, as well as a number of piping upgrades.
- Upgrading of the PSA plant capacity from 10 tpd to 15 tpd by the addition of a third 5 tpd module

The second part of P5M includes the following upgrades / additions:

- Installation of a mobile crushing plant and auxiliary equipment and conveyors to be located at the Esaase mine pit to allow for the crushing and transferring of the additional 2 Mtpa of Esaase Oxide material to the Overland Conveyor ("OLC")
- Installation of the OLC to transfer crushed material at -90 mm from the Esaase pit to the Obotan processing plant
- Installation of a stockpile facility to stockpile the Esaase material, together with auxiliary equipment and conveyors to feed and extract material from this stockpile and transfer the material to the existing mill feed conveyor
- Installation of pit dewatering, potable water and raw water infrastructure and equipment at the Esaase mine pit

The total connected electrical load, inclusive of all standby units, is 26MW for the upgraded Phase 1 CIL plant (current circuit post P5M upgrades), with a running power draw of 16MW.

17.2.2.2 P10M – Addition of a Second 5 Mtpa CIL Circuit

The process flowsheet for the P10M CIL circuit will be based on the P5M circuit, where feasible.

P10M will consist of a ROM handling and a closed circuit, two-stage crushing circuit located at the Esaase mine pit, followed by stockpiling and loading of the stockpiled material onto the overland conveying circuit to transport the crushed material to the Phase 1 processing site where the milling, gravity recovery, and CIL circuit would be located. Provision is made for intermediate stockpiling of the Esaase crushed material and interlinking conveyors between the P5M and P10M CIL circuits to allow the processing of Esaase material in either of the CIL circuits.

The crushed Esaase material will feed onto a live, intermediate stockpile from where it can either be fed to the P5M (upgraded Phase 1) milling circuit, or to the P10M milling circuit. Provision is made for sampling of the Esaase ore prior to milling.

The P10M ball milling circuit will operate in closed-circuit with a classification cyclone cluster. A gravity recovery circuit will be provided to treat the full cyclone underflow stream to maximize the recovery of coarse free gold from the recirculating load.

The milled product, (cyclone overflow) will gravitate to the P10M pre-leach thickener, via a trash removal screen. Thickener underflow will be pumped directly to a pre-oxidation stage followed by a seven stage CIL circuit, as per the existing plant.

Leached gold will adsorb onto activated carbon, which flows counter-currently to the gold-bearing slurry. Loaded carbon will be directed to the elution circuit while tailings will gravitate to a dedicated P10M cyanide destruction circuit. As per the existing circuit, provision will be made in the design to cater for the destruction of cyanide in the P10M CIL tailings using the SO_2 / Air process. WAD concentration will be reduced in a single tank by means of SMBS and air, after which the detoxified tailings will gravitate to the P10M CIL tails disposal system via a sampling system, from where it will be pumped to the expanded, common TSF.

As per the Phase 1 circuit, the absorbed gold will be eluted from the activated carbon by means of a heated solution of sodium cyanide and caustic soda via the split AARL procedure. Barren carbon from the batch elution process will be directed to a dedicated P10M carbon regeneration circuit, while the Pregnant Leach Solution (“PLS”) will be routed to an upgraded electrowinning circuit. A dedicated 5t elution facility will be provided for P10M.

The P5M electrowinning circuit and gold room will further be expanded as part of P10M to cater for the additional gold loading due to the inclusion of the second 5 Mtpa CIL circuit. After washing the gold sludge from the electrowinning cathodes, the sludge will be decanted and treated in either one of two drying ovens after which it will be mixed with fluxes and loaded into an induction smelting furnace. After smelting the gold bullion bars will be cleaned, labelled, assayed and prepared for shipping.

The P10M CIL plant will make use of the existing water treatment, reagent preparation, and reagent storage facilities where possible. Dedicated oxygen generation and supply, compressed air and water services will be provided for the P10M CIL plant.

The total connected electrical load, inclusive of all standby units and the overland conveying, is 22MW for the P10M CIL plant, with a running power draw of 18MW.

17.2.3 AGM DFS Process Design Criteria

The key process design criteria listed in Table 17-2 form the basis of the detailed PDC and MEL for the AGM DFS.

Table 17-2: AGM DFS Key Process Design Criteria (source: DRA 2017)

Parameter	Units	Value	
		P5M CIL Circuit	**P10M CIL Circuit**
Plant Capacity	t/a	5,000,000	5,000,000
LoM Average Head Grade	Au g/t	1.65	1.46
Nkran Sulphides	*Au g/t*	*1.80*	*1.60*
Nkran Oxides	*Au g/t*	*1.30*	*0.91*
Nkran Transitional	*Au g/t*	*1.67*	*1.26*
Esaase Sulphides	*Au g/t*	*1.47*	*1.45*
Esaase Oxides	*Au g/t*	*1.41*	*1.42*
Esaase Transitional	*Au g/t*	*1.61*	*1.60*
LoM Average Ore Split			
Nkran Sulphides	*%*	*55.3*	*3.2*
Nkran Oxides	*%*	*6.6*	*0.0*
Nkran Transitional	*%*	*4.3*	*0.0*
Esaase Sulphides	*%*	*22.3*	*62.2*
Esaase Oxides	*%*	*8.0*	*24.4*
Esaase Transitional	*%*	*3.6*	*10.2*
LoM Average Overall Recovery *	%	93.7	93.3
Nkran Sulphides	*%*	*93.7*	*93.4*
Nkran Oxides	*%*	*91.7*	*88.7*
Nkran Transitional	*%*	*93.3*	*91.5*
Esaase Sulphides	*%*	*94.8*	*94.2*
Esaase Oxides	*%*	*93.2*	*91.3*
Esaase Transitional	*%*	*92.7*	*92.5*
Gravity Recovery	%	52.5	52.2
Nkran Sulphides	*%*	*54.2*	*54.2*
Nkran Oxides	*%*	*37.3*	*37.3*
Nkran Transitional	*%*	*48.5*	*48.5*
Esaase Sulphides	*%*	*55.0*	*55.0*
Esaase Oxides	*%*	*48.7*	*48.7*
Esaase Transitional	*%*	*43.0*	*43.0*
Crushing Plant Running Time	hpa	5 125	5 833
Crushing Plant Feed Rate	tph	700**	1 200

Parameter	Units	Value	
		P5M CIL Circuit	P10M CIL Circuit
Milling & CIL Plant Running Time	hpa	7,996	7,996
Milling & CIL Plant Feed Rate	tph	625	625
ROM Feed Size (F_{100})	mm	800	800
Leach Feed Size (F_{80})	µm	106	106
Leach Residence Time	hr	16	16
Leach Slurry Feed Density	% w/w	50	50
Number of Pre-Oxidation Tanks	#	1	1
Number of CIL Tanks	#	7	7
LoM Average CIL Cyanide Consumption	kg/t	0.5	0.5
LoM Average Lime Consumption	kg/t	0.7	0.8
Elution Circuit Type		Split AARL	Split AARL
Elution Circuit Size	t	5	5
Frequency of Elution	batches/day	2	2
Cyanide Destruction Process		SO_2 / Air Process	SO_2 / Air Process

Notes

* Based on current operating and test work data and includes calculated discount factors to account for CIL solution gold losses and fine carbon gold losses as follows:

Concentrate CIL carbon fines losses at 40g carbon per ton milled at a grade of 50 g/t Au.

Solution gold losses based on 45% - 50% solids in the CIL tails stream and 0.01 g/l Au in solution.

A discount of 0.01% over the life of mine has been allowed for ramp up on the P5M CIL circuit to 5 Mtpa while a discount of 0.02% over the life of mine has been allowed for ramp-up and commissioning on the P10M CIL circuit.

** Mobile crusher unit procured and employed for additional capacity.

17.2.4 AGM DFS Plant Design

A simplified process flow diagram for AGM DFS is provided in Figure 17-2.

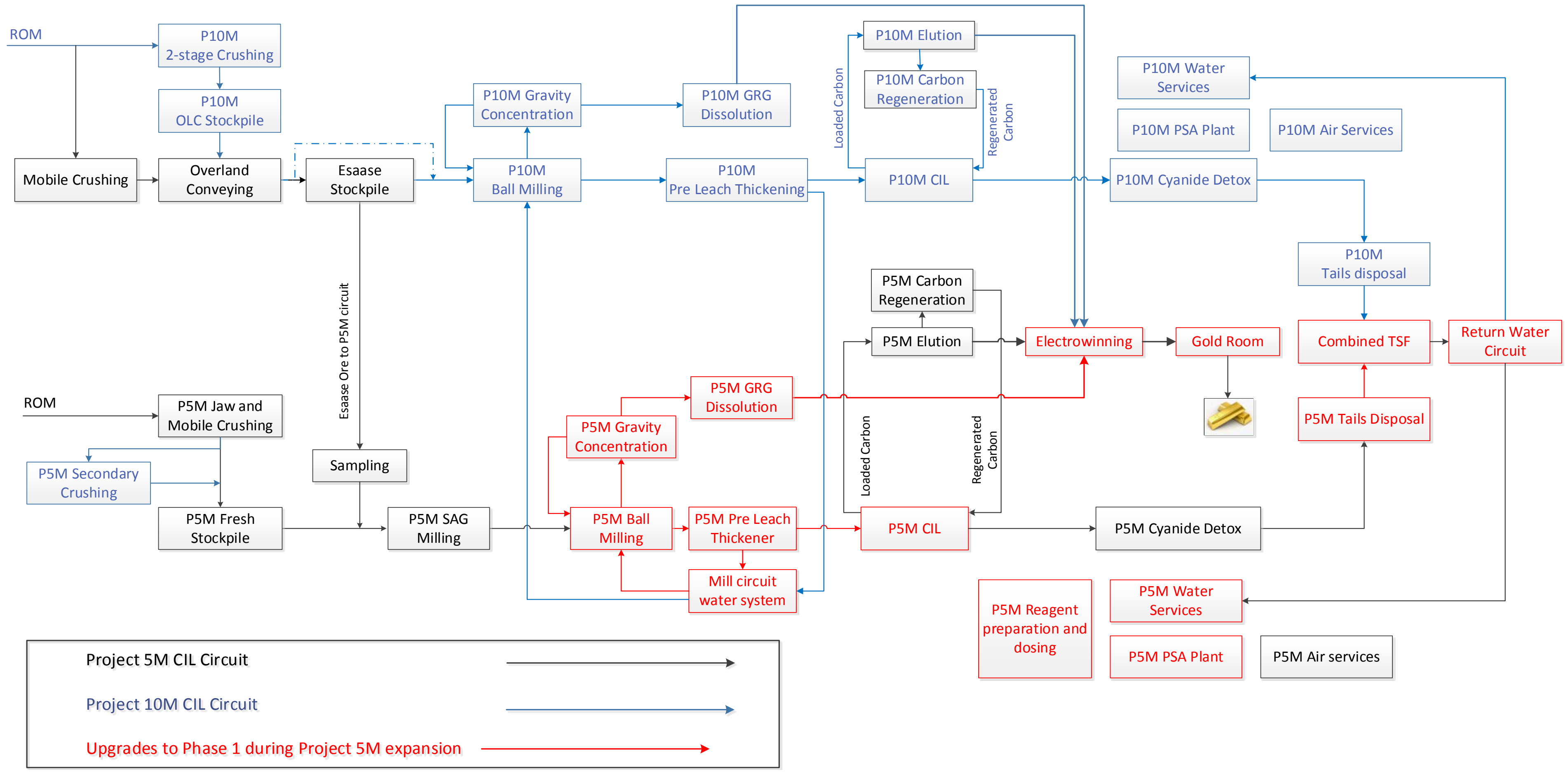


Figure 17-2: AGM DFS Block Flow Diagram (source: DRA 2017)

17.2.4.1 Plant 1 Primary Crushing Circuit

The existing primary crushing circuit consists of a single tip with a dedicated ROM bin and a single jaw crusher in open circuit. Primary crusher product reports to the crushed ore stockpile. The ROM ore (F_{100} 800 mm, F_{80} 500 mm) is loaded into a 100t ROM Bin by means of a Front End Loader (“FEL”), or by direct tipping from trucks.

The ROM ore is drawn from the ROM bin at a controlled rate by a single, variable speed apron feeder, and fed directly to the jaw crusher. The speed of the apron feeder is controlled to maintain crusher throughput. Fine material spillage from the apron feeder reports to the primary crushing conveyor, where it is combined with the primary crusher product (P_{100} 250 mm, P_{80} 160 mm). The primary crushing conveyor is fitted with a belt magnet to remove any tramp iron material. The primary crushing conveyor discharges the crushed material onto the overland conveyor.

Further to the above, a mobile crusher unit has been procured and is employed to assist with crushing of the ROM material, to maintain throughput in the crushing circuit.

Dust suppression as means for dust control is employed at the Phase 1 crushing circuit.

No modifications to the existing primary crushing circuit is envisaged as part of the AGM DFS.

17.2.4.2 Plant 1 Secondary Screening and Crushing Circuit

Provision is made for the addition of an open circuit secondary cone crushing installation as part of P10M, to crush the existing primary crusher product from P_{80} 160mm down to -90mm, prior to feeding to the 1,550t crushed ore stockpile.

The primary crushing conveyor is fitted with a belt magnet to remove any tramp iron material, prior to feeding the Obotan secondary crusher grizzly screen. The grizzly screen oversize material will discharge directly into the Obotan secondary cone crusher. The secondary crusher product will combine with the grizzly screen undersize material on the existing overland conveyor, from where it will be conveyed onto the 1,550t crushed ore stockpile.

Provision will be made for a bypass facility of this secondary crushing circuit, when softer material is being processed which does not require secondary crushing.

Dust suppression as means for dust control will be employed at the Obotan secondary screening and crushing installation.

17.2.4.3 Esaase Ore Receiving, Crushing, Overland Conveying and Stockpiling

The Esaase crushing circuit will be constructed in two phases.

During P5M of the AGM DFS, a two-stage mobile crushing unit will be used to crush the additional 2 Mtpa of Esaase Oxide material to -90mm, prior to loading onto the overland feed conveyor, via a bin and variable speed apron feeder arrangement, for transfer to the current processing site. The overland feed conveyor will be equipped with two-stage tramp-iron removal magnets, and will discharge the Esaase material onto the single flight overland conveyor.

The overland conveyor will deliver the crushed Esaase Oxide material onto the intermediate conveyor, which in turn will transfer the material onto the 3,500t live capacity Esaase crushed ore stockpile.

The material will be withdrawn from the stockpile at a controlled rate using two variable speed apron feeders, and loaded onto the Plant 1 Esaase ore conveyor, which will transfer the material onto the existing mill feed conveyor after sampling to combine with the Nkran sulphide ore as feed into the existing SAG mill.

During P10M, the mobile crushing unit will be replaced with a permanent crushing circuit, designed to crush 1,200 tph of Esaase material, as per Figure 17-3.

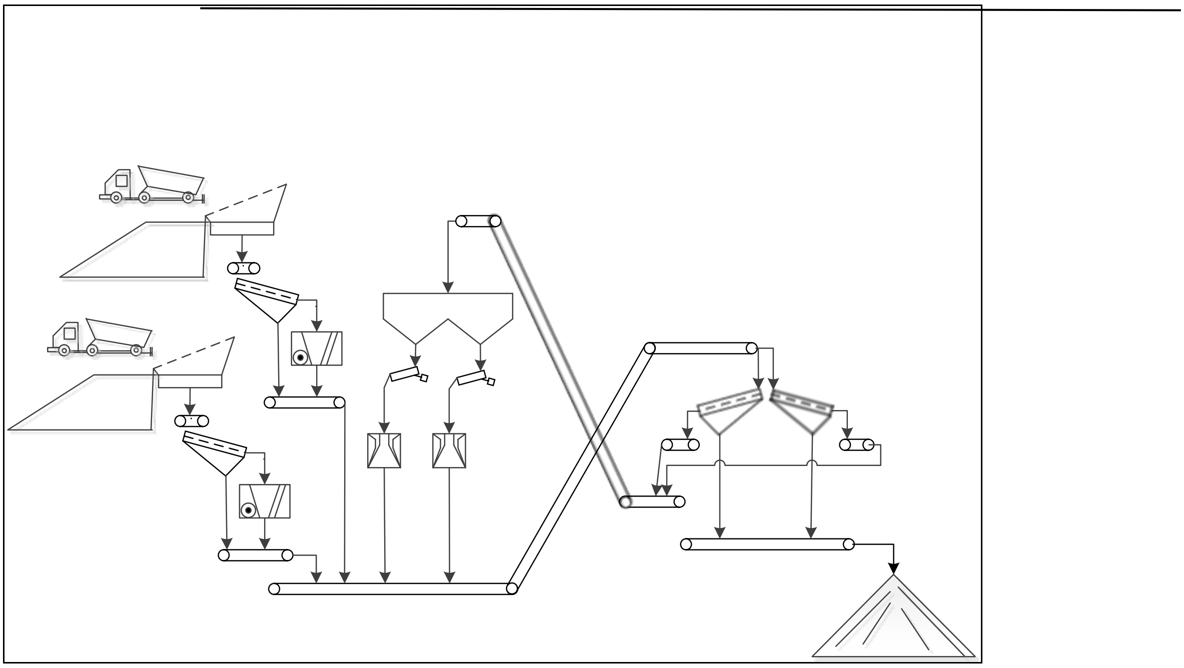


Figure 17-3: AGM DFS P10M Esaase Crushing Circuit (source: DRA 2017)

The Esaase ROM ore will be treated in a primary crusher operating in open circuit consisting of two tips, each with a dedicated ROM tip bin, vibrating grizzly screen and jaw crusher. The screens will be fed from the ROM bins with apron feeders. The product stream from the primary crushers (P_{80} of 190 mm), together with the screen undersized material will combine on the Esaase secondary screen feed conveyor from where it will be conveyed to the closed circuit secondary crushing area. The secondary screen feed conveyor will be equipped with a tramp iron removal magnet as protection to the secondary screens and crushers.

The secondary screen feed conveyor will transfer the primary crusher material to the secondary screen splitter chute, which will divide the material between the two secondary screen feeders feeding the secondary classification screens.

The secondary classification screening oversize material (+90 mm) will collect on dedicated screen oversize conveyors, which will transfer the screen oversize material to the Esaase secondary crushing feed conveyor. The classification screening undersize material (-90 mm) will be collected directly onto the Esaase crushed ore stockpile feed conveyor.

The Esaase secondary crushing feed conveyor will discharge the oversize material into the 500t Esaase secondary crusher feed bin. The material will be withdrawn from the bin using two variable speed vibrating feeders, and fed to two secondary cone crusher units. The secondary crushed material (-90 mm) will combine with the screen undersize material onto the Esaase crushed ore stockpile feed conveyor.

The -90mm Esaase material on the Esaase crushed ore stockpile feed conveyor will be stockpiled on the 1,500t live capacity overland feed stockpile. The material will be withdrawn from this stockpile at a controlled rate via two variable speed apron feeders, to ensure an equal loading onto the overland conveyor.

The Esaase crushed ore stockpiling area will be further expanded in P10M to include two additional apron feeders to extract and load Esaase material onto the Plant 2 mill feed transfer conveyor, which will discharge onto the Plant 2 mill feed conveyor, at a controlled rate.

This ore handling arrangement makes it possible to feed Esaase material from the Esaase pits to either the P5M, or P10M milling circuits.

Dust control on the Esaase screening and crushing circuits and stockpiling areas will be provided in the form of dust suppression.

Steel ball and quicklime addition facilities have been included on the Plant 2 mill feed conveyor.

17.2.4.4 Plant 1 SAG Mill Feed of Nkran Material

The Nkran crushed ore is conveyed from the existing crushing circuit to the 1,550t crushed ore stockpile area. A stockpile bypass facility has been incorporated to allow material to bypass the stockpile and report directly to SAG milling. The existing overland conveyor can discharges onto the stockpile feed conveyor.

The Nkran ore from the existing crushed ore stockpile is withdrawn at a controlled rate by means of a duty / standby apron feeder arrangement which discharges directly onto the existing Plant 1 mill feed conveyor. A weightometer indicates the instantaneous and totalised crushed ore mill feed tonnage and is used to control the SAG mill feed rate via the apron feeders as well as the supplementation rate of the feed with the Esaase Oxide material, to a total Fresh feed rate of 625 tph to the SAG mill.

Scats rejected by the SAG mill is conveyed to a pebble crusher and reintroduced into the system by discharging onto the Plant 1 mill feed conveyor.

Quicklime is stored in a 100t silo and is metered onto the Plant 1 mill feed conveyor using a variable speed screw feeder. Quicklime is delivered to site by tanker and pneumatically transferred to the lime silo using an off-loading blower.

A ball loading system is used for loading of grinding media into the SAG mill (via the Plant 1 mill feed conveyor).

Dust control is employed by a dust suppression system at the existing stockpile area.

17.2.4.5 Plant 1 Milling, Classification, Gravity Concentration, and ILR circuits

The Plant 1 grinding circuit consists of a primary SAG mill and secondary ball mill in closed circuit with a cyclone cluster.

The Plant 1 mill feed conveyor transfers the crushed material directly to the SAG mill, which operates in open circuit to deliver a primary milled product of roughly P_{80} 1 mm. Water is added to the SAG mill feed chute to control the in-mill density. The SAG mill discharge is screened via a 12 mm x 30 mm aperture trommel screen before gravitating to the mill discharge tank. Screen oversize is conveyed to a single pebble crusher, where it is crushed to below 12 mm prior to recycling back to mill feed conveyor. The pebble crusher feed conveyor is fitted with a weightometer for control purposes. A SAG mill pebble bunker is installed, in which any pebble overflow is stored for further handling.

No changes to the existing SAG mill circuit is envisaged as part of the AGM DFS.

The secondary ball mill is an overflow discharge mill, and operates in closed circuit with a classification cyclone cluster. Ball mill discharge is screened by means of a 12 mm aperture trommel screen before reporting to the mill discharge tank where it is combined with the SAG mill product. Trommel screen oversize material is collected in the ball mill scats bunker. Dilution water is added to the ball mill feed for in-mill density control, as well as the ball mill discharge tank for cyclone feed density control.

The combined SAG mill and ball mill products are pumped to the classification cyclone cluster via a duty / standby pumping arrangement. The classification cyclone overflow stream (P_{80} 106 µm) gravitates to the Plant 1 pre-leach thickening circuit, while a portion of the cyclone underflow stream gravitates to the gravity concentration circuit. The remainder of the cyclone underflow stream bypasses the gravity concentration circuit and combines with the gravity concentration circuit tailings product before reporting back to the secondary ball mill feed for re-grinding. The flow to each centrifugal concentrator unit is controlled with a weir type cyclone underflow box and gate valve arrangement.

Modifications to the existing Plant 1 ball mill circuit as part of P5M includes the installation of a new cyclone underflow splitter box feeding to the gravity concentrating circuit, a larger cyclone overflow pipeline, as well as the modification of the current cyclone cluster internals to ensure the correct cut-point is achieved.

The existing gravity recovery circuit will be upgraded to include a total of 3 gravity concentrator units, (as opposed to the two units currently installed). Space provision is further made for a fourth unit if required in future. Each of the units will be equipped with a horizontal vibrating, feed scalping screen upfront, to remove any debris from the feed. Scalping screen oversize material combines with the gravity concentrator tailings, and bypass stream. The gravity concentrators are operated on a semi-batch basis, with periodic recovery of the coarse, high SG concentrate being gravity fed to an expanded gravity dissolution circuit.

Provision is further made for the modification of the existing cyclone underflow box to allow for the feed distribution to the additional gravity concentrator units, as well as the expansion of the existing Plant 1 gravity dissolution circuit from a single ILR unit to two ILR units.

17.2.4.6 Plant 2 Ball Milling, Classification, Gravity Concentration and ILR circuits

The installation of the Plant 2 milling circuit will be conducted as part of P10M.

The Plant 2 ball milling circuit will treat material at a design feed rate of 625 dtph. The mill will operate in closed circuit with a classification cyclone cluster. The in-mill density will be controlled through the addition of process water to the mill hopper. A horizontal vibrating screen installed on the mill discharge sump will separate scats which will be discharged to a bunker for collection. The screen undersize material will be collected in the mill discharge sump before being pumped to the classification cyclone cluster.

The cyclone overflow product (P_{80} 106 µm) will gravitate to the Plant 2 pre-leach thickening circuit via a trash screening and two stage sampling installation, similar to the Plant 1 design.

The full cyclone underflow stream will report to the Plant 2 gravity concentration circuit, which will consist of 4 gravity concentrator units. Each of the units will be equipped with a horizontal vibrating, feed scalping screen upfront, to remove any debris from the feed. The gravity concentrators are operated on a semi-batch basis, with periodic recovery of the coarse, high SG concentrate being gravity fed to a single ILR unit.

Scalping screen oversize material will be combined with the gravity concentrator tailings and will report back to either the secondary ball mill feed or the mill discharge sump, depending on the in-mill density and water requirement around the circuit. The flow to each centrifugal concentrator unit will be controlled with a weir type cyclone underflow box and gate valve arrangement.

A mill liner handler and hydraulic hammer system have been included in the design to facilitate mill maintenance. Mill spillage will be pumped to the mill discharge screen for protection of the mill discharge pumps. Steel balls will be loaded into a ball loading kibble from drums using a forklift and added to the Plant 2 mill feed conveyor using a vibrating feeder.

17.2.4.7 Plant 1 Pre-Leach Thickening

The Plant 1 secondary ball mill classification cyclone overflow stream gravitates to a horizontal vibrating trash removal screen, to remove any miss-reporting coarse ore particles, wood fragments, organic material and plastics that would otherwise become locked up with the circuit carbon and block the CIL inter-tank screens. The trash screen oversize reports directly to a trash bin, whilst the underflow reports to the pre-leach thickener, via a two-stage sampling system.

The pre-leach thickener is a high rate thickener that produces an underflow product between 50% to 60% solids (w/w). The thickened underflow slurry will be pumped to the Plant 1 CIL circuit by means of an upgraded underflow pumping installation. The existing duty / standby pumping arrangement will be upgraded with larger motors during P5M. The existing thickener underflow pipeline to the CIL circuit will be upgraded to allow for the increase in throughput.

The thickener overflow product will gravitate to a new mill circuit water tank (to be installed in P5M).

Flocculant and lime is added to the circuit.

17.2.4.8 Plant 2 Pre-Leach Thickening

The installation of the Plant 2 pre-leach thickening circuit will be conducted as part of P10M.

As per the Plant 1 circuit design, the Plant 2 ball mill classification cyclone overflow stream will gravitate to a horizontal vibrating trash removal screen. The trash screen oversize will report directly to a trash bin, whilst the underflow will report to the Plant 2 pre-leach thickener, via a two-stage sampling system.

The Plant 2 pre-leach thickener will be designed to produce an underflow product between 50% to 60% solids (w/w). The thickened underflow slurry will be pumped to the Plant 2 CIL circuit by means of a duty / standby pumping arrangement.

The Plant 2 thickener overflow product will gravitate to mill circuit water tank for re-use as dilution water in the milling circuit.

Flocculant and lime will be added to the circuit as per the current operation.

17.2.4.9 Mill Circuit Water System

The Plant 1 and Plant 2 pre-leach thickening overflow products will be collected in a common mill circuit water tank, from where it will be circulated back to the milling circuits as dilution water via dedicated mill circuit water duty pumps. A common standby pump will be provided between the two milling circuits.

The installation of the mill circuit water system will be done as part of P5M.

Provision will be made to bypass this system if required, by transferring the pre-leach thickener overflow products to the process water circuits, as per current operation.

17.2.4.10 Plant 1 Leach and Carbon Adsorption Circuit

The existing leach and carbon adsorption circuit consist of a, mechanically agitated, pre-oxidation conditioning tank, followed by seven mechanically agitated CIL adsorption tanks, to provide a total leach time of 16 hours when processing 5 Mtpa.

The Plant 1 pre-leach thickener underflow is pumped into the CIL feed distribution box by means of a duty / standby pumping installation. The underflow material gravitates from the CIL feed distribution box to the pre-oxidation tank with a bypass option to enter the first adsorption tank, generally for maintenance purposes.

Currently, two pre-oxidation reactor pumps are operating in a duty / standby configuration to withdraw a recycle stream of slurry from the base of the pre-oxidation tank and pump it through two pre-oxidation reactors; also operating in a duty / standby configuration in current operations. This configuration will change to a two running configuration when processing 5 Mtpa, with a standby unit to be kept at the stores.

The seven adsorption tanks is connected with launders, and slurry flows by gravity through the tank train, while the carbon is transferred counter-current to the gold bearing slurry via carbon transfer pumps.

Oxygen (90% purity) from the Pressure Swing Absorption (“PSA”) plant is sparged into the pre-oxidation system and the first three CIL tanks, with provision for distribution to the remainder of the CIL tanks. Oxygen is sparged via lances into each of the CIL tanks. Each CIL tank is fitted with a vertical, mechanically swept wedge wire inter-tank screen to retain the carbon.

Each of the CIL tanks are further fitted with bypass facilities to allow any tank to be removed from service for maintenance.

Fresh and regenerated carbon is returned to the circuit at CIL tank 7 from where it is advanced counter-current to the slurry flow by pumping slurry and carbon from tank 7 to tank 6 to tank 5 and so on. The inter-tank screen in tank 6 will retain the carbon while slurry will flow by gravity back to tank 7. This counter current process is repeated until the carbon reaches the first CIL tank. Carbon is advanced at a rate of up to 10t/day.

A recessed impeller pump is used in each tank to transfer the slurry up the leach train and ultimately to the loaded carbon recovery screen. These CIL tank interstage screens will be upgraded during P5M to cater for the increased flow rate through the circuit.

Slurry from the final CIL tank gravitates to the linear, carbon safety screen (800 µm aperture cloth) to recover fine carbon from the tailings. The safety screen oversize is collected, while the underflow gravitates to the Plant 1 CIL tailings detox circuit.

17.2.4.11 Plant 2 Leach and Carbon Adsorption Circuit

The installation of the Plant 2 leach and carbon adsorption circuit is planned as part of P10M. The Plant 2 CIL circuit will be based on a similar design to the upgraded Plant 1 CIL installation, with the inclusion of an intermediate transfer station due to lay-out constraints.

As per Plant 1, the Plant 2 CIL circuit will consist of a pre-oxidation conditioning tank followed by seven CIL adsorption tanks, to provide a total leach time of 16 hours.

The Plant 2 pre-leach thickener underflow will be pumped to the Plant 2 CIL feed distribution box by means of a duty / standby pumping installation. The underflow material will gravitate to the pre-oxidation tank with a bypass option to enter the first adsorption tank, generally for maintenance purposes.

Provision will be made for two suitably sized pre-oxidation reactor pumps (duty / standby) to withdraw slurry from the base of the pre-oxidation tank and pump it through two suitably sized pre-oxidation reactors (duty / standby).

The seven adsorption tanks will be connected with launders so that slurry will flow by gravity through the tank train, while the carbon will be transferred counter-current to the gold bearing slurry via suitably sized carbon transfer pumps.

Oxygen (90% purity) from the Plant 2 Pressure Swing Absorption ("PSA") plant will be sparged into the pre-oxidation system and the first three CIL tanks, with provision for distribution to the remainder of the CIL tanks. Oxygen sparging will be conducted as per the Plant 1 design. Bypass facilities will be included for each tank as per the Plant 1 design.

Carbon will be advanced at a rate of up to 10t/day. Recessed impeller pumps will be provided to transfer the slurry up the leach train and ultimately to the Plant 2 loaded carbon recovery screen. Slurry from the final CIL tank will gravitate to the Plant 2 carbon safety screen to recover fine carbon from the tailings.

17.2.4.12 Plant 1 CIL Tailings Detoxification and Disposal

The tailings from the Plant 1 CIL circuit gravitates to the cyanide detoxification circuit. The detoxified tailings is combined with other process tailings and spillage streams before being pumped to the TSF.

As per EPA guidelines, the CIL tailings needs to be discharged with a final cyanide concentration of <50 g CN_{WAD}/m^3 at the TSF spigot.

Provision has been made in the Plant 1 design to allow for the use of the INCO process as means of cyanide detoxification, when required. The current cyanide detoxification circuit consists of a cyanide destruction feed box, gravity feeding into a single agitated tank, with a blower air sparging facility.

The detoxification process utilises SO_2 and air in the presence of a soluble copper catalyst to oxidise cyanide to the less toxic compound cyanate (OCN). Provision is made for SMBS as a SO_2 source, to be dosed into the cyanide destruction feed box as a 20% w/v solution, when required. The detox process further requires the presence of soluble copper to act as a catalyst and to ensure that any free cyanide present is bound to copper as a WAD cyanide. Provision is made for the preparation and dosing of a copper sulphate solution, for dosing to the cyanide destruction feed box as a 15% w/v solution when required. Oxygen required in the reaction will be supplied by sparging of blower air into the cyanide detoxification tank, when needed. The reaction is carried out at a pH of 8.0 which will be maintained by controlled lime addition to the cyanide destruction feed box, if required.

It is noted that the current operating conditions of the Plant 1 CIL circuit does not require the cyanide detox circuit to be operated, as the final cyanide concentration at the TSF spigot is <50 g CN_{WAD}/m^3.

The detoxified CIL tailings gravitates to the CIL tailings disposal tank via a two stage sampling system, where it is combined with the fine carbon from the fine carbon screen, eluted carbon screen undersize, cyanide destruction area spillage, ferric chloride area spillage, and dilution water.

Currently, the circuit tailings from the Plant 1 is pumped to the TSF by means of a duty / standby pump train installation, consisting of three pumps per train. In order to process 5 Mtpa, this system will be upgraded in P5M to allow for a third pump train consisting of three pumps, feeding into a dedicated pipeline to the TSF. This will allow the tailings disposal system to operate in a 2 duty / 1 standby configuration. The last pump in each existing train will further be upgraded with a larger motor.

17.2.4.13 Plant 2 CIL Tailings Detoxification and Disposal

The installation of the Plant 2 tailings detoxification and disposal systems are planned as part of P10M, and will be based on the Plant 1 design where feasible.

The tailings from Plant 2 CIL circuit will gravitate to the Plant 2 cyanide detoxification circuit. The detoxified tailings is combined with other process tailings and spillage streams before being pumped to the TSF via a dedicated disposal system.

Provision will be made in the Plant 2 design to allow for the use of the INCO process as means of cyanide detoxification, when required. The circuit will consist of a cyanide destruction feed box, gravity feeding into a suitably sized single agitated tank, with a dedicated Plant 2 blower air sparging facility.

The Plant 1 detox reagent preparation and storage facilities will be used.

The Plant 2 detoxified CIL tailings will gravitate to the Plant 2 CIL tailings disposal tank via a two stage sampling system, where it will combine with the fine carbon from the fine carbon screen, eluted carbon screen undersize, cyanide destruction area spillage, and dilution water.

The Plant 2 circuit tailings will be pumped to the TSF by means of a duty / standby pump train installation, consisting of three pumps per train transferring slurry to the TSF in a single tailings disposal pipeline.

17.2.4.14 Plant 1 Elution Circuit

Loaded carbon from the CIL circuit undergoes an acid wash to remove precipitated material (inorganic and organic) prior to elution to prevent contamination of the eluate and to restore carbon activity. Adsorbed gold is eluted from the activated carbon by means of a heated solution of sodium cyanide and caustic soda via the split AARL procedure. This elution process is followed by rinsing and cooling stages. Barren carbon from the batch elution process is directed to carbon regeneration circuit while PLS is routed to electrowinning.

The CIL carbon is batch treated in an elution circuit consisting of an acid wash hopper and a single 5t elution column with a heater facility. The CIL carbon is treated in 5t batches up to twice every 24 hours.

Acid Wash

The loaded carbon is dewatered on a 0.63 mm x 8 mm aperture vibrating, loaded carbon screen prior to reporting to the acid wash hopper. The loaded carbon screen oversize product (loaded carbon) gravitates into the acid wash hopper, while the undersize stream (slurry and wash water) gravitates back to the CIL pre-oxidation condition tank.

In the acid wash, every batch of carbon is treated to remove carbonated material by circulating a 3% hydrochloric acid solution through the carbon. 33% (w/v) Hydrochloric acid is pumped from the hydrochloric acid bulk container into the dilute acid make-up tank, where it is diluted to 3% w/v by adding filtered raw water. The diluted hydrochloric acid is pumped from the dilute acid make-up tank to the acid wash hopper in an up-flow direction to remove contaminants, predominantly carbonates, from the loaded carbon. This process improves the elution efficiency and has the beneficial effect of reducing the risk of calcium magnesium ‘slagging’ within the carbon during the regeneration process. The acid wash is performed for 30 minutes at ambient temperature. Following the acid wash, the carbon is soaked in the acid solution for a further period if required.

Rinse

The loaded carbon is rinsed with filtered raw water, for a duration of 120 minutes, to remove any residual acid from the carbon, which can interfere with the elution process.

Drain

Following the rinse cycle, the dilute acid and rinse water is drained from the acid wash hopper, to the acid wash effluent tank from where the spent acid is pumped to the tailings cyanide detoxification circuit, and is neutralised before being pumped to tailings.

Carbon Transfer

The loaded carbon is gravity fed, with the aid of transfer water, from the acid wash hopper to the 5t elution column for elution. The carbon transfer duration is typically done within 25 minutes.

Elution

The 5t elution column is constructed from stainless steel and has been designed to desorb gold from the loaded carbon at 125°C and a column pressure of 300 kPa. Once the carbon transfer from the acid wash hopper has been completed, the operator completes all of the relevant pre-start checks before initiating the elution sequence.

The caustic solution is pumped into the strip solution make-up tank from the caustic mixing tank and the cyanide solution is pumped from the cyanide dosing tank. The reagents are mixed with filtered raw water in the strip solution make-up tank to achieve the correct reagent concentrations. When the elution column is filled, the strip solution pump turns on and pumps the strip solution through the recovery heat exchangers followed by the primary heat exchangers before entering the bottom of the elution column at 125°C. The strip solution is recycled through the column via the strip solution pump, at a flow rate of 2 BV/h, for a total of 50 minutes to affect a strip. Eluate produced during the elution cycle is pumped to either one of the two eluate storage tanks located in the electrowinning area.

The Fresh strip solution cycle is followed by a spent solution cycle. During this cycle, the rinse solution from the previous elution (stored in the intermediate solution tank) is circulated through the elution column at 125°C a rate of 2 BV/h for 150 minutes. Once the cycle is complete, the spent solution is pumped to either one of the two eluate storage tanks.

Following this, the rinse cycle involves pumping water for 150 minutes at a rate of 2 BV/h through the elution column and storing the resulting solution in the intermediate solution tank for the spent solution cycle in the subsequent elution cycle. On completion of the elution cycle, cooling water will be pumped from the intermediate solution tank, through the elution column at a rate of 2 BV/h for 30 minutes and report to the CIL circuit.

Eluted carbon is removed from the elution column and transferred to the carbon regeneration kiln via the static sieve bend eluted carbon drainage screen, by means of pressurised water flow. Drained carbon gravitates to the carbon regeneration kiln feed bin from where it is fed to the carbon regeneration kiln. The regenerated carbon is collected in the barren carbon quench tank, from where it is pumped to the carbon dewatering screen for re-introduction into the CIL circuit.

No modification is envisaged for the Phase 1 elution circuit.

17.2.4.15 Plant 2 Elution Circuit

The Plant 2 elution circuit will be constructed as part of P10M, and will be based on the Plant 1 elution circuit design as discussed in Section 17.2.4.14.

17.2.4.16 Plant 1 Electrowinning and Gold Room Operations

Currently the PLS from the Plant 1 ILR is collected in the ILR pregnant solution storage tank. This pregnant solution is circulated through two dedicated electrowinning cells via a common steady head tank.

Pregnant solution from the Plant 1 elution circuit is collected in either one of the two eluate storage tanks. This solution is circulated through a dedicated electrowinning circuit consisting of four cells operating in parallel via a common steady head tank.

Gold is deposited on the electrowinning cell cathodes as a sludge while the solution is circulated until the desired barren gold concentration is achieved, or the cycle time has elapsed. After completion of an electrowinning cycle, barren solution is sampled before being pumped to the CIL feed circuit for disposal. Loaded cathodes are removed periodically from the cells, the gold sludge is washed off using a high pressure washer after which the washed solution is decanted. The gold sludge is treated in a drying oven before being mixed with fluxes and loaded into an induction smelting furnace. During smelting, metal oxides form slag and once the furnace crucible contents are poured into cascading moulds, gold will solidify at the bottom while slag separates easily from the gold. The gold bullion bars are cleaned, labelled, assayed and prepared for shipping. Slag is manually crushed, pulverized, and recycled to the SAG milling circuit together with the de-canted waste water from the gold room.

Hydrogen cyanide, ammonia, and hydrogen gas detection is installed, together with extraction systems, safes, scales and various security systems.

Changes to the gravity gold PLS electrowinning area in P5M includes the following:

- Modification to allow PLS from the second Plant 1 ILR to be collected in the existing ILR pregnant solution storage tank
- Upgrading of the two of the ILR electrowinning circuit feed pumps' suction and delivery piping to cater for a higher PLS circulating rate
- The existing ILR steady head tank feeding the ILR electrowinning cells will be replaced with an equal flow splitter box to ensure the incoming feed to the box is equally divided between the ILR electrowinning cells. The existing electrowinning cell feed manifold piping will be replaced with dedicated feed lines from the flow splitter box to each electrowinning cell, to ensure the maximum allowable flow to each cell is achieved
- Upgrading of the PLS return piping back to the ILR pregnant solution storage tank to cater for the increase in volumetric flow
- During the above piping upgrades, provision will be made for 4 electrowinning cells to be fed from the ILR equal flow splitter box

Changes to the CIL eluate electrowinning area in P5M includes the following:

- Upgrading of two of the existing eluate electrowinning circuit feed pumps to larger pumps, motors, and VSDs to cater for a higher eluate circulating rate in order to achieve 10 hour electrowinning cycles (required when Plant 2 is on-line). Provision is also made for the replacement of the current eluate electrowinning feed pump suction and delivery piping to cater for the increase in flowrates
- The existing eluate steady head tank feeding the eluate electrowinning cells will be replaced with an equal flow splitter box to ensure the incoming feed to the box is equally divided between the eluate electrowinning cells. The existing electrowinning cell feed manifold piping will be replaced with dedicated feed lines from the flow splitter box to each electrowinning cell, to ensure the maximum allowable flow to each cell is achieved
- Extension of the electrowinning building for the addition of a 12 cathode, 14 anode electrowinning cell, to be employed in the eluate electrowinning duty
- During the above piping upgrades, provision will be made for 4 electrowinning cells (3 off 6, 7 unit plus 1 off 12, 14 unit) to be fed from the eluate equal flow splitter box
- Upgrading of the eluate return piping back to the two eluate storage tanks to cater for the increase in volumetric flow

During P10M the electrowinning circuit will further be upgraded to include the following equipment, to be utilised by Plant 2:

- Two eluate storage tanks to receive CIL eluate from Plant 2
- Two (duty / standby) eluate electrowinning circuit feed pumps
- Two Plant 2 eluate electrowinning cells and rectifiers (12 cathode, 14 anode 1500A units)
- Dedicated equal flow splitter box to feed Plant 2 eluate electrowinning cells
- Associated piping

17.2.4.17 Reagents

Flocculant

Flocculant is delivered to site dry in 25 kg bags, and is added manually to the flocculant hopper. The flocculant is then fed into a venturi tube by a screw feeder, where it is pneumatically transferred into a wetting head. The dry flocculant is mixed with filtered raw water up to a 33% (w/v) solution and discharged into the flocculant mixing tank. After a suitable hydration period, the flocculant is pumped to the flocculant storage and distribution tank, from where it is dosed to the respective areas by means of a ring main system fed via a duty / standby variable speed pumping arrangement.

During P10M, a second storage tank will be included together with a flocculant dosing pump dedicated to the Plant 2 ILR and pre-leach thickener.

Copper Sulphate

The current installation allows for the delivery of copper sulphate in 1.25t bulk bags, and manual addition to the mixing tank using a hoist and a bag breaker system. Provision is made for the addition of filtered raw water to the mixing tank to dilute the copper sulphate to a 15% (w/v) solution. The copper sulphate solution would gravitate from the mixing tank to the dosing tank, from where it can be dosed directly to the Plant 1 CIL tailings cyanide detoxification circuit via a duty / standby variable speed pumping arrangement when required.

Copper sulphate spillage is pumped to the CIL tailings cyanide detoxification circuit.

During P10M, provision will be made for the addition of a third dosing pump to service the Plant 2 CIL tailings detoxification circuit. The existing standby pump would be shared between the dosing systems for Plant 1 and Plant 2.

Sodium Metabisulphite

The existing installation allows for the delivery of SMBS in 1.2t bulk bags and manual addition to the mixing tank using a hoist and a bag breaker system. Provision is made for filtered raw water addition to the mixing tank to dilute the SMBS to a 20% (w/v) solution. When required, the diluted SMBS solution will be pumped from the mixing tank to the dosing tank, from where it will be dosed directly to the cyanide detoxification circuit and RO plant via a duty / standby variable speed pumping arrangement.

No modifications to the existing pumping system is planned as part of the AGM DFS, other than the tie-in of the Plant 2 piping into the existing distribution network.

Diesel

Diesel is delivered to the plant site by the fuel tanker and stored in a diesel storage tank for distribution to the fire water system, elution circuit and the gold room.

Provision will be in P10M to allow the supply of diesel from the existing system to the second elution plant.

Caustic Soda

Caustic is delivered to site in 1t bags of ‘pearl’ pellets. Caustic soda bags are hoisted by a crane into the mixing tank via a bag breaker system. The caustic soda is diluted with filtered raw water up to a final solution concentration of 20% (w/v). The diluted caustic solution is pumped from the mixing tank to the dosing tank, from where it is dosed to the respective areas (ILR, elution, and electrowinning) by means of a duty / standby variable speed pumping installation.

No modifications to the existing pumping system is planned as part of the AGM DFS, other than the tie-in of the Plant 2 piping into the existing distribution network.

Sodium Cyanide

Sodium cyanide is delivered as dry briquettes in 1t boxes, and is added manually via a hoist and bag breaking system into the mixing tank. Filtered raw water is used to prepare a 20% (w/v) solution in the mixing tank. The diluted solution is pumped from the mixing tank to the dosing tank, from where it is dosed to the respective areas by means of dedicated variable speed dosing pumps.

During P10M, provision will be made for the addition of a third dosing pump to service the Plant 2 CIL circuit. The existing standby pump would be shared between the dosing systems for Plant 1 and Plant 2.

Hydrated Lime

Hydrated lime is delivered dry in 1t bulk bags, and is manually loaded to the lime make-up tank via a hoist and bag breaker system. The hydrated lime is fed into the lime make-up tank by means of a screw feeder. Filtered raw water is added to the make-up tank to produce a 20% (w/v) solution. The diluted milk of lime is distributed throughout the plant by means of a ring main system fed by a fixed speed duty / standby pumping installation.

No modifications to the existing pumping system is planned as part of the AGM DFS, other than the tie-in of the Plant 2 piping into the existing ring main network.

Ferric Chloride

Ferric chloride is delivered in 25 kg bags which are manually loaded via a hoist and bag breaking system into the mixing tank. Filtered raw water is added to the mixing tank to prepare a 20% (w/v) solution. The diluted solution is dosed directly from the mixing tank to the return water treatment circuit, by means of a variable speed, duty / standby pumping installation.

No modifications to the existing system is planned as part of the AGM DFS.

Hydrochloric Acid

Hydrochloric acid is delivered to site in 1,000L bulk containers at a solution strength of 33% w/v.

Quicklime

Quicklime is delivered in 36t bulk tankers and pneumatically off-loaded from the tanker into the lime silo. The lime is extracted from the silo using a variable speed screw feeder, and dosed directly onto the mill feed conveyor.

Provision has been made in P10M for the inclusion of a dedicated dosing system to service Plant 2.

Anti-Scaling Agent

The anti-scaling agent is delivered in 1t intermediate bulk containers, from where it is pumped to the de-scalant storage tank. The de-scalant reagent is pumped from the storage tank, through the elution heat exchangers, back to the storage tank.

Activated Carbon

Fresh activated carbon is delivered in 500 kg bulk bags. The Fresh carbon is added to the carbon quench tank using a hoist, as required for carbon make-up to the CIL inventory. The addition point will allow attrition of any friable carbon particles with subsequent fines removal on the sizing screen prior to entering the CIL tanks.

Grinding Media

15% Chrome, forged steel, 125 mm grinding media is being used in the SAG mill, while 60 mm grinding media is used in the secondary ball mill.

Grinding media is delivered in 200 kg drums. SAG mill balls are added to the mill using a hydraulic ball feeder which discharges directly onto the mill feed conveyor. Secondary ball mill media is added to the ball mill feed box by use of a specially designed kibble and hoist, which safely transports the media from the loading area to the feed box.

Provision will be made in P10M for the storage of 90 mm grinding media, required for the Plant 2 ball mill.

17.2.4.18 Plant Process Services

Filtered Raw Water

Raw water is currently supplied to the Plant 1 raw water storage tank from the pit dewatering boreholes and a number of borehole pumps. Additional raw water is sourced from the Adubiaso pit and pumped to the Plant 1 raw water tank, via the raw water treatment plant.

Provision has further been made on site to route tailings return water to the Plant 1 raw water storage tank via the discharge water treatment settling and RO plant.

The raw water is used for gland service, carbon transfer duties, elution, gravity concentrator circuit water, and reagent make-up.

A dedicated Plant 2 raw water storage tank will be installed as part of P10M, from where filtered raw water to the Plant 2 circuit gland seal water and spray water will be supplied by means of a duty / standby pumping installation. A second duty / standby pumping system will be provided to supply raw water primarily to the Plant 2 elution and ILR circuit.

The raw water storage tank has a reserve of water for firefighting purposes. This reserve is maintained by suitability positioned fire water and raw water pump suctions.

Fire Water

Firewater is drawn from the raw water tank. The firewater pumping system contains:

- An electric jockey pump to maintain fire water ring main pressure
- An electric fire water delivery pump
- A diesel driven fire water pump that automatically starts in the event that power is unavailable for the electric firewater pump

Fire hydrants and hose reels are placed throughout the process plant, fuel storage and plant offices at intervals that ensure coverage in areas where flammable materials are present.

No provision has been made in the AGM DFS to upgrade the existing fire water storage and pumping installation. Provision has been made to tie-in to the existing ring main system for distribution of fire water to Plant 2.

Potable Water

Potable water is taken from the borehole water line. It is pumped through a water treatment unit before being stored in the potable water tank. The potable water tank feeds the plant potable water tank, from where the plant and mining potable water is distributed. The contractors' camp potable water tank is also be fed from the potable water tank, and supplies potable water to the villages.

Provision is made in P10M for modification of the plant potable water distribution system to deliver potable water to Plant 2.

Process Water & Plant Run-off

Plant run-off is contained in the pollution control dam, from where it is pumped to the Plant 1 process water dam.

The Plant 1 process water dam collects the product from the pit-dewatering pumps, TSF return water, and any plant run-off from the pollution control dam. Provision is made for a raw water make-up stream, as required. Filtered water to the Plant 1 gravity concentration circuit is supplied by a dedicated pump system, while the remainder of the process water reticulation is done by means of a duty / standby pumping arrangement. Provision is made in the design for the treatment of excess process water prior to discharge to the environment. Other than the inclusion of the mill circuit water system discussed in Section 0, no modifications are envisaged to the Plant 1 process water circuit during the AGM DFS.

A dedicated Plant 2 process water circuit, similar to the Plant 1 circuit, will be installed during P10M. Raw water to the Plant 2 gravity recovery circuit will be supplied by a dedicated pump while the process water requirements to other Plant 2 areas will be supplied by a duty / standby process water pump installation. Raw water make-up to the Plant 2 process water circuit will be provided.

Discharge Water Treatment

The Plant 1 design allowed for the treatment of the excess process water in a mechanically agitated arsenic precipitation tank, where ferric chloride would be dosed to precipitate out arsenic from solution (in the presence of oxygen), at a pH of 6. Provision was made for lime addition and hydrochloric acid addition to this tank, as required for pH control. The current design allows for the treated water to overflow to an intermediated transfer tank, from where it is pumped to a RO water treatment plant, complete with pre-filters, prior to discharge. Filter cake product from the RO plant filters will be re-pulped with brine in the arsenic waste disposal tank, from where it will be pumped to the final tailings disposal circuit.

High Pressure (Compressed) Air Reticulation

Plant 1 instrument and plant air at 8Bar pressure are supplied by a dedicated, duty / standby compressor installation. The compressed air is stored in the instrument air receiver while a dedicated air receiver, located in the milling area, is provided for plant air storage, and is fed from the main instrument air receiver.

All the air is dried and filtered prior to storage in the instrument air receiver, from where it is reticulated throughout the plant for instrument air requirements.

Provision is made for the installation of the similar plant and instrument air supply system for Plant 2 as part of P10M.

Low Pressure (Blower) Air Reticulation

A total of three low pressure blowers supply air to the Plant 1 tailings detoxification circuit and water treatment circuits.

Provision is made for the installation of two more low pressure blowers to supply air to the Plant 2 CIL tailings detoxification circuit as part of P10M. A common standby unit will be shared between Plant 1 and Plant 2.

Oxygen Reticulation

Plant 1 currently utilises a 10 tpd oxygen plant (comprising 2 modules of 5 tpd each) to generate oxygen at 90% purity and 300 kPa pressure, for use in the ILR, pre-oxidation, CIL, and arsenic precipitation circuits. The oxygen is stored in the oxygen plant air receiver from where it is distributed.

The existing system will be upgraded during P5M to include a third 5tpd module to increase the Plant 1 PSA capacity to 15tpd.

Further to the above, a second 15 tpd oxygen plant (comprising 3, 5 tpd modules) will be installed as part of P10M to service Plant 2.

17.2.4.19 Return Water and Return Water Treatment

Currently, TSF return water is pumped from the TSF via a duty / standby pumping installation to the Plant 1 process water circuit. Site personnel has also made provision for the routing of the TSF return water to the discharge water treatment and RO plant for treatment prior to discharge to the Plant 1 raw water tank, to supplement the raw water requirement.

Provision has been made in the Plant 1 design to allow for the treatment of excess water during net-positive conditions. Refer to Section 17.2.4.18 for details.

The existing circuit will be upgraded as part of P10M to include a third return water pump and second return water pipeline, to allow the system to operate in a 2 duty / 1 standby configuration as soon as Plant 2 is on-line.

17.2.4.20 Esaase Mine Site Services

Provision has been made in P5M for run-off water collection at the Esaase pit area, and diversion to a new storm water dam, from where it will be pumped to the intermediate head balancing dam using a submersible pump. The mine curtain dewatering product will also be pumped to the intermediate head balancing dam via a number of borehole pumps (some of which will be installed in P5M with the remainder only to be installed in P10M). The water quality in the pit balancing dam will be checked on a continuous basis to determine arsenic levels in the dam. In the event that the arsenic levels are below the EPA requirements, water will be released directly to the environment; however, if the arsenic levels are not met the water will be transferred to the buffer dam, using a duty / standby pumping installation. Water stored in the buffer dam will be utilised for dust suppression purposes. Excess water will be pumped to a water treatment plant which will make use of the NXT-2 arsenic removal media.

The Esaase water treatment system will utilise NXT-2 arsenic removal media. The design will include the conditioning of the contaminated water in an agitated tank, after which the conditioned water will overflow to a second mixing tank where the NXT-2 media will be added manually. The second mixing tank will be suitably sized to allow for the required residence time prior to overflowing to a transfer sump from where the treated water will be discharged to environment. The spent media will be pumped from the second mixing, to a dewatering screen, and the dewatered media will be collected and drummed prior to disposal.

Raw water sourced from boreholes will serve as raw water make-up supply to the Esaase raw water tank from where it will be used as supply to the Esaase potable water plant. The potable water plant product will be transferred to an elevated storage tank, from where the potable water will be distributed.

Provision is made at the Esaase area for fire water storage and distribution.

The above facilities are planned in P5M with the exception of the buffer dam which will only be installed during P10M, once the sulphide material is being mined.

18 PROJECT INFRASTRUCTURE

18.1 Phase 1 - Existing Infrastructure and Services

Current site infrastructure at Obotan consists of:

- A Clients administration block, training facilities, exploration offices, core storage area, clinic and laboratory
- Senior and Junior accommodation facilities located to the west of the Obotan mine
- An exploration camp and office at Esaase
- An established mining operation at Obotan with various structures like offices, stores, workshops and fuel storage facilities
- A new CIL Plant at Obotan (3.6 Mtpa capacity) with various structures like offices, stores, workshops and reagent buildings
- A TSF
- Multiple boreholes sunk for water supply
- A 161 kV incoming power line fed from the Asawinso sub-station
- Communications currently available at the site are good due to the erection of an additional Vodafone tower at the Obotan camp.

18.2 Roads and Site Access

18.2.1 General

A range of road types will be required within the project site to meet a wide range of duties. The hierarchy of road types includes dedicated mine haul roads, main access roads, general access roads and minor use roads and tracks. Some of the roads will border service corridors and therefore road alignments need to consider service routes in addition to transport requirements.

18.2.2 Project Access

Existing road access to the site is available from the west, south and east, but the main access used will be from the ports of Tema and Takoradi to the south via Kumasi, or Obuasi. The total distance from Tema to the project site, via Kumasi is approximately 400 km.

The Esaase site is accessed by existing public roads from two directions:

- Kumasi / Sunyani road to the north-east (sealed road)
- Kumasi / Obuasi road to the south (sealed road)
- Roads from both directions become gravel topped, (fair to poor condition) for the last 20 km to the Esaase site.

The Obotan Plant is also accessed by existing public roads:

- Kumasi / Manso Nkwanta road (sealed road)
- With the last 12 km on the private mine road

18.2.3 Re-locate Public Roads (Esaase – P10M)

The existing main road at Esaase will be affected by pit development, and will require re-alignment. Sections of the road between Manhiya and Tetkaaso, as well as between Tetkaaso and Mpatoam will be included in the re-alignment.

Development of the WRDF that ultimately covers the village of Tetrem will result in the public road being shifted between the WRDF and the 500m pit basting radius. This road will be suitably upgraded to ensure safety and usability for locals and mine vehicles, (light and supply vehicles only).

Based on the current pit and waste rock dump configuration, it is anticipated that the re-aligned public road will therefore cross the haul road to the Tetrem waste rock dump. A bridge will be constructed to limit the interaction between heavy mine vehicles and public road users.

18.2.4 Plant Access Road (Obotan)

The existing plant roads will be utilised and no additional roads will be required.

18.2.5 Haul Roads (Esaase)

The haul road design for the project is based on a cross section approximately 20m wide, (including drains and shoulders) and allows for dual lane traffic, as well as road drains. Safety berms and barriers, (in line with legislated requirements), will be in addition to this cross section.

Mine haul roads will be designed and constructed by the mining contractor.

18.2.6 Service Roads

Conveyor belts and pipelines will be serviced via a gravel track running alongside the facility. This will be constructed by clearing existing vegetation, ripping and compacting the in-situ material for a 3m wide cross-section.

Where available the existing exploration, community and district roads will be rehabilitated for construction road access whilst formal road construction is still underway.

18.3 Power

18.3.1 Power Supply

Power to the existing Obotan Plant is generated by the Volta River Authority ("VRA") and transmitted from the Asawinso sub-station via a 161 kV overhead line, owned and operated by GRIDCo. The capacity of the overhead line feeding the plant is 150 MW, which far exceeds the estimated power requirements for both P5M and P10M.

A memorandum of understanding is currently in place between Asanko Gold and GRIDCo for the supply of power to the existing AGM Plant. This memorandum states that the existing Asanko AGM Plant has an "estimated maximum demand of 18.6 MW" and an "estimated average demand of 14 MW".

Based on the current load estimates, the maximum demand, as stated in the above memorandum would be exceeded when P5M comes online.

VRA have been informally contacted regarding the increases in Notified Maximum Demand ("NMD") for both P5M and P10M and have indicated that they have available generating capacity to meet the total DFS additional demand.

Currently a 25/33MVA ONAN/ONAF 161 kV / 11.5 kV Transformer installed in the GRIDCO Obotan 161/11 kV Outdoor Yard receives power from the 161 kV overhead line.

The 161 kV / 11.5 kV transformer has sufficient capacity to supply the additional P5M demand. Depending on final demand levels the transformer might however be required to supply more than 25 MVA and thus operate with forced cooling (ONAF) to increase its capacity to 33 MVA.

It is anticipated that a 33 MVA demand will be exceeded when P10M comes on line. This will require the installation of an additional 25/33MVA ONAN/ONAF 161 kV / 11.5 kV transformer in the spare transformer bay of the outdoor yard.

The DFS phases medium voltage distribution design on the Obotan site follows the same philosophy that was used for Phase 1. 11 kV is distributed via cable to Motor Control Centres ("MCCs"), servicing particular plant areas from an 11 kV consumer sub-station, which is fed by the 161/11 kV outdoor yard.

The main busbars of the existing consumer sub-station 11 kV switchboard are sufficiently rated to supply power to both the Phase 1 and P5M loads. The existing consumer sub-station building does, however, not have sufficient space to house additional switchgear panels to feed the P5M plant loads. The 11 kV design therefore makes provision for connection of the existing 11 kV switchboard to a new 11 kV switchboard, with the same ratings as the existing switchboard (2500A, 11 kV, 31.5 kA), dedicated to P5M loads and housed in a separate building. This new 11kV switchboard will be extended for P10M loads. Part of the extension will include a new incomer to receive power from the additional 25/33MVA 161 kV /11.5 kV transformer mentioned previously.

The Esaase site, as well as the overland conveyor and any loads along the overland conveyor route, receive a 33 kV supply by means of a 33 kV overhead line between Obotan and Esaase. The overhead line follows the overland conveyor route and is fed from the new P5M 11 kV switchboard by means of a single feeder. The study phase design steps the voltage up to 33 kV using a 9 MVA, 11/34 kV transformer. The overhead line capacity is also specified at 8 MW.

Due to the physical separation of loads along the overland conveyor route and at Esaase, a central sub-station is not used for medium voltage distribution, as is the case at Obotan. 33 kV is tapped directly off the line using either Auto-Reclosers (“ARC”) or fused isolators to feed ground, or pole mounted step-down transformers respectively.

Late changes to the Process Plant design (conversion from Flotation Plant to CIL Plant in P10M) were not captured in the Electrical, Control and Instrumentation portions of this DFS. This is not expected to have an impact on power demand estimates, but may have implications on the budget and it is therefore recommended that this be reviewed in the FEED Phase of the project.

18.3.2 Estimated Loads

The power requirements for the DFS are shown below:

18.3.2.1 P5M

- The estimated maximum demand of the existing plant at Obotan will increase to 18.8 MVA (no PFC)
- The estimated maximum demand at Esaase is 2.7 MVA (no PFC)
- The total estimated maximum demand after completion of P5M is 21.5 MVA (no PFC)

18.3.2.2 P10M

- The estimated maximum demand of the existing plant at Obotan will increase to 30.8 MVA
- The estimated maximum demand at Esaase will increase to 5.7 MVA
- The total estimated maximum demand after completion of P10M is 36.5 MVA

18.4 Potable Water

18.4.1 General

Potable water for human consumption at Obotan and the Esaase exploration camp is currently being sourced from a local bore and is sufficient for the respective facilities requirements.

A new potable water network for the Esaase MSA area will be supplied from ground water boreholes.

It is proposed that potable water requirements for the construction and operational phases of the project will be similarly supplied by additional ground water boreholes. Ten community boreholes have been sunk by Asanko Gold up to 40m in the Bonte Valley for baseline monitoring of potential mining impacts on communities. This is to the base of the oxidation/transition zone. It is understood that half of these boreholes yield 3 l/s, or better.

Three boreholes have been sunk in locations on the northern Bonte Valley slope (away from mining activities) see Table 18-1 below.

Table 18-1: Locations of Drilled Potable Water Boreholes at Esaase (source: Asanko Gold Technical Report 2014)

Borehole ID	Location		Depth (m)	Yield (l/m)	Remarks
	East	North			
PWW 01	620228	725998	81	77.4	Located adjacent to Camp Accommodation
PWW 02	621686	726184	60	150.0	Located adjacent to Aboabo Community
PWW 03	622781	726593	50	800.0	Located north west of North Pit

Pump tests were undertaken on the three boreholes with the following summarized results:

- PWW 01 – Quality within Ghanaian Drinking Water standards (As< 0.002 ppm)
- PWW 02 – Slightly exceeded in as (0.011 ppm). Requires re-sampling to confirm results
- PWW 03 – Exceeds As limit (0.04 ppm) – Not suitable for drinking water
- PWW 01 is suitable for drinking water supply. Depending on the resampled test results for PWW 02, this may be acceptable for drinking water standards.
- PWW 03 is not considered suitable for drinking water requirements. It is anticipated that these locations will be able to deliver the projects required potable water demand.

18.4.2 Potable Water Treatment and Storage

The treatment and storage infrastructure for Obotan and the Esaase exploration camp is already in place.

A new potable water network will be installed for the Esaase MSA area. This supply line will feed a MSA potable water treatment plant. The treatment plant will purify the water from where it will be pumped to a centralised potable water storage facility with a 2 day storage capacity of 60 m^3. From there it will be gravitate fed to address the respective human consumption requirements of the mining operations.

18.4.3 Ground Water

A ground water investigation at Esaase was previously undertaken by Knight Piésold ("KP") to collect sufficient hydrogeological data to allow study design and costing of a production bore field. The investigation took place from April to June 2012 and included the drilling of eight investigation holes followed by the pump testing of six bores.

18.5 DFS Project - Site Raw / Process Water Balance

A water balance was conducted by KP to model process water supply scenarios using historical rainfall records for wet, dry and average rainfall scenarios. Given the change in overall site layout and process philosophy, the water balance was subsequently updated. This is described in their report, Asanko Gold Phase 2 Expansion – Feasibility Study Water Balance Modelling Update "PE301-00370/22" (2 December 2016).

The main outcomes of the report can be summarised as follow:

- There is sufficient storm water storage capacity in the TSF to accommodate all design storm events and rainfall sequences, with sufficient freeboard to comply with Ghanaian Mining Regulations, which require 1,0m freeboard over a 1 in 100 year recurrence interval, 24 hours storm event pond volume
- A recycle waste shortfall occurs during each year of operation under average conditions, resulting in a make-up water demand. The maximum annual make-up demand volume for average conditions is 4.0 Mm³ in Year 4, representing 33% of total annual plant water requirements.
- The maximum annual make-up demand for 1 in 100 year dry rainfall sequence is 5.0 Mm³ in Year 4, representing 40% of total annual plant water requirements
- There are water shortfalls in years 4 and 5 under average conditions
- The Esaase dewatering supply is an important component of the site water supply. If it were not included in the Nkran water balance, an alternative water supply would be required
- Process water shortfall occurs under design dry conditions in year 4 to 6 (2019 to 2021)
- This coincides with the active mining period for the Adubiaso pit and subsequent re-filling after completion of mining
- The maximum annual total water shortfall for average conditions is 0.5 Mm³ in Year 5 (occurring over 2 months). This may be reduced as follow:
    - By relocating the water currently stored in the Adubiaso pit to an alternive storage location during mining of the Adubiaso pit
    - By allowing some storage of surplus water within the pit sumps if practicable

- The maximum annual total water shortfall for 1 in 100 year dry rainfall sequence is 1.38 Mm³ in Year 5 (occurring over 4 months). This may be reduced as follows:
  - By relocating the water currently stored in the Adubiaso pit to an alternive storage location during mining of the Adubiaso pit
  - By allowing some storage of surplus water within the pit sumps if practicable

The water balance includes arsenic concentrations for all flows based on baseline water quality monitoring results and advice from EIS. It forms an integral part of the projects water management strategy and it will be maintained and calibrated throughout the LoM.

The model is operated to maximize containment of process water on site (i.e., avoid the need for discharge). However, a water treatment plant will be constructed at the Obotan process plant to treat excess process water in the event that water captured in the TSF and elsewhere is more than can be practically stored. Another treatment plant is also planned for Esaase at the Buffer dam.

The models used in the analyses presented adopted a limit on arsenic concentrations from areas affected by mining of 0.050 mg/l dissolved arsenic. If water from mine affected areas is at higher arsenic concentration, it will be captured for use within the process circuit, or possibly for treatment to reduce arsenic concentration

With the treatment plant operational, the treated water released from the mine will be at a concentration of 0.010 mg/l, and this water will be released at a rate of 65000 m³/month (100 kl/hour adjusted for likely plant efficiencies) for both plants.

There will be no run-off from mine affected areas when run-off has ceased from areas unaffected by mining. Therefore run-off from mine affected areas will always be diluted within the mine by the very much larger run-off occurring from areas unaffected by mining.

The water balance studies and development of the water balance model have demonstrated that water volumes and water quality can be managed within the boundaries of the mining as illustrated in Figure 18-1 below.

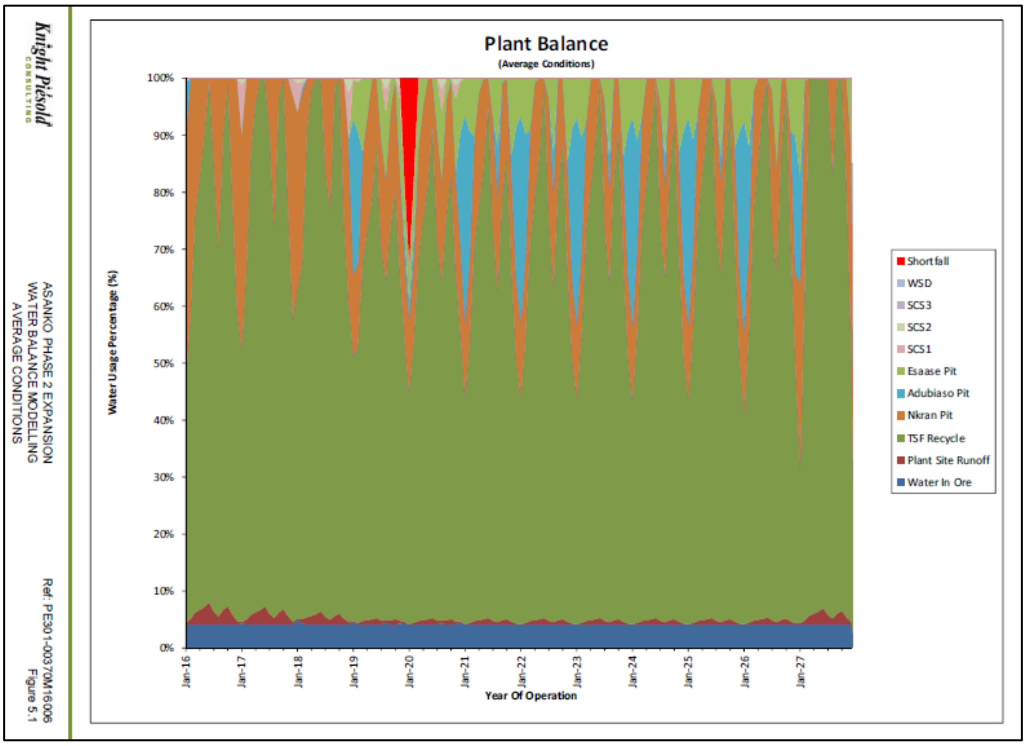


Figure 18-1: Water Balance Summary (source: DRA 2017)

18.6 Sewage Handing

18.6.1 General

Sewage and waste water will be generated at both Obotan and Esaase accommodation camps, exploration camps, mine services areas and crushing plant.

Total sewage loading is a function of the potable water usage at each area for the expected manpower requirements for construction and operational phases.

18.6.2 Sewage Treatment Plants ("STP")

The sewage treatment infrastructure for Obotan site and the Esaase exploration camp is already in place.

A new package STP will be installed for the Esaase MSA. The selection, sizing and location of required STP has been optimized to consider the main contributing areas of mine services area.

Sewage treatment will be undertaken through vendor supplied Biological Sewage Treatment Plant ("BSTP") based on a recycled activated sludge process consisting of inlet screen, anaerobic phase (denitrification) where effluent is allowed to settle, bioreactor phase (aerobic nitrification) followed by clarification and disinfection.

A 30 m^3/d STP will be installed at the MSA to handle the expanded loading during operations which is designed for a compliment of 200 people. Treated effluent will be discharged to the Bonte River.

The STP will be fully containerized and completely pre-assembled for installation above ground on pre-constructed concrete slabs. The STP is designed to treat the sewage loading for effluent discharge to meet the Ghanaian EPA discharge standards. Samples will be collected and tested by STP operational personnel to ensure compliance with applicable legislative criteria.

18.7 Tailings Storage Facility

18.7.1 Tailings Storage Facility Expansion

The TSF will consist of a multi-zoned downstream perimeter embankment, comprising a total footprint area of 377.4 ha (basin area 277.8 Ha) for the final TSF. The current TSF basin will be expanded to incorporate the current water dam basin (upstream of the TSF).

The TSF will operate with the Phase 1 configuration for years 1 to 3 of operation. Two raises (Stage 2 and 3) will be constructed prior to the expansion of the TSF basin. Full basin works in the water dam reservoir (including surface preparation and basin liner construction) will be completed during the Stage 4 works. Tailings will then be deposited from the eastern extents of the basin to fill the expanded TSF basin. Subsequent to Stage 4 construction, the TSF will be raised as a single cell as required during the operation. The Stage 3 east embankment will be abandoned and allowed to overtop.

A design summary is provided below.

Table 18-2: TSF Design Summary for P5M and P10M (source: DRA 2017)

Parameter	Unit	Value
Capacity	Mt	94.6
Maximum Final Embankment Crest	m RL	196.7
Embankment Maximum Height	M	44.9
Total Embankment Volume	Mm^3	12.1
Embankment Crest Length	m	7,500
Embankment Footprint Area	Ha	100
TSF Basin Area	Ha	278

The elevation of the highest point on the existing ground in proximity to the TSF is approximately RL 196m, therefore the proposed final TSF is similar in elevation to the surrounding natural topography.

All embankments will have an upstream slope of 3H:1V (to facilitate 1,5 micron HDPE geomembrane liner installation), an operating downstream of 3H:1V, together with 5m wide benches located at 10m height intervals, for an overall slope profile of 3.5H:1V and a minimum crest width of 10m. The final downstream embankment profile will consist of an overall slope of 3.5H:1V. A cut-off trench will be located beneath the upstream toe of the embankments and will be excavated to extend through to competent low permeability foundation material.

The design will utilise the existing basin under-drainage system, comprising a network of collector and finger drains. The under-drainage system drains by gravity to a collection sump located at the lowest point in the TSF. Solution recovered from the under-drainage system will be released to the top of the tailings mass via submersible pump, reporting to the supernatant pond. Above the current (Stage 1) TSF east embankment crest, the expanded TSF basin (incorporating the current water dam) under-drainage system will tie into the current under-drainage system, with a small under-drainage system below this elevation, reporting to a temporary sump and riser pipe at the east embankment toe. The existing ground water collection system will operate for the Expansion project.

The TSF basin area will be cleared, grubbed and topsoil stripped and unsuitable material removed from the valley base. Existing unsuitable stockpiles will be removed from the TSF footprint. A 200 mm thick compacted soil liner will be constructed over the entire TSF basin area, comprising either reworked in-situ material, or imported Zone A material. A 1.5mm smooth HDPE geomembrane liner will be installed over the entire basin area, overlying the compacted soil liner.

Supernatant water will be removed from the TSF via submersible pumps mounted on a floating barge (by others) located within the supernatant pond throughout operation. An additional temporary barge will be required during initial filling of the current water dam basin with tailings.

The current downstream seepage collection system will be maintained within and downstream of the west embankment to allow monitoring and collection of seepage from the TSF in the collection sump, located downstream of the TSF downstream toe.

An operational emergency spillway will be available at all times during the TSF operation. At closure, the final TSF emergency spillway will be deepened to allow full water shedding from the tailings beach.

The current pipeline containment trench (TDRT) and access road between the plant site and TSF will be utilised for the P10M expansion.

The TSF seepage assessment indicates that the estimated rate of seepage from the TSF with operational underdrainage is 1.5 L/s, resulting in an equivalent basin permeability of 4.4×10^{-10} m/s (considerably better than the Ghanaian Mining Regulations requirement of 1×10^{-8} m/s).

The TSF stability analysis indicates that factors of safety for the TSF embankments comply with the latest Ghanaian Mining Regulations.

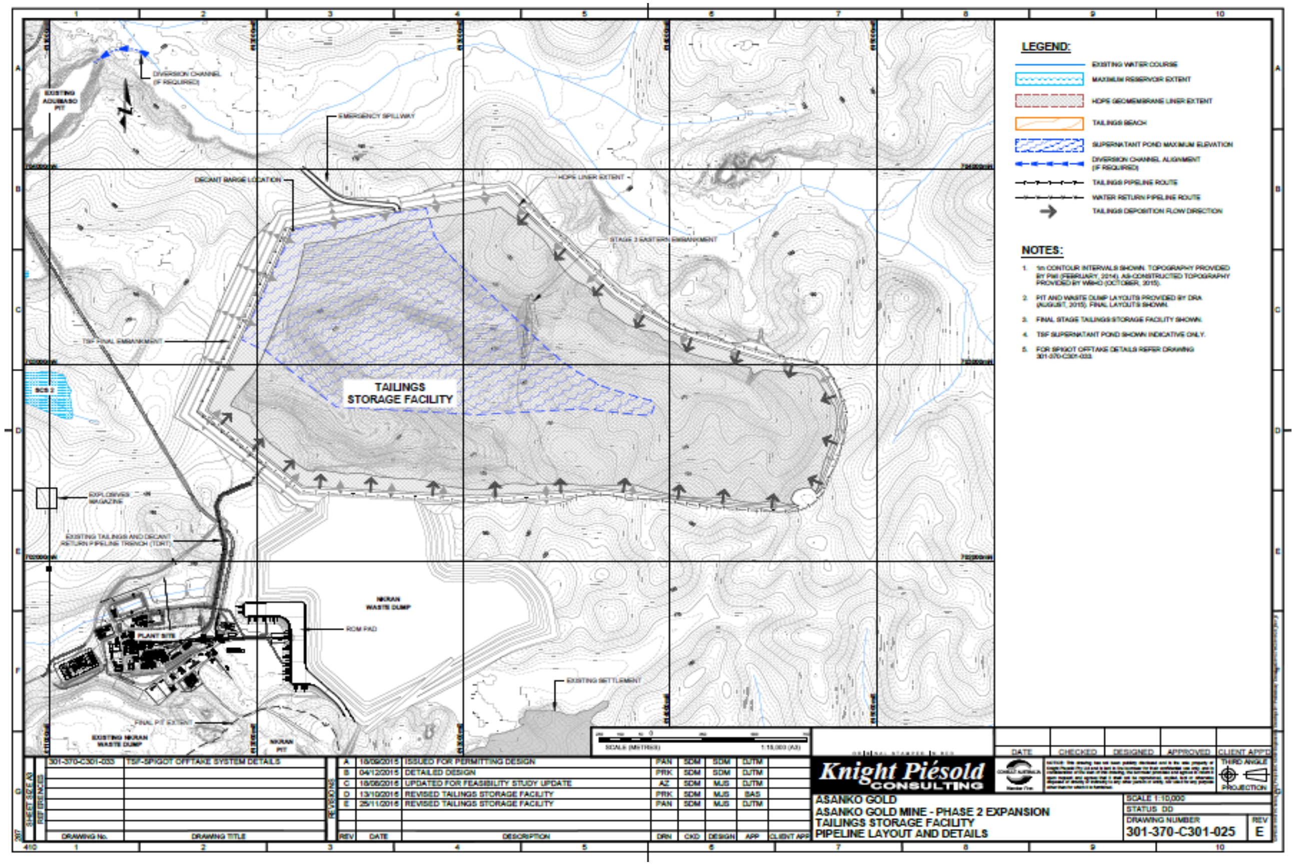


Figure 18-2: Final TSF Expansion (source:KP 2017)

18.7.2 Monitoring

A monitoring programme for the TSF will be developed to monitor for any potential problems which may arise during operations.

Two additional ground water monitoring stations will be installed downstream of the TSF to facilitate early detection of changes in ground water level and/or quality, both during operation and following decommissioning.

Vibrating wire piezometers will be installed on the TSF embankment crest to monitor pore water pressures within the embankment to ensure that stability is not compromised. The piezometers will be monitored at regular intervals and any rises in water level noted.

Survey pins will be installed at regular intervals along the TSF embankment crest in order to monitor embankment movements to assess the effects of any such movement on the embankment.

The TSF will undergo quarterly operational audits and annual technical audits by a suitably qualified geotechnical engineer to ensure that the facility is operating in a safe and efficient manner.

18.7.3 Rehabilitation

At the end of the TSF operation, the downstream faces of the embankment will have a slope of 3H:1V, together with 5m wide benches located at 10m height intervals, for an overall slope profile of 3.5H:1V. The adopted downstream profile should be inherently stable under both normal and seismic loading conditions.

The final TSF emergency spillway will be extended to allow full water shedding of the tailings surface after the remaining supernatant is proven to be suitable for release, to facilitate discharge downstream of the TSF, via a lined channel excavated into a ramp constructed on the final west embankment downstream face, into the northern drainage course and reporting to the existing Adubiaso pit via a diversion channel.

Rehabilitation of the tailings surface will commence upon termination of deposition into the TSF. The following covering for the tailings beach subsequent to decommissioning is proposed:

- Low permeability fill layer (300 mm)
- Topsoil growth medium layer (200 mm)

The finished surface will be shallow ripped and seeded with shrubs and grasses.

18.7.4 Geotechnical Investigation

This study included a site reconnaissance and desktop review of existing geotechnical investigations and site knowledge. A geotechnical site investigation for the existing TSF was completed in two phases in order to assess:

- Foundation and excavation conditions for the TSF
- Availability and suitability of construction materials on site

The following key conclusions were based on the site investigation findings:

- The TSF location is considered suitable for the proposed infrastructure
- In situ soils in the TSF basin are generally suitable for construction of an in situ soil liner
- The near surface residual soils in the TSF may be problematic with regards to reworking and plant movement
- Low permeability materials for the TSF embankment should be sourced from local borrows within the TSF basin
- Structural fill for the TSF embankments should be sourced from near surface materials (Colluvium) from local borrows within the TSF basin
- Drainage medium (sand) for use in the TSF can be sourced from a number of potential borrows

The following notes from the current (Stage 1) TSF construction are pertinent to the expanded TSF basin and provide a general update to the previous site investigation:

- The foundation material on the valley sides, (outside of the unsuitable area) was suitable as both Zone A embankment fill and imported soil liner (HDPE geomembrane liner subgrade) in all instances
- After removal of unsuitable material in the valley base, the foundation material is not suitable for use as compacted soil liner, (or HDPE subgrade) and therefore some subgrade will need to be imported

18.7.5 Tailings Physical Characteristics

Tailings physical testing of representative samples (of oxide and primary tailings) for the Nkran orebody was completed by KP in 2012, and in 2015 testing was conducted for the Esaase orebody.

For the Nkran tailings, it is expected water (supernatant) release will be in the order of 47% of the water in slurry for oxide tailings, and in the order of 42% of the water in slurry for primary tailings. Under-drainage release will typically average around 5 to 10% depending on the arrangement of under-drainage collection and basin treatment.

For the Esaase tailings, it is expected water (supernatant) release will be in the order of 53% of the water in slurry for oxide tailings, and in the order of 47% of the water in slurry for primary tailings.

Under-drainage release may reach up to 20% depending on the arrangement of underdrainage collection and basin treatment.

The dry densities of tailings to be deposited into the facility can be predicted from laboratory testing. Assuming the TSF is efficiently operated a typical achievable density range of between 1.40 t/m^3 and 1.50 t/m^3 is expected within the overall tailings mass.

18.7.6 Tailings Geochemical Characteristics

Tailings geochemistry testing of representative samples (of oxide and primary tailings) for the Nkran orebody was completed by Genalysis in 2012.

Both oxide and primary Nkran tailings samples were classified as Non Acid Forming. The samples had a moderate number of elemental enrichments, with arsenic and mercury classed as highly enriched in both samples and boron found to be highly enriched in the oxide sample. Several metals were found to be present at elevated levels. As such, a cover system should be constructed on closure to isolate the tailings from the environment.

The Nkran supernatant water quality was compared with reference water quality standards for release from mining operations, livestock and wildlife drinking water. Both samples were found to exceed the guideline concentrations for arsenic, and the primary sample was found to exceed the guideline values for TDS, cyanide total, cyanide WAD, iron and sulphate, with the oxide sample exceeding the guideline values for fluoride and molybdenum.

Geochemical testing of three samples of tailings from the Esaase orebody was conducted by Environmental Geochemistry International ("EGi") including both static testing and column leaching. The results of the static acid-base accounting testing indicate that the samples are all non-acid forming (NAF). The only element which was found to be significantly enriched was arsenic. The column leaching indicates that the leachate from the columns remained alkaline with a low salinity, and that the majority of highly soluble arsenic was removed in the processing stage at high pH with limited ongoing leaching of arsenic.

Based on the proposed environmental control measures (i.e. HDPE geomembrane liner and under-drainage system), seepage rates through the base of the facility should not have a negative effect on shallow surface water or ground water. However, as a precaution the TSF should be fenced to prevent access by terrestrial animals.

18.8 Esaase Buildings and Facilities

The new buildings and miscellaneous facilities allowed in the capital estimate are discussed below.

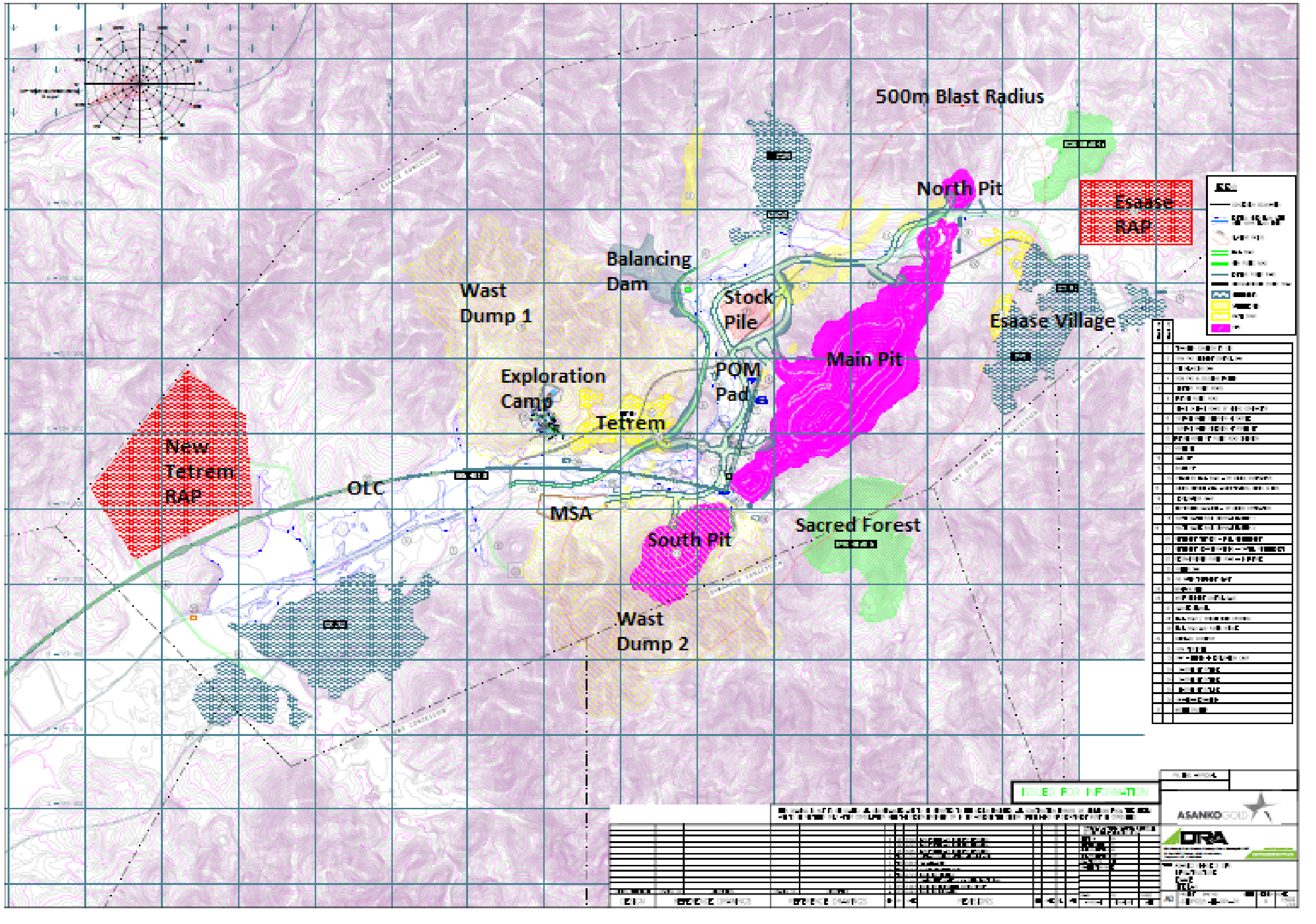


Figure 18-3: Esaase Block Plan (P5M and P10M) (source:DRA 2017)

18.8.1 Mine Services Area (P5M)

The mine services area will house the facilities and services necessary to fully support the mining operation. This includes HMV workshops, fuel bays, wash bays, tyre bays, stores, administrative functions and a bulk fuel farm.

All the building, structures and facilities required during operations will be part of the mining contractors' scope.

In addition the following infrastructure will also be provided for:

- Mining terrace platform
- Power supply
- Potable water boreholes, reticulation, treatment and storage
- Sewer reticulation and treatment
- Storm water control

18.8.2 Explosives Magazine (P5M)

Provision was made for the bulk earthworks, a concrete slab and fencing around the facility. The explosive supplier will be responsible for the storage containers and relevant structures to store the explosives during operations.

18.8.3 Strategic Stockpile (P5M)

The Strategic Stockpile will be constructed and maintained by the mining Contractor.

18.8.4 Waste Dumps Sediment Control Dam (P5M and P10M)

Part of the environmental requirements is the installation of sediment control dams. Provision was made for four unlined dams, two for the WRDF1 and two for WRDF2.

18.8.5 ROM Pad and Tip (P10M)

Two jaw crushers are located at the ROM pad area, at the base of a 3150 m^2, 21m high reinforced retaining wall.

Reinforced concrete tip aprons, safety berms and associated features will be constructed at the top of the reinforced wall.

The ROM pad is seen as a contaminated area and as such will also receive its own silt trap and PCD dam. The water from this dam will be pumped to the buffer dam.

18.8.6 Buffer Dam (Esaase – P10M)

Water collected from the in-pit sumps and pit dewatering boreholes will be pumped to a pit balancing dam, where it will be tested for compliance to the EPA Sector Specific Effluent Guidelines for mining. If it's not compliant it will be pumped to buffer dam for further treatment before being discharged into the environment.

The buffer dam will be constructed from material borrowed within the dam basin, together with a clay core. The dam will then be lined with 1,5 micron HDPE. The public diversion road will run over the dam wall and will share bulk earthworks in the area.

18.9 DFS - Obotan Buildings and Facilities

The new buildings and miscellaneous facilities allowed in the capital estimate are discussed below.

18.9.1 Sub-station Design (P5M)

One new combined main consumer sub-station will be constructed for P5M and P10M.

The consumer sub-station will be an elevated brick building on steel columns with a concrete QC decking floor. This is to allow installation of electrical cables from the bottom of the control panels. The concrete floor needs to be to a specific tolerance as specified by the electrical department and panel suppliers. Complete with lighting, small power, fire extinguishers and warning signs.

The sub-stations would be equipped with correctly sized split unit air conditioners, applying the principle of N +1 unit to control the temperature inside to protect sensitive equipment.

18.9.2 Electro Winning & Gold Room Expansion (P10M)

The existing electro winning and gold room will be extended during P10M to cater for the increased gold production. The construction will be similar to the existing with concrete slabs, block work and IBR sheeted roofs.

18.9.3 Changehouse Expansion (P10M)

The existing change house will be extended by another 15m to cater for the increased labour requirements during P10M. The construction will be similar to the existing with concrete slabs, block work and IBR sheeted roofs.

18.9.4 Processing Plant and Supporting Infrastructure (P10M)

The new process water dam will be constructed next to the existing one on top of the terrace.

Now new terrace will be required for the P10M brownfields upgrades / expansion as the existing space on the Obotan plant terrace proof to be sufficient.

18.10 Accommodation

For the project, construction accommodation has been provided for in the existing senior camp and a new 500 man junior Contractors camp at Obotan, including the exploration camp at Esaase.

The available rooms in these camp is, however, limited due to current operations and only expats and senior staff members will be housed in these camps. The Contractors have been instructed to make provision for their own accommodation and catering requirements for their local and general staff compliments during construction.

Contractors will be expected to source most of their junior staff and unskilled labour from the nearby communities and to bus them in on a daily bases.

18.10.1 Senior Camp at Obotan

The existing village has accommodation for 234 persons in existing blockwork buildings and new containerised accommodation units. It is available for the construction and operational phases with overflow of senior personnel to be housed in local communities.

A local specialist facilities management company ("NAFHAS") has been appointed to manage the camp including catering, cleaning and servicing requirements.

18.10.2 Junior Camp at Obotan

In addition to the existing mine village catering for senior personnel, a new Contractor's camp have been constructed to accommodate junior, (skilled and semi-skilled) personnel that are not housed in the local communities during the construction phase. The Contractor's camp is sized to cater for 380 skilled and 120 semi-skilled personnel and is to be constructed as a permanent camp. This will be used during the operational phase of the project for Asanko Gold junior personnel whilst seeking housing in the local communities. It will also be used to house Contractor staff during subsequent phases in the project development.

A specialist facilities management company, NAFHAS is managing the camp including catering, cleaning and servicing requirements.

18.10.3 Esaase Exploration Camp

Provision for senior staff accommodation at Esaase is currently at the existing exploration camp located adjacent to the Tetrem village. This camp was developed previously and currently has 65 rooms available for accommodation, of which 6 are connected to the existing dry mess and laundry.

The Esaase exploration camp will be decommissioned during the mine operations phase as the area has been identified for use as a waste rock facility later in the project.

Further investigation and trade-offs are suggested between relocation the exploration camp, or moving / re-permitting the waste dumps facilities.

18.11 Resettlement

18.11.1 Demographic & Socio-Economic Survey Analyses

Based on the 2006 census, the 2009 projected population was obtained from the Amansie West District Assembly.

Table 18-3: Resettlements Population (source:DRA 2017)

Community	Projected Population 2009	Total Number of Households
Project Area		
Tetrem	1,030	200
Esaase	3,350	600
Manhyia	594	120

According to the household survey, 49% of the studied population is male and 51% female, almost mirroring the District's figures. Nearly 60% of the surveyed population is under the age of 26. More than two-thirds (69%) of households surveyed are headed by men, in keeping with national statistics. The majority of household heads (89%) identified their ethnicity as Ashanti. Household heads from the north of Ghana constitute the next largest ethnic group (6.54%) followed by Bono (1.55%). Approximately 39% of the population is younger than 16 years of age (49% female). There are 1.6 females and 1.7 males younger than 19, per household.

18.11.2 Estimated Population to be Affected during Execution

It is currently estimated that the Tetrem village has increased to 250 structures, while the affected portion at the Esaase-Manhyia village is approximately 105 households.

18.11.3 Rapid Asset Survey ("RAS")

A RAS will be undertaken as the moratorium has been declared and will record the following information:

- Recording of immovable structures
- Basic owner / occupier information
- Basic structural observations
- Photographs of building number and structure owner
- GPS coordinates of building
- Issuance of notice regarding moratorium, including verbal explanation
- Sign-off by relevant owners (where present) and witnesses
- Satellite imagery or aerial photography on the date of the moratorium

The following parties will be represented during the survey to ensure transparency and compliance with the relevant standards:

- Community Liaison Officer
- Compensation Negotiation Committee representative
- Land Valuation Division ("LVD") representatives

Following the completion of the RAS, the following tasks will be undertaken:

- Full Built Asset Survey
- Land, Farm and Crops Surveys
- Assessment of Crops for Compensation
- Socio-Economic Survey
- Analysis of Baseline Data
- Analysis of Secondary Data Sources
- Thematic Mapping
- Building Survey Analyses
- Valuation of Built Assets
- Crop Survey Analysis

18.11.4 Project Related Impacts

Project impacts specifically relevant to the resettlement of communities as well as Economic Displacement are listed below and are associated with the establishment of mining infrastructure, as well as the implementation of a buffer zone around the pit to ensure the safety of community members.

- Loss of dwellings
- Loss of farm buildings, and other structures (wells, boreholes, fish ponds)
- Loss of institutional buildings/public facilities
- Loss of agricultural land, trees, and standing crops
- Impeded, or lost access to community resources including forest and woodland
- Loss of business income during transition
- Reduced income resulting from these losses
- Loss of access to section of the Bonte River for fresh water

A description of the community infrastructure is summarised below. The baseline surveys show that the project area contains the following:

Table 18-4: Resettlements Community Structure Survey (source:DRA 2017)

Tetrem Community (approximately 1,000 people)	
Schools	
Roman Catholic School	JHS (Single stream four classroom block). Primary (Single stream six classroom block) and Kindergarten (2 room building).
Churches	
	African Faith/Apostles Continuation Church. Pentecost Church. Seventh Day Adventist Church. Catholic Church and Has significant Moslem community but did see any mosques.
Health	None. They use the Esaase Catholic Health Centre.
Infrastructure	Boreholes for water. Over 80% of buildings have electricity. Street lights exist. One house with septic tank and overhead water tank. Untarred roads with no drains. Hilly terrain and poor road linkage to houses so not all homes have vehicular access. This could be good selling point for re-settlement.
Commercial	Cocoa shed. Drinking bars and no market but a few table top traders.
Housing	**Condition:** Grouped into main community, Zongo 1 and Zongo 2. The main community is linear along road to Mpatuom but Zongos are located behind the community to the left and right of the road. About 10- 20% housing in sandcrete. 60% housing in metal roofing. Some thatch. Glass louvre blades on just a few buildings. Average size will be four bedroom houses. Assume six bedrooms or bigger to be about 30%. About 70% of houses typical Ashanti courtyard houses, about 10-15% typical urban houses.

Full analysis of the asset and socio-economic surveys will confirm actual numbers of households and individuals normally resident, or farming in the project area, providing detailed information on project-affected people and primary impacts.

18.12 Integration of Esaase and Obotan by Overland Conveyor – P5M

ELB were appointed to conduct a DFS for an overland conveyor between Esaase and Obotan. The duty of the overland conveyor is to transport gold ore from the Esaase mine to the processing plant at Obotan over a distance of approximately 27 km.

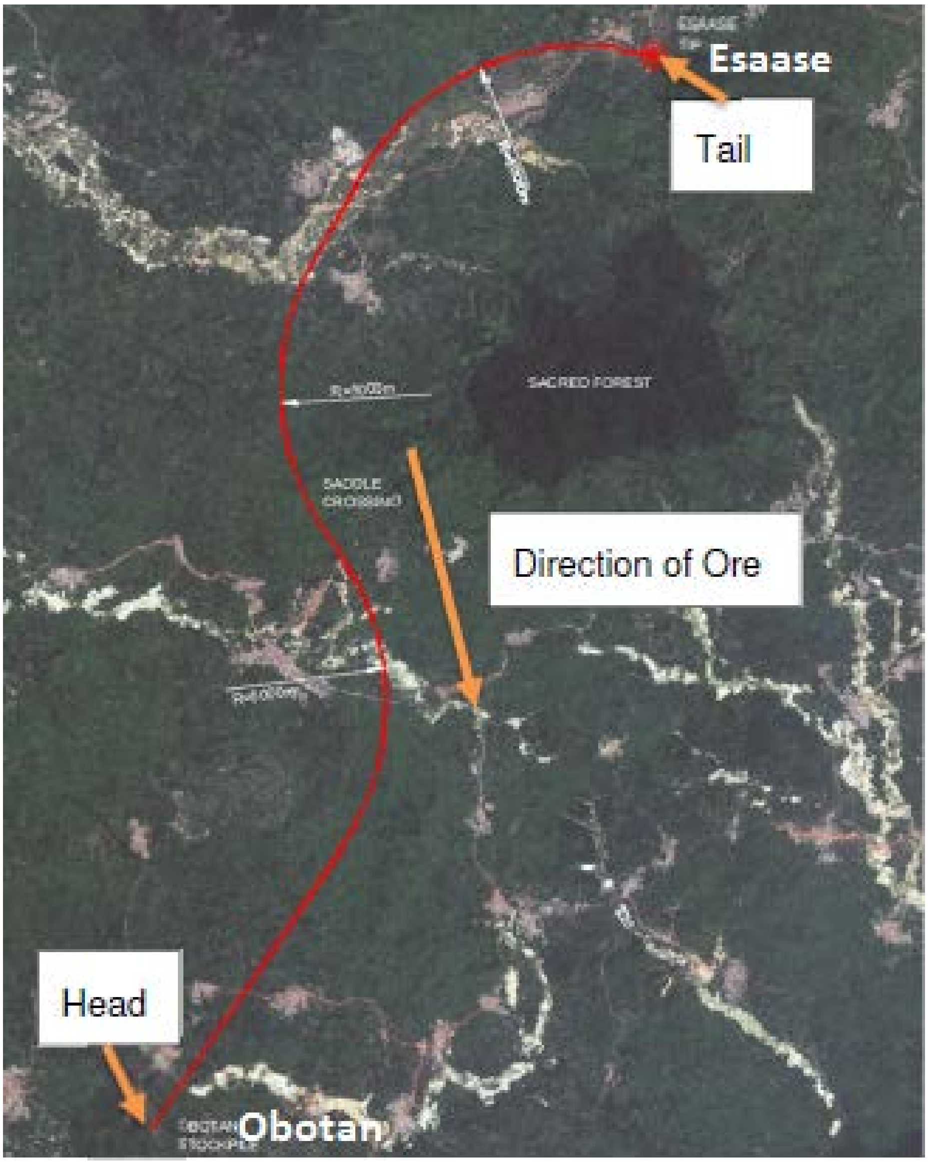


Figure 18-4: Overland Conveyor Route (source: ELB 2017)

The overland conveyor was designed taking into account the requirements for the phased development of the mine and process plants. The overland conveyor has therefore been designed with the ability to operate at varying capacity between 600 ph and 1200 tph.

18.12.1 Conveyor Routing

The selected western route starts at the Esaase tip and travels in a westerly direction before turning south along the western side of the valley. The route passes over a mountain saddle due to constraints of passing to the north of the existing Abore pit and avoiding existing settlements. The overland conveyor then turns onto a south-west alignment before arriving at the Obotan stockpile interface point.

The overland conveyor has an overall length of 27,029m and an elevation of -64.8m which generally classifies this as true overland conveyor where friction resistance dominate.

The overland conveyor generally follows the existing ground terrain profile whilst complying with the vertical curvature constraints as specified in the basis of design.

Figure 18-5: Curved Overland Conveyor (source: courtesy of ELB 2017)

18.12.2 Design Criteria

Primary crushed ore from Esaase to Obotan (original basis).

The design criteria for the overland conveyor is summarised below:

- Material Conveyed .... Gold Ore crushed
- Design Tonnage .... 1200 tph

- Bulk Density .... 1600-1900 kg/m³
- Surcharge Angle .... 20°
- Maximum Lump Size .... 90 mm
- Maximum Temperature Range .... 20℃ to 40℃
- Strength ST-2400N/mm
- Width .... 800 mm
- Speed .... 6 m/s
- Cover (top x bottom) .... 6.0 mm Grade M x 6.0 mm SLRR
- Weight (estimated) .... 24 kg/m
- Tape length (excluding splice allowance .... 54,403 m

- Belt class: .... ST 3 150
- Start-up on full load: ....Yes
- Soft Start: ....VFD
- Take-up type: ....Gravity with Take-up trolley
- Holdback: .... External Low Speed
- Belt Turnovers: ....Before tail pulley and after
- secondary drive pulley
- Carry Idlers: .... 3 roll carry idler configuration
  - Trough Angle: 45°
  - Roll diameter: 178 mm
  - Bearing Diameter: 40 mm
  - Rotational Speed: 697 rpm
  - Idler Spacing: 4.5m (average)
- Return Idlers

- Trough Angle: 10
- Roll diameter: 178 mm
- Bearing Diameter: 40 mm
- Rotational Speed: 697 rpm
- Idler Spacing: 9m (average)

The super low rolling resistance bottom cover is a requirement to ensure frictional resistance is kept to a minimum, thereby reducing the overall capital cost of the system.

18.12.3 Drive Specifications

Table 18-5: Overland Conveyor Drives (source: ELB 2017)

	Motor Nameplate (kW)	Motor Running (kW)	Starter Type	Torque Limit (% np)	RPM	Gear Ratio	Flywheel Inertial (kg.m)	Hold-Back (kN.m)	Brake Required
Drive 1	2 x 700	2x568	VFD	120	1500	13.2	2x50	25	No
Drive 2	1 x 700	568	VFD	120	1500	13.2	50	N/A	No
Drive 3	1 x 700	623	VFD	120	1500	13.2	N/A	N/A	No

Drive 1 and 2 are located at the head end (Obotan) and Drive 3 is located at the tail (Esaase).

The drives are all VSD controlled to facilitate start-up and shut down and to enable the phased capacity requirement.

18.12.4 Mechanical Arrangement

The conveyor stringer structure is a simple support system for the idlers and belt loaded with material.

An integrated low profile stringer-hood completely encloses the ore removing the need for side cladding on the gallery to prevent the release of dust into the environment.

The conveyor will have drives located at the head and the tail end of the conveyor. Power will be supplied from Esaase side at the tail end and from Obotan at the head end.

Belt turnovers have been allowed for. They are located at the head and tail end of the conveyor on the return strand. The belt turnover prevents any material that has adhered to the carry strand and was not removed with the belt cleaners, from falling to the ground, thus significantly reducing spillages.

The entire length of the conveyor requires a large take-up length (about 70m) to account for both active (starting / stopping and variations in load) and permanent belt stretch. The conveyor uses a dual take-up / belt storage system.

The horizontal gravity system at the head end of the conveyor is used as the active take-up and accounts for belt stretch / contraction during starting and stopping and during load variations. An additional fixed take up system is provided for at the tail end. This would be a winch activated trolley system which would be locked out (fixed in place) during normal operation.

The conveyors will be equipped with all instrumentation and devices necessary to operate the conveyors. All devices necessary to enable operability shall be included.

18.12.5 Overland Conveyor Bulk Earthworks and Infrastructure

Allowance was made for the cut to fill bulk earthwork operations. The designs makes provision for 30% more cut than fill to insure that material does not get haul for more than a 2 km free haul distance.

Over and above the earthworks, allowance was also made for 1.8m high galvanised diamond mesh medium security fence on both sides of the overland conveyor, 2 haul road crossings, 10 public crossing and 27 pedestrian crossing.

Galamsey areas will be encountered for 4.30 km along the 27.03 km overland conveyor route. Soil improvement will be required in these area and have been allowed for.

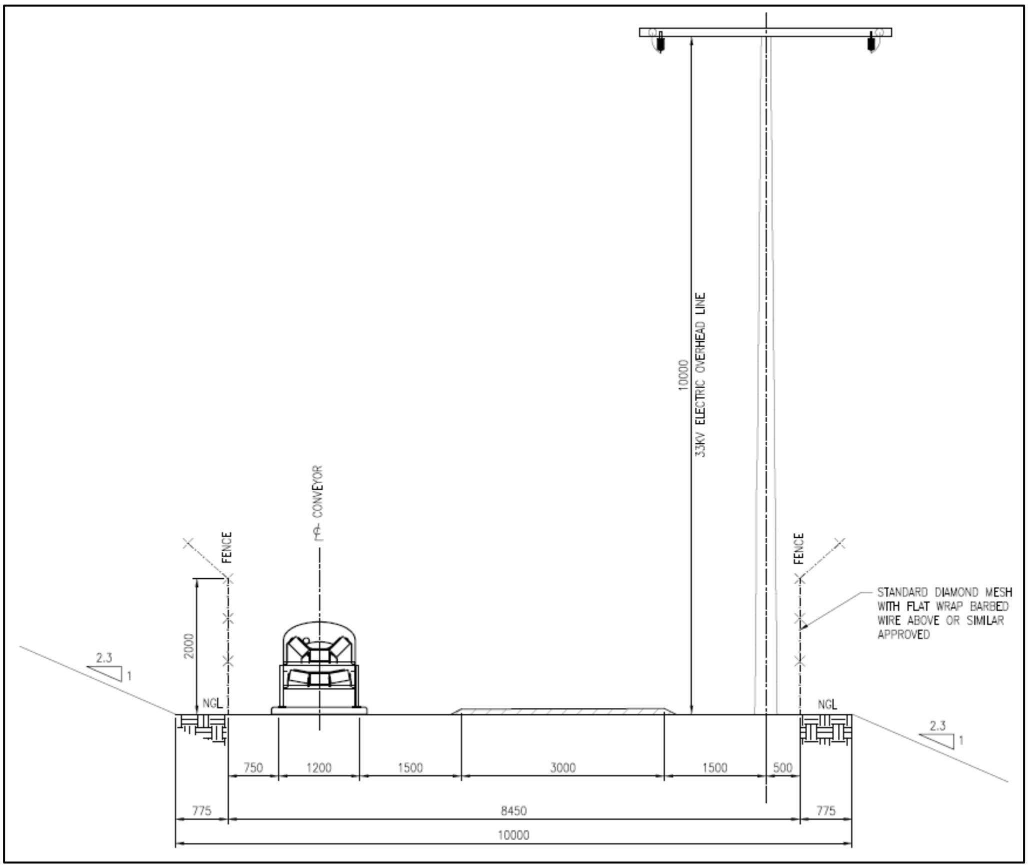


Figure 18-6: Illustration of the Overland Conveyor Servitude (source: ELB 2017)

19 MARKET ANALYSIS

The only commodity anticipated to be produced at the AGM is gold, which is widely and freely traded on the international market with known and instantly accessible pricing information.

In conjunction with debt financing for the Project, the Company has entered into an off-take agreement to sell 100% of the future gold production from the Project to its lender, up to a maximum of 2.22 million ounces. Under the off-take agreement, the lender will pay for 100% of the value of the gold nine business days after shipment. A provisional payment of 90% of the estimated value will be made one business day after delivery. The gold sale price will be a spot price selected during a nine day quotational period following shipment.

Should the Company wish to terminate the off-take agreement, a termination fee will be payable according to a schedule dependent upon the total funds drawn under the loan, as well as the amount of gold delivered under the offtake agreement at the time of termination.

A gold price of US$1,250/oz was used in all the financial models and trade-offs as a prudent view looking forward based on the past three year average gold price. The average gold price for the past three years was US$1,210/oz.

Table 19-1: Three Year Average Gold Price (source: Venmyn Deloitte)

Year	Average Gold Price US$/oz
2014	1,266
2015	1,160
2016	1,249
2017 – year-to-date	1,235
3 Year Average	1,210

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section provides an overview of the environmental legislation and guidelines applicable to the AGM, summarises the permitting process and also provides an overview of stakeholder engagement conducted in respect of the Expansion project.

20.1 Ghanaian Legislation and Guidelines

20.1.1 Environmental and Social

The key environmental and social legislation in Ghana is the Environmental Protection Agency Act 1994 (Act 490) and the Environmental Assessment Regulations 1999 (LI 1652). The Environmental Protection Agency ("EPA") is the regulatory body that administers these laws.

The Environmental Protection Agency Act 1994 (Act 490) establishes Ghana's EPA and defines the functions of the EPA, including, but not limited to the following:

- Prescribing standards and guidelines relating to the pollution of air, water and land
- Ensuring compliance with environmental impact assessment procedures in the planning and execution of development projects
- Any undertaking that has the potential to have an adverse impact on the environment can be required by the EPA to submit an Environmental Impact Statement ("EIS") under Part II of the *Environmental Protection Agency Act 1994 (Act 490).* The EIS covers both the biophysical and the socio-economic aspects and impacts of the project

The Environmental Assessment Regulations 1999 (LI 1652) support the Environmental Protection Agency Act 1994 (Act 490) and describe the process of environmental assessment in Ghana.

Submission of an EIS is mandatory for any mining project where the mining lease covers a total area in excess of 10 hectares (25 acres). The regulations outline the environmental and social aspects that must be addressed in an EIS. This includes addressing the possible direct and indirect environmental impacts of the proposed undertaking during pre-construction, construction, operation, decommissioning (i.e., mine closure) and post-decommissioning phases.

An Environmental Scoping Report must be prepared and approved by the EPA prior to submitting an EIS. The purpose of the scoping document is to determine an agreed scope of works for the EIS and must include a draft terms of reference.

The regulations also prescribe a number of activities that must be carried out once an Environmental Permit is obtained.

These include:

- Submit, and have approved, an Environmental Management Plan (“EMP”) within 18 months of commencement of operations and thereafter every 3 years
- Submit an annual Environmental report 12 months after the commencement of operation and every 12 months thereafter
- Obtain an Environmental Certificate from the EPA within 24 months of commencement of operations
- Mining businesses are required to submit closure plans to the EPA and obliged to post reclamation bonds. The Environmental Protection Agency Act, 1994 (Act 490) and the Environmental Assessment Regulations, 1999 (LI 1652) also contain provisions for community engagement.
- The Water Resources Commission Act, 1996 (Act 522) and the subsequent Water Use Regulations, 2001 (LI 1692) govern the abstraction, impoundment, and discharge of water

20.1.2 Minerals and Mining

6 Minerals and Mining Legislative Instruments (“LIs”) were promulgated in 2012 to govern mining operations in Ghana.

These are:

1. Minerals and Mining (General) Regulations 2012.
2. Minerals and Mining (Licensing) Regulations 2012.
3. Minerals and Mining (Support Services) Regulations 2012.
4. Minerals and Mining (Compensation and Resettlement) Regulations 2012.
5. Minerals and Mining (Explosives) Regulations, 2012
6. Minerals and Mining (HSLP and Technical Regulations 2012

The Minerals Commission is the principal regulatory body that administers these laws. It was established under the Minerals Commission Act, 1993 (Act 450) for the “regulation and management of the utilisation of the mineral resources (of Ghana) and the co-ordination of policies in relation to them”.

The Minerals and Mining Act, 2006 (Act 703) aims to:

- Develop a national policy on mining and consolidate the disparate laws on mining in force prior to 2006
- Increase investment by foreign mining companies in Ghana
- Remove the uncertainty concerning the availability and conditionality of mining rights as well as the bureaucratic gridlock
- The Act requires that an application for a mineral right (e.g., mining lease) be accompanied by a statement providing:
    - Particulars of the financial and technical resources available to the applicant

- An estimate of the amount of money proposed to be spent on the operations
- The proposed programme of mineral operations
- A detailed programme with respect to the employment and training of Ghanaians

Once granted, the holder of a mining lease must notify the Minister Lands and Natural Resources, who is the sector minister, of amendments the holder intends to make to the programme of mining operation. Under Section 72 of the Minerals and Mining Act, 2006 (Act 703) the holder of a mineral right must have due regards to the effects of mineral operations on the environment and must take whatever steps necessary to prevent pollution of the environment as a result of mineral operations.

The Minister may, as part of a mining lease, enter into a Stability Agreement with the holder of the mining lease to ensure that the holder will not, for a period of up to 15 years, be adversely affected by a new enactment, changes to an enactment, or be adversely affected by subsequent changes to the level of, and payment of, royalties, taxes, customs or other related duties. The Stability Agreement becomes effective upon ratification by Ghana's Parliament.

Where the proposed investment to be made by the mining company will exceed US$500,000,000, the Minister may, on the advice of the Minerals Commission, enter into a Development Agreement under the mining lease.

The Development Agreement may contain provisions relating to:

- The mineral right or operations to be conducted under the mining lease
- The circumstance or manner in which the Minister will exercise discretion conferred by, or under, the Minerals and Mining Act, 2006 (Act 703)
- Stability terms under a Stability Agreement
- Environmental management expectations and obligations of the holder to safeguard the environment in accordance with the Minerals and Mining Act 2006, or another enactment
- Settlement of disputes

The Development Agreement is also subject to the country's Parliamentary ratification in order to make it effective.

The Minerals and Mining (Health Safety and Technical) Regulations provide mining, health, safety and environmental requirements that have to be met by a mining lease holder.

20.1.3 Compensation

Acquisition and access to land in Ghana for development activities, including mining, may be undertaken either through the State's power of eminent domain, or by private treaty. The taking of land requires the payment of due compensation. The regulatory oversight of private sector land acquisition and resettlement related to mining activities and actions is governed by the Constitution of Ghana and two legislative acts:

The 1992 Constitution of Ghana ensures protection of private property and establishes requirements for resettlement in the event of displacement from State acquisition (Article 20 [1, 2 and 3])

The State Lands Act 1962 (Act 125) and its subsequent amendment, *State Lands (Amendment) Act 2000 (Act 586)*, mandates compensation payment for displaced persons and sets procedures for public land acquisitions

The *Minerals and Mining Act, 2006 (Act 703)* vests all mineral rights in land to the State and entitles landowners or occupiers to the right for compensation. In particular, Section 74 [1] requires compensation for:

- Deprivation of the use or a particular use of the natural surface of the land, or part of the land
- Loss of, or damage to immovable property
- In the case of land under cultivation, loss of earnings, or sustenance suffered by the owner, or lawful occupier, having due regard to the nature of their interest in the land
- Loss of expected income, depending on the nature of crops on the land and their life expectancy

20.1.4 Health, Safety and Labour

The principal health, safety and labour laws applicable in the mining industry include:

- The Minerals and Mining Act, 2006 (Act 703)
- Workmen's Compensation Act, 1987 (PNDCL 187)
- Labour Act, 2003 (Act 651)
- Minerals and Mining (Health Safety and Technical) Regulations (LI 2182)

Provisions in the mining law state in part that a holder of a mineral right shall give preference in employment to citizens of Ghana "to the maximum extent possible and consistent with safety, efficiency and economy."

As with other sectors, a foreign employee in the mining sector needs a work and residence permit in order to work.

However, under the mining laws of Ghana, there are immigration quotas in respect of the approved number of expatriate personnel mining companies may employ.

20.2 Project Permitting Process

20.2.1 Expansion Project Permitting Process

Two key regulatory permits are required for development of the Esaase Gold Project.

These are:

- The Mine Operating Permit issued by the Minerals Commission
- The Environmental Permit issued by the EPA

Following the required engagements, regulatory site visits, and submission of the relevant project details, the Minerals Commission has issued the Mine Operating Permit for the Project. This Permit was received in January 2017.

The Environmental Permit was received in January 2017.

The permitting processes for the two regulatory bodies are summarised in the next sections.

20.2.2 Minerals Commission Permitting Process

Asanko Gold (formerly known as Keegan) acquired the Esaase Concession in 2006 and, under an Exploration Permit issued by the Minerals Commission, conducted an extensive geological survey and drilling programme to define its mineral reserves.

Following completion of this work stream and preliminary establishment of a business case, a Mining Area application was submitted to the Minerals Commission in 2012 which defined the location of the proposed mine on the concession as well as locations of the pits, waster rock dumps and other related facilities.

The Mining Area application was approved by the Minerals Commission and a Temporary Mine Operating Permit issued that same year.

In 2014 further work was conducted to optimise the project. The Minerals Commission was regularly updated on the Project and a formal application was submitted to the Minerals Commission in December 2016 which led to issuance of the permanent Mine Operating Permit for the Esaase Concession in January 2017.

20.2.3 EPA Permitting Process

The Permitting Process followed was as per the EPA's approved EIA process, which is shown in Figure 20-1. In line with the Company's commitment towards utilising local resources and supporting local business, Asanko Gold appointed a Ghanaian environmental management consulting firm, the African Environmental Research & Consulting Company ("AERC"), to carry out the required work on its behalf.

The process commenced with formal consultations with the EPA on the proposed plan to develop the Esaase Gold project followed by submission of an EIA application for the project which included its basic technical details.

In this regard, the EPA's form EA2 application was filed with the Agency on 12th June, 2015, for the proposed mining development at Esaase, as well as the 27 km long overland conveyor from Esaase to Obotan.

In line with the permitting process, the EPA responded to the EA2 submission by requesting Asanko Gold to conduct an EIA in respect of the proposal and submit an EIS in line with the requirements of their EIA procedures.

A Scoping Report, with draft Terms of Reference for the EIA, in respect of the proposed project proposal was prepared and subsequently submitted to the EPA in August, 2015.

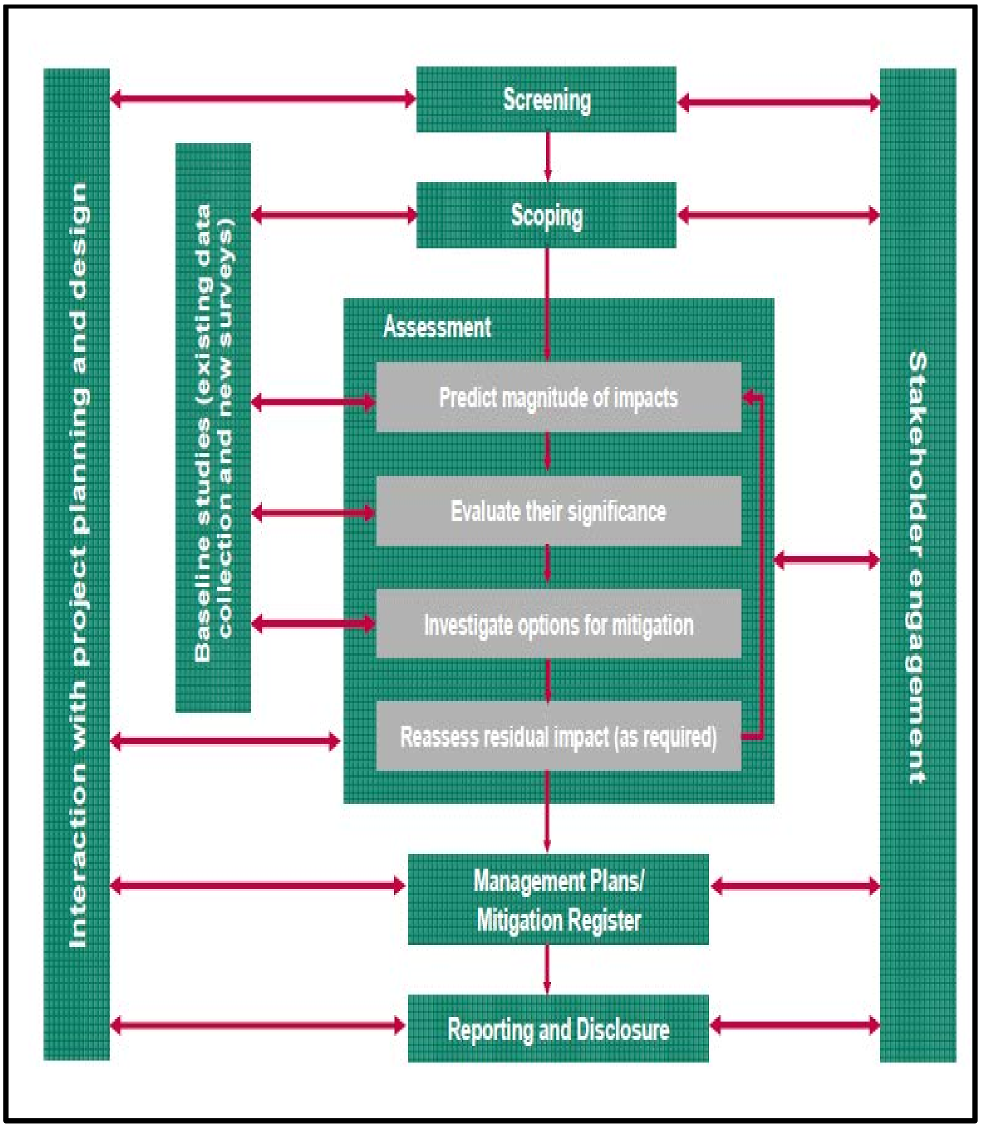


Figure 20-1: EIA Approach for the Esaase Gold Project (source: Asanko Gold 2017)

Following this, work was done on the EIA which entailed:

- Tour(s) by AERC to the project sites for familiarisation with pit locations, conditions, access and general relief of area
- Technical meetings between Asanko Gold and AERC discuss and clarify the project scope as a basis for determining the spatial and time boundaries of the EIA assignment

- Identification and review of all appropriate Ghanaian Environmental, Mining and Allied Act, Regulations, Standards, Conditions and Guidelines
- Field investigations comprising environmental, socio-economic and cultural surveys within the project area of influence to determine existing baseline conditions
- Development of an inventory of all proposed mine infrastructure within the vicinity of the project area
- Collection, sorting and review of company documentation relevant to the proposed undertaking, including concept descriptions, independent study reports, design drawings and maps, etc.
- Holding of consultations with all Governmental (and Non-governmental) Institutions
- Holding of consultations with traditional authorities and all impacted communities

In line with the EPA's permitting process, the Agency held a Public Hearing on 19th April 2016 at Esaase to collate the views and opinions of all stakeholders, especially potentially impacted communities, on the project.

In attendance at that were:

- Officials of the EPA
- Officials of Asanko Gold
- Officials of AERC, Asanko's environmental management consultants
- A representative of the Asantehene, the King of the Ashanti Kingdom
- The chiefs and members of all 12 communities within the catchment area of the proposed Esaase mining project as well as those along the overland conveyor corridor
- The Member of Parliament for the area
- The District Chief Executive and officials of the Amansie West District
- A representative of the District Chief of the Atwima Nwabiagya District Assembly
- Religious leaders from the communities
- The media

The ceremony was chaired by Dr. Richard Amankwah Kuffour, a lecturer of the University of Education (UEW), Winneba, Ghana.

The AGM General Manager for Operations, Mr. Charles Amoah, gave an overview of the project highlighting its impacts and the interventions to be implemented by the company to mitigate these. He further enumerated the financial and socio-economic benefits of the project to all stakeholders including the Government of Ghana and the local communities.

In an open forum, members of the community, as well as the chiefs of each of the 12 communities, publicly declared their support for the project and expressed their expectation that the expansion projects will create jobs for the youth in the community and also lead to socio-economic development of the catchment area.

The positive outcome of this key EIA activity (i.e. the Public Hearing) was pivotal to the permitting process and an account of the event formed an integral part of the Draft EIA developed in respect of the project.

Figure 20-2: Community Members Reviewing Details of the AGM expansion projects at the EPA Public Hearing (source Asanko Gold 2016)

Figure 20-3: A cross-section of Chiefs and Members of the Community at the EPA Public Hearing (source: Asanko Gold 2016)

The various findings of the respective EIA activities were subsequently compiled into the Draft Environmental Impact Statement ("Draft EIS") which was submitted to the EPA as a sequel to the Scoping Report on 30th September, 2016. The Draft EIS outlined the project description, its potential impacts and mitigations, proposed environmental monitoring action plans, provisional environmental management plans and reclamation and closure alternatives of relevance to the mining undertaking. A summary of the baseline studies conducted as part of this process is presented in section 20.3 of this report.

The EPA reviewed the Draft EIS and reverted with their comments and queries as well as invoices for the permit processing fees. This effectively marked technical approval of the project by the EPA.

The Draft EIS was revised incorporating the EPA's comments with the final version being submitted to the Agency on 30th November, 2016. Payment has also been effected in respect of the EPA's invoice and the environmental permit for the project is expected to be issued shortly.

20.3 Stakeholder Engagement

20.3.1 Guiding Principles of Stakeholder Engagement

Extensive interactions were held with various stakeholder groups including the government, regulatory authorities and, particularly, members of communities that will be impacted by the development of Essase and the expansion projects.

As per Figure 20-2, these interactions were guided by Asanko Gold's principles of conducting stakeholder engagement in a way that is:

- Respectful and sensitive to local culture and societal norms
- Transparent and honest in deliberations over issues of concern
- Based on continuous engagement and keeping stakeholders updated, and their opinions sought, every step of the way
- Aimed at building mutually beneficial long-term partnerships

Further to these, the engagements followed the lines of Free, Prior and Informed Consent ("FPIC") so as to ensure that, apart from legal and regulatory consent to the project, affected communities were fully informed about the project, its potential technical and socio-economic impacts on them, interventions to mitigate these impacts, among others, so the communities could make the decision on whether or not to allow the project to be implemented on their land.

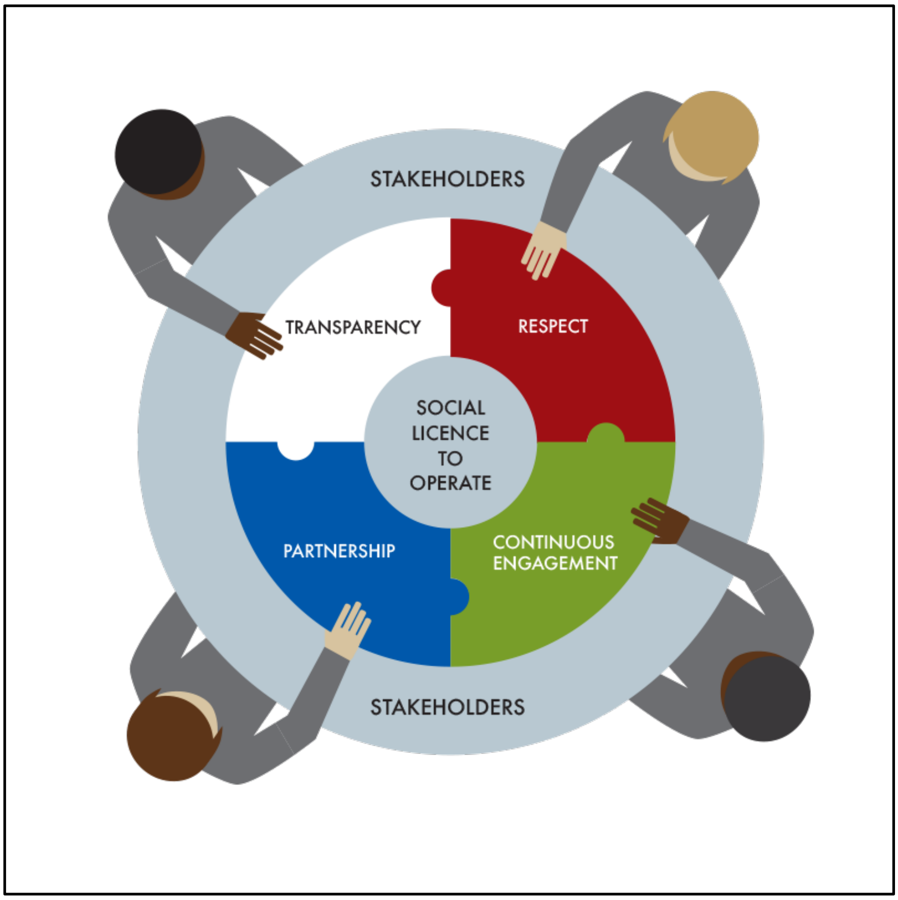


Figure 20-4: Asanko's Principles for Stakeholder Engagement (source: Asanko Gold 2016)

20.3.2 Engagement with Communities

The Esaase Project Area has 12 communities within its catchment area, the locations of which are shown in Figure 20-5.

Figure 20-5: Esaase Project Area villages (source: Asanko Gold 2017)

It was, therefore, of critical importance that these communities were fully engaged, as well as the 25 communities within the Obotan project area, given that the conveyor would traverse both the Obotan and Esaase project areas.

Principal engagement methods and venues to date have included:

- Multi-stakeholder forums
- Village level Community Liaison Committees
- Establishment of staffed Community Information Centres ("CIC") as an ongoing access point for village residents
- Durbars
- Individual and small group meetings
- Open door policy at the project site offices

A Grievance Management process was also instituted to ensure all community concerns were documented, reviewed, necessary actions taken and timely feedback provided to affected community members.

The AGM further engaged additional Community Liaison Officers to enhance the frequency and quality of interactions, particularly with the DFS communities, and also to build trusting relationships with stakeholders even before commencement of the project.

In line with the foregoing, a comprehensive Stakeholder Engagement and Action Plan was developed, with broad stakeholder groups and committees established in the communities, to keep members of the communities fully updated on the project and to deepen their relationship with Asanko, thereby building a strong linkage with the local population. This approach ensured effective information flow between the Company and the catchment communities and provided the platform for building strong and collaborative working relationships with project stakeholders.

The composition of the committees and stakeholder groups that were consulted on the project, and continue to be engaged, is shown in Table 20-1 below.

Table 20-1: Stakeholder Groups and Committee Membership (source: Asanko Gold 2017)

No	Stakeholder Group	Membership
1	Community Consultative Committee ("CCC")	40
2	Traditional Authority Council	30
2	Assemblymen and Unit Committee Reps	25
3	Women Consultative Committee ("WCC")	28
4	Asanko Community Development Committee ("OCDC")	50 (Community based)
5	Crop Rates Review Committee ("CRRC")	25
6	Youth Associations	Community Based

No	Stakeholder Group	Membership
7	Resettlement Negotiation Committee	17
8	District Assembly and Heads of Government Institutions	15
9	Small Scale Miners and Opinion Leaders	10
10	Religious Clergies and Imams	25
11	Social Responsibility Forum ("SRF")	50

The above stakeholder groups and committees met with the AGM's representatives on a monthly basis to either receive updates on the project, deliberate on matters of mutual concern, as well as to present their concerns and grievances with the aim of working with the Company to amicably resolve any issues or concerns. Since 2014, the AGM has recorded between 100 and 120 stakeholder meetings each year and such interactions are expected to increase as the Company transitions into implementation of the expansion projects.

In guiding these stakeholder interactions, Asanko Gold developed a well-defined communications plan for the development of Esaase and the associated overland conveyor with key discussion items as follows:

- Project development activities
- Planned mining activities and any associated changes
- Proposed conveyor route, its impacts and mitigatory interventions
- Rehabilitation works and post-closure land use requirements of stakeholders
- Development of partnerships with stakeholders for community development
- Proposals for company sponsored livelihood and agricultural land improvement programmes
- Determination and review of crop compensation and deprivation of land use rates
- Sustainable development and community assistance projects
- Social Responsibility Forum update

20.3.3 Governmental Stakeholders

On the governmental and regulatory side, engagement sessions were held with the various regulators and government departments to discuss the project (both technical and social aspects) with a view to obtaining the necessary regulatory consents and licences required to pave way for implementation.

To this end, the following governmental stakeholders were fully informed, their opinions and inputs sought and actively updated on the project with formal notifications, submissions and applications made as required.

These were:

- The Ministry of Lands and Natural Resources
- The Minerals Commission
- The Inspectorate Division of the Minerals Commission
- The Ministry of Environment, Science, Technology and Innovation
- The Environmental Protection Agency
- The Water Resources Commission
- The Forestry Commission
- The Ashanti Regional Coordinating Council
- The Amansie West District Assembly
- The Ministry of Food and Agriculture – Amansie West District
- The Ghana Health Service – Amansie West District
- The Land Valuation Board – Ashanti Region

The relevant consents, regulatory permits and approvals have since been obtained from all the governmental and regulatory bodies.

As highlighted in section 20.2.3, the EPA has technically consented to the project with the permit processing fees already paid by Asanko and the final DFS Environmental Impact Statement duly submitted.

To this end, the Agency's administrative processes, and finalisation of permit conditions, are being progressed towards imminent issuance of the Environmental Permit for the Esaase pit and overland conveyor.

20.3.4 Industry Group Stakeholder

Asanko Gold Ghana is a duly registered and active member of the Ghana Chamber of Mines, which is the umbrella body that represents the interests of mining companies in the country.

The Chamber enhances collaboration among its members and plays an advocacy role for the industry in its engagement with the Government of Ghana on policy issues that impact mining in the country.

21 CAPITAL AND OPERATING COSTS

21.1 Economic Scenarios

In order to present the economic results of the incremental additions to the AGM, the following scenarios have been reported upon:

- Base Case, defined by:
  - Existing CIL Plant runs at 3.6 Mtpa
  - Ore feed is from Nkran and the Satellite pits
- Base Case and P5M and P10M, defined by:
  - Existing CIL Plant at Obotan upgraded to 5 Mtpa
  - Overland conveyor constructed
  - Esaase brought into production
  - Second CIL plant constructed, adding another 5 Mtpa
  - Ore feed is from Esaase, Nkran, and Satellite pits
- Base Case and P5M, defined by:
  - Existing CIL Plant at Obotan upgraded to 5 Mtpa
  - Overland conveyor constructed
  - Esaase brought into production
  - Ore feed is from Esaase, Nkran, and Satellite pits

The incremental economic results of P5M + P10M are shown by subtracting the formulaic results of Base Case from the respective Base Case and Project scenarios.

21.2 The AGM – Capital and Operating Costs (NI 21)

Capital and operating cost estimates for the above scenarios have been prepared by Asanko Gold and their appointed independent consultants, with the basis of the estimates detailed in their respective sections of this document, which in turn has been based on a completed DFS document on the Project. The estimate meets the required accuracy criteria of + 10% - 15%. The base date for the capital cost estimate is June 2016.

21.2.1 Capital Costs

The Project capital expenditure (“capex”) estimations are derived from the studies on mining, processing, mine infrastructure, TSF designs, dam construction, electrical supply, owner’s costs and indirect costs.

The general estimation approach was to measure/quantify each cost element from the engineering drawings, Process Feed Diagrams, mechanical equipment list, infrastructure equipment list, and motor lists. Quotations from three, or more vendors were obtained for the major equipment whereas minor equipment, in general, was single sourced. The estimate for the plant has been based on an assumption of a continuous engineering, procurement and construction effort with no interruption of the implementation programme after funding approval has been obtained. The estimate is based on a project execution strategy whereby major units of construction work will be allocated to a number of contractors. The cost estimates assume that all material and equipment acquisition and installation sub-contracts will be competitively tendered and no allowance for delays is included. A separate contingency allowance of 3% on the overall project was allowed to address the unforeseen risks applicable to this project.

The capital costs, broken down in major capital cost items, provided for in the DCF model (Base Case) are summarised in Table 21-1 and the timing of the capex spend is illustrated in Figure 21-1. The total Base Case capex over the LoM is estimated to be US$66 million. The largest contributors to the Base Case capex over the LoM is for the TSF (approx. 55%) and closure costs (approx. 39%).

Table 21-1: Base Case - Total Capital Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)*
Total Installation Capital	**-**
Total SIB	**66**
On-Going Rehabilitation	4
Closure Cost	26
Tailings Dam	36

*Rounding applied

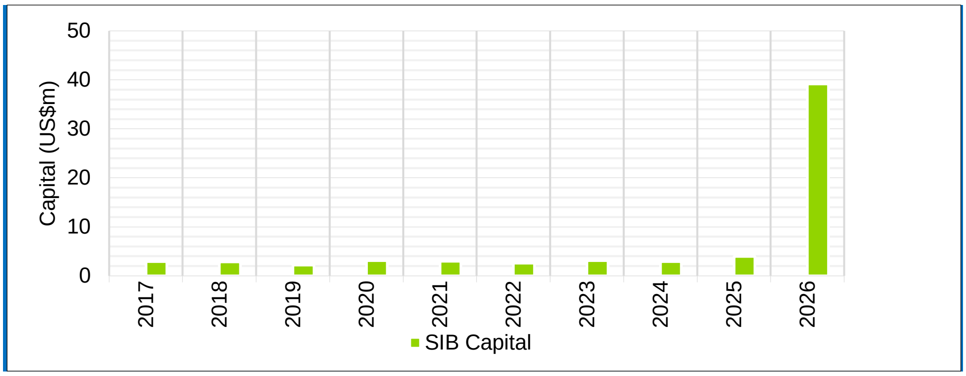


Figure 21-1: Base Case - Capital Scheduling (source: Venmyn Deloitte 2017)

The capital costs, broken down in major capital cost items, provided for in the DCF model (Base Case + P5M + P10M) are summarised in Table 21-2 and the timing of the capex spend is illustrated in Figure 21-2. The total Base Case + P5M + P10M capex over the LoM is estimated to be US$474 million. The largest contributors to the Base Case + P5M + P10M capex over the LoM is for the process plant and infrastructure (approx. 33%) and the overland conveyor (approx. 16%).

Table 21-2: Base Case + P5M + P10M - Total Capital Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)*
Total Installation Capital	**349**
Process Plant	100
Overland Conveyor	78
Process Plant Infrastructure	55
RAP Project	24
Mining	8
Owners Cost	32
Project Indirect	29
Design Development	14
Contingency	9
Total SIB	**125**
On Going Rehabilitation at Obotan	8
Closure Cost (Obotan)	19
Tailings Dam	56
On Going Rehabilitation at Esaase	14
Closure Cost (Esaase)	12
Non-Mining Infrastructure	5
RAP Project	11

- * Rounding applied.

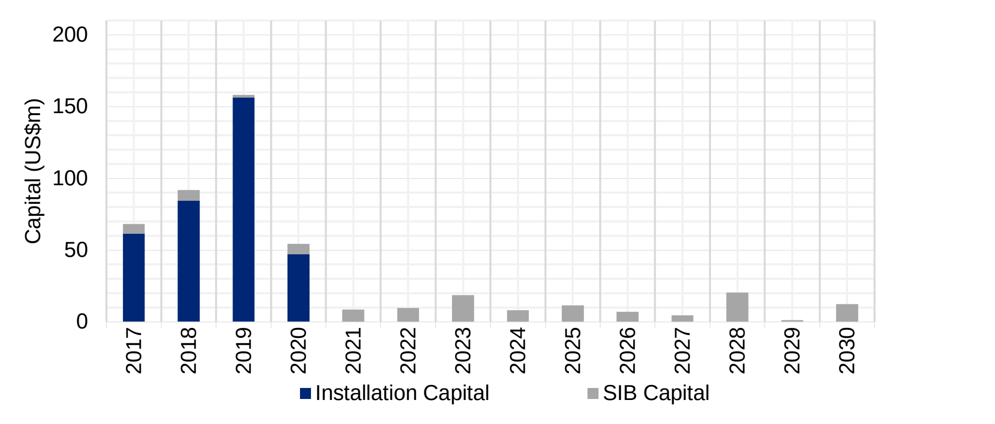


Figure 21-2: Base Case + P5M + P10M - Capital Scheduling (source: Venmyn Deloitte 2017)

The capital costs, broken down in major capital cost items, provided for in the DCF model (Base Case + P5M) are summarised in Table 21-3 and Figure 21-3.The total Base Case + P5M capex over the LoM is estimated to be US$322m. The largest contributions to the Base Case + P5M capex over the LoM are the overland conveyor (approx. 28%) and tailings dam (approx. 21%). It must be noted that the difference of US$12 million on the overland conveyor between P5M and P5M+P10M has been investigated, and found to be a total capex reporting difference. Capital costs covering the discharge of the conveyor at the P5M+P10M processing facility has been included in the Process Plant Infrastructure costing element while it has not been classified as such in the P5M scenario.

Table 21-3: Base Case + P5M - Total Capital Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)*
Total Installation Capital	**147**
Process Plant	6
Overland Conveyor	90
Process Plant Infrastructure	13
RAP Project	-
Mining	1
Owners Cost	14
Project Indirect	13
Design Development	6
Contingency	4
Total SIB	**175**

Aspect	Amount (US$m)*
On Going Rehabilitation at Obotan	8
Closure Cost (Obotan)	19
Tailings Dam	67
On Going Rehabilitation at Esaase	23
Closure Cost (Esaase)	12
Non-Mining Infrastructure	10
RAP Project	36

- *Rounding applied.

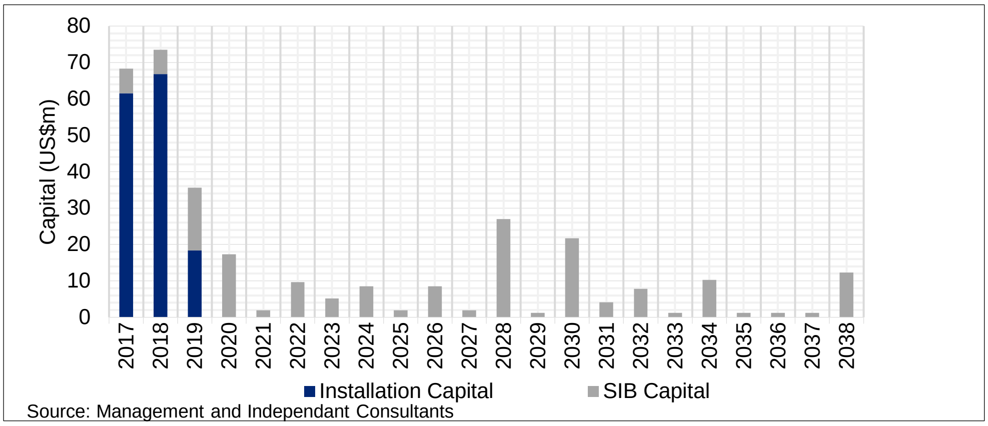


Figure 21-3: Base Case + P5M - Capital Scheduling (source: Venmyn Deloitte 2017)

21.3 Operating Costs

Operating cost ("OPEX") estimates were developed from each of the Project component studies and include mine design criteria, process flow sheet, plant consumable studies, mass and water balance, mechanical and electrical equipment lists, and in-country labour cost data. The cash operating costs are defined as the direct operating costs including contract mining, processing, and tailings storage, and water treatment, general and administrative and refining costs.

The operating costs accounted for in the financial model for Base Case over the scheduled LoM are summarised in Table 21-4, Figure 21-4 and Figure 21-5. The total Base Case opex/oz over the LoM is estimated to be US$797/oz. The largest contributors to the Base Case opex over the LoM is the mining operation accounting for 56%.

Table 21-4: Base Case - Total Operating Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)	Amount (US$/oz)
Mining (Obotan)	832	443
Mining (Esaase)	-	-
Processing	424	226
Other (Refining and G&A)	241	128
Total OPEX	**1,497**	**797**

*Rounding applied

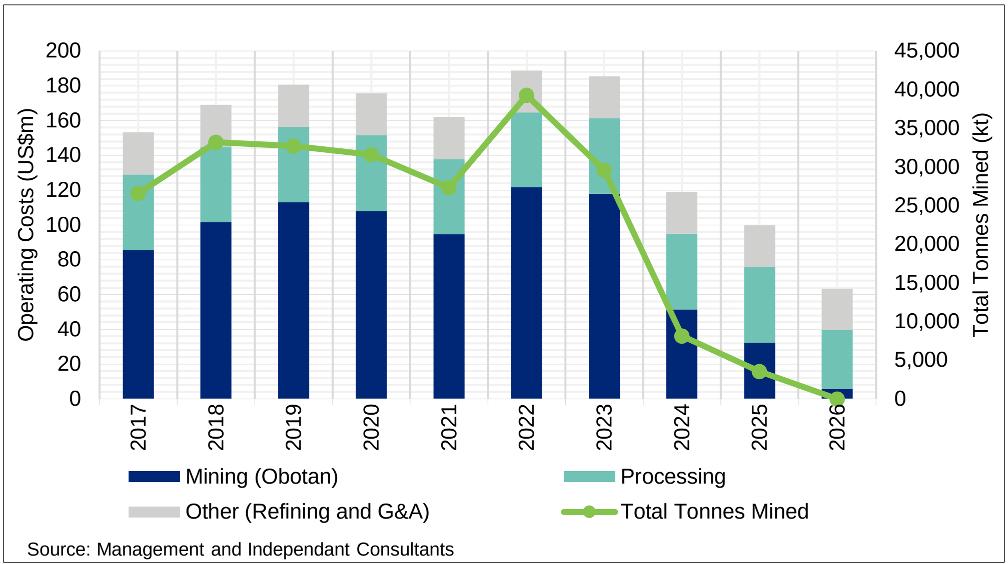


Figure 21-4: Base Case - Operating Cost Scheduling (source: Venmyn Deloitte 2017)

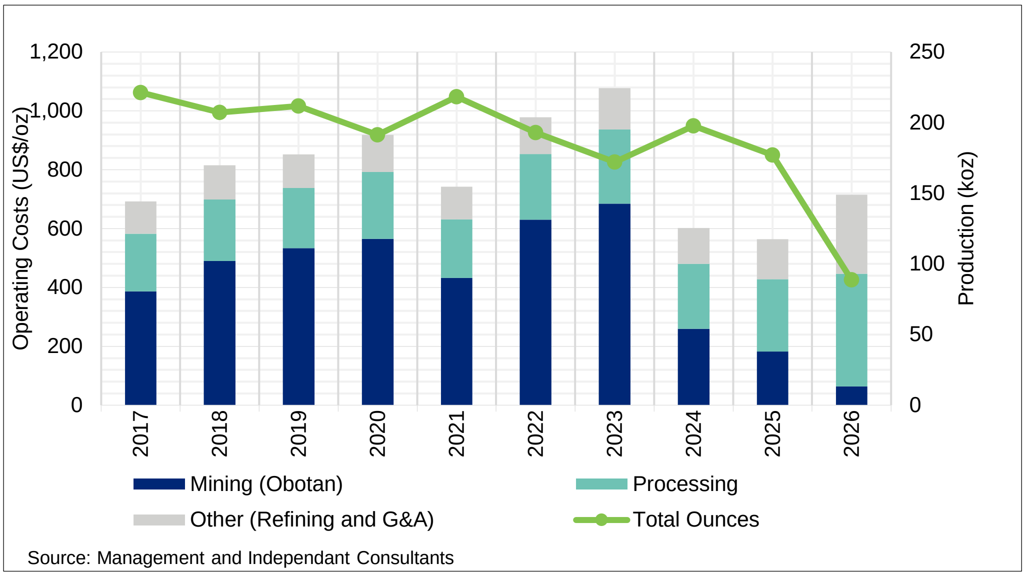


Figure 21-5: Base Case - Operating Cost US$/oz (source: Venmyn Deloitte 2017)

The operating costs accounted for in the financial model for Base Case + P5M + P10M over the scheduled LoM are summarised in Table 21-5, Figure 21-6 and Figure 21-7. The total Base Case + P5M + P10M opex/oz over the LoM is estimated to be US$761/oz. The largest contributors to the Base Case + P5M + P10M opex over the LoM is the Esaase mining at 37%, however this is closely followed by processing at 30% and Obotan mining at 24%.

Table 21-5: Base Case + P5M + P10M - Total Operating Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)	Amount (US$/oz)
Mining (Obotan)	903	186
Mining (Esaase)	1,351	279
Processing	1,109	229
Other (Refining and G&A)	323	67
Total OPEX	**3,686**	**761**

*Rounding applied

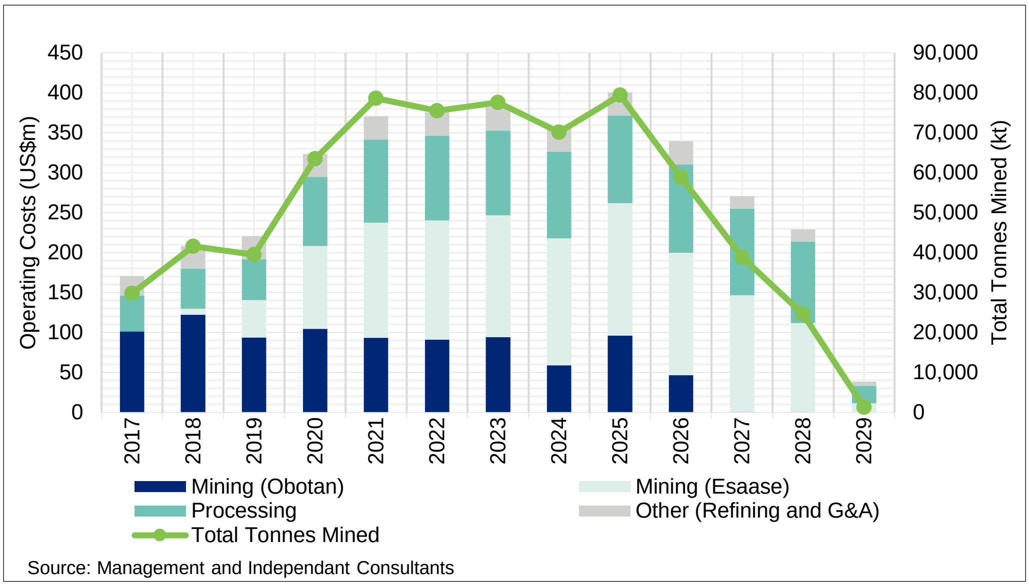


Figure 21-6: Base Case + P5M + P10M - Operating Cost Scheduling (source: Venmyn Deloitte 2017)

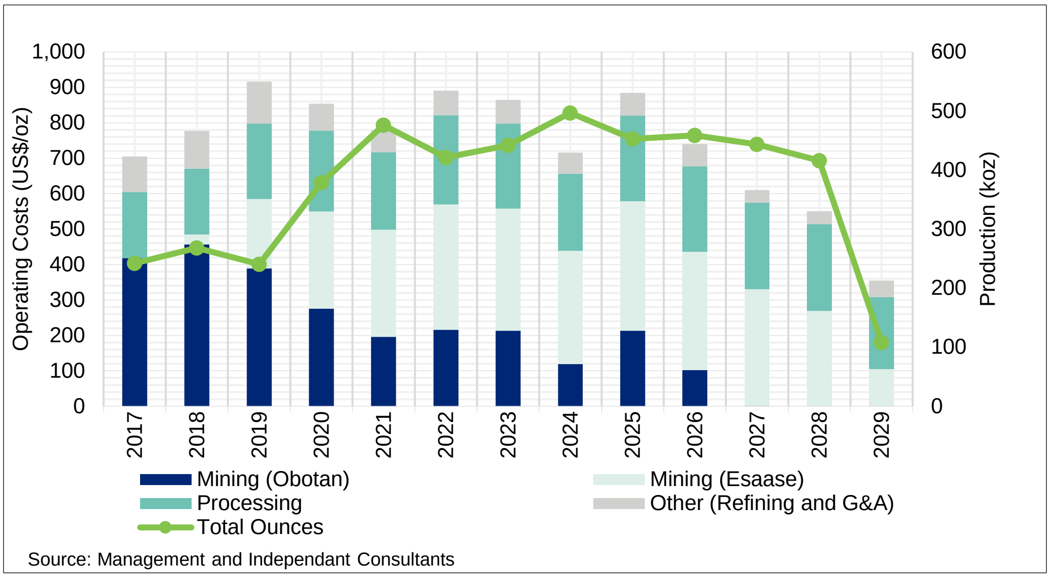


Figure 21-7: Base Case + P5M + P10M - Operating Cost US$/oz (source: Venmyn Deloitte 2017)

The operating costs accounted for in the financial model for Base Case + P5M over the scheduled LoM are summarised in Table 21-6, Figure 21-8 and Figure 21-9. The total Base Case + P5M opex/oz over the LoM is estimated to be US$837/oz. The largest contributors to the Base Case + P5M OPEX over the LoM is the Esaase mining at 36%, however this is closely followed by processing at 29% and Obotan mining at 22%. It must be noted that because of the increase in the LoM for the Base Case + P5M versus the Base Case + P5M + P10M, an increase of US$37/oz in the total opex/oz is observed.

Table 21-6: Base Case + P5M - Total Operating Costs (source: Venmyn Deloitte 2017)

Aspect	Amount (US$m)	Amount (US$/oz)
Mining (Obotan)	903	186
Mining (Esaase)	1,460	301
Processing	1,192	246
Other (Refining and G&A)	505	104
Total OPEX	**4,060**	**837**

*Rounding applied

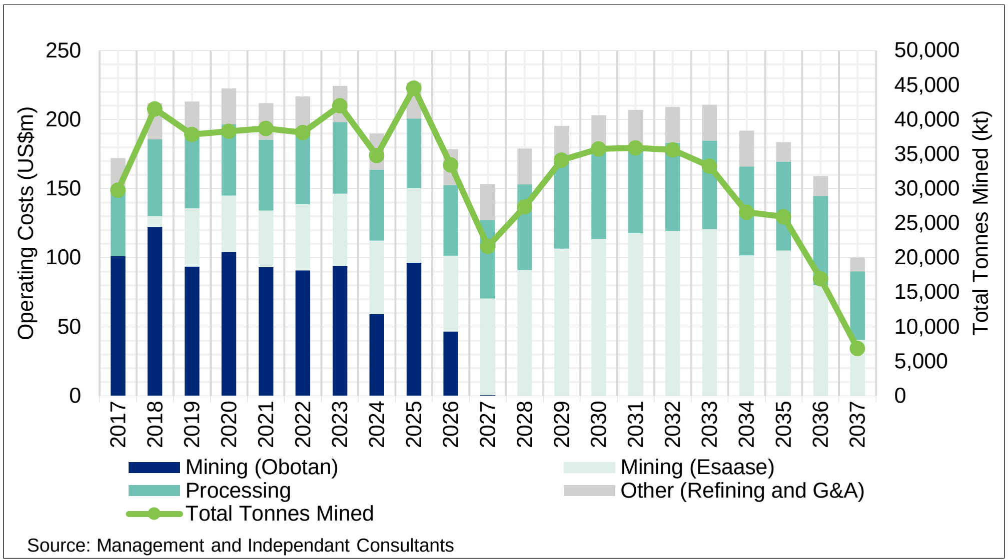


Figure 21-8: Base Case + P5M - Operating Cost Scheduling (source: Venmyn Deloitte 2017)

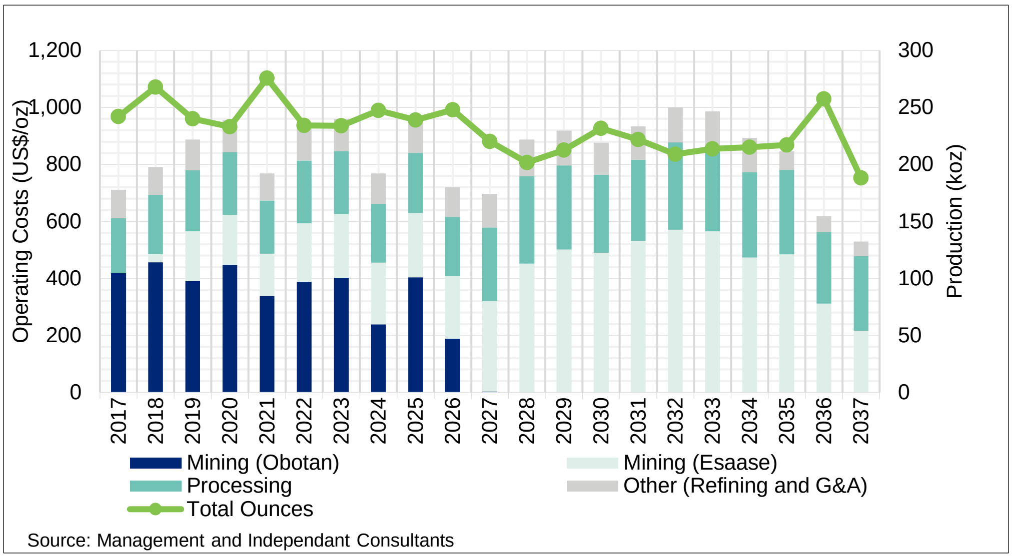


Figure 21-9: Base Case + P5M - Operating Cost US$/oz (source: Venmyn Deloitte 2017)

22 ECONOMIC ANALYSIS

22.1 The AGM – Economic Analysis

Venmyn Deloitte constructed a Discounted Cash Flow ("DCF") model for the purposes of the economic analysis of the Project. The DCF model was constructed in Excel and was based on techno-econimoc input assumptions that were derived from the studies on mining, processing, mine infrastructure, TSF designs, dam construction, electrical supply, owner's costs and indirect costs, which are summarised in Section 22. The DCF model assesses the post-tax real cash flows for the Project.

The results of the economic evaluation would be an indicator of the NPV of the Project given the quality and quantity of information provided by the contributing specialists and the quality of the estimates made on some inputs of the respective models.

22.1.1 Principle Assumptions

The principle economic assumptions employed in the economic analysis of the Project are presented in Table 22-1. The DCF model assumes that both revenue and costs, as well as royalty and taxes, are incurred in US$, therefore, no exchange rate assumptions are necessary. For the purposes of the economic analysis, Venmyn Deloitte used a discount rate of 5% and a Gold Price of US$1,250/oz. However, in the sensitivity analysis, the results are reported over a range of discount rates and commodity prices.

Table 22-1: Principle Economic Assumptions / Inputs (source: Venmyn Deloitte 2017)

Techno-Econoimc Assumptions/ Inputs	UOM	Base Case	P5M + P10M	P5M
Total Tonnes Mined	t	231,996.78	679,246.98	679,246.98
Ore Processed	t	35,143.00	102,744.59	102,744.59
Waste Mined	t	196,853.78	576,502.39	576,502.39
Stripping Ratio*	calc	5.60	5.61	5.61
Plant Grade* (CIL – Plant 1)	g/t	1.77	1.65	1.57
Plant Grade* (CIL - Plant 2)	g/t	-	1.46	-
Recovery* (CIL - Plant 1)	%	94	94	94
Recovery* (CIL - Plant 2)	%	-	93	-
Total Gold Recovered	Moz	1.88	4.84	4.85
Opening TAX Shield	US$m	417.20	417.20	417.20
Corporate Tax Rate	%	35.00%	35.00%	35.00%

Techno-Econoimc Assumptions/ Inputs	UOM	Base Case	P5M + P10M	P5M
Obotan Royality Rate	%	5.00%	5.00%	5.00%
Esaase Royality Rate	%	5.50%	5.50%	5.50%
Gold Price	US$/oz	1,250.00	1,250.00	1,250.00
Discount Rate	%	5.00%	5.00%	5.00%
Capital – Base	US$m	-	-	-
Capital – Project	US$m	-	349.42	146.79
Sustaining Capital	US$m	65.80	125.28	174.58
Cash Operating Costs*	US$/oz	796.95	760.81	837.20
Total Tonnes Mined	t	231,996.78	679,246.98	679,246.98

*Rounding applied

22.1.2 Cash Flow Approach

The annual cash flows for the Project for:

Base Case (source: Venmyn Deloitte 2017)

			2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
			JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN
	UNIT	SUMMATION										
MINING DRIVERS												
Obotan Total	(kt)	**231,997**	**26,583**	**33,211**	**32,707**	**31,581**	**27,346**	**39,274**	**29,618**	**8,151**	**3,525**	**0**
Esasse Total	(kt)	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
PROCESSING DRIVERS												
Material Processed	(kt)	**35,143**	3,590	3,601	3,600	3,600	3,601	3,602	3,600	3,600	3,600	2,749
Total Mineralised Tonnes Mined	**(kt)**	**33,195**	**3,975**	**4,190**	**4,438**	**2,797**	**4,185**	**3,994**	**4,046**	**3,055**	**2,515**	**0**
Total Tonnes Mined	**(kt)**	**231,997**	**26,583**	**33,211**	**32,707**	**31,581**	**27,346**	**39,274**	**29,618**	**8,151**	**3,525**	**0**
PROCESSING OUTPUTS												
Plant 1	(toz)	**1,878,728**	221,368	207,284	211,863	191,383	218,406	192,849	172,110	197,789	177,058	88,619
Plant 2	(toz)	**0**	0	0	0	0	0	0	0	0	0	0
Total Recovered Material		**1,878,728**	**221,368**	**207,284**	**211,863**	**191,383**	**218,406**	**192,849**	**172,110**	**197,789**	**177,058**	**88,619**
OPERATING INCOME												
Gold Revenue	(US$m)	**2,348**	277	259	265	239	273	241	215	247	221	111
Gold Price	(US$/oz)		1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Total Operating Income	**(US$m)**	**2,348**	**277**	**259**	**265**	**239**	**273**	**241**	**215**	**247**	**221**	**111**
OPERATING EXPENDITURE												
OBOTAN												
Mining Cost	(US$m)	**(827)**	(83.22)	(101.53)	(112.99)	(108.09)	(94.53)	(120.31)	(116.65)	(51.44)	(32.21)	(5.63)
Sustaining Costs	(US$m)	**(5)**	(2.37)	0.00	0.00	0.00	0.00	(1.31)	(1.29)	0.00	0.00	0.00
ESASSE			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mining Cost	(US$m)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Sustaining Costs	(US$m)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PROCESSING			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Processesing Costs	(US$m)	**(424)**	(43.33)	(43.24)	(43.38)	(43.47)	(43.36)	(42.96)	(43.42)	(43.52)	(43.52)	(33.98)
Other												
GnA	(US$m)	**(234)**	(23.40)	(23.40)	(23.40)	(23.40)	(23.40)	(23.40)	(23.40)	(23.40)	(23.40)	(23.40)
Refining	(US$m)	**(8)**	(0.89)	(0.83)	(0.85)	(0.77)	(0.87)	(0.77)	(0.69)	(0.79)	(0.71)	(0.35)
Total Operating Expenditure	**(US$m)**	**(1,497)**	**(153)**	**(169)**	**(181)**	**(176)**	**(162)**	**(189)**	**(185)**	**(119)**	**(100)**	**(63)**
CAPITAL EXPENDITURE												
Phase 1	(US$m)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
P5M	(US$m)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
P10M	(US$m)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
SIB	(US$m)	**(66)**	(2.89)	(2.88)	(2.19)	(3.13)	(2.99)	(2.57)	(3.11)	(2.91)	(3.98)	(39.15)
Total Capital Expenditure	**(US$m)**	**(66)**	**(3)**	**(3)**	**(2)**	**(3)**	**(3)**	**(3)**	**(3)**	**(3)**	**(4)**	**(39)**
Royalty Payment	(US$m)	**(117)**	(13.84)	(12.96)	(13.24)	(11.96)	(13.65)	(12.05)	(10.76)	(12.36)	(11.07)	(5.54)
Tax Payment	(US$m)	**(101)**	(9.18)	0.00	0.00	0.00	0.00	0.00	(3.95)	(39.47)	(37.56)	(11.04)
Post Tax Cash Flow	**(US$m)**	**567**	**98**	**74**	**69**	**48**	**94**	**38**	**12**	**73**	**69**	**(8)**
Net Cash Flow		**567**	**98**	**74**	**69**	**48**	**94**	**38**	**12**	**73**	**69**	**(8)**
Discounted Cash Flow	**(US$m)**	**482**	**98**	**71**	**62**	**42**	**77**	**30**	**9**	**52**	**47**	**(5)**

Base Case + P5M + P10M (source: Venmyn Deloitte 2017)

			2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
			JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN
	UNIT	SUMMATION														
MINING DRIVERS																
Obotan Total	(kt)	**268,069**	**29,802**	**41,558**	**29,082**	**28,265**	**28,030**	**23,988**	**25,516**	**17,931**	**27,579**	**16,318**	**0**	**0**	**0**	**0**
Esasse Total	(kt)	**411,178**	**0**	**0**	**10,376**	**35,247**	**50,604**	**51,528**	**52,095**	**52,232**	**51,809**	**42,560**	**38,876**	**24,590**	**1,260**	**0**
PROCESSING DRIVERS																
Material Processed	(kt)	**102,745**	3,950	5,012	5,000	8,450	10,000	10,000	9,890	10,000	10,000	10,000	10,000	8,713	1,729	0
Total Tonnes Mined	**(kt)**	**679,247**	**29,802**	**41,558**	**39,458**	**63,512**	**78,634**	**75,517**	**77,610**	**70,163**	**79,388**	**58,878**	**38,876**	**24,590**	**1,260**	**0**
PROCESSING OUTPUTS																
Plant 1	(toz)	**2,903,839**	242,007	267,866	240,572	234,113	269,967	222,638	227,345	257,664	243,374	245,107	227,507	174,130	51,549	0
Plant 2	(toz)	**1,940,977**	0	0	0	144,554	206,242	198,965	214,690	239,184	209,319	213,473	216,230	241,887	56,433	0
Total Recovered Material		**4,844,816**	**242,007**	**267,866**	**240,572**	**378,667**	**476,209**	**421,603**	**442,035**	**496,848**	**452,692**	**458,580**	**443,738**	**416,017**	**107,982**	**0**
OPERATING INCOME																
Gold Revenue	(US$m)	**6,056**	303	335	301	473	595	527	553	621	566	573	555	520	135	0
Gold Price	(US$m)		1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Total Operating Income	(US$m)	**6,056**	**303**	**335**	**301**	**473**	**595**	**527**	**553**	**621**	**566**	**573**	**555**	**520**	**135**	**0**
OPERATING EXPENDITURE																
OBOTAN																
Mining Cost	(US$m)	**(898)**	(98.62)	(121.44)	(93.68)	(104.38)	(93.20)	(90.93)	(92.48)	(59.05)	(96.31)	(46.70)	(0.63)	(0.16)	0.00	0.00
Sustaining Costs	(US$m)	**(5)**	(2.58)	(0.91)	0.00	0.00	0.00	0.00	(1.58)	0.00	0.00	0.00	0.00	0.00	0.00	0.00
ESASSE			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mining Cost	(US$m)	**(1,326)**	0.00	(5.86)	(43.69)	(100.89)	(141.67)	(146.05)	(148.92)	(156.24)	(164.11)	(151.47)	(144.94)	(111.28)	(11.03)	0.00
Sustaining Costs	(US$m)	**(25)**	0.00	(1.68)	(3.32)	(2.85)	(2.50)	(3.28)	(3.73)	(2.51)	(1.53)	(1.69)	(0.94)	(0.35)	(0.33)	0.00
PROCESSING			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Processesing Costs	(US$m)	**(1,109)**	(45.12)	(49.71)	(51.31)	(86.38)	(103.96)	(106.05)	(106.13)	(108.42)	(109.39)	(110.49)	(108.71)	(101.89)	(21.87)	0.00
Other																
GnA	(US$m)	**(304)**	(23.40)	(27.51)	(27.51)	(27.52)	(27.66)	(27.67)	(27.66)	(27.66)	(27.46)	(27.47)	(13.74)	(13.74)	(4.67)	0.00
Refining	(US$m)	**(19)**	(0.97)	(1.07)	(0.96)	(1.51)	(1.90)	(1.69)	(1.77)	(1.99)	(1.81)	(1.83)	(1.77)	(1.66)	(0.43)	0.00
Total Operating Expenditure	(US$m)	**(3,686)**	**(171)**	**(208)**	**(220)**	**(324)**	**(371)**	**(376)**	**(382)**	**(356)**	**(401)**	**(340)**	**(271)**	**(229)**	**(38.3)**	**0**
CAPITAL EXPENDITURE																
Phase 1	(US$m)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
P5M	(US$m)	**(150)**	(61.43)	(69.79)	(18.47)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
P10M	(US$m)	**(200)**	0.00	(14.70)	(137.86)	(47.13)	(0.05)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
SIB	(US$m)	**(125)**	(6.76)	(7.38)	(1.90)	(7.24)	(8.45)	(9.62)	(18.72)	(8.21)	(11.43)	(6.98)	(4.69)	(20.40)	(1.20)	(12.30)
Total Capital Expenditure	(US$m)	**(475)**	**(68)**	**(92)**	**(158)**	**(54)**	**(9)**	**(10)**	**(19)**	**(8)**	**(11)**	**(7)**	**(5)**	**(20)**	**(1)**	**(12)**
Royalty Payment	(US$m)	**(320)**	(15.13)	(16.74)	(15.45)	(25.01)	(31.47)	(27.95)	(29.15)	(32.96)	(30.09)	(30.53)	(29.63)	(28.46)	(7.42)	0.00
Tax Payment	(US$m)	**(416)**	(7.07)	0.00	0.00	0.00	0.00	0.00	(22.61)	(74.32)	(43.35)	(67.22)	(85.51)	(88.25)	(28.10)	0.00
Post Tax Cash Flow	(US$m)	**1,159**	**41**	**18**	**(93)**	**70**	**184**	**114**	**100**	**150**	**80**	**129**	**164**	**154**	**60**	**(12)**
Net Cash Flow		**1159**	**41**	**18**	**(93)**	**70**	**184**	**114**	**100**	**150**	**80**	**129**	**164**	**154**	**60**	**(12)**
Discounted Cash Flow	(US$m)	**811.39**	**41**	**17**	**(85)**	**61**	**152**	**89**	**74**	**106**	**54**	**83**	**101**	**90**	**33**	**(7)**

Base Case + P5M (source: Venmyn Deloitte 2017)

			2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2037
	UNIT	SUMMATION	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN	JAN
MINING DRIVERS																								
Obotan Total	(kt)	**268,069**	**29,802**	**41,558**	**29,082**	**28,265**	**28,030**	**23,988**	**25,516**	**17,931**	**27,579**	**16,318**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Esasse Total	(kt)	**411,178**	**0**	**0**	**8,764**	**10,051**	**10,685**	**14,112**	**16,501**	**16,874**	**16,989**	**17,108**	**21,654**	**27,401**	**34,146**	**35,751**	**35,902**	**35,630**	**33,244**	**26,603**	**25,925**	**16,984**	**6,853**	0
PROCESSING DRIVERS																								
Material Processed	(kt)	**102,745**	3,950	5,012	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	3,783	0
Total Mineralised Tonnes Mined	**(kt)**	**177,312**	**4,710**	**5,228**	**6,940**	**7,435**	**7,330**	**6,839**	**8,028**	**9,214**	**11,180**	**7,655**	**5,899**	**7,243**	**8,822**	**9,870**	**11,426**	**12,297**	**12,244**	**10,590**	**10,459**	**9,065**	**4,836**	**0**
Total Tonnes Mined	**(kt)**	**679,247**	**29,802**	**41,558**	**37,845**	**38,316**	**38,715**	**38,100**	**42,017**	**34,804**	**44,568**	**33,426**	**21,654**	**27,401**	**34,146**	**35,751**	**35,902**	**35,630**	**33,244**	**26,603**	**25,925**	**16,984**	**6,853**	**0**
PROCESSING OUTPUTS																								
Plant 1	(toz)	**4,849,653**	242,024	267,943	240,166	233,062	275,765	234,247	233,883	247,305	238,951	247,855	220,260	201,734	212,697	231,582	221,808	208,849	213,764	214,986	217,144	257,518	188,110	0
Plant 2	(toz)	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Recovered Material		**4,849,653**	**242,024**	**267,943**	**240,166**	**233,062**	**275,765**	**234,247**	**233,883**	**247,305**	**238,951**	**247,855**	**220,260**	**201,734**	**212,697**	**231,582**	**221,808**	**208,849**	**213,764**	**214,986**	**217,144**	**257,518**	**188,110**	**0**
OPERATING INCOME																								
Gold Revenue	**(US$M)**	**6,062**	303	335	300	291	345	293	292	309	299	310	275	252	266	289	277	261	267	269	271	322	235	0
Gold Price	(US$/oz)		1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	0
Total Operating Income	**(US$M)**	**6,062**	**303**	**335**	**300**	**291**	**345**	**293**	**292**	**309**	**299**	**310**	**275**	**252**	**266**	**289**	**277**	**261**	**267**	**269**	**271**	**322**	**235**	**0**
OPERATING EXPENDITURE																								
OBOTAN																								
Mining Cost	**(US$M)**	**(898)**	(98.62)	(121.44)	(93.68)	(104.38)	(93.20)	(90.93)	(92.48)	(59.05)	(96.31)	(46.70)	(0.63)	(0.16)	0	0	0	0	0	0	0	0	0	0
Sustaining Costs	**(US$M)**	**(5)**	(2.58)	(0.91)	0.00	0.00	0.00	0.00	(1.58)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ESASSE																								
Mining Cost	**(US$M)**	**(1,436)**	0.00	(5.86)	(38.25)	(37.60)	(38.59)	(45.83)	(50.64)	(51.91)	(52.54)	(53.44)	(68.64)	(89.78)	(105.63)	(112.73)	(117.37)	(119.09)	(120.45)	(101.39)	(105.01)	(80.19)	(40.67)	0.00
Sustaining Costs	**(US$M)**	**(25)**	0.00	(1.94)	(3.76)	(3.00)	(2.34)	(2.10)	(1.72)	(1.54)	(1.40)	(1.28)	(1.33)	(1.18)	(0.94)	(0.83)	(0.43)	(0.14)	(0.28)	(0.28)	(0.14)	(0.06)	0.00	0.00
PROCESSING																								
Processesing Costs	**(US$M)**	**(1,192)**	(46.63)	(55.67)	(51.43)	(51.59)	(51.38)	(51.68)	(51.75)	(51.26)	(50.48)	(51.11)	(56.81)	(61.97)	(62.93)	(63.46)	(63.38)	(63.89)	(64.12)	(64.38)	(64.30)	(64.52)	(49.48)	0.00
Other																	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
GnA	(US$M)	**(486)**	(23.40)	(25.07)	(25.07)	(25.07)	(25.19)	(25.19)	(25.19)	(25.19)	(25.07)	(25.07)	(25.07)	(25.07)	(25.07)	(25.07)	(25.07)	(25.07)	(25.07)	(25.07)	(13.37)	(13.37)	(8.69)	0.00
Refining	(US$M)	**(19)**	(0.97)	(1.07)	(0.96)	(0.93)	(1.10)	(0.94)	(0.94)	(0.99)	(0.96)	(0.99)	(0.88)	(0.81)	(0.85)	(0.93)	(0.89)	(0.84)	(0.86)	(0.86)	(0.87)	(1.03)	(0.75)	0.00
Total Operating Expenditure	(US$M)	**(4,060)**	**(172)**	**(212)**	**(213)**	**(223)**	**(212)**	**(217)**	**(224)**	**(190)**	**(227)**	**(179)**	**(153)**	**(179)**	**(195.4)**	**(203)**	**(207)**	**(209)**	**(211)**	**(192)**	**(184)**	**(159)**	**(100)**	**0**
CAPITAL EXPENDITURE																								
Phase 1	(US$M)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
P5M	(US$M)	**(147)**	(61.56)	(66.84)	(18.38)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
P10M	(US$M)	**0**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
SIB	(US$M)	**(175)**	(6.76)	(6.61)	(17.18)	(17.31)	(1.90)	(9.62)	(5.20)	(8.49)	(1.90)	(8.49)	(1.90)	(26.99)	(1.20)	(21.72)	(4.13)	(7.79)	(1.20)	(10.29)	(1.20)	(1.20)	(1.20)	(12.30)
Total Capital Expenditure	**(US$M)**	**(321)**	**(68)**	**(73)**	**(36)**	**(17)**	**(2)**	**(10)**	**(5)**	**(8)**	**(2)**	**(8)**	**(2)**	**(27)**	**(1)**	**(22)**	**(4)**	**(8)**	**(1)**	**(10)**	**(1)**	**(1)**	**(1)**	**(12)**
Royalty Payment	(US$M)	**(320)**	(15.13)	(16.75)	(15.47)	(15.09)	(17.70)	(15.18)	(15.13)	(15.83)	(15.10)	(15.78)	(14.13)	(13.83)	(14.62)	(15.92)	(15.25)	(14.36)	(14.70)	(14.78)	(14.93)	(17.70)	(12.93)	0.00
Tax Payment	(US$M)	**(337)**	(6.54)	0.00	0.00	0.00	0.00	0.00	0.00	(14.13)	(17.99)	(38.04)	(35.92)	(17.44)	(16.70)	(20.47)	(15.29)	(8.86)	(12.08)	(18.53)	(23.76)	(49.24)	(41.86)	0.00
Post Tax Cash Flow	(US$M)	**1,023**	**40**	**33**	**36**	**36**	**113**	**51**	**48**	**81**	**37**	**69**	**70**	**15**	**38**	**28**	**35**	**21**	**28**	**33**	**48**	**95**	**80**	**(12)**
Net Cash Flow		**1,023**	**40**	**33**	**36**	**36**	**113**	**51**	**48**	**81**	**37**	**69**	**70**	**15**	**38**	**28**	**35**	**21**	**28**	**33**	**48**	**95**	**80**	**(12)**
Discounted Cash Flow	**(US$M)**	**658**	**40**	**31**	**33**	**31**	**93**	**40**	**36**	**57**	**25**	**44**	**43**	**9**	**21**	**15**	**18**	**10**	**13**	**14**	**20**	**37**	**30**	**(4)**

22.1.3 NPV, IRR and Capital Payback Period

Net Present Value (NPV), internal rate of return (IRR) and payback time are typically used as indicators of project performance and for evaluation. As the AGM is an operating mine, the NPV and IRR are considered to be the most appropriate indicator of economic performance for this mineral asset.

In consideration of the above statement and basis of estimate assumption detailed in this document, the following NPVs and IRRs were reached, at a discount rate of 5% and gold price of US$1,250/oz:-

Base Case:

NPV: US$481.82m

IRR: N/A as net cash flow of Base Case starts positively

Base Case + P5M + P10M:

NPV: US$811.39m

IRR: N/A as net cash flow of Base Case + Project starts positively

Base Case + P5M:

NPV: US$657.72m

IRR: N/A as net cash flow of Base Case + Project starts positively

Project (P5M + P10M Incremental):

NPV: US$329.57m

IRR: 20.37%

Project (P5M Incremental):

NPV: US$175.89m

IRR: 13.22%

In all of the scenarios investigated for the AGM, where a 5% discount rate and US$1,250/oz was applied, a resultant positive NPV is shown. This highlights positive performance of the asset under differing scenarios. P5M + P10M shows a better NPV and IRR period than that of P5M, however P5M + P10M requires approximately US$202 million more installation capital than P5M.

In consideration of the above assumptions, the payback period for P5M + P10M is four years, and P5M is five years. P5M + P10M shows a better payback period than that of P5M, however P5M + P10M requires approximately US$202 million more installation capital than P5M.

22.1.4 Taxes, Royalties and Other Government Levies

22.1.4.1 Taxes

While the general corporate income tax (“CIT”) in Ghana is 25%, mining companies are required to pay a CIT of 35%. Depreciation of depreciable assets of a business is not a permissible deduction in deriving taxable profits. In its stead, capital allowances at prescribed statutory rates for mining companies currently are marked at 20% straight line. Both of these, CIT and taxable profits calculations, have been applied into the DCF model.

22.1.4.2 Royalties

The Ghanaian government has implemented a 5% mineral royalty, which has been taken into account for this Project.

22.1.4.3 Other Government Levies

By agreement with BLC, there is an additional 0.5% royalty applicable to the Esaase operations, resulting in the royalty applied being 5% for Obotan and 5.5% for Esaase.

22.2 Sensitivity Analyses

The cash flow model was subjected to a sensitivity and scenario analysis in order to assess the effect of changes in significant cost and revenue drivers on the NPV. The results of the sensitivity analysis against the respective scenarios are summarised in **Figure 22-1**, **Figure 22-2** and **Figure 22-3** after the sensitivities represented in **Table 22-2** were applied. Differences in the results of the scenario analysis and further sensitivities may be due to rounding and/or non-linear relationships modelled.

Table 22-2: Sensitivity Factors Applied (source: Venmyn Deloitte 2017)

Aspect	Sensitivity
Commodity Price	10%
Operating Costs	10%
Grade	0.1g/t
Discount Rate	1% Differential
Recovery	2%
Grade	0.1 g/t

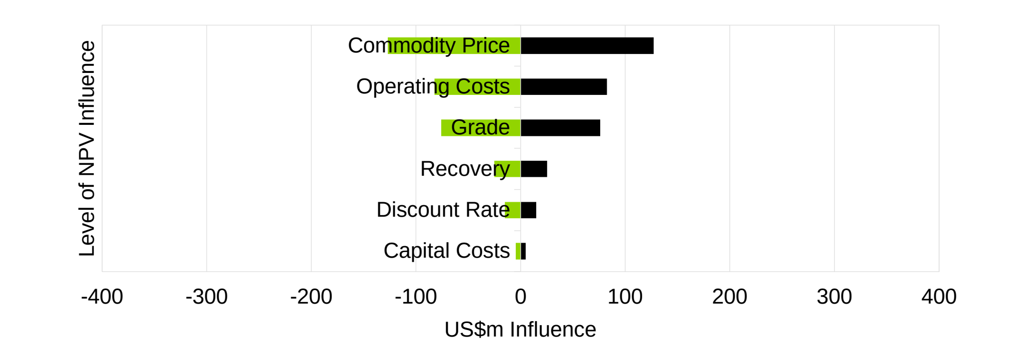


Figure 22-1: Base Case Sensitivity Analysis (source: Venmyn Deloitte 2017)

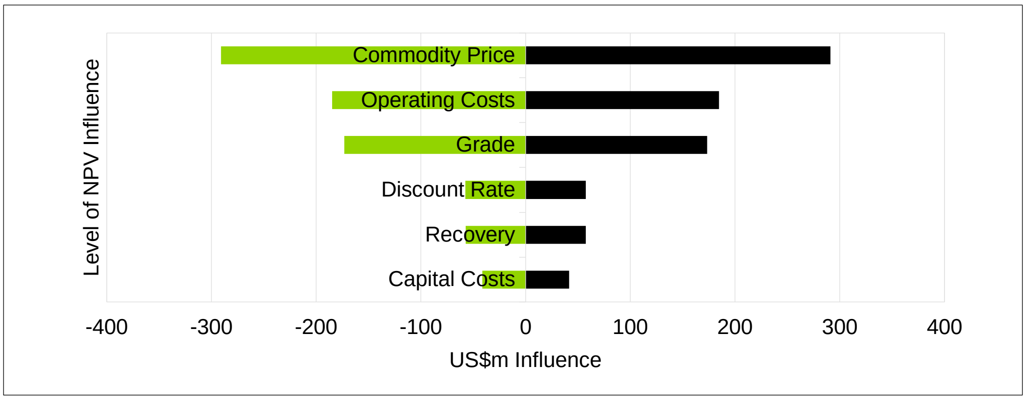


Figure 22-2: Base Case + P5M + P10M Sensitivity Analysis (source: Venmyn Deloitte 2017)

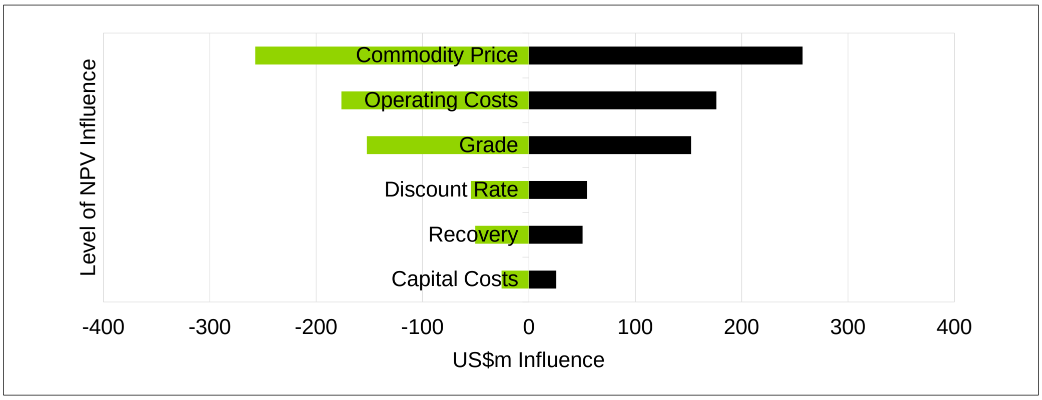


Figure 22-3: Base Case + P5M Sensitivity Analysis (source: Venmyn Deloitte 2017)

Venmyn Deloitte investigated the sensitivity of the Project NPV to different input parameters, namely commodity pricing, operating costs, grade, discount rate, recovery and capital costs associated with the Project. From this analysis the NPV generated from the DCF models proved to be most sensitive to changes in parameters affecting revenue, most notably commodity pricing. Following commodity pricing next most influential parameters are operating costs and grade. The impact that changes in the combined discount rate, recovery and capital costs had on the Project is smaller than that resulting from changes in grade alone. It was observed that capital costs had the smallest level of influence in the sensitivity analysis as compared to the other investigated factors.

To further test the impact of the commodity price, an iterative scenario analysis was conducted. This scenario analysis takes into consideration both changes in Discount Rate and the gold price applied. The results of the scenario analysis are summarised in **Table 22-3**, **Table 22-4**, **Table 22-5**, **Table 22-6** and **Table 22-7**.

Table 22-3: Base Case Scenario Analysis (source: Venmyn Deloitte 2017)

Gold Price (US$/oz)	NPV (US$m) at Discount Rate (%)				
	3	**4**	**5**	**6**	**7**
1,050	278	268	259	251	243
1,150	406	393	380	369	358
1,250	513	497	482	468	455
1,350	617	598	579	562	546
1,450	720	697	675	655	636

*Rounding applied

Table 22-4: Base Case + P5M + P10M Scenario Analysis (source: Venmyn Deloitte 2017)

Gold Price (US$/oz)	NPV (US$m) at Discount Rate (%)				
	3	**4**	**5**	**6**	**7**
1,050	415	375	338	305	275
1,150	677	626	580	538	499
1,250	932	869	811	759	711
1,350	1,183	1,107	1,037	974	915
1,450	1,433	1,344	1,262	1,187	1,118

*Rounding applied

Table 22-5: Base Case + P5M Scenario Analysis (source: Venmyn Deloitte 2017)

Gold Price (US$/oz)	NPV (US$m) at Discount Rate (%)				
	3	4	5	6	7
1,050	291	258	229	205	183
1,150	540	494	453	417	386
1,250	775	712	658	610	567
1,350	1,006	926	857	795	741
1,450	1,235	1,138	1,053	978	912

*Rounding applied

Table 22-6: P5M + P10M Scenario Analysis (source: Venmyn Deloitte 2017)

Gold Price (US$/oz)	Incremental NPV (US$m) at Discount Rate (%)					Incremental IRR (%)
	3	4	5	6	7	
1,050	137	106	79	54	32	8.65%
1,150	271	233	199	169	141	14.50%
1,250	419	372	330	291	256	20.37%
1,350	566	510	458	412	369	25.79%
1,450	713	647	586	532	482	31.06%

*Rounding applied

Table 22-7: P5M Scenario Analysis (source: Venmyn Deloitte 2017)

Gold Price (US$/oz)	Incremental NPV (US$m) at Discount Rate (%)					Incremental IRR (%)
	3	4	5	6	7	
1,050	13	-11	-30	-46	-60	3.52%
1,150	134	101	73	48	28	8.62%
1,250	262	216	176	142	112	13.22%
1,350	389	329	277	233	195	17.29%
1,450	515	441	377	323	276	21.06%

*Rounding applied

A US$100/oz decrease in revenue, as a result of a gold price reduction, reduces the NPV by approximately US$131m for the P5M + P10M Scenario and by US$103m for the P5M Scenario.

In a similar fashion a US$100/oz decrease in revenue, as a result of a gold price reduction, reduces the IRR by approximately 5.87% for the P5M + P10M Scenario and by 4.60% for the P5M Scenario.

The analysis summarised in **Table 22-3**, **Table 22-4**, **Table 22-5**, **Table 22-6** and **Table 22-7** highlights that P5M + P10M is more resilient than P5M with regards to movement in gold price per ounce. This resilence is highlighted by the fact that a 16% reduction in the gold price would still yield a positive NPV and IRR result, while for the P5M Scenario it would result in a negative NPV and IRR.

23 ADJACENT PROPERTIES

Properties adjacent to the Asanko Gold Project area and tenements are shown below in Figure 23-1. The AGM properties are shown in bright red and are named. The property listing is shown in Table 23-1. These properties are all located within the Kumasi basin, and share similar underlying deformed siliciclastic metasediments as the primary rock type, with a range of syn- to late tectonic granite intrusives mainly to the east of the AGM tenements. None of these adjacent properties host mineral resources that are NI 43-101 compliant.

Table 23-1: Adjacent Property Listing (source: Asanko Gold 2016)

Tenement / PL Number	Tenement Owner
137	Tropical Minerals Co. Ltd
91	Moseaso Co. Ltd
155	Joam Enterprise Ltd

169	Rock and Rivers
19	AngloGold Ashanti
234	Triple Key Co. Ltd
150	Romex
138	U & N Ltd
145	Westminister
257	Star Gold Ltd

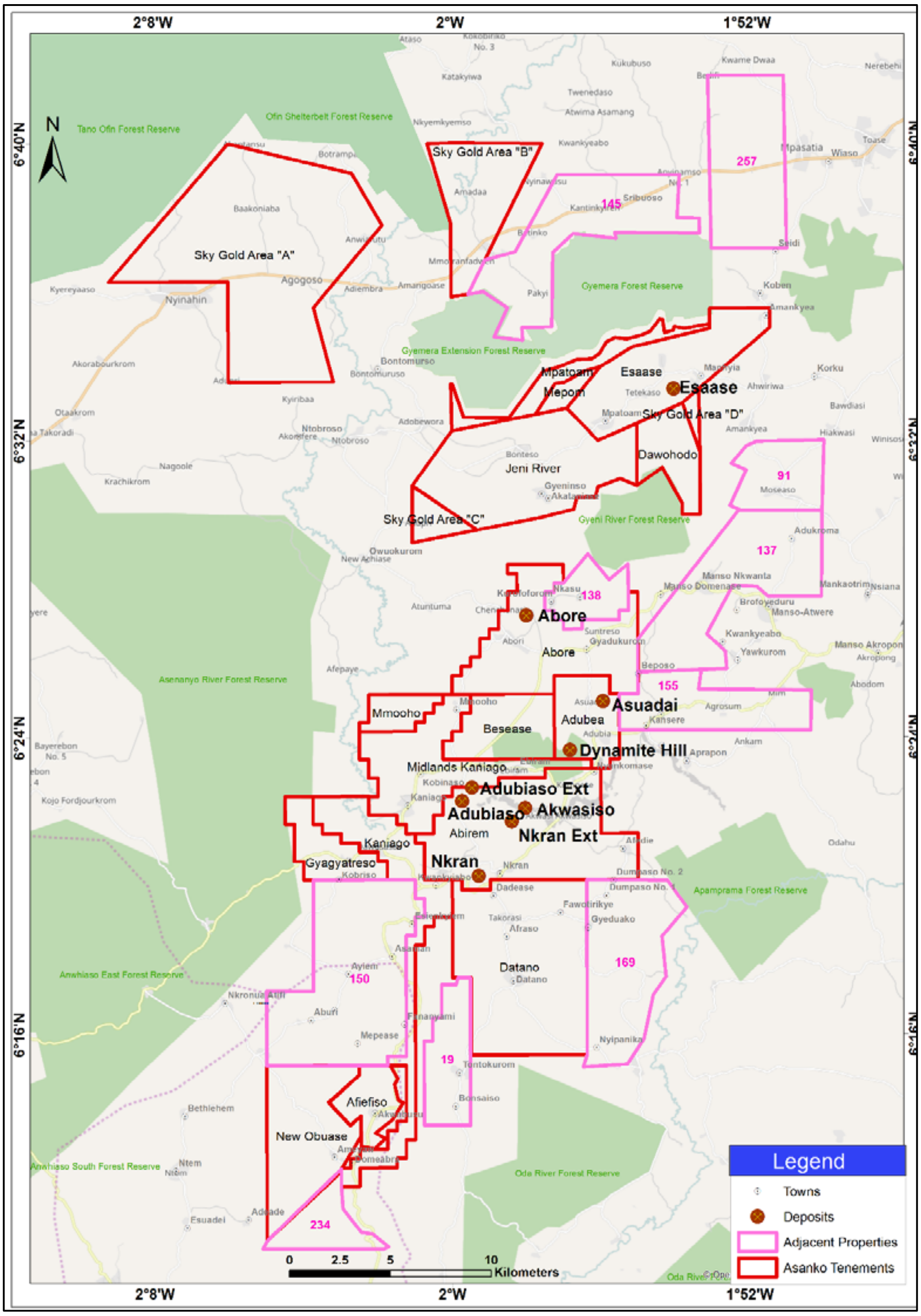


Figure 23-1: AGM tenements and adjacent properties. (source: Asanko Gold 2017, Ghana Minerals Commission 2016)

24 OTHER RELEVANT DATA AND INFORMATION

All relevant data and information has been reported in other sections of this report.

25 INTREPRETATIONS AND CONCLUSION

The start of the P5M FEED has been approved by the Asanko Gold Board.

25.1 Project Risks

A risk assessment has been conducted as part of the study to generate a risk register for on-going risk management with respect to the Project. The risk assessment conducted only included mining, processing, approvals and environmental, financial, social, infrastructure and corporate issues.

The remaining high risks, after controls are:

- Lower than expected grades
- Project scope creep during execution, especially regards community expectations
- Tropical diseases during construction
- Insufficient supply of water to operate the P10M plant at full capacity

25.2 Geology & Resource Estimates Risks

The bulk of the DFS MREs have been erected by CSA and have de-risked the basis of the MRevs for the DFS.

The MRE tabled in this disclosure are all located in the Amansie West District of southwest Ghana. External risks pertaining to these MRE are related to factors including:

- Ghanaian socio-political issues which are not under the control of Asanko Gold, and which include the stability of local communities, security of tenure and Government intervention on the right to mine, punitive environmental legislation, and increased taxes and duties
- A significant down turn in the gold price which could render the ore bodies sub economic. Notably the current operations have a break-even grade of around 0.45 g/t Au, compared to a LoM Reserve grade of 1.93 g/t Au
- Asanko Gold is a publicly listed company. Market pressures can impact on the ability of Asanko Gold to conduct normal business due to short positions taken by speculative investment research companies or capital hedge funds
- Geotechnical failure of open pit sidewall/s, which would potentially delay profitable mining operations and impact on the extraction of resources

25.3 Mining & Reserves

The Nkran pit has been in production since February 2015 with a well established mining contractor – PW Mining (Ghana). The DFS is predicated on PW Mining (Ghana) continuining to mine the major pits of Nkran and Esaase, thereby materially de-risking mining deliverables.

In addition, MRevs have been calculated based on standard and conventional mining methodologies as described in section 15.

25.4 Processing

25.4.1 Process Flowsheet

The proposed AGM P10M processing flowsheet is based on the current AGM Phase 1 operating plant which will be expanded to P5M utilising a conventional gravity-CIL flow sheet with low technical risk. The design, to a large degree, reflects the flowsheet of the process plant that had previously successfully been operated at Obotan under Resolute.

25.4.2 Mill Throughput

The throughput capacity of the AGM Phase 1 SAG and ball mill installation is dependent on the BBWi and feed top size of the SAG mill feed. The 85th percentile of the BBWi data from various comminution test campaigns was used to confirm treatment rates of the existing SAG and ball mill (for the P5M upgrade). A SAG mill feed blend BBWi in excess of 15.1 kWh/t will result in a coarser than targeted product grind. This can be managed by blending of the softer Oxide material with the more competent Fresh material in suitable blends. Provision is further made in the design to include a secondary crushing stage in the Phase 1 crushing circuit, to provide suitably sized SAG mill feed in order to process 5 Mtpa when processing competent material.

25.4.3 Recovery

The existing AGM Phase 1 circuit has been in operation since February 2016 and as a consequence emperical performance data is available and forms the basis of the DFS.

Testing for variability between the Nkran, Adubiaso, Abore, and Dynamite Hill pits on Fresh ore was conducted at SGS Booysens and the results showed there was little variation between the pits and ore types except for the Adubiaso pit.

It is noted that no independent test work data is available on the processing of Oxide material from the Akwasiso, Asuadai, Nkran Extension, and the Adubiaso Extension satellite pits. Internal testing at AGM on the Akwasiso and Adubiaso Oxide material returned recoveries over 95%. It is, however, noted as a low risk, seeing that Oxide material from these satellite pits contributes to less than 2% of the overall ounces treated over the LoM of the AGM DFS. The recoveries of these satellite pits were benchmarked against the test work data available for the Oxide material tested from other deposits including Dynamite Hill and Nkran, as well as historical operating data on Abore.

Limited test work was done on the processing of Esaase Fresh material in a gravity-CIL at a grind of 106 µm, however, when benchmarked to the Nkran Fresh material data, the results indicated similar responses between the two deposits (see Section 13.4.3.2). The whole ore leach test on the Esaase Fresh material highlighted the requirement for an extensive gravity recovery circuit, which has been included in the design.

25.5 Infrastructure

The overland conveyor route in the DFS is now informed by a full Lidar and Geotechnical survey, significantly reducing the construction and capital risk.

A history of stable power supply during Phase 1 operations under pins the performance of P5M and P10M.

25.6 Economic Outcomes

The key driver of project value is the gold price. Under the P5M and P10M scenarios, a zero pre-tax NPV is only reached if prices fall by nearly 25% from the DFS price of US$1,250/oz.

This suggests that the project is relatively robust and significant reductions in gold price are required to endanger the project.

26 RECOMMENDATIONS

26.1 Geology and Resources

The following recommendations are noted:

- The MRE process is now handled on an in-house basis, and an independent 3rd party QP is used for auditing and sign-off / compliance issues. The next step to be achieved is for all the AGM mineral deposits to be modelled and estimated in-house. The target is during Q4 2017, in time for the December 2017 disclosure
- Geological models for all of the mining operations require systematic updating at the same time as Grade Control model updates
- Continued development of the density database for Oxide, Transition and Fresh material
- Conduct a detailed review of duplicate sampling methods and upgrade all the drilling rigs to have the same rotary splitter
- The integration of a spectrometer in the borehole assay and logging procedures(for alteration mineral diagnosis and identification) will improve the geological modelling, and provide further information to enable vectoring economic zones of mineralisation

26.2 Mining

No further recommendations.

26.3 Processing

It is recommended that further variability test work be conducted on the Esaase sulphide material to further substantiate the benchmarking against the Nkran material and the similarities between the two deposits.

26.4 Economic Analysis

No further recommendations.

26.5 Project

No further recommendations.

27 REFERENCES

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27.2 Geology

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27.3 Geotechnical

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27.4 Processing

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27.5 Infrastructure

Knight Piésold Consulting (2014). Obotan Gold Project – Tailings Storage Facility Expansion to 78 MT

Knight Piésold Consulting (2017) Tailings Storage Facility Feasibility Study update

28 CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE of AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a) Name, Address, Occupation:

Charles Johannes Muller

CJM Consulting (Pty) Ltd

Suite 3, Building 1

Ruimsig Office Estate

199 Hole-in-one Road

Ruimsig, South Africa

Director – Mineral Resources

b) Title and Effective Date of Technical Report

Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c) Qualifications

I graduated with a B.Sc. (Geology) degree from the Rand Afrikaans University - Johannesburg in 1988. In addition I have obtained a B.Sc. Hons (Geology) from Rand Afrikaans University in 1994 and attended courses in geostatistics through the University of the Witwatersrand.

I am a Registered Professional Scientist with the South African Council for Natural Scientific Professions, Pr. Sci. Nat. Reg. No. 400201/04.

My relevant experience, includes 21 years as a consulting mineral resource estimation geologist in gold, uranium and platinum for a number of early and advanced listed underground and open cast gold mining companies throughout Africa. I have resource estimation experience on deposits within Ghana including Tarkwa gold Mine 2017, Goldfields Damang 2016, Keegan Resources 2012 and other gold resource estimation reports including Sibanye Gold 2014 in South Africa.

I have read the definition of “qualified person” set out by National Instrument 43101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.

d) Site Inspection:

I visited Esaase project in October 2012 and Obotan projects 1st to 3rd of September 2014

e) Responsibilities

I am responsible for the following sections: 1.8, 11, 12, 14.

f) Independence:

I am independent of Asanko Gold Mine and Asanko Gold Incorporated in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I have been involved with the development of the resource model at Esaase and Obotan.

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

Charles J Muller Director

Mineral Resources CJM Consulting (Pty) Ltd

CERTIFICATE of AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a. Name, Address, Occupation

   David J T Morgan

   Knight Piesold Pty Ltd, Level 1, 184 Adelaide Terrace, East Perth, Western Australia, Australia

   Managing Director

b. Title and Effective Date of Technical Report

   Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

   Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c. Qualifications

   I am a graduate of University of Manchester, (B.Sc. Civil Engineering, 1980), and the University of Southampton (MSc. Irrigation Engineering 1981). I am a member in good standing of the Australasian Institute of Mining and Metallurgy (Australasia 202216) and a Chartered Professional Engineer and member of the Institution of Engineers Australia (Australia, 974219).

   My relevant experience as tailings dam project director includes (i) in Ghana, the Akyem Gold Mine as Tailings dam Project Director from 2002 to 2014, the Ahafo Gold Mine from 2002 to 2014, as Tailings dam Project Director for Asanko Gold from 2013 and ongoing; (ii) in Burkina Faso, the Hounde Gold Mine from 2012 and ongoing, as well as the Ity Gold Mine from 2012 and ongoing; (iii) in Cote d'lvoire the Agbaou Gold Mine from 2014 and ongoing; and (iv) in Australia the Boddington Gold Mine from 2000 and ongoing.

   I have read the definition of "qualified person" set out by National Instrument 43101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.

d. Site Inspection

   I have visited the Project site 22nd – 23rd Feb 2016

e. Responsibilities

I am responsible for sections 1.10, 18.7

f. Independence

I am independent of Asanko Gold Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

g. Prior Involvement

I am responsible for the design and construction of the tailings dam at Obotan.

h. Compliance with NI 43-101

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i. Disclosure

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

David Morgan

Managing Director

CERTIFICATE of AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a. Name, Address, Occupation

Doug Heher

DRA Projects (Pty) Ltd, DRA Minerals Park, 3 Inyanga Close, Sunninghill, 2157, South Africa

Project Manager

b. Title and Effective Date of Technical Report

Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c. Qualifications

I graduated with a B.Sc. Engineering (Mechanical) degree from the University of KwaZulu Natal - Durban in 1992; I am a member of the following professional associations: Registered Professional Engineer with the Engineering Council of South Africa (ECSA – Reg. No. 990333).

Since 1997 I have worked in the mining industry as a mechanical Engineer on various process plant installations within South Africa. I have worked as an execution and feasibility project study manager for 10 years on Pekoa Zinc, Burkina Faso - 2008, BMC Copper DRA 2010, Energizer Graphite in Madagascar 2012 and Asanko Gold from 2013 and ongoing.

I have read the definition of “qualified person” set out by National Instrument 43101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.

d. Site Inspection

I have visited the Project site from21 -25 March 2016

e. Responsibilities

I am responsible for sections 1.1, 1.13, 1.2, 1.3, 1.11, 1.16, 2, 3, 18, 19, 20, 24, 25.1, 25.5, 26.5 and 27.5

f. Independence

I am independent of Asanko Gold Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

g. Prior Involvement

I have been involved in all feasibility studies for Asanko as Study manager from 2013.

h. Compliance with NI 43-101

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i. Disclosure

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

Doug Heher.

Doug Heher

Senior Project Manager

CERTIFICATE of AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a. Name, Address, Occupation

   Malcolm Titley

   46 Barttelot Road, Horsham, England, RH12 1DQ

   Geologist – Principal Consultant

b. Title and Effective Date of Technical Report

   Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

   Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c. Qualifications

   BSc Geology and Chemistry, University of Cape Town, South Africa, 1979

   Member AIG and AusIMM

   I have over 36 years' experience in mineral resource estimation, mineral reserve estimation and third-party audits, open pit operational roles and project development as principle consultant on a variety of mineral deposits including gold projects within West Africa. I am Technical services consultant to Asanko Gold Ghana from 2016 and ongoing, resource estimation and technical support for feasibility studies for the Hummingbird Yanfolila (Mali) and Oklo Resources Dandoko (Mali) gold projects from 2015 and ongoing.

   In am involved in the ongoing technical review of the Forest Gold and RAN exploration projects in Zimbabwe.

   I have read the definition of "qualified person" set out by National Instrument 43101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101. My relevant experience, includes 30 years as a mineral resource geologist.

d. Site Inspection

Regular site visits since 1st September, 2016. My last site visit was from 6th November – 23rd November 2017. I undertook a total of 9 visits since July 2016.

Deposits visited: Nkran, Dynamite Hill and Akwasiso

Reviewed: Drill data; QAQC; Drill Core; Site surface geology and available drill collars; Production reconciliation data for Nkran; 3rd party MRE's and in-house MRE preparation work.

e. Responsibilities

I am responsible for sections 12 and 14 relating to Deposits Nkran, Dynamite Hill and Akwasiso.

f. Independence

I am independent of Asanko Gold Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

g. Prior Involvement

I have had no prior involvement with Asanko Gold Inc or the deposits being reported prior to September 2016.

h. Compliance with NI 43-101

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i. Disclosure

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

Electronic signature. Not for duplication. Electronic signature. Not for duplication. Electronic signature. Not for duplication. Electronic signature. Not for duplication. Electronic signature. Not for duplication.

Malcolm Titley
BSc Geology and Chemistry
MAIG; MAusIMM

CERTIFICATE of AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a. Name, Address, Occupation:

Thomas Kwabena Obiri-Yeboah

DRA Projects SA (Pty) Ltd, DRA Minerals Park, 3 Inyanga Close, Sunninghill, 2157, South Africa

Senior Mining Engineer

b. Title and Effective Date of Technical Report:

Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c. Qualifications

I am a graduate of University of Mines and Technology Tarkwa, Ghana, BSc / Post Graduate Diploma (Mining), 1991/1992, and I have carried out my profession continuously since then. I am a member in good standing of the Engineering Council of South African (ECSA), Registration number 20100340.

My relevant experience, with respect to the Mining portion of the Obotan Gold Project Feasibility Study Technical Report, includes 24 years in the mining sector covering production, planning and project work. I have worked in Ghana for Ashanti Goldfields from 1992 to 2006 as senior Mine Engineer working on whittle design and minable reserves and production schedules, for Mantra Resources in Tanzania from 2010 to 2014 on open pit mine design and mine production schedule and for Asanko gold since 2013 and ongoing on mine reserve and open pit design.

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

d. Site Inspection

I visited the Project site 25th-29th September 2017

e. Responsibilities

I am responsible for sections: 1.9, 15, 16, 25.3, 26.2 and 27.3

f. Independence

I am independent of Asanko Gold Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

g. Prior Involvement

I have been involved with the design of the Nkran pit and reserve statement from 2013.

h. Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i. Disclosure

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

Thomas Kwabena Obiri-Yeboah (Pr Eng)

CERTIFICATE of AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a. Name, Address, Occupation

Glenn Bezuidenhout

DRA Projects SA (Pty) Ltd, DRA Minerals Park, 3 Inyanga Close, Sunninghill, 2157, South Africa

Process Director

b. Title and Effective Date of Technical Report

Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c. Qualifications

I am a graduate of Witwatersrand Technicon in Johannesburg, South Africa, National Diploma (Extractive Metallurgy), 1979, and I have carried out my profession continuously since then. I am a Fellow member in good standing of the South African Institute of Mining and Metallurgy (FSIAMM), Membership # 705704.

My relevant experience, with respect to the Metallurgical portion of the Obotan Gold Project Feasibility Study Technical Report, includes 22 years' engineering involvement in 18 mineral processing and mining projects and 13 years of plant operations experience. The most recent gold project that I have completed was Edikan Gold Mine for Perseus in 2011.

I have read the definition of "qualified person" set out by National Instrument 43101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.

d. Site Inspection

I have not visited the Project site. I have visited all laboratories involved in the testwork program annually.

e. Responsibilities

I am responsible for sections 1.12, 13,17, 25.4, 26.3 and 27.4

f. Independence

I am independent of Asanko Gold Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

g. Prior Involvement

I have been involved as lead metallurgist in all metallurgical testwork and plant design from 2013.

h. Compliance with NI 43-101

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i. Disclosure

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

Glenn Bezuidenhout

CERTIFICATE of AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a. Name, Address, Occupation

Phil Bentley

9a Trilby Street, Oaklands, Johannesburg 2192 South Africa

Asanko Gold Executive : Geology and Mineral Resources

b. Title and Effective Date of Technical Report

Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c. Qualifications

MSc (Economic Geology) Victoria University of Wellington, New Zealand

MSc (Mineral Exploration) Rhodes University, Grahamstown, RSA

I am a Registered Professional Scientist with the South African Council for Natural Scientific Professions, Pr. Sci. Nat. Reg. No. 400208/05.

I have over 30 years' relevant experience in mineral resource estimation and exploration for a number of listed early development and production gold mining companies throughout Africa and New Zealand. In West Africa, I have worked in Mali (Mining & Exploration) – Syama, Morila, Loulo, Medinandi/Papillon from 2007 to 2013 and in Ghana (Mining & Exploration) – Bibiani, Asanko Gold (Nkran & Esaase) from 2013 to 2017.

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

d. Site Inspection

I have visited the Asanko site on a number of occasions , My last visit was from 4th September – 8th September 2017.

e. Responsibilities

I am responsible for sections 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 4, 5, 6, 7, 8, 9, 10, 11, 12, 23, 25.2, 26.1, 27.1, 27.2 and oversight on compilation of 14.

f. Independence

I am not independent of Asanko Gold Inc.

g. Prior Involvement

Active Executive employee of Asanko Gold Feb 2015 to September 2017.

h. Compliance with NI 43-101

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i. Disclosure

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

Phil Bentley

Executive Geology and Resources (SACNASP)

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects ,effective May 9, 2016 Part 8.1.

a. Name, Address, Occupation

   Dr Godknows Njowa

   Building 33, The Woodlands Office Park, 20 Woodlands Drive, Woodmead, Sandton

   Senior Manager: Venmyn Deloitte

b. Title and Effective Date of Technical Report

   Asanko Gold Definitive Feasibility Study, National Instrument 43-101 Technical Report.

   Effective date: Master Document Rev 6.1 dated 18 July 2017 amended and Restated December 13, 2017

c. Qualifications

   B. Sc Hons (Mining Engineering), University of Zimbabwe, 2003

   Corporate Governance and Financial Accounting, Institute of Chartered Secretaries and Administrators, 2004

   GDE (Graduate Diploma in Mining Engineering specialising in Mineral Resources Management and Mineral Asset Valuations), University of the Witwatersrand, 2005

   M.Sc (Mining Engineering specialising in Mineral Resources Management), University of the Witwatersrand, 2007

   Securities Investment Analysis, Investment Analyst Society, 2008

   Mining Tax Law Certificate, University of the Witwatersrand, 2012

   Doctorate (Mineral Asset Valuation and Financial Reporting), University of the Witwatersrand, 2017

   I have experience in the review and development of financial models for mining feasibility studies and production businesses. I have acted for confidential mining companies as independent mineral asset evaluator and independent fair and reasonable evaluator on a number of company transactions and fund raisings in Southern Africa since 2007.

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

d. Membership and Association

Professional Engineer, Engineering Council of South Africa, 2009

Member, Australian Institute of Mining and Metallurgy, 2008

Member, South African Institute of Mining and Metallurgy, 2006

e. Site Inspection

A site inspection was not undertaken.

f. Responsibilities

I am responsible for sections 1.14, 1.15, 1.16, 21,22, 25.6 and 26.4.

g. Independence

I am independent of Asanko Gold Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

h. Prior Involvement

I have not been involved with Asanko prior to the Asanko Gold Expansion Definitive Feasibility Study, National Instrument 43-101 Technical Report.

i. Compliance with NI 43-101

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

j. Disclosure

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated: 20 December 2017

Dr Godknows Njowa M.Sc Eng (Mining), PrEng