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National Instrument 43-101 Independent Technical Report for Asanko Gold Incorporated's Pre-Feasibility Study on the Esaase Gold Project in Ghana

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National Instrument 43-101 Independent Technical Report for Asanko Gold Incorporated's Pre-Feasibility Study on the Esaase Gold Project in Ghana

Synopsis

NI 43-101 Item 1

Venmyn Deloitte (Proprietary) Limited (Venmyn Deloitte) was requested by DRA Minerals Projects (DRA), on behalf of Asanko Gold Incorporated (Asanko Gold), to prepare an Independent Technical Report (ITR) on the results of an new Preliminary Feasibility Study (PFS) on the Esaase Gold Project (Esaase Project or the Project) in Ghana, West Africa.

Asanko Gold is a Toronto Stock Exchange (TSX) and New York Stock Exchange (NYSE) listed junior gold development company with two gold assets in Ghana, namely the Esaase and Asumura Projects, held by Keegan Resources Ghana Inc (Keegan Resources). Asanko Gold is a well-financed Canadian exploration and development company, strategically focused on near term gold production.

Property Description and Terms of Reference

The Esaase Project is an advanced gold exploration project at the point of conversion to a development project. The Project has been the subject of several historic technical studies and two prior economic studies, the results of which are available in the public domain. A Preliminary Economic Analysis (PEA) was completed in May 2010 and a PFS by Lycopodium Minerals Proprietary Limited (Lycopodium) was published in September 2011 in the form of a Canadian National Instrument 43-101 ITR by Coffey Mining Limited (Coffey Mining). The 2011 PFS provided Mineral Resource estimations reported in compliance with NI 43-101 and "Joint Ore Reserves Committee" (JORC) ("Australasian Code for Reporting of Exploration Results, Mineral Resource and Ore Reserves") codes. In addition, the 2011 PFS investigated capital and operational costs for an open pit mine, concentrator and whole-ore leach processing plant.

Asanko Gold made the strategic decision in June 2012 to defer the completion of a Definitive Feasibility Study (DFS) based on the 2011 PFS scope and to instead undertake a re-scoped PFS. In August 2012, Asanko Gold completed a Concept Study with the revised project scope and in September 2012 appointed DRA to undertake the new PFS (2013 PFS). The 2013 PFS would permit the publication of an updated 2012 Mineral Resource and Mineral Reserve estimate, a new mine design based on the new resource estimate with the specific focus on the feasibility and optimisation of a new process flow design. Re-estimation of the capital, and operational costs, at current gold market conditions would be undertaken.

The purpose of this ITR therefore will be to summarise and document the results of the 2013 PFS in terms of technical parameters, Mineral Resources and Mineral Reserves, new mine and infrastructure design, re-designed and improved process flow, environmental review and economic analysis of the Esaase Project in a manner fully compliant with the requirements of:-

- the Canadian National Instrument 43-101 Standards for Disclosure for Mineral Projects (Form 43-101F1) and the Companion Policy Document 43-101CP (NI 43-101);
- the disclosure and reporting requirements of the TSX as stipulated in the 2007 TSX Company Manual;
- the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2005); and
- CIMVAL Standards and Guidelines for Valuation of Mineral Properties (2003).

Esaase Project PFS Concept

The 2011 PFS was based on an open pit mining methodology with a "whole-ore" leach processing plant. The study included an unlined Tailings Storage Facility (TSF) with initial mill throughputs of 9 million tonnes per annum (Mtpa), reducing to 7.5Mtpa once mining exploited fresh rather than oxide/transition zone material. The 2011 PFS resulted in a positive Net Present Value (NPV) for the Project but additional studies indicated that better economic outcomes could be achieved with alternative processing methodologies.

The re-scoped 2013 PFS would permit the following changes to the original 2011 PFS:-

- incorporation of the updated 2012 Mineral Resource and Mineral Reserve estimate which includes increased resources in the Indicated category and new Measured Mineral Resources;
- the 2012 Mineral Resource estimate would support a new optimised mine design based on the new resource estimate;
- the feasibility and economics of a gravity-flotation-carbon-in-leach (CIL) plant could be investigated;
- the process plant could be designed to a target throughput of 5Mtpa run-of-mine (RoM);
- optimisation of the proposed mine layout could be achieved in order to efficiently accommodate the required infrastructure including waste rock dumps, buffer dam and tailings storage facilities (TSFs);
- optimisation of the power supply could be undertaken;
- a target life-of-mine (LoM) of at least 10years with an average gold production of 200,000oz per annum (ozpa) could be achieved; and
- the 2013 PFS would specifically focus on the capital and operational costs at 2013 gold market conditions.

The various PFS component studies were undertaken by numerous independent specialist consultants. Each consultancy provided teams of technical advisors, with an overall Qualified Person in terms of NI 43-101 definitions, who has signed-off the appropriate sections of the ITR. Venmyn Deloitte has undertaken independent reviews of the processing and environmental studies and has reviewed at a high level all the sections of the ITR. The results of the specialist studies conducted for the 2013 PFS on the Esaase Project have been incorporated into an economic analysis of the Project and the following summary, conclusions and interpretations were highlighted in the review.

Property Location, Ownership and Legal Tenure

The Esaase Project is located in southwest Ghana in West Africa within the Amansie West District of the Ashanti Region, approximately 35km southwest of the regional capital Kumasi. The Esaase Project comprises two contiguous mining leases (the Jeni and Esaase mining leases) and four prospecting licences that cumulatively cover an area of approximately 99.10km^2 in the prospective gold mining Ashanti region. The Esaase and Jeni concessions contain alluvial gold deposits in the Bonte River valley which have been exploited historically by artisanal miners. The Esaase Project however, is focussed on the primary host Birimian aged (2.17Ga) meta-sediments and not on evaluation of the alluvial deposits.

The Jeni River and Esaase mining leases were originally granted in March 1990 to the Jeni River Development Company Limited and to Bonte Gold Mines Limited respectively. Both of these companies filed for bankruptcy in 2002 and the Esaase mining lease was acquired from the Bonte Liquidation Committee by Sametro Company Limited, a private Ghanaian company. Asanko Gold entered into an agreement with Sametro in May 2006 to earn 100% of the mineral right and transfer of the Esaase mining lease to Asanko Gold was completed in June 2007.

The mining leases are valid until 2020 and can be renewed for an additional thirty year period. The mining leases permit Asanko Gold to undertake mining activities provided certain conditions and fee payments are maintained with the Ministry of Lands, Forestry and Mines;.

Geology and Mineralisation

The Esaase Project is located in the Archaean (Birimian aged) Asankrangwa Belt within the Kumasi Basin. World renowned gold mines such as AngloGold Ashanti Limited's Obuasi deposit, which began production in 1898, are located in the neighbouring Ashanti and Sefwi belts. Similar vein deposits in the Ashanti belt, such as Prestea and Konongo, have robust production histories.

The gold quartz vein mineralisation in these deposits is structurally controlled along faults and shear zones resulting from compressional tectonic episodes, particularly along the margins of greenstone belts such as the Ashanti and Sefwi Belts.

The target Esaase Project mineralisation is classified as a mesothermal orogenic quartz vein deposit. Such mesothermal orogenic deposits are associated with Precambrian and Phanerozoic convergent plate boundaries and are hosted in sequences generally metamorphosed to greenschist facies. The Esaase Project area comprises a sequence of intensely folded and faulted meta-sediments and the mineralisation comprises a system of gold-bearing quartz veins hosted by the tightly folded meta-sediments.

The mineralisation is considered to have been produced by a series of fluid influxes which were channelled along lithological, rheological and structural boundaries. The target gold mineralisation comprises free gold and very fine grained gold in association with sulphides in quartz-carbonate-sulphide veins hosted within parallel, northeast trending, moderately to steeply, west dipping units of extremely deformed siltstone/shale.

The Esaase Project mineralisation is structurally controlled by Eburnean age compressional tectonic events which produced fold-thrust patterning followed by a late stage strike-slip deformation event. The deformational intensity increases systematically from the northwest to the southeast across the Project area. The regional structural synthesis suggests that most of the structural elements are compatible with a single, extended and progressive phase of regional deformation involving substantial northwest-southeast compression. Towards the close of the tectonic event, the stress direction changed from northeast-southwest to north-south. The change in stress direction caused left lateral strike-slip movement along pre-existing faults which created a northsouth dilational opening permitting the emplacement of late stage non-mineralised northsouth oriented veins.

The weathering profile on the Esaase Property is strongly influenced by topography and influences the proposed mine site layout and mining methodology. The weathered profile generally comprises surface laterite and ferruginous duricrust, followed by saprolite, which gradationally merges with oxidised bedrock. In regions of higher elevation at the Esaase Project, the laterite and underlying saprolite have been removed by erosional processes so that only the oxidised bedrock remains which provides adequate conditions for foundation and construction stability.

Exploration Concept and Status

Since mid-2006, Asanko Gold through its subsidiary Keegan Resources Ghana Limited (Keegan Resources) has undertaken an extensive exploration programme combining soil geochemistry, Induced Polarisation (IP) resistivity and Versatile Time Domain Electromagnetic (VTEM) geophysical surveys, followed by diamond and reverse circulation exploration and resource drilling. The geophysical surveys delineated the resistivity of the host and bedrock at Esaase Project and the >6,100 soil sample geochemistry programme clearly supports the mineralisation model of parallel northwest trending mineralised structures.

The drilling programme focused mainly on the northwest striking gold bearing structures and a total of 1,496 reverse circulation (RC) and diamond drillholes were completed on the Project area. A geotechnical drilling programme of 28 diamond drillholes was also completed. Sample recovery from the drilling programme was generally acceptable except in the moderate to highly weathered saprolite and highly fractured/brecciated zones but recovery factors are unlikely to materially affect the accuracy and reliability of the sampling results. Quality control and twin drillhole verification showed no negative bias in the diamond drilling due to the use of water. Reviews of the drilling and sampling protocols by independent Qualified Persons from Coffey Mining and Minxcon provide confidence that the data is spatially well represented and that the methodologies employed are within international standards and the resultant information is suitable for Mineral resource estimation purposes.

Suitable specific gravity measurements were undertaken to support the Mineral Resource estimation process. Assay sample preparation and analysis was undertaken in three independent internationally accredited laboratories and a single umpire laboratory. Field and laboratory Quality Assurance/Quality Control (QA/QC) measures to monitor accuracy, contamination, precision and accuracy are appropriate for the style of mineralisation. Detailed analysis of the quality control data was undertaken by Coffey Mining and Minxcon with the concluding opinion that the Esaase Project quality control and assurance is adequate and within international standards.

Mineral Resource and Mineral Reserve Estimates

Minxcon (Pty) Limited (Minxcon) conducted the 2013 PFS Mineral Resource estimate on a refined block model utilising existing data with an additional 19,598 assay results to more precisely delineate the mineralised zones within the resource area.

Grade estimation was undertaken using Ordinary Kriging, with Indicator Kriging chosen to delineate the areas with continuous grade. Mineralised domains were defined and modelled on an approximate lower cut-off grade of 0.3g/t. A total of 46 zones, including waste, were identified and included in the statistical analysis and resource estimation. Outlier analysis and variogram determinations were undertaken and the final NI 43-101 compliant Mineral Resource (June 2013) estimate at a 0.6g/t cut-off grade is:-

- Measured Mineral Resources of 23.4Mt at a grade of 1.49g/t Au for 1.12Moz contained gold;
- Indicated Mineral Resources of 71.3Mt at a grade of 1.44g/t Au for 3.29Moz contained gold;
- Total Measured plus Indicated Mineral Resources of 94.6Mt at a grade of 1.45g/t Au for 4.41Moz contained gold; and
- Inferred Mineral Resources of 33.6Mt at a grade of 1.40g/t Au for 1.51Moz contained gold.

The Mineral Reserve estimate was based on the 2013 PFS mining study at a cut-off grade of 0.6g/t Au, a USD1,400/oz gold price and a mining recovery of 97%. Gold resources which occur in satellite pits and which are currently deemed uneconomic to extract were excluded from Mineral Reserve estimate, together with Mineral Resources which are affected by permitting, environmental, logistic and socio-political issues such as proximity to villages or forest reserves. The 2013 PFS Mineral Reserve estimate is

- Proven Mineral Reserves of 22.9Mt at a grade of 1.43g/t Au for 1.05Moz contained gold;
- Probable Mineral Reserves of 29.5Mt at 1.40g/t Au for 1.32Moz contained gold; and
- Total Proven and Probable Reserves of 52.34Mt at a grade of 1.41g/t Au for 2.37Moz contained gold.

Mining Study

DRA Mining (Pty) Limited (DRAM) undertook the mining study for the 2013 PFS, which included mine design on Indicated and Measured Mineral Resources only, pit optimisation, mine production and scheduling and costing. The appropriate mining methodology for the Esaase Project comprises conventional open pit drill and blast mining, followed by load and haul to various stockpiles. The drilling and blasting would be performed on benches between 10m and 20m in height and total material movement over the 11yr LoM is estimated between 30Mtpa and 35Mtpa. For the purpose of the 2013 PFS, Asanko Gold opted for the mining study to be conducted on a contract mining basis. The mining costs for the optimisation study were based on a mining operational cost of USD3.23/t obtained by combining DRA's database of costs, for similar operations and average costs from contractor mining quotes from the 2011 PFS, escalated for inflation. As the PFS progressed and more detail became available, the costs were optimised and refined but these optimisations will be included in the DFS.

Geotechnical studies confirmed slope angles of 52° in fresh material and 35^0 in oxide material. Pit optimisation produced a main pit with various satellite pits but only the Main Pit and South Pit were included in the mining study. A series of pits shells were generated at a range of cut-off grades with Whittle Pit financial optimiser to isolate the optimum mine plan for the Project. Once the optimal Whittle shell was selected, detailed mine designs, waste dump designs and LoM mining schedules were then completed using Studio 5 Planner and EPS Scheduling to determine the optimal long term mine plan.

The selected pit shell reaches fulfils the Asanko Gold criteria for the average RoM grade and LoM. The pit shell extends to a depth of -19mamsl, or approximately 290m below the Bonte river valley surface and contains 52.34Mt RoM at 1.4g/t Au for 2,37Moz of gold. A total of 224Mt of waste are contained within the pit, equating to a waste to ore stripping ratio of 4.28. A pit dewatering programme has been designed and grade control planning is essential. Bench and face mapping, for grade control as well as for geotechnical reasons, will be a routine task in finalising the ore and waste blocks to be marked out for excavation.

The mine plan includes pre-production stripping and waste provision for starter dams. Thereafter, the Main Pit remains the principal source of RoM, with South Pit contributing, on average, a third (±31%) of the overall tonnes mined for the first four years until it's mined out and ready to be backfilled. The mine site plan was optimised during the 2013 PFS and the final plan includes the free blast zone, various waste dumps, two TSFs located in a valley to the west of the Main Pit, a plant site in an elevated valley west of the Main Pit, contractor miner camp near the Bonte River and an upgraded and relocated public road.

Metallurgical Testwork and Process Design

The 2011 PFS for the Esaase Project was based on an open pit mining methodology with an associated "whole-ore" leach processing plant. The 2013 PFS was initiated with a revised scope which included investigation of a conventional crushing, milling and gravity recovery plant, followed by flotation, with the flotation concentrate being reground and then leached in a standard CIL circuit using AARL elution technology.

Under the supervision of Lycopodium in Australia four extensive phases of metallurgical testwork were completed for the 2011 PFS and DRA undertook a Phase V testwork programme in 2012 to support the new process design. The final process recommended by DRA comprises:-

- an open circuit primary and secondary crushing followed by run-of-mine ball milling. The mill will operate in closed circuit with a cluster of cyclones to produce a p80 (80% passing) grind of 75µm;
- a primary gravity recovery from the mill circuit comprising two independent 400t/h Knelson concentrators with the gravity concentrate reporting to a high intensity batch dissolution reactor;
- a flotation circuit from the mill circuit, comprises a single bank of seven 130m^3 rougher cells in series. with regrind and secondary gravity recovery of the flotation concentrate;
- a CIL recovery circuit comprising seven 330m^3 tanks in series with cyanide detoxification and arsenic removal circuits to produce a tailings stream with <1.0ppm As in solution; and
- the pregnant solution from the intensive leach reactor and the eluate solution stripped from the loaded carbon in the CIL circuit will pass through the electrowinning circuit and the resultant gold sludge washed, dried and smelted into dore bars for transfer to Rand Refineries.

Trade-off comparisons in the metallurgical testwork showed that the performance of a flotation circuit gave similar LoM recoveries as a "whole-ore" leach circuit but at significantly lower operating costs. Testwork results indicated an optimal grind size of 75µm with a LoM recovery of 90.06%. The recoveries include a 1.09% recovery discount over to allow for practical processing limitations in a full-scale operating plant environment.

The final design has the benefit that the flotation tailings, comprising approximately 85% to 90% of the feed, are benign and can be disposed to a non-HDPE lined waste TSF, whilst the CIL and downstream plants can be downsized accordingly.

Esaase Proposed Mine Infrastructure

Two options exist for the power supply to the proposed Esaase mine, namely an off-grid connection to the national state utility company Gridco or an Independent Power Producer (IPP) option. The Ghanaian utility company, GridCo can supply 34MW at a total cost of USD32m, which maybe significantly reduced due to a redesign of the connection by placing a new 161kV substation on the main line and utilising a single 11km 161kV overhead line to feed the mine. The costs of the Gridco option for power supply to the proposed Esaase mine are high and Asanko Gold is investigating various alternative options. A quotation from an American based company, USP&E Global, for a dual diesel/heavy fuel oil power plant has been obtained which indicates a capital expenditure of USD22.5m (excluding reticulation and instrumentation) will be required with an operating cost of USD0.15/kWh.

The initial Project start–up water requirement of 1.4Mm^3 will be sourced from the Mpatoam weir and Bonte river in the wet season and stored initially in the empty CIL Leach Tailings Storage Facility (L-TSF). Potable water demand during construction and steady state production will reach a maximum of 168m^3 per day to be supplied by two boreholes. The water balance model is based on a closed circuit for beneficiation process water and the TSFs, with make-up water supply from pit dewatering and abstraction of water from the Bonte river at the beginning of the wet season.

A process water buffer dam has been designed to capture any contaminated process water that cannot be discharged. The Esaase mine will eventually be a net water positive situation and at steady state production, the total water in slurry discharged to the mine residue disposal facilities will be 920m^3/hr of which 60% to 90% is returned to the plant process water circuit.

The surface water management system for the Project will consist of two separate systems namely, a clean water diversion system to control the uncontaminated run-off from the higher lying natural environment, and a dirty storm water system to capture the contaminated storm water from plant, operational and processing areas.

The design of the mine residue disposal facilities for the 2013 PFS was undertaken by independent design specialists Epoch Resources Pty Limited and included two facilities namely a clay lined storage facility for the tailings arising from the gravity-flotation circuit of the beneficiation plant (F-TSF) and a HDPE lined facility designed to accommodate the tailings from the CIL circuits of the process plant (L-TSF).

The 196.2ha F-TSF comprises an in situ clay compacted, fully-contained, valley dam, ring-dyke facility catering for a depositional tonnage rate of 4.5Mtpa over a 11 year LoM to be constructed out of waste rock in four, phased, wall raises to a maximum height of 65.0m. The facility to accommodate the tailings arising from the CIL circuit comprises a heavy duty polyethylene (HDPE) lined, fully-contained, valley dam L-TSF catering for a depositional tonnage rate of 0.5Mtpa over a 11 year LoM. The embankment is to be constructed out of waste rock in two, phased, downstream wall raises to a maximum height of 42.4m.

The designs and costings of the mine infrastructure have been completed to PFS levels of accuracy and include haul roads, mine accommodation buildings, contractor miner camp and other required structures. The various costings of the infrastructure have been included in the overall capital cost estimate.

Environmental and Social Study

The environmental and social pre-feasibility study was undertaken by an independent consultant, Epoch Resources (Pty) Limited and reviewed by Venmyn Deloitte. The study results indicate that in the context of the specifications of Ghanaian legislative requirements, and internationally accepted standards of practice, including the Equator Principles and IFC Performance Standards, the environmental aspects of the Project PFS have been acceptable at this stage of project development. The Equator Principle Finance Institutes would categorise the Esaase Project as a Category A project. Formal consultation and stakeholder engagement regarding the Project have been on-going since 2007 and extensive baseline environment studies have been completed in key areas including climate, air quality, visual and noise impact, hydrology and hydrogeology, biodiversity, cultural and archaeological, as well as socio-economic.

Key areas of potential environmental impact have been identified and measures to mitigate these risks are to be assessed in the DFS and the Environmental Impact Assessment (EIA) process. Critical authorisations that are required for the development of the proposed Esaase mine include various water use licences in terms of the Water Resources Commission and an approved Environmental Impact Statement (EIS). The EIS to be submitted in July 2013 to the Environmental Protection Agency (EPA) for approval of the mining permit and the issue of the required environmental permit, will be in compliance with the Environmental Protection Agency Act (1994) and the Environmental Assessment regulations (1999).

A conceptual rehabilitation and mine closure plan for the proposed Esaase mine and its associated infrastructure was undertaken. An estimate of the required financial provision is USD30.01m including on-going re-habilitation and aftercare.

Esaase Project Capital and Operational Expenditure Estimates

The Esaase Project capital expenditure estimation (excluding escalation) is derived from the studies on mining, processing, mine infrastructure, TSF designs, dam construction, electrical supply, owners' costs and indirect costs. The capital costs for the mining operation, process plant and TSFs. The total initial capital cost estimate is USD260.50m, which increases to USD286.50m including a 10% contingency allowance as summarised below:-

CAPEX		COST (USDm)
Initial	Plant including EPCM	93.2
	Infrastructure including mining	111.00
	Indirect costs	53.6
	Sub-total	**260.5**
	Contingency	26.00
Total initial capital		**286.5**
Deferred capital		12.95
Sustaining capital		51.90
Closure capital		29.60

The process and mining capex estimates are USD82,9m (excluding EPCM and contingency) and USD13.7m respectively. The initial capital cost estimate for the mine residue disposal facilities is USD34.4m and a total of USD78.86m over the LoM.

The operating cost 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 average LoM cash operating cost is estimated at USD736/oz Au based on the treatment of 5Mtpa producing an average of 200,000oz/a based on power costs of USD0.15/kWh.The mining opex for ore is USD4.64/t RoM, with USD12.24/t applicable to the waste mining. The process opex is USD10.37/t RoM and the general and administration costs are USD2.53/t as summarised in the table below. The resultant total cash operating cost (excluding royalties) is USD30.00/t milled.

Summary Operational Expenditure

PROJECT COMPONENT DESCRIPTION	OPEX (USD/oz Au)	OPEX (USD/tonne milled)
Waste mining	300.26	*12.24
Ore mining	113.84	4.64
Processing	254.19	10.37
General and administrative	62.14	2.53
Refining	5.35	0.22
Total cash operating costs	735.78	30.00
Royalties	77	3.14
Total cash costs	812.78	33.14
Sustaining and deferred capex	30.36	1.24
Pre-tax sustaining cash cost	843.14	34.38
Taxation	146.71	5.99
Post tax inclusive sustaining cash cost	**989.85**	**40.37**

Source : DRA 2013
*Applying the LoM stripping ratio of 4.28

Economic Analysis

Venmyn Deloitte constructed a Discounted Cash Flow (DCF) model for the purposes of the economic analysis of the Esaase Project at a gold price of USD1,400/oz Au. The DCF model was constructed in Excel and was based on input assumptions generated from the 2013 PFS mining schedule, processing schedule, operating costs and capital expenditure estimates. Venmyn Deloitte received input from DRA and Asanko Gold on the timing of the various inputs, including working capital requirements. The DCF model assesses the post-tax real cash flows for the Project at a 5% real discount rate. The economic analysis indicates a positive Net Present Value (NPV) of the Esaase Project of USD354.7m with a post-tax IRR of 23.2% at a gold price of USD1,400 for a 5% real discount rate. The payback period is 3.8 years from first production (commissioning). The Project NPV at various gold prices and discount rates are summarised as follows:-

Summary Economic Analysis of the Esaase Project at Various Discount Rates and Gold Prices

GOLD PRICE (USD/oz)	PROJECT NPV (USDm) at various discount rates (%)					IRR (%)
	3%	5%	6%	7%	8%	
1,100	95.07	50.75	31.24	13.29	-3.27	7.8
1,200	209.73	155.14	131.03	108.77	88.20	13.4
1,300	321.08	255.51	226.50	199.70	174.91	18.5
1,400	431.50	354.71	320.71	289.27	260.17	23.2
1,500	541.65	453.58	414.55	378.45	345.02	27.6
1,600	651.61	552.19	508.10	467.31	429.53	31.7
1,700	761.53	650.77	601.62	556.15	514.01	35.8

The Project NPVs generated from the DCF model proved to be most sensitive to changes in parameters affecting revenue. A 10% change in revenue changes the NPV by 39.1%. The NPV is less sensitive to changes in operating expenditure (opex) with a 10% change in opex translating into a 21.9% change in the NPV. A 10% change in capital expenditure changes the NPV by 17.4%.

Esaase Project Risk Assessment

The development of any mining operation entails risks associated with geological confidence, grade and operational parameters but these risks are inherent in all advanced exploration and development mining projects. The specific risk assessment of the Esaase Project identified 83 risks, of which 2 are extremely high, 44 high, 31 moderate and 6 low to very low. At the current project study stage these risks are without mitigating controls. The DFS will address the risks and with mitigating controls, the risks are likely to be reduced.

The two extremely high risks relate to the haul roads cross-over public roads at level crossings. These risks will require appropriate controls in order to decrease them to appropriate levels. Delays in attaining an environmental permit and appropriate water use licences pose a risk to timelines and cost of capital.

Esaase Project Execution Plan

The 2013 PFS included a full project execution plan which will be undertaken and managed by DRA. Front-end engineering and early work on site is scheduled for the start of the dry-season in November 2013 and main construction is planned to begin in March 2014. The construction schedule is 18 months with first gold produced in H1 2015 and steady-state production in H2 2015.

Concluding Remarks

Venmyn Deloitte concludes that the 2013 PFS has fulfilled its scope of optimising the process flow design, improving project economics and has succeeded in providing a robust basis for the DFS going forward. The Mineral Resource and Reserve basis is founded in international standard exploration and analytical results and the process flow design is based on reasonable and appropriate metallurgical testwork. The mine design has already identified optimisation opportunities for the DFS which provides positive upside potential for the Project. No environmental or risk factors that cannot be mitigated have been identified and the Project site layout plan adequately accommodates all Project components without impacting areas of environmental sensitivity. The economic analysis shows that the Esaase Project is robust, with an attractive positive NPV even at gold prices less than the current June gold spot price of USD1,379/oz.

Recommendations

The Venmyn Deloitte review of the 2013 PFS for the Esaase Project has highlighted that the Project is robust and economically viable and Venmyn Deloitte concurs with Asanko Gold's decision to progress the Project to the DFS level of investigation. Several opportunities exist to improve upon the economic results of the 2013 PFS which should be investigated in the DFS, namely:-

- a metallurgical testwork program focusing on the optimisation of the flotation reagent suite which could result in the reduction of the concentrate mass pull through the float plant to further optimise the process flow sheet and associated costs;
- a detailed mine design using the PFS developed modifying factors to improve the conversion of Mineral Resources to Mineral Reserves;
- a detailed mine design reflecting higher mining and processing rates whilst treating softer oxide RoM in the early years of the Project LoM; and
- further geotechnical engineering studies to determine if steeper pit slope angles can be introduced into the design of the open pit.

The metallurgical testwork can commence immediately on the remaining material from the previous metallurgical testwork programme. The detailed mine design will include the deeper sections of the orebody, classified as Measured and Indicated Resources, which would extend the LoM and thereby improve the Project economics further. The proposed costs for the DFS are presented in the table below:-

Estimated Proposed DFS Costs

DFS PROJECT COMPONENT	ESTIMATED COSTS (USD)
EPCM	1,210,239
Mine Residue storage facilities and water storage dams	31,595
Environmental Studies	115,006
Metallurgical testwork	119,518
Geotechnical studies	269,549
Land survey	63,189
IPP	324,977
DFS implementation and management	541,629
Contingency	273,636
Additional	60,662
TOTAL	**3,010,000**

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1. Introduction

NI 43-101 Item 2(a), 2(b)

Venmyn Deloitte (Proprietary) Limited (Venmyn Deloitte) was requested by DRA Minerals Projects (DRA), on behalf of Asanko Gold Incorporated (Asanko Gold), to prepare an Independent Technical Report (ITR) on the results of an Update Preliminary Feasibility Study (PFS) on the Esaase Gold Project (Esaase Project or the Project) in Ghana, West Africa.

Asanko Gold is a Toronto Stock Exchange (TSX) and New York Stock Exchange (NYSE) listed junior gold development company with two gold assets in Ghana, namely the Esaase and Asumura Projects, held by Keegan Resources Ghana Inc (Keegan Resources). Asanko Gold trades on the TSX and the New York Stock Exchange (NYSE) under the ticker symbol AKG. This ITR summarises the technical and economic aspects of its wholly owned, flagship Esaase Project only.

The Esaase Project is an advanced gold exploration project at the point of conversion to a development project with 4.4 million (m) ounces (oz) of gold (Au) in the Measured and Indicated Mineral Resource categories with an average grade of 1.45 g/t Au. The Esaase Project has been the subject of several historic technical studies and two prior economic studies, the results of which are available in the public domain. A Preliminary Economic Analysis (PEA) was completed in May 2010 and a PFS by Lycopodium Minerals Proprietary Limited (Lycopodium) was published in September 2011 in the form of a Canadian National Instrument 43-101 ITR by Coffey Mining Limited (Coffey Mining). The 2011 PFS provided Mineral Resource estimations reported in compliance with NI 43-101 and "Joint Ore Reserves Committee" (JORC) ("Australasian Code for Reporting of Exploration Results, Mineral Resource and Ore Reserves") codes. In addition, the 2011 PFS investigated capital and operational costs for an open pit mine, concentrator and whole-rock leach processing plant.

Asanko Gold made the strategic decision in June 2012 to defer the completion of a Definitive Feasibility Study (DFS) based on the 2011 PFS scope and to instead undertake a re-scoped PFS. In August 2012, Asanko Gold completed a Concept Study with the revised project scope and in September 2012 appointed DRA to undertake the new PFS (2013 PFS). The 2013 PFS would permit the incorporation of the updated 2012 Mineral Resource and Mineral Reserve estimate which included increased resources in the Indicated category and new Measured Mineral Resources. The 2012 Mineral Resource estimate would support a new mine design and the 2013 PFS would specifically focus on the feasibility and optimisation of a new process flow design. Re-estimation of the capital, and operational costs, at current gold market conditions would be undertaken.

The purpose of this ITR will be to summarise and document the results of the new PFS (2013 PFS) in terms of technical parameters, Mineral Resources and Mineral Reserves, updated mine design, improved and revised process design, environmental review and economic analysis of the Esaase Project in a manner fully compliant with the requirements of:-

- the Canadian National Instrument 43-101 Standards for Disclosure for Mineral Projects (Form 43-101F1) and the Companion Policy Document 43-101CP (NI 43-101);
- the disclosure and reporting requirements of the TSX as stipulated in the 2007 TSX Company Manual;
- the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2005); and
- CIMVAL Standards and Guidelines for Valuation of Mineral Properties (2003).

Each section of the ITR is designated with the relevant NI 43-101 Item number (NI Item) and the guidelines are considered by Venmyn Deloitte to be a concise recognition of best practice due diligence methods and accord with the principles of open and transparent disclosure that are embodied in internationally accepted Codes for Corporate Governance. These standards of disclosure are the minimum standard for Venmyn Deloitte techno-economic due diligence and embody current trends in technical and economic evaluation of mineral properties. Venmyn Deloitte employs a proprietary checklist that ensures that an internationally acceptable process is completed for the reporting of the mineral assets.

1.1. Corporate Structure

Asanko Gold was originally named Keegan Resources Incorporated and changed its name to Asanko Gold in February 2013, to reflect its transformation from an advanced exploration company to a development and ultimately a gold producing company.

Asanko Gold holds several wholly owned subsidiaries as illustrated in Figure 1. The Esaase Project is held by Barbados registered Keegan Ghana Incorporated which in turn holds a 90% shareholding Keegan Resources Ghana Limited. The Ghanaian state holds the remaining 10% of Keegan Resources Ghana Limited. Throughout the ITR the term Keegan Resources refers to Keegan Resources Ghana Limited.

1.2. Sources of Data

NI 43-101 Item 2 (c)

The original 2009 Mineral Resource estimate undertaken by Coffey Mining was based on the following key digital data provided by Asanko Gold:-

- drillhole database containing collar location, downhole survey, assay and geological data;
- 3-dimensional model of the topography (digital terrain model);
- a representative selection of the original assay sheets;
- quality control procedures and database;
- internal and external quality control data;
- bulk density dataset derived from drillcore; and
- representative geological cross-sections.

The information included in the 2013 PFS was sourced from the following project specific studies, information in the public domain, individual specialist studies that comprise the various sections of the ITR and additional sources which are all listed in the Reference List (Section 26) :-

- Esaase Gold Project Pre-Feasibility Study dated September 2011;
- 2011 Coffey, Esaase Gold Project, Ghana NI 43-101 Technical report dated 22 September, 2011;
- 2009 Coffey, Esaase Gold Deposit Resource Estimation, dated 24 April, 2009;
- The 2012 Mineral Resource estimate by Minxcon (Proprietary) Limited (Minxcon); and
- Revised Eburnean geodynamic evolution of the gold-rich southern Ashanti Belt, Ghana, with new field and geophysical evidence of pre-Tarkwaian deformations.

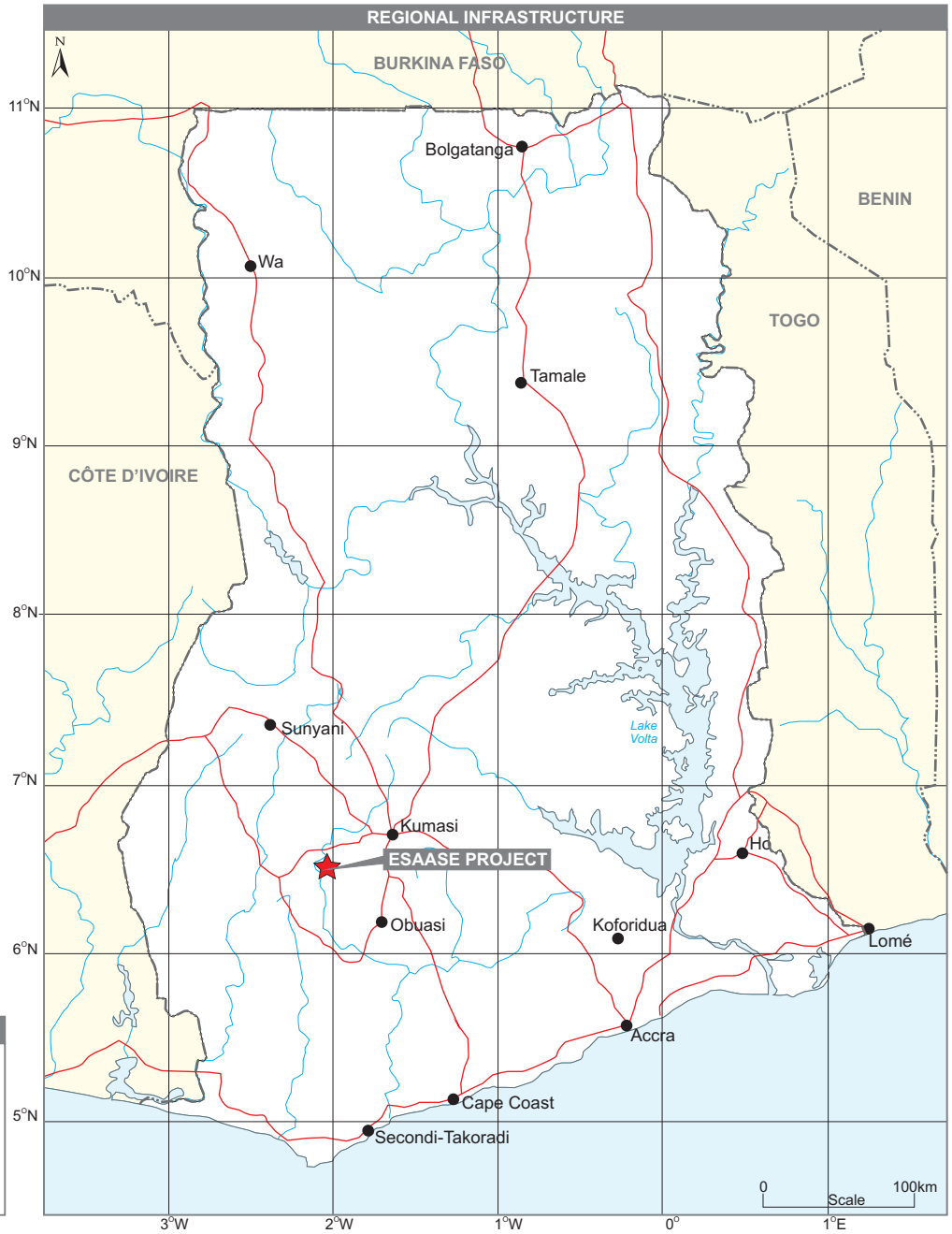
1.3. Scope of the Opinion and Statement of Independence

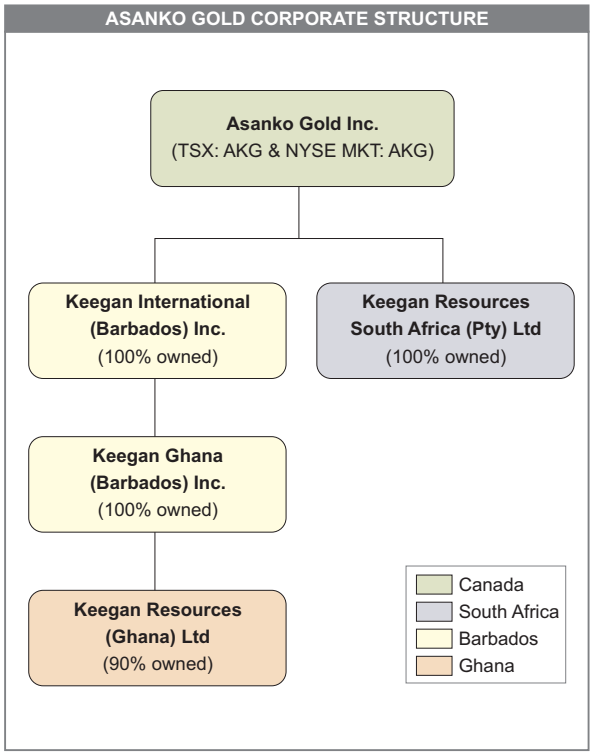
NI 43-101 Item 2(c)

Venmyn Deloitte's primary obligation in preparing mineral asset reports for the public domain is to describe mineral projects in compliance with the reporting codes applicable under the jurisdiction in which the company operates. In this case, it is the Canadian NI 43-101 code, as discussed in Section 1.

In the execution of its mandate, Venmyn Deloitte and the contributing consultants undertook the 2013 PFS, in order to identify the factors of both a technical and economic nature, which would impact the future viability of the Esaase Project. Venmyn Deloitte prepared this ITR for potential investors and their advisors. Venmyn Projects considered the strategic merits of the asset utilising the best practise due diligence methodologies. The ITR has been compiled in order to incorporate all available and material information that will enable potential future finance providers to make balanced and reasoned judgements regarding the economic merits of the Esaase Project.

LOCATION AND REGIONAL INFRASTRUCTURE - ESAASE PROJECT GHANA





Source: Minxcon 2012, Asanko Gold 2013

D1237_DRAKeeganResources_2013

VennmynDeloitte.

Asanko Gold

Figure 01

Venmyn Deloitte is an independent advisory company. Its consultants have extensive experience in preparing Technical Reports, technical advisers' and valuation reports for mining and exploration companies. Venmyn Deloitte advisors have, collectively, more than 100 years of experience in the assessment and evaluation of mining projects and are members in good standing of appropriate professional institutions. The signatories to this report are qualified to express their professional opinions on the Project and qualify as Qualified Persons, as defined by the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects. To this end, Qualified Persons Certificates are presented in Appendix 2.

Neither Venmyn Deloitte nor its staff, have or have had, any interest in any of Asanko Gold projects capable of affecting their ability to give an unbiased opinion, and, have not and will not, receive any pecuniary or other benefits in connection with this assignment, other than normal consulting fees. Neither Venmyn Deloitte, nor any of the authors of the ITR, hold any interest in Asanko.

1.4. Contributing Qualified Persons and Personal Inspection

NI 43-101 Item 2 (d)

The 2013 PFS was undertaken by independent specialist consultants under the management of DRA and a summary of the contributing consultancies is presented in Table 1. Most consultancies provided teams of technical advisors who contributed to the various studies and each consultancy has nominated an overall Qualified Person in terms of the definition in NI 43-101 Standards of Disclosure Part 1.1, who has signed-off the appropriate sections of the ITR. The Qualified Person from each team, has had a technical team member as a representative visit the Esaase Project or has personally visited the site, as summarised in Table 1:-

Table 1 : Qualified Persons' ITR Responsibility and Site Visits

CONSULTANT	2013 PFS COMPONENT	QUALIFIED PERSON	SITE VISIT	ITR SECTION	NI 43-101 Item
DRA Mineral Projects (Pty) Limited (DRA)	Study Management, Power Supply, Infrastructure and Capital and Operating Costs	Mr D. Heher	01 July 2012	Section 17.1, 17.2, 17.3, 17.4, 17.6, 20, 23	Item 18, 21,
DRA	Process Plant and Metallurgy	Mr G. Bezuidenhout	Several visits in 2012-2013 to Amdel Laboratories in Australia to supervise metallurgical testwork	Section 12 , 12.1 to 12.4, 16 to 16.2	Item 13, 17
DRA Mineral Projects (Pty) Limited - Mining Division (DRAM)	Reserve Estimation and Mine Design	Mr T. Yeboah	DRA mining team site visit 1 September 2013	Section 14, 15	Item 15, 16, 18
Minxcon Proprietary Limited	Mineral Resource Estimate	Mr C. Muller	29 September to 2 October 2012	Section 3, 4, 5, 6, 7, 8, 9, 10, 11,13	Item 4, 5 ,6, 7, 8, 9, 10, 11, 12, 14
Epoch Resources (Proprietary) Limited	Mine Re-habilitation and Closure Mine Residue Disposal Facilities	Mr G Wiid	Epoch team site visit 16 to 18 April 2013	Section 17.5, 19.5, 20.1.3	Item 18, 20
Epoch	Social and Environmental Impact Assessment	Mr F. Coetzee	26 to 30 November 2012	Section 19. to 19.4	Item 20
Venmyn Deloitte Proprietary Limited (Venmyn Deloitte)	43-101 including Financial Model Development and Related Analysis	Mr A Clay,	No site visit	Section 1, 2, 18, 21, 22, 24, 25, 26, 27	Item 1, 2, 19, 22, 24, 25, 26, 27

1.5. Esaase Project PFS Concept

NI 43-101 Item 2 (b)

In September 2011 Asanko Gold commissioned Coffey Mining and Lycopodium to undertake a PFS for the Esaase Project based on the criteria listed in Table 2. The 2011 PFS was based on an open pit mining methodology with a "whole-ore" leach processing plant. The study included an unlined Tailings Storage Facility (TSF) with initial mill throughputs of 9 million tonnes per annum (Mtpa), reducing to 7.5Mtpa once mining of the fresh zone began. The PFS resulted in a positive Net Present Value (NPV) for the Project but additional studies indicated that better economic outcomes could be achieved with alternative process design methodologies.

Table 2 : Criteria and Scope 2013 Esaase Project PFS

PFS CRITERIA	APRIL 2013
Mineral Resource Base (inclusive of Mineral Reserves)	Measured + Indicated Mineral Resources of 94.62Mt at 1.45g/t Au for 4.41Moz contained gold
Mineral Reserve Base	Proven and Probable Mineral Reserves of 52.43Mt at 1.41g/t Au for 2.37Moz contained gold
Mine Design Optimisation-Pit size	277Mt
Mine Design Optimisation-Waste	224Mt
Mine Design Optimisation-Ore	52Mt
Stripping Ratio	4.28
Average Grade	1.41g/t
In situ Au	2.4Moz
Process Design	Flotation/Carbon-in-leach (CIL)
Mill Throughput (Oxide Zone)	5Mtpa
Mill Throughput (Fresh Zone)	5Mtpa
Tailings Storage Facility	Clay lined float TSF and separate lined CIL TSF
Initial Water Storage	1.43Mm^3
Gold Price	USD1,400

Source: DRA 2013 and Coffey Mining 2011

Asanko Gold made the strategic decision in June 2012 to defer the initiation of a Definitive Feasibility Study (DFS) based on the 2011 PFS concept scope and criteria, and to instead undertake a re-scoped update PFS. In August 2012, Asanko Gold completed a Concept Study with the revised project scope and in September 2012 appointed DRA to undertake the new 2013 PFS.

The 2013 PFS would permit the following changes (Table 2):-

- publication of an updated 2012 Mineral Resource and Mineral Reserve estimate which includes increased resources in the Indicated category and new Measured Mineral Resources;
- the 2012 Mineral Resource estimate would support a new optimised mine design based on the new resource estimate;
- the feasibility and economics of a flotation with carbon-in-leach (CIL) plant could be investigated;
- the PFS was to target a life-of-mine (LoM) of at least 10years with an average gold production of 200,000oz per annum (ozpa);
- the 2013 PFS would specifically focus on the capital and operational costs at 2013 gold prices.

The potential for optimisation of various criteria became obvious as the 2013 PFS progressed and a 2013 PFS base case was identified as a result of these optimisation studies with the following specific criteria changed from, or added to, the initial 2012 PFS scope:-

- 5Mtpa run-of-mine (RoM) material processed through the process plant;
- addition of a flotation plant which would entail primary and secondary crushers feeding to a RoM ball mill. The flotation concentrate would be reground before leaching;
- only the Southern Pit and Main Pit would be developed;
- initially the waste rock was planned to be trucked to a waste rock dump located approximately 3km to the west of the main pit as opposed to the south (Table 2) but further optimisation indicated that the best economic results were obtained from dumping the waste rock within the 500m blast exclusion zone as far as possible to reduce haulage costs;
- the tailings would be processed into two independent streams feeding differing TSFs:-
- a CIL tailings stream to be deposited in a lined leach TSF (L-TSF) which would accommodate 10% of the RoM; and
- flotation tailings stream to be sent to a clay lined TSF (F-TSF) accommodating approximately 90% of the RoM. The site of the F-TSF to be relocated to maximise efficiency;

- the Project start up water would initially be stored in the empty L-TSF to reduce infrastructure expenditure;
- a buffer dam was relocated from the original plans; and
- the power supply was changed from supply from the national utility company (Gridco) to Independent Power Producer (IPP).

1.6. Terminology

The Canadian National Instrument Companion Policy 43-101 (Section 2.3) states "We consider the use of the word "ore" in the context of mineral resource estimates to be misleading because "ore" implies technical feasibility and economic viability that should only be attributed to mineral reserves". In compliance with Section 2.3 of the Companion Policy, the term "ore" is not used in the Mineral Resource context of this ITR (Section 13). The term "whole-ore" leach is used in the processing sections of the ITR and implies no demonstration of economic viability.

2. Reliance on Other Experts

NI 43-101 Item 3 (a)

Venmyn Deloitte is not qualified to provide extensive comment on legal issues, including the status of tenure associated with the Project. In the assessment of these aspects it has relied upon information provided by Keegan Resources, which has not been independently verified by Venmyn Deloitte. The 2013 PFS has been prepared on the understanding that the property is, or will remain, lawfully accessible for evaluation, development, mining and processing.

Minxcon has relied on information supplied by Keegan Resources to be valid and complete, which applies to and is not limited to the:-

- Esaase Gold Project Pre-Feasibility Study dated September 2011;
- 2011 Coffey, Esaase Gold Project, Ghana NI 43-101 Technical report dated 22 September 2011; and
- 2009 Coffey, Esaase Gold Deposit Resource Estimation, dated 24 April, 2009.

Minxcon has scrutinised all the information provided by these companies and is satisfied that the information is sound and could be used in the preparation of an NI 43-101 compliant Mineral Resource estimate.

Venmyn Deloitte fully relied on the geophysical expertise of Geotech Limited for the geophysical survey and interpretation of results by Condor Consulting Incorporated (Condor) reported in Coffey Mining 2011 and summarised in the exploration results Section 8 of this ITR.

3. Property Description and Location

NI 43-101 Item 4(a), (b)

The Esaase Project is located in southwest Ghana in West Africa as illustrated in Figure 1, within the Amansie West District of the Ashanti Region, approximately 35km southwest of the regional capital Kumasi (Figure 1). The Project comprises two contiguous mining leases and four prospecting licences that cumulatively cover an area of approximately 99.10km^2 (Figure 2). The Esaase mining lease area is referred to throughout the ITR as the Esaase concession and the Jeni mining lease area as the Jeni concession. The Esaase and Jen concessions contain alluvial gold deposits in the Bonte River valley which have been exploited historically by artisanal miners. The Esaase Project however, is focussed on the primary host Birimian aged meta-sediments and not on evaluation of the alluvial deposits.

3.1. Location, History and Country Profile of Ghana

The Republic of Ghana is a West African country approximately 239,000km^2 in size and is one of the five African nations along the northern coastline of the Gulf of Guinea.

During the first half of the twentieth century Ghana was called the Gold Coast which at that time was a British colony. Ghana was the first sub-Saharan country in colonial Africa to be granted independence, which took place on 6 March 1957. Following a national referendum in July 1960 Ghana became a republic. Between 1966 and 1992 periods of democratic rule alternated with military rule. By 1992 the economy had stabilised, a new constitution was inaugurated and Ghana returned to democracy with the election of Jerry Rawlings as president. Rawlings' National Democratic Congress party continued in power throughout the 1990s, being replaced by the New Patriotic Party in the 2000 democratic election. Ghana has been under continuous democratic rule for the last seventeen years and the last general election was held in December 2012 and was won by the National Democratic Congress.

Ghana has a developing mixed economy based largely on agriculture and mining and is one of the most developed countries in tropical Africa. The gross national product (GNP) is keeping pace with population growth and the domestic economy of Ghana is dominated by subsistence agriculture, which accounts for approximately 25% of the gross domestic product (GDP). 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. The most important source of foreign exchange is gold mining, followed by cocoa and timber products. Ghana is the world's tenth and Africa's second largest producer of gold, with gold production of 2.97Moz in 2010.

Inflation, decreasing currency exchange rate and high interest rates have caused concern in recent years, but are improving with more stringent fiscal and monetary policies. The transport infrastructure within Ghana is comparatively good for the region. Since the early 1990's multiple large, medium and small gold mining operations have been developed in Ghana with both their construction and ongoing operational logistic requirements being met by the two main ports and the internal road network. Ghana has no natural harbours and two artificial harbours were built at Takoradi and Tema to accommodate Ghana's shipping requirements.

A distinguishing geographic feature of Ghana is the Volta River, which was dammed in 1964 (Akosombo Dam) to create the enormous Lake Volta which occupies a sizeable portion of Ghana's south-eastern territory.

3.2. Physiography of Ghana

Ghana is characterised by low physical relief as the Precambrian basement sequences have undergone considerable erosion (Figure 2). The highest elevation in Ghana, Mount Afadjato in the Akwapim-Togo Ranges, rises only 880m above sea level (mamsl).

The country is divided into four distinct geographical regions:-

- **the Low Plains** which cover the southern part of the country and comprise the coastal savannah, the Volta Delta, the Accra Plains and the Akan lowlands.
- The Volta Delta, extends into the Gulf of Guinea and is a flat featureless region which consists of thick sandstone, limestone, and silt deposits. The coastline consists of a low sandy coastal plain, except in the west, where the forest extends to the ocean;
- **the Ashanti Uplands** and the Kwahu Plateau, which are located north of the Akan lowlands and continue into Cote d'Ivoire. The forest belt, which extends 320km northward from the western coast and eastward for a maximum of 270km, is broken up into heavily wooded hills and steep ridges. The Uplands form a physical divide for the major river systems (Figure 2) i.e. the Afram and Pru rivers flow into the Volta river, whilst from the opposite side, the Pra, Birim, Ofin, Tano, and other rivers flow south toward the sea (Figure 2). Apart from the Volta river, only the Pra and the Ankobra rivers permanently cross the coastal dunes as most of the other rivers terminate in brackish lagoons. The Ashanti Uplands forms the northern limit of the forest zone (Figure 2);
- the **Volta Basin** occupies the central regions of Ghana and covers 45% of the land surface (Figure 2). The northern section of the basin lies above the upper part of Lake Volta, and rises to a height of 150m to 215mamsl whilst to the south and the southwest the elevation is less than 300m. Lake Volta, formed by the impoundment of the Volta behind Akosombo Dam, is the world's largest manmade lake with a surface area of 8,485km^2; and

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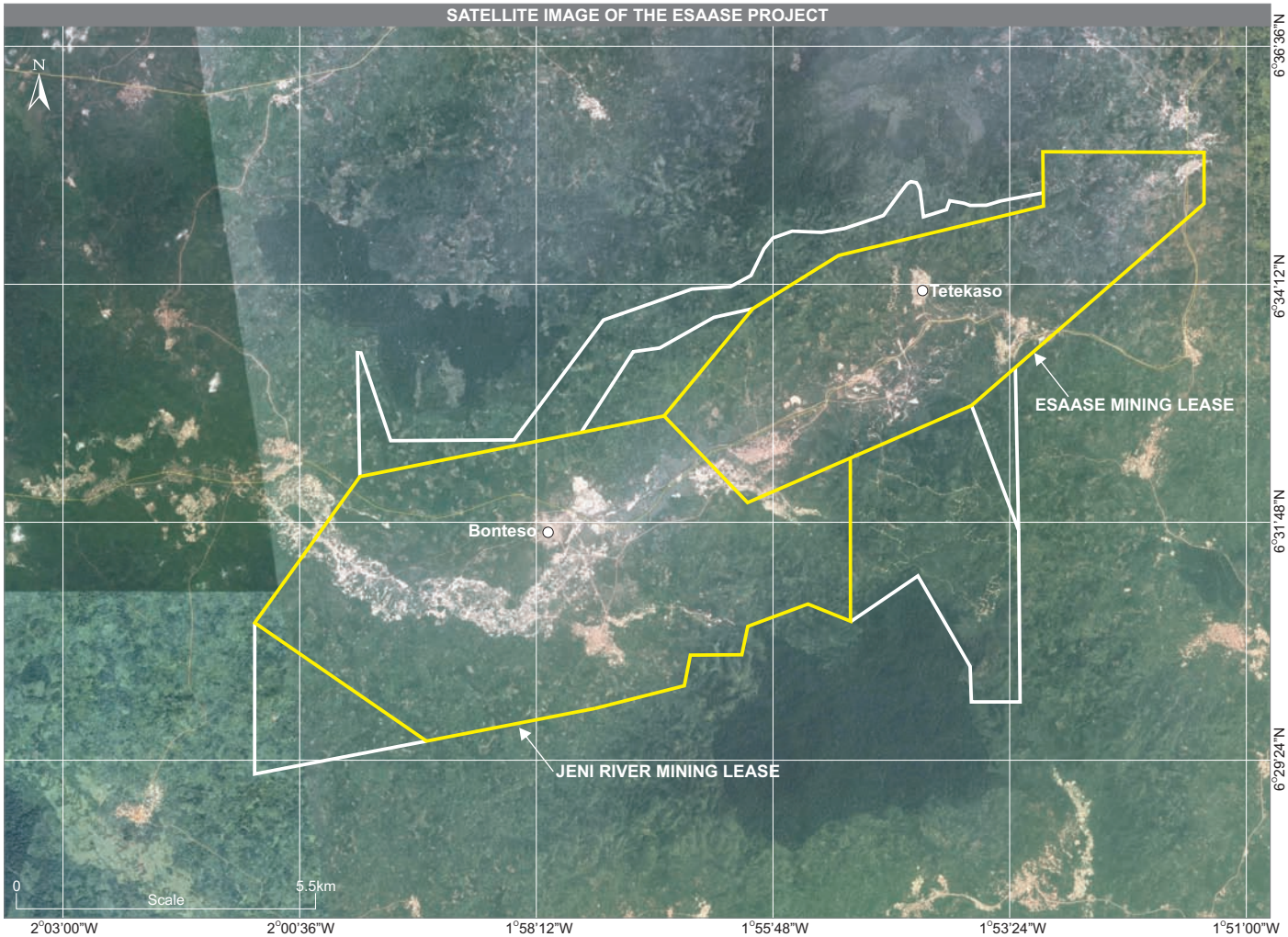
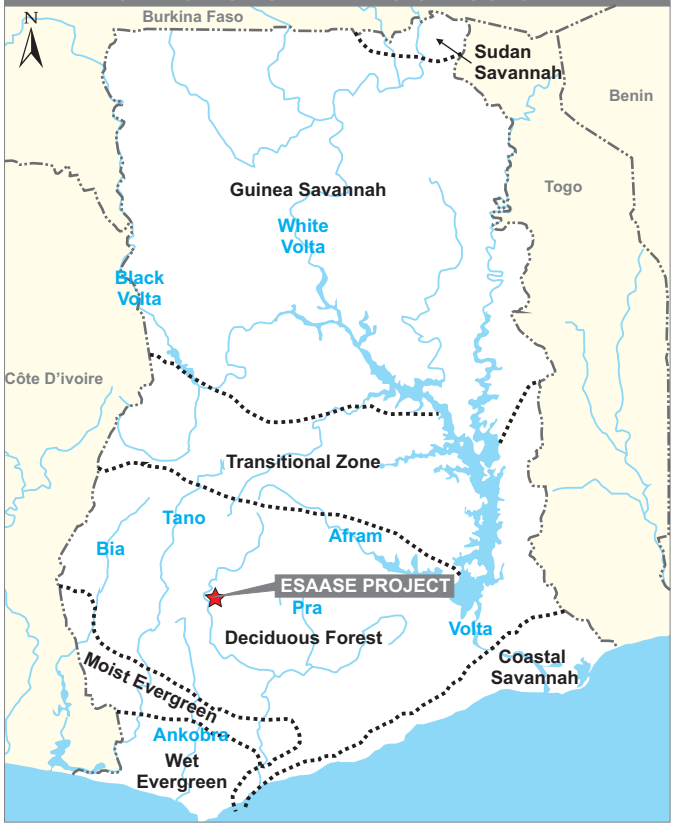
PHYSIOGRAPHY OF GHANA AND THE ESAASE PROJECT

SATELLITE IMAGE OF THE ESAASE PROJECT



TOPOGRAPHY OF GHANA

VEGETATION ZONES AND RIVER SYSTEMS OF GHANA



Source: Venmyn Deloitte 2013

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- the **High plains** which occupy the northern and northwestern region of Ghana and comprise a dissected plateau, which averages between 150m and 300mamsl in elevation. Soils in the high plains are more arable than those in the Volta Basin, and the population density is considerably higher.

The climate throughout most of Ghana is either a wet or dry tropical climate type, marked by warm to hot temperatures throughout the year, and abundant rainfall in only one season. This characteristic is especially noticeable in northern Ghana where less annual rainfall and strictly seasonal precipitation is prevalent. Although this region receives 750mm to 1,000mm of rain annually, severe dry spells occur from November to March. Most of Ghana receives 1,000mm to 1,500mm annual precipitation typical of savannah type climates.

A second climatic region exists in southwestern Ghana which is a wet tropical climate with hot temperatures throughout the year and abundant rainfall (over 2,000mm), well distributed throughout the year.

Ghana experience warm to hot temperatures throughout the year because of its proximity to the Equator and its relatively low elevation. The average annual temperature in Accra, Ghana, is 27ºC. The northern section of Ghana has hotter temperatures and some seasonal temperature variations because it is farthest from the moderating influence of the ocean, and closest to the Sahara desert.

3.3. Legal Aspects and Tenure

NI 43-101 Item 4(c)

3.3.1. Ghanaian Mining Law

NI 43-101 Item 4 (e)

The Ghanaian mining industry is regulated by the Ministry of Mines and Energy within the overall legislative framework of the Minerals and Mining Law of 1986 (Law 153) which was as amended in 1994 and 2005. The Minerals and Mining Law stipulates the payment of royalties and standard corporate tax rates by mining companies. The royalty payable by the mining sector to the Ghanaian government was increased from 3% to 5% in 2010.

The Minerals and Mining Act of 2006 (Act 703) embodies the conditions pertaining to the application and retention of mineral rights in Ghana. In terms of the Act, all minerals are the property of the state and are vested in the President in trust for the nation. The approval of mineral right applications is subject to recommendations by the Minerals Commission and rests with the Minister on behalf of the President.

The Ministry of Mines and Energy oversees all aspects of Ghana's mineral sector and within the Ministry:-

- the Minerals Commission has responsibility for administering the Mining Act, recommending mineral policy, promoting mineral development, advising the Government on mineral matters, and serving as a liaison between industry and the Government;
- the Ghana National Petroleum Corporation (GNPC) is the Government entity responsible for petroleum exploration and production;
- the Precious Minerals Marketing Corporation (PMMC) is the Government entity responsible for promoting the development of small-scale gold and diamond mining in Ghana and for purchasing the output of such mining, either directly or through licensed buyers; and
- the Mines Department has authority in mine safety matters. All mine accidents and other safety problems also must be reported to the Ghana Chamber of Mines, which is the private association of operating mining companies.

The Minerals and Mining Act 2006 provides for the following mineral rights which are only granted to person's incorporated under the Ghanaian Companies Code 1963 (Act 179):-

- reconnaissance licence: granted initially for one year which permits exploration but no drilling or excavation. The reconnaissance licence can be extended once for a period of one year and can apply to an area not more than 5,000 contiguous blocks of 21ha each;
- prospecting licence: granted for three years which permits exploration activities according to an approved programme. The licence can be extended for a further three years with a compulsory simultaneous reduction in area of 50%. A prospecting licence cannot exceed an area comprising 750 contiguous blocks of 21ha each;
- mining lease: granted for thirty years or a lesser period dependent on the expected Life of Mine (LoM) of the project. A mining lease can be extended for an additional thirty years. The lease is applicable to specific minerals and permits exploitation of the mineral, erection of necessary infrastructure and storage of waste material according to the specification of an approved Environmental Impact Assessment (EIA). Mining leases require adherence to the Water Resources Commission Act (Act 552), forestry and environmental protection regulations and regular reporting of activities to the Minerals Commission. All geological information must be submitted to the Geological Survey and copyright of such information resides with the state.

The holder of a mineral right can only exercise the rights under this Act subject to the limitations relating to surface rights. Surface rights owners are permitted to continue agricultural activities if these do not hamper exploration and exploitation. Surface rights owners may apply to the Minister for compensation or re-settlement arrangements.

The holder of mineral rights in Ghana is required to pay annual mineral right fees, annual land rental fees and royalties at a maximum rate of 6% of the total revenue from minerals obtained by the holder.

3.3.2. Esaase Project Mineral Tenure

NI 43-101 Item 4(a), 4(b), 4(C), 4(d), 4(e)

The Esaase Project comprises two contiguous mining leases and four prospecting licences that cumulatively cover an area of approximately 99.10km^2. The two mining leases, namely Esaase and Jeni River mining leases, are elongate contiguous concessions, a total of approximately 20km in length, trending in a northeasterly direction, as illustrated in Figure 3.

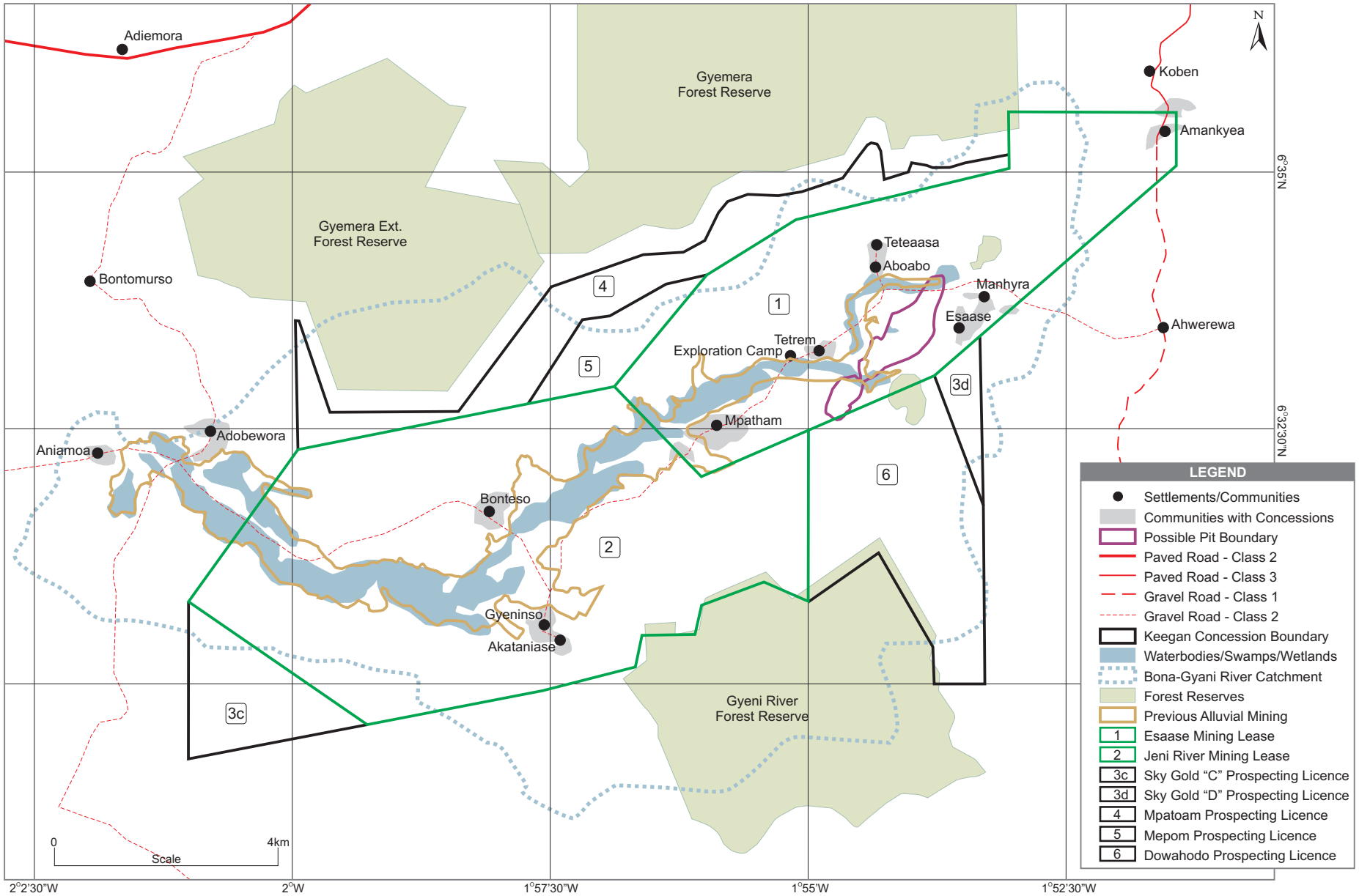
The mining leases are bordered in the north and south by smaller prospecting licences (the Mpatoam, Mepom, Dawohodo and Sky Gold prospecting licences), which constitute approximately 25% of the Project area (Table 3, Figure 3). The Asanko Gold legal tenure is summarised in Table 3:-

Table 3 : Legal Tenure for the Esaase Project

MINERAL RIGHT	REFERENCE NUMBER	AREA (km^2)	CO-ORDINATES	EXPIRY DATE	HOLDER
Esaase Mining Lease	EPA/PR/PN/804	27.03	1° 53' west, 6° 34' north	03 September 2020	Keegan Resources Ghana Limited
Jeni River Mining Lease	EPA/PR/PN/805	43.41	1° 98' west, 6° 52' north	21 March 2020	
Mpatoam PL*	EPA/PR/PN/806	9.83	1° 57' west, 6° 33' north	30 November 2011	
Mepom PL **	LVB19490/09	2.37	1° 56' west, 6° 33' north	17 February 2011	
Dawohodo PL*	EPA/PR/PN/733	10.36	1° 54' west, 6° 32' north	08 March 2011	
Sky Gold C PL***	RL3/22	4.60	2°00' west, 6° 30' north	20 June 2012	
Sky Gold D PL***	RL3/22	1.50	1° 53' west, 6° 32' north	20 June 2012	
TOTAL		**99.10**			

Source : Asanko Gold 2013
The state of Ghana holds 10% free carried interest in Keegan Resources
EPA - Environmental protection Agency Reference Number (see Sections 3.3.2.2 and 19)
LVB - Land Valuation Board Reference Number
RL – Reconnaissance Licence Reference Number
*Renewal of PL application submitted and fees paid
**Terminal reports submitted to the Mining Commission. Renewal of licence pending
***Awaiting approval of application for conversion to prospecting licences

LEGAL ASPECTS AND TENURE - ESAASE PROJECT



Source: Minxcon 2012, Asanko Gold 2013

D1237_DRAKeeganResources_2013

VennmynDeloitte.

Asanko Gold

Figure 03

The Jeni River mining lease was originally granted in March 1990 to the Jeni River Development Company Limited and the Esaase mining lease was granted to Bonte Gold Mines Limited in September 1990. Both of these companies filed for bankruptcy in 2002.

The Esaase mining lease was acquired from the Bonte Liquidation Committee by Sametro Company Limited, a private Ghanaian company. Asanko Gold entered into an agreement with Sametro in May 2006 to earn 100% of the mineral right and transfer of the Esaase mining lease to Asanko Gold was completed in June 2007.

Asanko Gold acquired the Jeni River mining lease in March of 2008 from the Bonte Liquidation Committee and the Minerals Commission of Ghana. The leases are valid until 2020 and can be renewed for an additional thirty year period. The mining leases permit Asanko Gold to undertake mining activities provided certain conditions and fee payments are maintained with the Ministry of Lands, Forestry and Mines.

The prospecting licences were acquired by Asanko Gold as follows:-

- the Mepom prospecting licence was purchased from a private Ghanaian company, Mepom Mining Company, and transferred to Asanko Gold on in June 2009;
- the Mpatoam prospecting license is a new concession granted to Keegan Resource in November 2009;
- Sky Gold reconnaissance licence granted July 2011 over 91.5km^2;
- Dawohodo prospecting licence 10.6km^2 in January 2011

The Ghanaian government has a standard 10% free carried interest in all permits within the country but this interest only comes into effect once exploitation and production commences. The Minerals and Mining Act 2006 also makes provision for a royalty on mining revenue which was increased in 2010 to a maximum of 6%. The royalties applicable to gold mining are 5%. In addition, the mining leases contain provision for a 0.5% royalty payment to the Bonte Liquidation Committee.

The surface rights in the Project area are held by the head of the Ashanti tribe. As soon as exploitation and revenue production commences, the Manso-Nkwanta Paramountcy Stool has the right to approach the state for compensation by way of a share in the royalties. The amount of the compensation is subject to approval by the Minister of Mines in consultation with the Land Valuation Board.

3.3.2.1. Environmental Liabilities

NI 43-101 Item 4 (f)

The historic alluvial mining activities by Bonte Gold Mines Limited, resulted in extensive land disturbances and silting of the drainage systems which has caused a diversion of the relatively small Bonte and Jeni Rivers. A substantial percentage of the disturbed land was successfully re-vegetated, however the shallow mined areas, particularly on the Jeni River mining lease, were not rehabilitated by Bonte Gold Mines Limited at the time of closure of the operation.

A series of shallow impoundments were constructed on the Esaase concession as settling ponds for clay-rich sediments and these have naturally vegetated since closure to become a series of wetlands.

The Bonte Liquidation Committee acts on behalf of the Ghanaian Government and its agreement with Asanko Gold, stipulates that Asanko Gold assumes no environmental liability arising from the historic operations on the Esaase mining lease. In terms of the Jeni River mining lease, Asanko Gold has agreed to remediate the existing environmental disturbance should any large-scale mining operation be undertaken on the concession.

3.3.2.2. Permitting

NI 43-101 Item 4(g)

Asanko Gold's defined gold resources and identified exploration targets are fully encompassed by and authorised by the Esaase and Jeni River mining leases.

The primary permits required for the development of the Project include the following:-

- Environmental Protection Agency (EPA) permit issued by the Minister of Environment;
- an amendment to the existing mining leases issued by the Minister of Lands, Forestry and Mines on the advice of the Minerals Commission for intended amendments to the programme of mining operations;
- various other permits from other government departments including water abstraction permit and surface storage facilities; and
- renewal of the mining leases in 2020.

In terms of the above requirements, Asanko Gold has a valid EPA permit to the end of 2013. In addition, Asanko Gold has the right to mine under the existing mining leases; however, before mining activities can be undertaken, the submission of a mine plan to the EPA will be required and on approval, a mining permit will be issued.

3.3.2.3. Other Significant Risks

NI 43-101 Item 4(h)

To the best of Asanko Gold's knowledge, no additional risks exist above those normally associated with an exploration and mining venture and those specifically discussed in the preceding sections.

4. Access, Climate, Local Resources, Infrastructure and Physiography

NI 43-101 Item 5

4.1. Topography, Elevation and Vegetation

NI 43-101 Item 5 (a)

The Esaase Project is located within the Ashanti Uplands of central Ghana (Figure 2) and the main topographic feature of the area is the Bonte River valley which traverses the properties in an approximate east-west direction. The Bonte River is bounded by steep valley sides which reach an elevation of 500mamsl. The river valley has been extensively exploited by artisanal miners seeking alluvial gold and the mining activity is clearly visible on the satellite image as overburden and tailings dumps (Figure 2).

The Esaase Project area primarily supports subsistence farming producing mainly food crops such as plantain, corn, cassava, yam, tomatoes and some cash crop such as cocoa and oil palm. Approximately half of the Project area comprises secondary forest and thick natural vegetation.

4.2. Accessibility and Proximity to Population Centres

NI 43-101 Item 5 (b), (c)

The Esaase Project is located in the southern-central area of Ghana within a region of developed local infrastructure and reasonably dense population.

The Project is located in the Ashanti Province 35km southwest of the regional capital Kumasi. The renowned Obuasi Mine in the Ashanti Belt lies approximately 30km to the southeast of Esaase Project and the Bibiani Mine 40km directly to the west.

The Esaase Project area is accessed from Kumasi via 10km of tarred road, westwards towards Bibiani Junction at the town of Asenemuso. The access route continues southwest for 10km to the village of Wiaso, where a secondary asphalt road accesses the village of Amankyea 8km to the south. The Project area is reached via 11km of gravel roads through the villages of Ahewerwa and Tetrem as shown in Figure 3. The access within the prospecting and mining licences is via a series of secondary roads constructed either by the former Bonte Gold Mines or by Keegan Resources. Regions of the Project area undergoing surface exploration are accessed by both dirt roads and footpaths.

4.3. Climate, Vegetation and Operating Season

NI 43-101 Item 5 (d)

The Esaase Project is located within the deciduous forest region of Ghana as shown in Figure 2 and the annual rainfall in the Ashanti Province ranges between 1,500mm to 2,000mm and temperatures range from 22°C to 36°C. The major wet season occurs from April to July followed by a minor rainy season from September to October. Asanko Gold has operated without cessation or delay throughout both of the rainy seasons.

4.4. Local Resources, Infrastructure and Available Surface Rights

NI 43-101 Item 5 (e)

The Esaase Project is located within a 45km radius of the well-developed Sefwi and Ashanti Belts gold mining regions. The mature mining industry in the region ensures local availability of both skilled and unskilled personnel. The area is well serviced by the Ghana national power grid with at least two alternate points of supply within a 50km radius of the potential plant site and open pit mine and surrounding villages are connected to the national power supply. Mobile telephone communication is accessible in most parts of the concession and a satellite dish is installed in the current exploration camp for internet access.

The nearest medical clinic and police station are located at Toase-Nkawie, on the Bibiani Highway, 20km from the exploration camp. Hospitals and most government offices are available in Kumasi. Food and general supplies are available in Kumasi.

The surface rights available for the Esaase Project TSFs, plant site and waste rock disposal dumps are adequate for the Project requirements as demonstrated in Section 17 (Figure 15). A detailed discussion of the availability of power, water and mining personnel is presented in Section 17.

5. Historic Ownership and Exploration

NI 43-101 Item 6

5.1. Ownership and Historic Exploration

NI 43-101 Item 6(a), (b)

The Bonte River region alluvial gold deposits have been exploited by artisanal miners for decades. Evidence exists of adits excavated by European settlers, between 1900 and 1939, however, no documented records remain of the mining activity. Drilling was conducted on the Bonte River valley alluvial sediments during 1966 and 1967 to determine alluvial gold potential.

In 1990, the original Bonte mining lease was granted to Akrokerri- Ashanti Gold Mines and was later transferred to a local subsidiary namely Bonte Gold Mining Company (Table 4). Bonte Gold Mining Company reportedly recovered approximately 200,000oz of alluvial gold on the Esaase mining lease area and another 300,000 oz downstream on the Jeni River mining lease area, prior to entering into receivership in 2002.

The Esaase concession, including the camp facilities at Tetrem, was acquired from the Bonte Liquidation Committee by Sametro Company Limited, a private Ghanaian company and in May, 2006, Keegan Resources signed a letter of agreement with Sametro to earn 100% of the Esaase concession over a three year period of work commitments and option payments. Since mid-2006, Keegan Resources has undertaken an extensive exploration programme combining soil geochemistry and Induced Polarisation (IP) geophysical surveys followed by diamond and reverse circulation (RC) exploration and resource drilling.

Table 4 : Historic Ownership of the Esaase Project

DATE	OWNERSHIP APPROVALS AND TRANSFERS
March 1990	Jeni River mining lease granted to Jeni River Development Company Limited
September 1990	The Esaase lease was granted to Bonte Gold Mining
November 2002	Dawohodo Manufacturing and Marketing Limited, a Ghanaian incorporated private company granted the prospecting licence over the area now covered by the Esaase Project (Dawohodo-Esaase prospecting licence)
2003	Jeni River Development Company Limited and Bonte Gold Mining declared bankrupt
	Esaase Mining Lease, including the camp facilities at Tetrem, was acquired from the Bonte Liquidation Committee by Sametro Company Limited, a private Ghanaian company
May 2006	Keegan Resources entered into an option agreement with Sametro to earn 100% of the Esaase Mining Licence
June 2007	Transfer of the Esaase Mining Lease to Keegan Resources
June 2007	Sky Gold Limited, a Ghanaian incorporated private company granted reconnaissance licence
March 2008	Keegan Resources acquired the Jeni River mining lease for a consideration of USD50,000 to BLC and USD50,000 to the Minerals Commission of Ghana
March 2008	Transfer of Jeni River mining lease to Keegan Resources
June 2009	Mepom prospecting licence acquired from Mepom Mining Company
November 2009	Mpatoam prospecting licence granted to Keegan Resources
July 2010	An agreement was entered into between Keegan Resources and Sky Gold Limited
July 2011	Ministerial consent to the option agreement between Sky Gold Limited and Keegan Resources
November 2010	An agreement was entered into between Keegan Resources and Dawohodo Manufacturing and Marketing Limited
January 2011	Ministerial approval for the assignment of the Dawohodo-Esaase prospecting licence to Keegan resources
October 2012	Land swop with Mpatoam Small Scale Mining Company to acquire 10.3km^2 southwest of the Esaase Main zone and relinquish the western portion of the Jeni concession

Source : Minxcon 2012

5.2. Historic Mineral Resource and Mineral Reserve Estimates

NI 43-101 Item 6 (c)

Venmyn Deloitte is unaware of any estimate of the quantity, grade, or metal or mineral content that was prepared before Asanko Gold acquired an interest in the Project therefore no historic Mineral Resource estimates are available for the Esaase Project.

Previous Mineral Resource estimates, as defined by Section 2.4 NI 43-101 Standards of Disclosure, were compiled for Asanko Gold by Coffey Mining in 2007, 2009 and 2011. The previous Mineral resource estimate by Coffey Mining in 2011 is presented in Table 5 and the categories are consistent with NI43-101 Section 1.2 and 1.3 categories. The Mineral Resource estimate by Minxcon 2012 is considered the current Mineral Resource basis for the 2013 PFS (see Section 13).

5.3. Historic Production

NI 43-101 Item 6 (d)

The alluvial gold deposits on the Esaase and Jeni concession areas were exploited in the early 1990s by the Bonte Mining Company until bankruptcy entering receivership in 2002. The Bonte Gold Mining Company recovered an estimated 200,000oz of alluvial gold on the Esaase concession and another 300,000oz downstream on the Jeni River concession.

Table 5 : Previous Mineral Resource Estimate - 2011

RESOURCE CATEGORY	CUT-OFF GRADE (g/t Au)	TONNAGE (Mt)	GRADE (g/t Au)	CONTAINED METAL (Moz)
Measured	0.4	5.03	1.2	196
	0.5	4.66	1.3	191
	0.6	4.21	1.4	183
	0.7	3.73	1.4	173
	0.8	3.26	1.5	162
	0.9	2.83	1.7	150
	1.0	2.44	1.8	139
Indicated	0.4	93.71	1.1	3,441
	0.5	83.03	1.2	3,288
	0.6	72.04	1.3	3,096
	0.7	61.67	1.5	2,882
	0.8	52.51	1.6	2,663
	0.9	44.72	1.7	2,451
	1.0	38.14	1.8	2,251
Inferred	0.4	45.90	1.1	1,553
	0.5	40.54	1.1	1,476
	0.6	34.82	1.2	1,375
	0.7	29.39	1.3	1,265
	0.8	24.79	1.4	1,153
	0.9	20.61	1.6	1,039
	1.0	17.12	1.7	932

Source : Coffey Mining 2011
Compliant with NI 43-101 Standard of Disclosure
Inconsistencies in computation due to rounding

The 2011 Mineral Reserve Estimate (Table 7) was carried out by Coffey Mining in 2011 and is presented in Table 6.

Table 6 : Previous Mineral Reserve Estimate at 0.4g/t Au Cut-off Grade – 2011

RESERVE CATEGORY								
PROVEN			PROBABLE			TOTAL		
Tonnage (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnage (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnage (Mt)	Grade (g/t Au)	Contained Gold (Moz)
5.1	1.2	0.199	74.3	1.1	2.68	79.4	1.1	2.88

Source : Coffey Mining 2011
Compliant with NI 43-101 Standard of Disclosure
Inconsistencies in computation due to rounding

6. Geological Setting and Mineralisation

NI 43-101 Item 7 (a), (b)

6.1. Regional Geological Setting

NI 43-101 Item 7 (a)

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. The core of Archaean sequences is surrounded by younger Palaeo-Proterozoic (2,2Ga-2,0Ga) sedimentary and volcanic units that form narrow (20km to 60km 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 (Figure 4). At least five greenstone belts are identified, namely the Ashanti, Asankrangwa, Sefwi, Kibi and Bui belts (Figure 4), which are described in detail in Section 7. The intervening sedimentary basins include from east to west; the Cape Coast Basin, Kumasi Basin and the Sunyani Basin (Figure 4).

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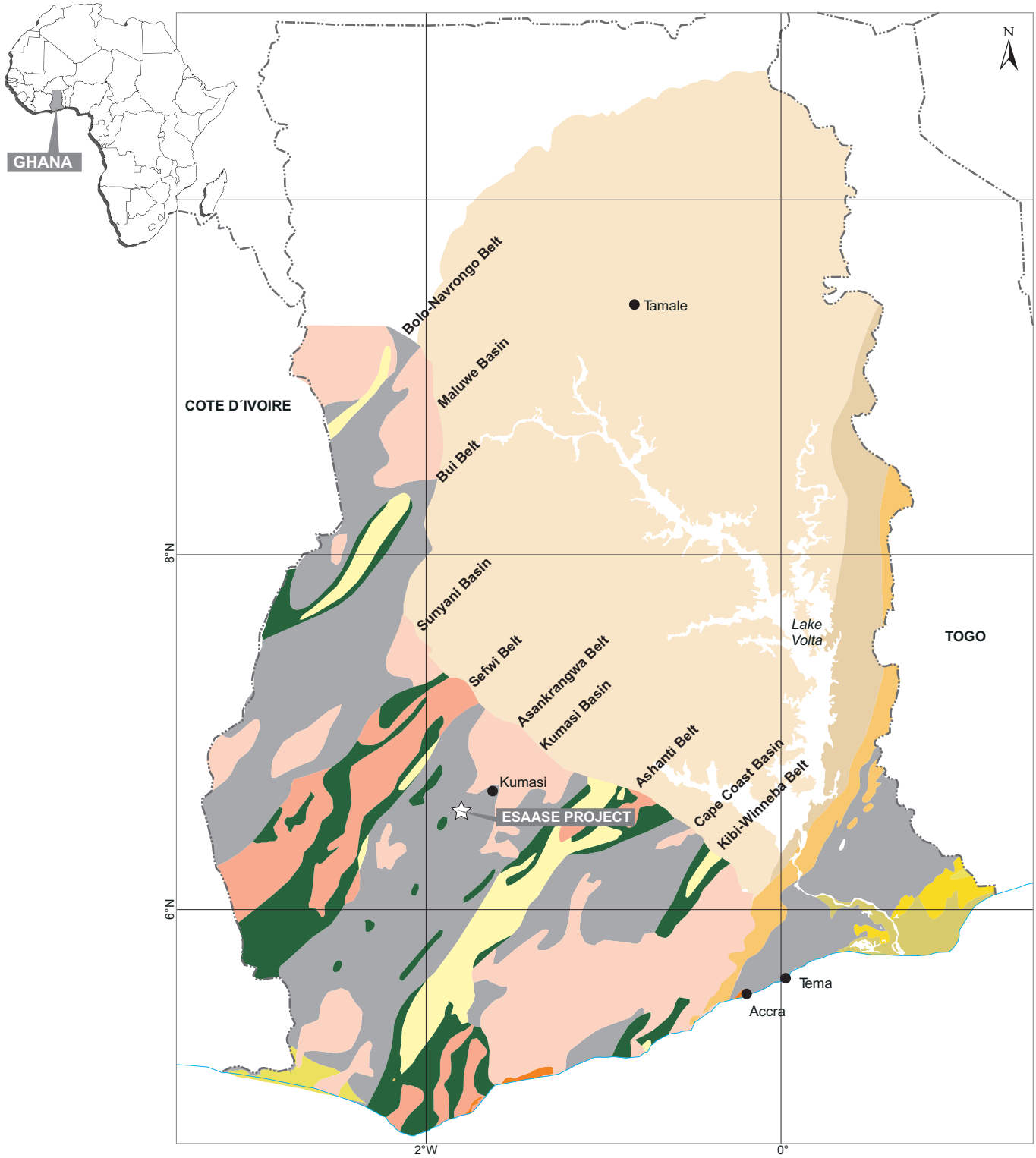
REGIONAL GEOLOGY OF GHANA



LEGEND		
CENOZOIC-CRETACEOUS	*Holocene-Pleistocene*	Sands, clays, gravels
	Neogene and Paleogene	Red continental deposits, mainly limonitic sands, sandy clays, gravels
	Eocene-Cretaceous	Marine shales, sandstones and limestones
	Cretaceous-Devonian	Sandstones, grits, conglomerates, shales, mudstones of the Sekondian and **Accraian Groups**
PALAEOZOIC	*Palaeozoic-Neoproterozoic*	Quartzites, shales, arkoses, mudstones, limestones, conglomerates of the **Voltaian Supergroup**
PROTEROZOIC	*Neoproterozoic*	Shales, sandstones, arkoses, lavas of the **Buerm Formation**
		Quartzites, shales, phyllites of the **Togo Series**
		Acidic and basic gneisses and schists of the **Dahomeyan Group**
PRECAMBRIAN	*Palaeoproterozoic*	Quartzites, phyllites, grits, conglomerates of the **Tarkwaian Group**
		Metamorphosed lavas and pyroclastic rocks of the **Upper Birimian System**
		Phyllites, schists, tuffs, greywackes of the **Lower Birimian System**
		Belt type granitoids
		Basin type granitoids

Source: Venmyn Deloitte 2013

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The five alternating low grade meta-volcanic and meta-sedimentary sequences are classified as the Palaeo-Proterozoic Birimian Supergroup (2,17Ga-2.10Ga), with the Lower Birimian comprising largely meta-sedimentary units and the Upper Birimian comprising mainly meta-volcanics (Figure 4). The meta-volcanic belts are separated by basins containing isoclinally folded dacitic volcaniclastics, wackes and argillitic sediments. The volcanic sequences and sedimentary units were deposited contemporaneously as lateral facies equivalents. The transition zone between greenstone belt volcanics and basin sediments is marked by a chemical facies, which comprises cherts, manganiferous and carbon-rich sediments and carbonates. The margins of the belts exhibit faulting on local and regional scales and these structures played a significant role in the genesis of the gold deposits for which the region is renowned.

Subsequent to the Birimian volcano-sedimentary basin development, the Eburnean tectono-thermal event took place between 2.2Ga and 2.0Ga, which caused deformation and metamorphism of the Birimian Supergroup. The Eburnean orogeny or Eburnean cycle comprised a series of compressional tectonic, metamorphic and plutonic events which resulted in regional faulting, folding of the pre-existing volcano-sedimentary sequences, intrusion by various types of granitoids, upliftment and erosion, as evidenced by the major unconformity dated between 2.15Ga and 2.11 Ga (Figure 4). The erosion products were deposited as sediments of the Tarkwaian Group (2.12-2.14Ga) in long, narrow intra-montane grabens which formed due to rifting preferentially in the central portions of all five Birimian volcanic belts (Figure 4). Each of the Birimian meta-volcanic belts is partially overlain by these younger, Tarkwaian Group clastic meta-sediments consisting of quartzite, shale and quartz pebble conglomerates. In the type area of the Tarkwa District, the 2.5km thick sequence hosts important palaeo-placer gold deposits. Relative stratigraphic relations suggest that deposition of Tarkwaian Group sediments took place as the underlying and adjacent volcanic and sedimentary rocks were undergoing the initial stages of compressional Eburnean deformation.

The Birimian sediments and volcanics were extensively metamorphosed to greenschist facies, although in many areas, higher temperatures and pressures are indicated by amphibolite facies mineral assemblages.

The Eburnean aged granitoids are classified into two major and one minor types, identified as follows (Figure 4):-

- basin granitoids (Cape Coast type) which occur as syn-orogenic, foliated batholiths chiefly in the central portions of Birimian sedimentary basins. The basin granitoids are peraluminous and generally granodioritic in composition;
- belt granitoids (Dixcove type) which are late-orogenic, unfoliated intrusions in volcanic belts. The belt granitoids are metaluminous in chemistry and commonly are tonalitic; and
- minor, post Tarkwaian age potassium (K) rich granitoids.

A genetic model to explain the formation of the volcano-sedimentary basins involves small-scale, equi-dimensional, parallel, and contemporaneously operating convection cells in the upper mantle. The convection cells caused rifting of a highly attenuated proto-crust, as well as linear eruptions of tholeiitic magmas to form the Birimian volcanics.

Clastic shallow water sediments of the Neo-Proterozoic Volta Sedimentary Basin cover the northeast of the country (Figure 4) and a small strip of Palaeozoic and Cretaceous to Tertiary sediments occur along the coast and in the extreme southeast of Ghana

6.2. Local Project Geology

NI 43-101 Item 7 (a)

The Esaase Project area comprises a sequence of intensely folded and faulted Birimian aged meta-sediments. Geological units on the Esaase property have been interpreted through a combination of airborne geophysical resistivity mapping (Versatile Time-Domain Electromagnetic Surveying or VTEM), resource definition drilling and associated outcrop mapping. Stratigraphic correlation between geological units is difficult due to the similarity of lithologies and the intense folding and deformation. However, broad subdivision of the meta-sediments into units with relatively high or low electrical conductivity and resistivity characteristics is possible.

The Esaase mineralisation comprises a system of gold-bearing quartz veins hosted by the tightly folded Birimian age sedimentary rocks. Lithologically the host sequences comprise massive, thinly layered phyllite; interlayered phyllite and siltstone with substantial carbonate in the matrix and massive, thick-bedded sandstone/greywacke. The mineralisation associated with the phyllite host occurs predominantly in the hanging wall of a resistivity break, whilst that associated with the sandstones/greywacke occurs in the footwall of the resistivity break (Figure 5).

6.2.1. Structural Aspects

The structural features of the Esaase Project area have resulted from Eburnean age compressional tectonic events which produced fold-thrust patterning followed by a late stage strike-slip deformation event. The deformational intensity increases systematically from the northwest to the southeast across the Project area as illustrated in Figure 6.

The folding in the deformed Birimian siltstone/shale sequences has a general axial plane strike direction of 020º to 035º and the folds plunge northeast 30º to 70º (Figure 6). The asymmetric folding ranges from open to tight, with isoclinal folds developed locally and the general dip direction is northwest. The folding tightens and deformation increases systematically towards the southeast of the Project area as shear zones are approached and this characteristic is observable on a 10m to a 100m scale. The asymmetry of the folding tilts towards the southeast, which is compatible with a regional interpretation of overall tectonic transport to the southwest (Figure 6).

The intensity of the folding increases with increasing crustal shortening and in these high strain zones, shear zones and thrust faults were developed. The strain within the shear zones is commonly eased by the shearing and the pattern of increasing strain to a point of shearing is repeated throughout the Project area on 10m to 50m scale (Figure 6).

The northeast striking, northwest-dipping syn-kinematic shears roughly parallel fold axial planes. Commonly, the basal shear/thrust separates the more deformed, altered, mineralised and electrically conductive hanging wall siltstone shale unit from the more massively bedded, and less deformed sandstone/greywacke in the footwall. Brecciated carbonaceous material at this contact indicates late brittle fracturing. The resistivity contrast marks the shears on a property wide scale.

The regional structural synthesis by Eisenlohr (1989) concluded that most of the structural elements in the region have common features which are compatible with a single, extended and progressive phase of regional deformation involving substantial northwest-southeast compression.

However, considerable heterogeneity exists locally, and at Esaase Project additional structural evidence indicates that towards the close of the tectonic event, the stress direction changed from northeast to southwest to north to south. The change in stress direction caused left lateral strike-slip movement along pre-existing faults (. The strike-slip movement created a northsouth dilational opening permitting the emplacement of late stage northsouth oriented veins (Figure 6).

The Esaase Project meta-sediments are intruded post-kinematically by younger dykes of intermediate to felsic composition (tonalite to granodiorite) which, towards the southern portion of the Project, are intensely brecciated and occur at or near the footwall of the mineralised units.

The weathering profile on the Esaase Property is strongly influenced by topography. The typical oxidised zone in tropical West Africa consists from surface downwards of surface laterite and ferruginous duricrust followed by saprolite which gradationally merges with oxidised bedrock. In regions of higher elevation at the Esaase Project, the laterite and underlying saprolite are removed by erosional processes and only the oxidised bedrock remains. At intermediate elevations, the weathering profile is mostly preserved 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.

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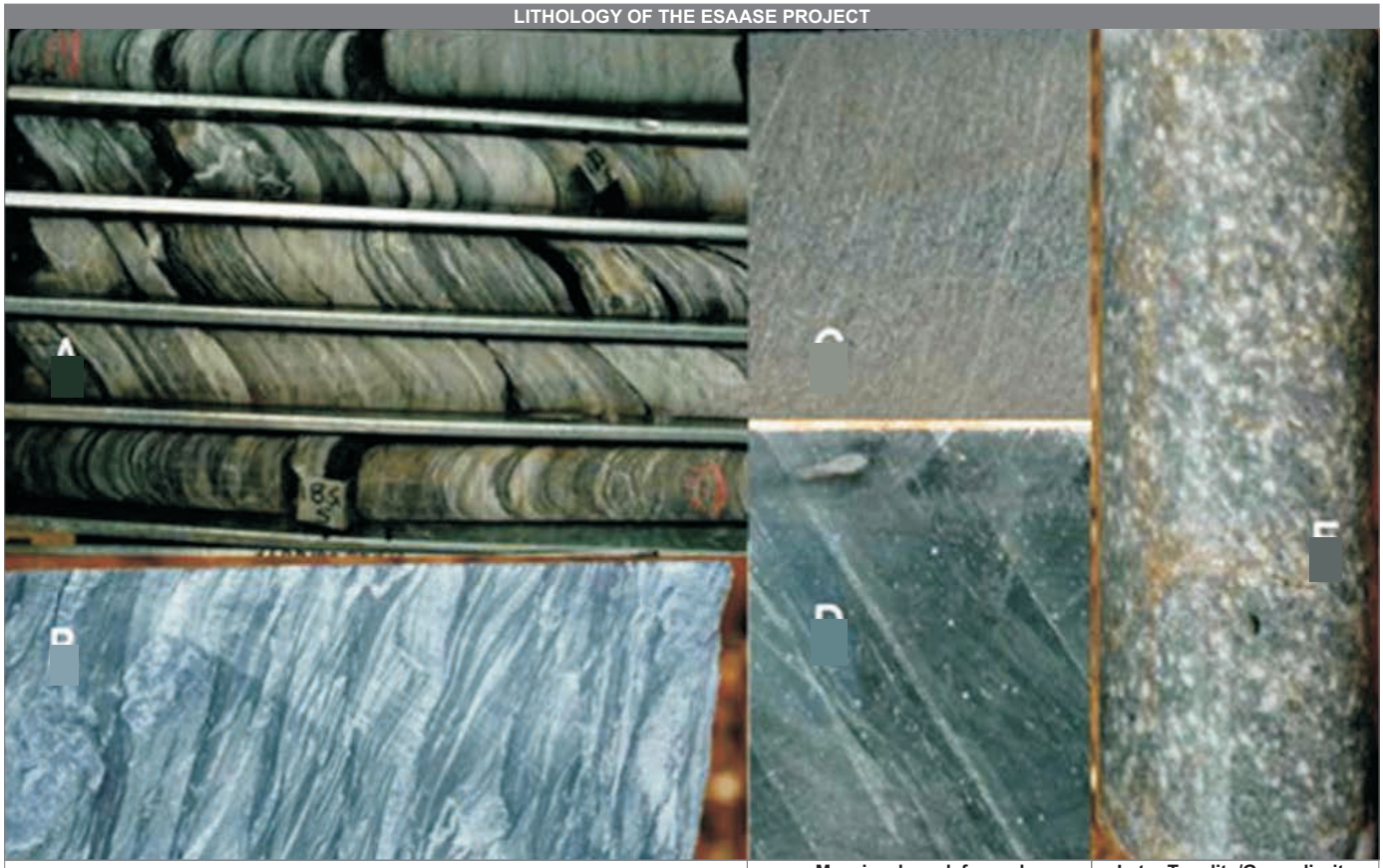
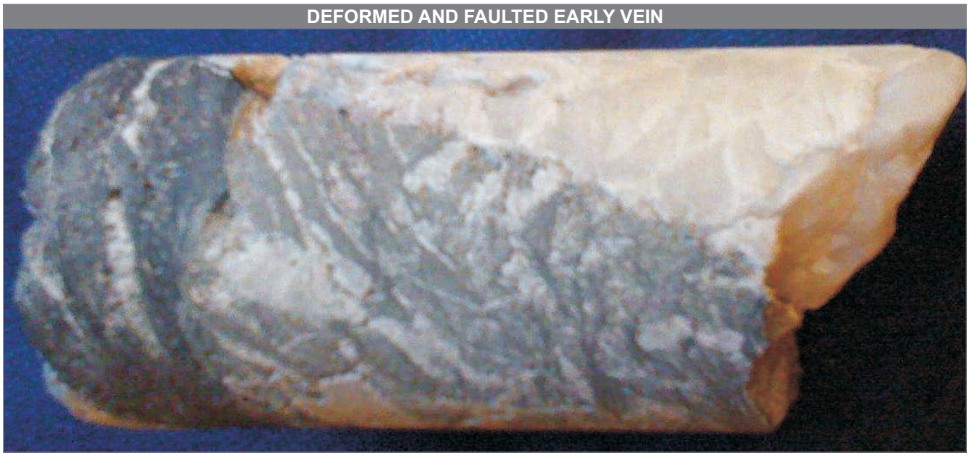
LITHOLOGY AND MINERALISATION OF THE ESAASE PROJECT

LITHOLOGY OF THE ESAASE PROJECT



Deformed Siltstone/Shale Hangingwall	Massive, less deformed Sandstone/Greywacke Footwall	Later Tonalite/Granodiorite Intrusions

DEFORMED AND FAULTED EARLY VEIN



MINERALISED VEIN WITH VISIBLE GOLD

Source: Minxcon 2012

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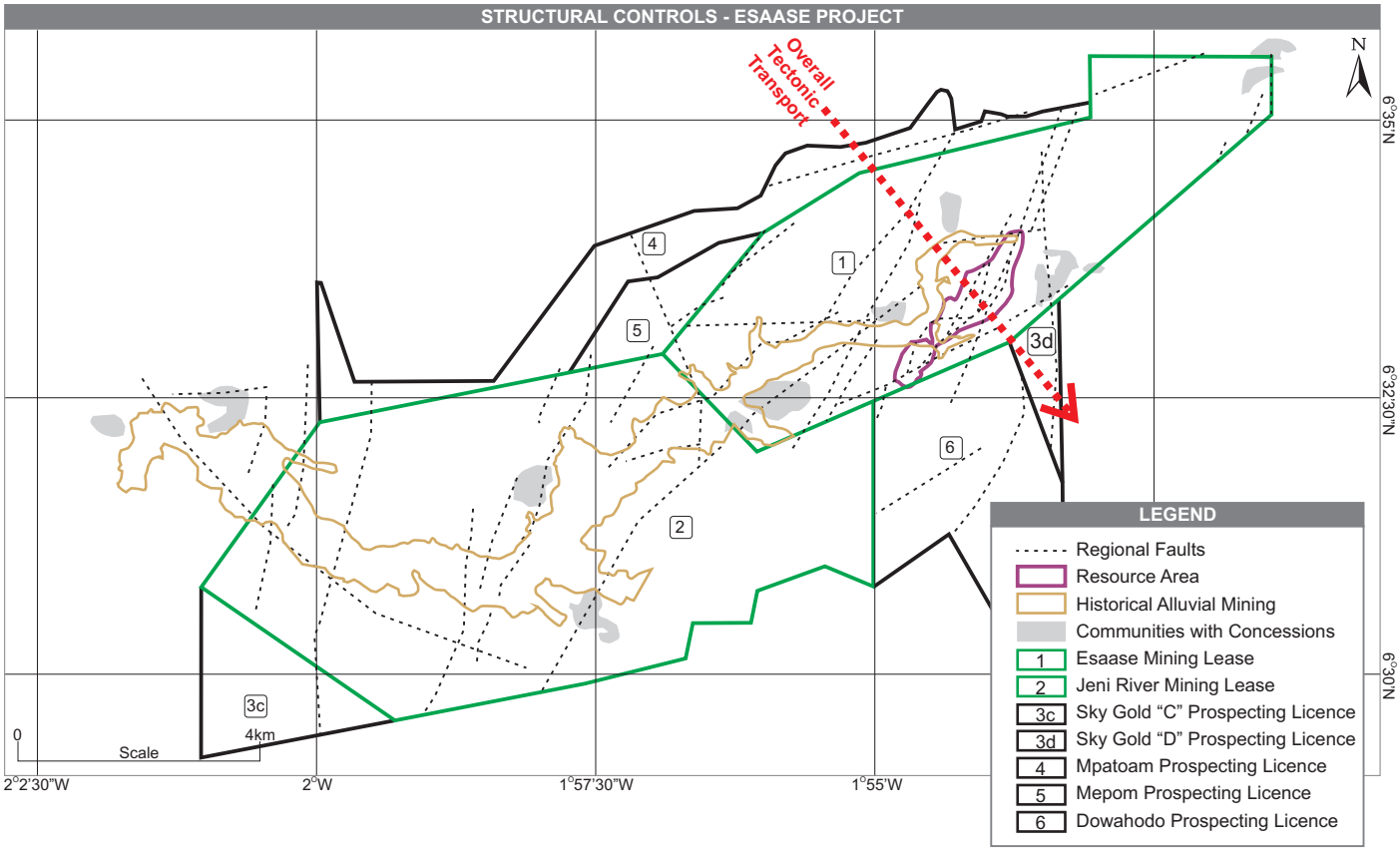
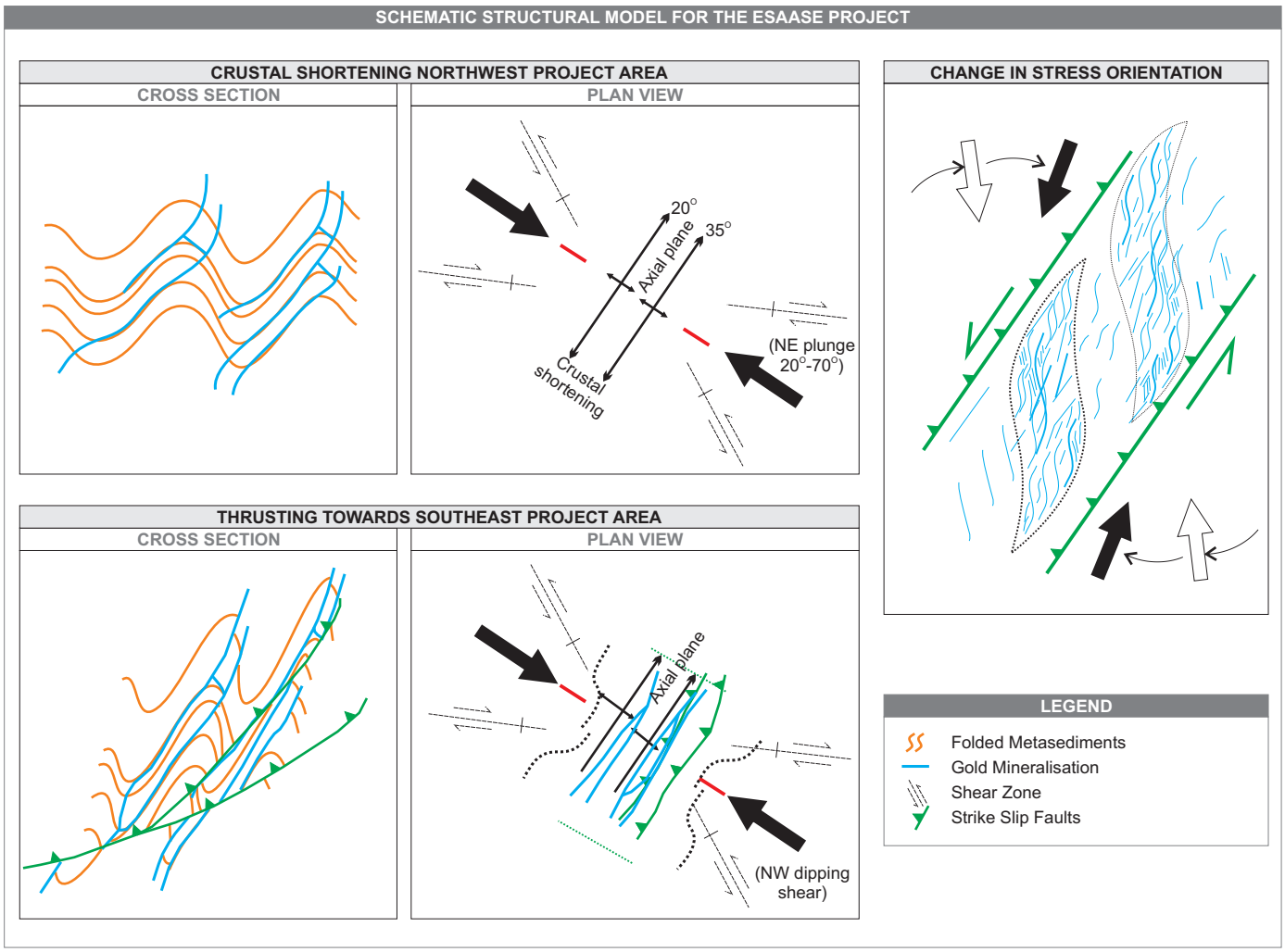
STRUCTURAL ASPECTS OF THE ESAASE PROJECT

STRUCTURAL CONTROLS - ESAASE PROJECT



SCHEMATIC STRUCTURAL MODEL FOR THE ESAASE PROJECT



Source: Venmyn Deloitte 2013

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6.2.2. Mineralisation

NI 43-101 Item 7 (b)

The target gold mineralisation on the Esaase property occurs in association with quartz-carbonate veins hosted within parallel, northeast trending, moderately to steeply west dipping units of extremely deformed siltstone/shale. The mineralisation is considered to be mesothermal-orogenic lode type veining produced by a series of fluid influxes which were channelled along lithological, rheological, lithological and structural boundaries.

Mineralisation occurred in a series of episodes over most of the duration of the extensive fold, thrust and strike slip deformation events. Four stages of vein development can be identified and these include (Figure 5):-

- an early un-mineralised quartz-only episode which has undergone later deformation and brecciation;
- a second, early vein episode which consists of numerous fine quartz-carbonate veins which display deformation and truncation by later veining;
- a third stage which consists of quartz-carbonate-sulphide veins with visible free gold. The associated sulphide assemblage is generally pyrite, but up to 15% can comprise chalcopyrite and minor arsenopyrite; and
- late stage post-mineralisation calcite veins crosscut all previous features

Fluid movement throughout the Eburnean tectonic event is evidenced by veining along lithologic boundaries, concentration in fold hinges, veining associated with the shear zones and thrust faults and late stage dilational en echelon veining (Figure 6).

The significantly mineralised veins strike generally 350º to 020º with sub vertical dips and are either planar or S-shaped in form. The veins are associated with the shear zones and thrust faults as illustrated in Figure 6 and late stage emplacement in dilation openings to produce en echelon vein sets. The latter veins were emplaced during a transition from fold thrust deformation to left lateral strike slip deformation (Figure 6).

Various alteration assemblages are present in the Esaase Project lithological units. A disseminated alteration most commonly present in the oxidised zone comprises a quartz-sericite-pyrite alteration which can be difficult to detect visibly in fresh core. Surface weathering converts the sericite to white kaolinite creating a white alteration product easily distinguished in trenches, road cuts, and drill pads.
On a smaller scale, pyrite pseudomorphs can be distinguished. A second alteration stage consists of pervasive carbonate alteration in the form of carbonate porphyroblasts, particularly after andalusite in phyllitic rocks.

7. Deposit Type

NI 43-101 Item 8

Most Ghanaian Birimian aged gold occurrences and mines are concentrated in narrow volcanic belts approximately 10km to 15km wide, separated by deep sedimentary basins (Figure 4). The volcanic belts comprise:-

- the Ashanti Belt which has been a world-renowned gold mining region for over 100 years. The main gold deposit is the AngloGold Ashanti Limited's Obuasi deposit, which began production in 1898. Underground mining operations have exploited vein mineralisation along 7kms of strike to a depth of 1.6km and the total historic and projected in situ gold resources are estimated at 60Moz. Similar vein deposits such as Prestea and Konongo have robust production histories and new million ounce deposits have been discovered along and adjacent to the Ashanti Belt. Tarkwaian Supergroup sedimentary rocks in the southern portion of the Ashanti Belt, host palaeo-placer deposits formed from the pre-existing Birimian aged vein deposits and new discoveries suggest that some Tarkwaian sequences also host vein mineralisation;

- the Asankrangwa Belt is a 5km to 10km wide belt of northeast-striking, steeply northwest dipping shear zones and minor intrusions that host gold-bearing quartz-carbonate vein systems. The shear zones are similar to those of the Kumasi Basin and are considered the structurally higher continuation of equivalent shear zones in the Kumasi Basin. The shear zones host gold mineralisation in a manner similar to that in gold deposits along the margins of the Ashanti and Sefwi Belts;
- the Kibi Belt is the eastern-most Birimian aged volcanic belt in Ghana, and the belt least explored by modern exploration techniques. The belt is similar to the Ashanti Belt in composition and structure, including the presence of Tarkwaian sedimentary rocks. Gold occurs in veins along shear zones near the margins of the belt in the historic Kibi Mine and the Abu Asafo concession straddles the southwest shear zones in the margin of the belt;
- the Sefwi Belt is a 40km to 60km wide, 220km long, Birimian volcanic belt located north of, and parallel to, the Ashanti Belt. Currently the Sefwi Belt hosts three gold mines, namely the Newmont Limited's Ahafo Mine, Redback Limited's Chirano Mine and the historic Bibiani Mine, which is now owned by Central African Gold. The Sefwi Belt is dominated by mafic volcanics, metasediments and intrusive granitoids which are bounded by adjacent the adjacent Sunyani sedimentary basin to the west and the Kumasi Basin to the east. Northeast marginal faults are traceable along the full length of the belt which are the prospective structures for mineralisation; and
- the Bui Belt is a 10km to 25km-wide, 150km-long belt located on the western border of Ghana which is parallel to and north of the larger Sefwi Belt. The Bui Belt is bounded by the Maluwe sedimentary basin to the north and the Sunyani Basin to the south. The Bui Belt is defined by a northeast-southwest trending syncline of Tarkwaian meta-sediments. The total thickness is estimated to be 9,000m which is more than 3 times thicker than the comparable units observed in the Tarkwa District.

The Birimian aged gold mineralisation within the belts occurs principally as:-

- disseminated mineralisation associated with sulphides, which is generally chemically controlled in specific lithologies; and
- quartz vein mineralisation which is exclusively structurally controlled along faults and shear zones resulting from compressional tectonic episodes, particularly along the margins of belts such as the Ashanti and Sefwi Belts.

The target Esaase Project mineralisation belongs to the latter category and is classified as a mesothermal orogenic quartz vein deposit. Such mesothermal orogenic deposits are associated with Precambrian and Phanerozoic convergent plate boundaries and are hosted in sequences generally metamorphosed to greenschist facies.

The mineralisation is carried by gold-sulphide enriched fluids which are strongly structurally controlled in crustal scale deformation features such as shears and thrust faults. The fluids responsible for the mineralisation can be generated by deep circulating meteoric water, magmatic fluids associated with tonalite/granodiorite intrusions and fluids associated with metamorphic events.

Orogenic gold deposits are the most significant style of mineralisation in West Africa and are commonly referred to as Ashanti-type deposits as the Obuasi area, within the Ashanti belt, is the type locality and the largest gold deposit in the region. The deposits are confined to tectonic corridors that are often >50km long and up to several kilometres wide and display complex, multi-phase structural features, which control the mineralisation. The most common host rock is fine-grained meta-sediment, often in close proximity to graphitic, siliceous, or manganiferous chemical sediments. However, mafic volcanics and later intrusions are also known to host significant gold occurrences.

Refractory deposits feature early-stage disseminated sulphides in which pyrite and arsenopyrite host gold mineralisation. The early mineralisation is overprinted by extensive late stage quartz veining in which visible gold is quite common and accessory polymetallic sulphides are frequently observed. These deposits include important lode/vein deposits in Ghana such as at Obuasi, Prestea, Bogosu, Bibiani and Obotan.

A 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, lack the arsenopyrite that is common in the refractory type deposits. Such deposits include the Chirano and Ahafo type deposits.

The Esaase Project exploration programme was specifically designed to investigate the structural features of the area as this is the primary control on mineralisation. The combination of geophysical approaches together with the drilling programme successfully delineated the Esaase mineralisation in parallel northwest trending mineralised structures.

8. Exploration

NI43-101 Item 9 (a), (b), (c), (d)

Minimal exploration was conducted on the Esaase Project area prior to the exploration programmes conducted by Keegan Resources in 2006.

8.1. Geophysical Programmes

An IP survey was completed in 2006 which successfully identified significant faults which are considered to be significant mineralisation boundaries (Figure 6 and Figure 7). In order to identify additional structures, Keegan Resources contracted Geotech Limited (Geotech) to perform an airborne VTEM geophysical programme over the Project area. The survey was conducted between 11 October and 25 October 2007. The principal geophysical sensors included Geotech's VTEM system and ancillary equipment included a Global Positioning System (GPS) and a radar altimeter. A total of 2,266 line-kms were flown at nominal traverse line spacings of 200m and flight line directions were N130°E/N50°W. The helicopter maintained a mean terrain clearance of 122m.

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 data was processed and interpreted by Condor Consulting Incorporated (Condor) which 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 characterised the 2D and 3D nature of the survey.

The 10 channel map shown in Figure 7 is a 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.

8.2. Soil Sampling Programme and Sampling Methodology

Keegan Resources commenced a soil sampling programme of the Esaase and Jeni Concessions in June 2006 which produced an assay database of 6,100 soil samples relating to these concessions. Sampling was undertaken on northeast oriented grid lines, spaced 100m to 400m apart with samples taken at 25m intervals. In addition 1,630 soil samples were collected from the Dowahodo prospecting licence.

The sampling methodology included the following protocols:-

- soil samples were taken only in areas where no obvious disturbance by alluvial miners was evident;
- care was taken to sample below the organic horizon;
- the soil samples below the organic horizon on ridge tops or steep slopes from higher elevations comprises weathered bedrock, whereas the samples nearer to the alluvial stream bed comprises colluvium and/or saprolite; and

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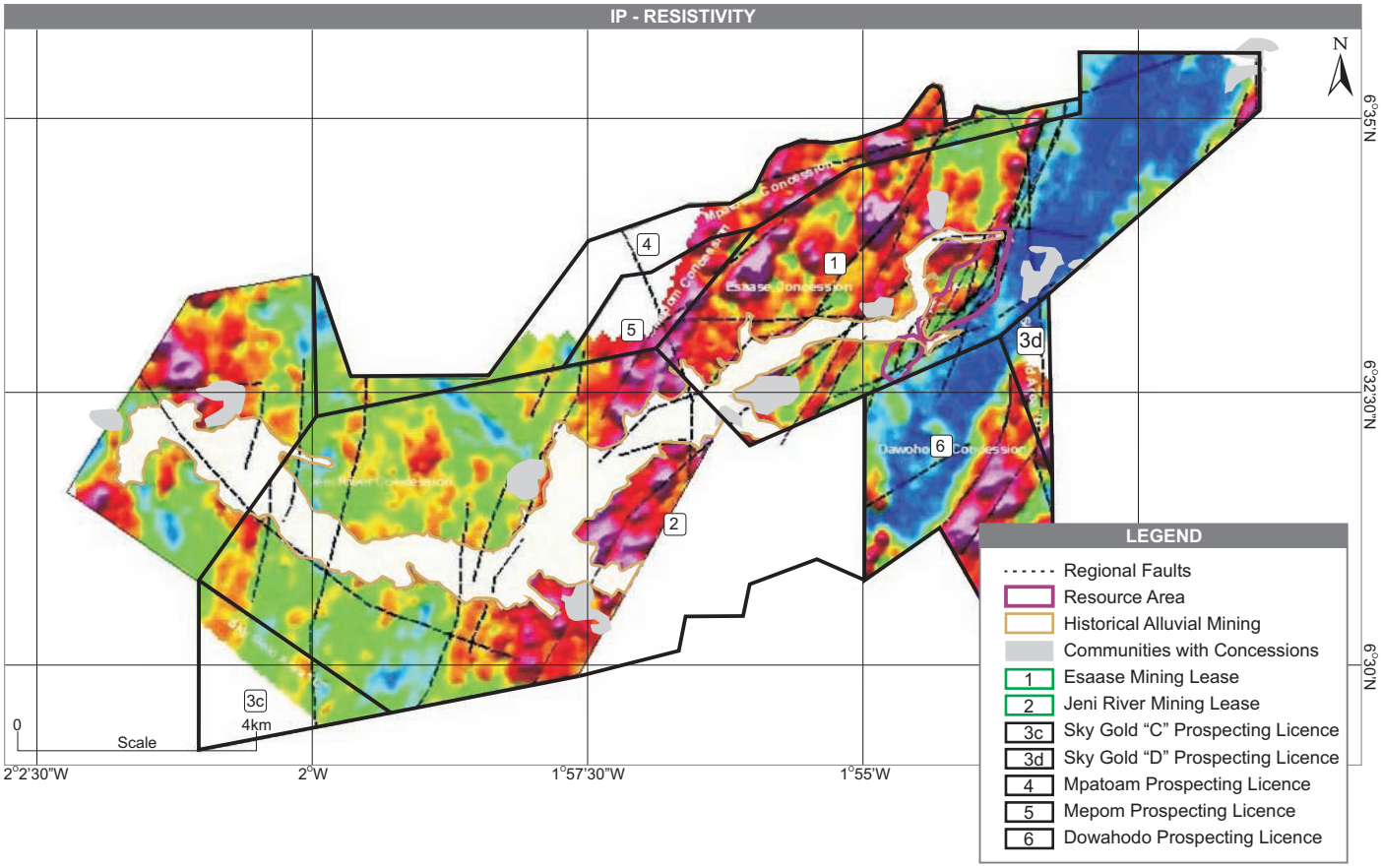
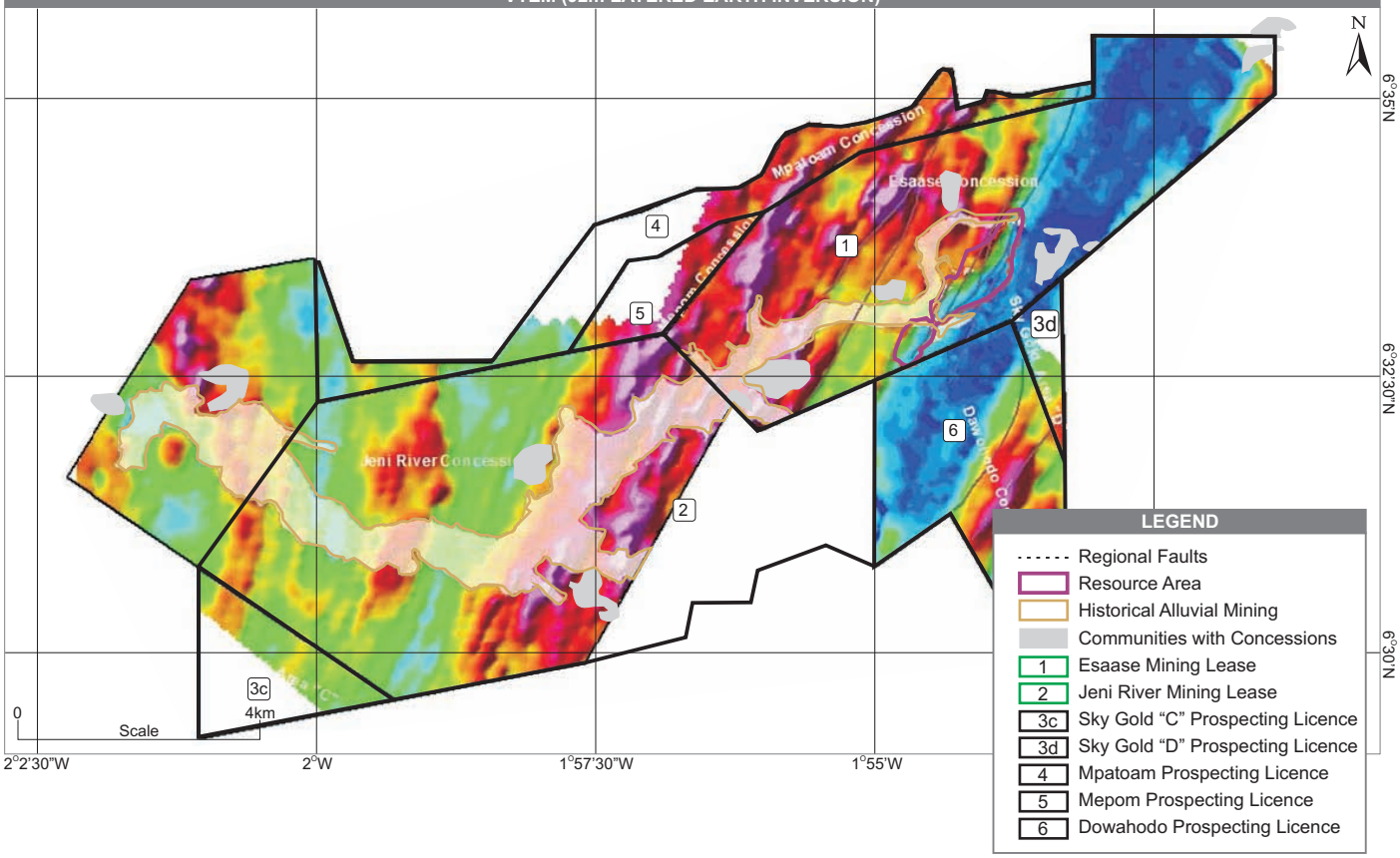
EXPLORATION OF ESAASE PROJECT - GEOPHYSICS

IP - RESISTIVITY



VTEM (92m LAYERED EARTH INVERSION)



Source: Minxcon 2012

- drilling and trenching indicate that soil samples from weathered bedrock, on average, have gold contents within an order of magnitude of the underlying bedrock values. Soil samples from non-bedrock sources (i.e. alluvial) tend to have much lower gold values than the underlying bedrock. As a result of this observation, Keegan Resources used auger sampling programs in order to get samples at or closer to the saprolite/soil interface.

The summarised results of the programme are presented in Figure 8 and clearly support the mineralisation model of parallel northwest trending mineralised structures.

8.3. Sampling Methodology for the Drilling Programme

See Section 9

8.4. Bulk Density Determinations

Bulk density determinations have been undertaken over a range of lithologies and oxidation states. The procedure is typical of bulk density determinations based on the Archimedes Principle of weight "in-air" versus weight "in-water". A custom set of bulk density scales with a weighing hook located underneath supplied by Corstor (South Africa) was utilised for the measurements:-

- 10cm billet of clean dry (dried in an oven for 4 hours at 60° C) core is weighed;
- core is immersed in paraffin wax then reweighed to establish weight of the wax; and
- core is then suspended and weighed in water to determine the volume.

The bulk density is then calculated as:-

Bulk density core = [Mass core] / [(Mass air – Mass water) – (Mass wax / 0.9)].

9. Drilling

NI 43-101 Item 10

The drilling programme conducted at the Esaase Project focused mainly on the northwest striking gold bearing structures in the Esaase concession but in addition, targets on the Jeni, Dawohodo and Mpatoam concessions were drilled. The drilling programme entailed both surface reverse circulation (RC) and diamond core (DC) drilling methods focused on the targets identified by soil sampling, trenching and geophysical interpretations. The DC and RC drilling programmes were conducted by independent Eagle Drilling Contractors and Geodrill Contractors both of which are reputable Ghana-based companies providing RC and DC drilling services consistent with current industry standards. The drilling programmes were supervised by Keegan Resources permanent staff qualified geologists.

A total of 1,496 drillholes were completed on the Project area. The drillhole collar positions are presented in Figure 8 and the vast majority of the drillholes into the west dipping mineralisation were collared at an orientation of approximately 100º (UTM). A small number of drillholes were drilled towards approximately 300º. Of these, 1,187 drill holes in the currently defined resource area were used for the resource estimation study.

9.1. Diamond Core Drilling Procedures

NI 43-101 Item 10 (a)

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

9.2. RC Drilling Procedures

NI 43-101 Item 10 (a)

The RC drilling was completed by Geodrill Contractors using a UDR KL900-02 multipurpose track mounted rig. The RC rods were 4½ inch diameter and the drill bit used was a standard 140mm diameter face sample hammer.

9.3. Topographic Control

Topographic models of the Esaase Project area have been generated as follows:-

- a total station survey was completed by Coffey Mining surveyors in 2007 over the primary deposit to an accuracy of +/-30cm and the survey compares well with the drillhole collar survey data; and
- a topographic model to a 2m contour interval was generated in several stages for the entire group of concessions by Photosat Information Limited using stereo pairs of IKONOS satellite images collected in December 2007 and July 2008. These images were ortho-rectified to control points including all drillhole collar points at the time surveyed by surveyors working for Asanko.

9.4. Downhole Surveying Procedures

The drillholes were 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.

9.5. Sampling Procedures

9.5.1. RC Sampling and Logging

The drill chips from the RC drilling programme were collected in 1m intervals downhole via a cyclone which discharged into PVC bags. The collected samples were weighed prior to splitting and then were riffle split using a three tier Jones riffle splitter. A final sample of approximately 3kg was collected for submission to the laboratory for analysis. The RC chip material was stored in trays which were systematically compiled and logged with all bulk rejects stored at the Asanko Gold's exploration camp in the village of Tetrem. All 1m interval samples were submitted for analysis.

9.5.2. Diamond Core Sampling and Logging

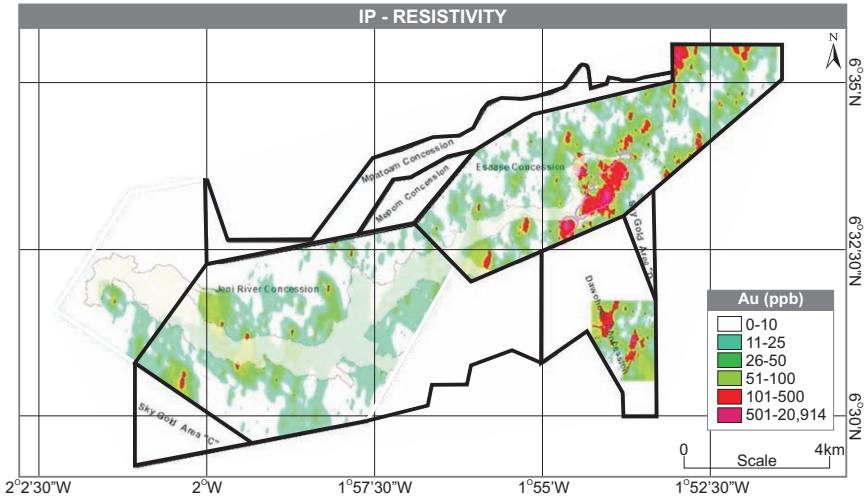
The sampling of the core was subject to the discretion of the geologist completing the geological logging. Early in the exploration, nominal 2m intervals samples were taken unless otherwise dictated by geological or structural features. After December 2006, the sample interval was 1m intervals with the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

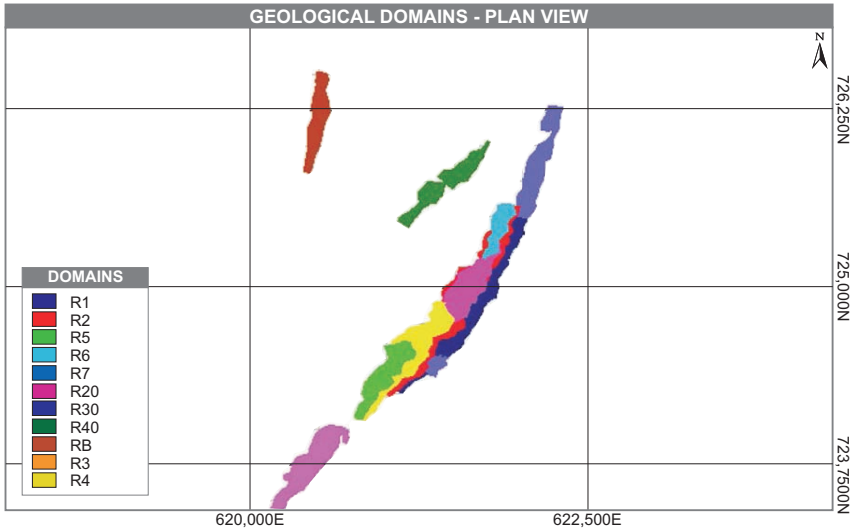
The sampling intervals are significantly smaller than the true width of overall mineralised zones, which is variable throughout the deposit, but is typically in excess of 30m.

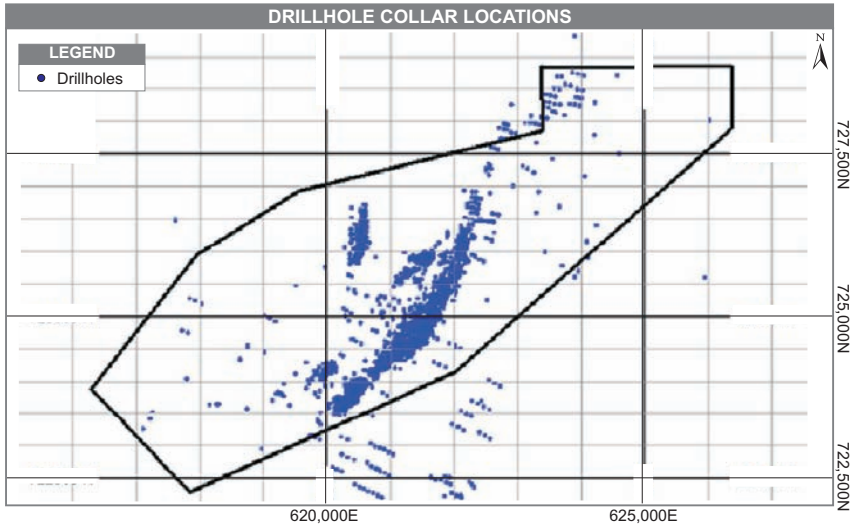
The required interval was marked on the core and the sample cut in half by electric diamond blade core saw. The standard protocol is that the cut is made 1cm to the right in a downhole direction of the orientation line, with the left side being retained and the other half broken up for assay. In the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

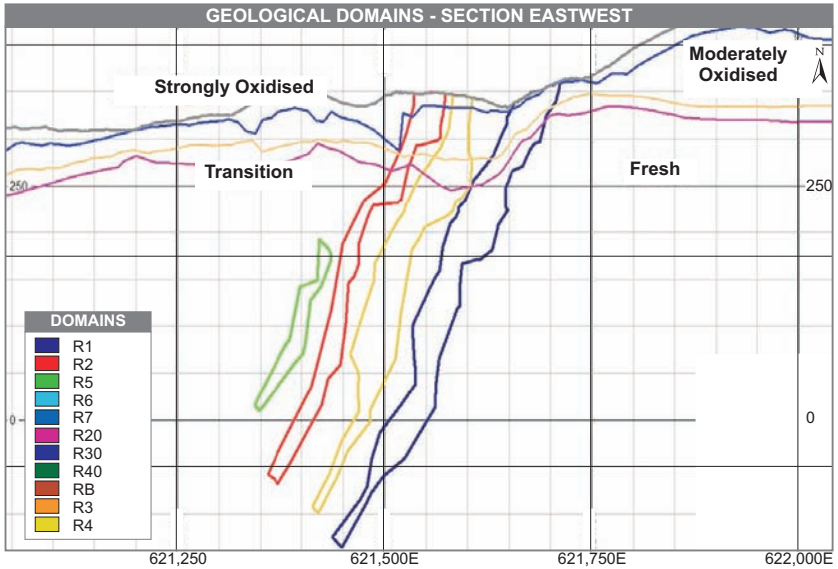
The structure orientations noted in the core were routinely recorded to assist in determining the controls on mineralisation, in establishing a reliable geological model for resource estimation, and to provide additional geotechnical information to determine likely blast fragmentation and pit stability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

EXPLORATION OF ESAASE PROJECT - GEOLOGICAL MODEL









Source: Minxcon 2012, Asanko Gold 2013

D1237_DRAKeeganResources_2013

Deloitte.

Asanko Gold

Figure 08

The core was transferred from the trays and pieced together on a V-rail (angle iron) rack. The orientation line (bottom of hole), determined by the orientation tool recorded during drilling, was 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 metallurgical drillholes of approximately 28 oriented HQ3 core drillholes were drilled radially outward from within the deposit through depths beyond an assortment of potential pit wall limits.

9.5.3. Recovery

The sample recovery for the RC drilling averages approximately 34kg per metre drilled. Bulk sample weights (on a per metre basis) have been recorded in the database for approximately two thirds of all RC samples drilled. Sample recovery in DC drillholes was good although in the moderate to highly weathered saprolite and highly fractured and brecciated zones poor recoveries were experienced. Asanko Gold began utilising HQ3 drilling to minimise the core loss in the weathered and transition zones after July 2008.

Recovery factors are unlikely to materially affect the accuracy and reliability of the results.

9.5.4. Sample Quality

The Keegan Resources sampling procedures adopted for the drilling programmes are consistent with current industry best practise. Samples collected by DC drilling within the highly weathered zones are of moderate quality, with the remainder being of good quality. Sample recoveries and quality for the RC drilling are high with drilling switching to diamond core once wet samples were noted.
A quality control twin drillhole exercise was undertaken to determine if any negative bias resulted in the DC drilling due to the use of water. A number of the DC drillholes had poor recovery in the highly weathered zone and potential exists to wash out fine gold and therefore underestimate the gold content. Four DC and RC drillhole pairs were suitable for comparison and results indicate comparable intervals of mineralisation with broadly equivalent grades between DC and RC drilling.

10. Sample Preparation, Analysis and Security

NI 43-101 Item 11

Sample preparation and assaying of samples from the Esaase Project have to date been undertaken at three independent laboratories:-

- SGS Tarkwa (SGS) (from April 2007): part of the global group of SGS laboratories with ISO/IEC 17025 accreditation;
- Transworld Tarkwa (TWL) (from October 2006): TWL Tarkwa was acquired by Intertek Minerals Group in October 2008. Intertek Minerals Group includes Genalysis Laboratory Services Pty Limited of Australia and operates in accordance with ISO/IEC 17025, which includes the management requirements of ISO 9001:2000; and
- ALS Kumasi (from November 2007) ALS Laboratory Group with ISO 9001:2000 accreditation:

10.1. Sample Security

NI 43-101 Item 11 (a)

Asanko Gold sampling protocols require samples to be collected in staple-closed bags, transported to the Project camp to be collected by the laboratory vehicle, at which point the laboratory assumes responsibility and transports the consignment to the laboratory directly. The samples submission procedures are supervised by Asanko Gold technical staff and the rapid submission of samples 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.

10.2. Field Quality Control Measures

NI 43-101 Item 11 (a), (b)

Duplicate samples are routinely collected by Asanko Gold to assess field sampling error or bias and acceptable reproducibility of RC field duplicate data indicates no bias in the sampling. Standard internationally certified reference material (CRM) is used as a measure of accuracy and 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 (Coffey Mining 2009, Minxcon 2012). The following Quality Control Measures are standard Asanko Gold protocols:-

- insertion of 16 (Geostats Standards and CDN Resource Standards) internationally CRMs (5% of samples);
- insertion of blank material (5% of samples);
- RC field duplicates taken (5% of samples);
- diamond drillhole duplicates from second split at the 3mm jaw crushing stage;
- submission of selected umpire samples; and
- review of the Asanko Gold and the internal laboratory QC data on a batch by batch basis.

10.3. Laboratory Sample Preparation Methodology

NI 43-101 Item 11 (b)

The various sample preparation techniques undertaken at the three independent laboratories are summarised and compared in Table 7:-

Table 7 : Sample Preparation Methodologies

ASPECT OF SAMPLE PREPARATION	SGS TARKWA LABORATORY	ASL KUMASI LABORATORY	TRANSWORLD TARKWA LABORATORY
Sample size	3kg sample	3kg	3kg sample
Drying	Dried and disaggregated	Dried and disaggregated	Dried and disaggregated
Crushing	Jaw crushed to 3mm	Jaw crushed to 2mm	Jaw crushed to 3mm
Pulverising	Pulverised to nominal 95% passing -75µ using LM2 pulveriser	Pulverised to nominal 85% passing -75µ using LM2 pulveriser	Pulverised to nominal 95% passing -75µ using LM2 pulveriser
Pulp samples	Two samples retained for analysis and storage	Two samples retained for analysis and storage	Two samples retained for analysis and storage
Analysis	50g charge, fire assay fusion, lead collection, AAS determination to 0.1ppm detection limit	50g charge, fire assay fusion, lead collection, AAS determination to 0.1ppm detection limit	50g charge, fire assay fusion, lead collection, AAS determination to 0.1ppm detection limit

Source ;: Coffey Mining 2011, Minxcon 2012

The sample preparation and analytical procedures adopted to date are consistent with current industry practise and are considered by Minxcon to be acceptable for the style of mineralisation identified at Esaase Project. Venmyn Deloitte concurs with this opinion.

10.4. Laboratory Quality Control and Quality Assurance Measures

NI 43-101 Item 11 (a), (b), (c)

The quality control measures undertaken at each of the internationally accredited laboratories are summarised in Table 8:-

Table 8 : Laboratory Quality Control Measures

CONTROL MEASURES	SGS TARKWA	ASL KUMASI	TRANSWORLD TARKWA
Sample identification	Cross referencing of sample identification tags and client submission sheets	Cross referencing of sample identification tags and client submission sheets	Similar to SGS Tarkwa
Cleaning equipment	Compressed air gun used to clean crushing and milling equipment between samples	Compressed air gun used to clean crushing and milling equipment between samples	
Contamination prevention	Barren 'wash' material applied to the milling/pulverising equipment at between sample preparation batches	Barren 'wash' material applied to the milling/pulverising equipment at between sample preparation batches	
Contamination assessment	Quartz washes assayed prior to use to determine the level of cross contamination	Quartz washes assayed prior to use to determine the level of cross contamination	
Sieve tests	Sieve tests carried out on pulps at the rate of 1:50 to ensure adequate size reduction	Sieve tests carried out on pulps on a regular basis to ensure adequate size reduction	
CRM	Assaying of certified standards at the rate of one per batch of 20	Assaying of certified standards at the minimum rate of one per batch (dependant on batch size and assay technique)	
Duplicate samples	A minimum of 5% (1:20) of the submitted samples in each batch are subject to repeat analysis	A minimum of one of the submitted samples in each batch are subject to repeat analysis	
Blank samples	Blank samples inserted at the rate of approximately 1:30	Blank samples inserted at the beginning of each batch	
CRM	Industry recognised certified standards disguised and inserted at a rate of 1:30		
Internal CRM	Assaying of internal standards data		
Umpire checks	Participation in two international round-robin programs; LQSi of USA and Geostats of Australia	Participation in a number of international round-robin programs which include CANMET of Canada and Geostats of Australia	

Source ;: Coffey Mining 2011, Minxcon 2012

10.5. Quality Control Analysis

NI 43-101 Item 11 (c)

Detailed independent analysis of the quality control data was originally undertaken by Coffey Mining 2011 and reviewed by Minxcon 2012. The data included:-

- field and laboratory standards and blanks for RC and diamond drillhole data;
- laboratory duplicates;
- re-assayed pulps and
- umpire assaying.

The quality control data was statistically assessed to determine relative precision and accuracy levels various sets of assay pairs and the quantum of relative error. The statistical analysis included the use of the following comparative statistical methodologies and a summary of the results is presented in Table 9:-

- Thompson and Howarth Plot showing the mean relative percentage error of grouped assay pairs across the entire grade range, used to illustrate precision levels by comparison 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 illustrate 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 which provides an overall visualisation of precision and bias over selected grade ranges. Correlation coefficients are also used; and
- Quantile-Quantile (Q-Q) plots which are a means whereby the marginal distributions of two datasets can be compared. Similar distributions should be noted if the data is unbiased.

10.5.1. Umpire Laboratory Assay Results

NI 43 – 101 Item 11 (c)

In January and February 2007 a total of 1,197 RC samples were re-split and sent for analysis at SGS Tarkwa. Only assays >0.1g/t Au were considered in the analysis and a total of 481 assay pairs were available for analysis. Results show a significantly lower mean (by 15.6%) for analysis completed at SGS, although this 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 >0.1g/t Au were 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

10.5.2. Quality Control and Assurance Conclusions

NI 43-101 Item 11 (d)

Minxcon is of the opinion that the quality control and assurance undertaken by Coffey Mining is adequate and that the current protocols 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:

- use of CRM has shown a significant relative low bias for SGS Laboratories, Tarkwa;
- use of CRM has shown a relative high bias for Transworld Laboratories, Tarkwa and this interpretation is supported by the umpire analysis programme;
- repeat analyses have confirmed that the precision of sampling and assaying is generally within acceptable limits for sampling of gold deposits; and
- umpire analysis at Genalysis in Perth has shown a lack of precision between the various laboratories. This is currently unexplained and requires investigation.

Table 9 : Quality Control and Assurance Data Analysis

CONTROL MEASURES	SGS TARKWA LABORATORY	ASL KUMASI LABORATORY	TRANSWORLD TARKWA LABORATORY
Duplicate samples/splits	RC (339) and diamond core (73) duplicates - second split at the 3mm jaw crushing stage of the sample preparation. Also four random repeats in every batch of 50 samples	RC (176) and diamond core (62) duplicates a-second split at the 3mm jaw crushing stage	Every 20 samples repeated. Only assays >10xs the detection limit are included in the analysis.
Results	Results for both duplicate sets show equivalent means and a high level of precision between the original and the re-assay for both diamond core and RC samples.	Results show equivalent means and a high level of precision between the original and the re-assay for the diamond core samples however the second mean is 7.5% lower than the original assay.	Results show equivalent means between the duplicate repeats and precision within acceptable limits for both diamond core and RC samples
AAS Pulp re-spray			Every 10 samples are repeated. Only assays >10xs detection limit (0.1ppm Au) are included. 1202 assays in dataset
Results			Results show equivalent means between the duplicate repeats and precision well within acceptable limits
Random repeat assays	Random repeat assay with four random repeats completed from each batch of 50 samples. A total of 582 Diamond core and 2,392 RC analyses are available for analysis.	Random repeat assays RC (892) DD (223)	
Results	Results show equivalent means and an acceptable level of precision between the original and the re-assay	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 diamond core and RC samples.	
High grade check repeats			50g samples from the original sample envelope. Check analyses to 2007 available.
Results			Results show equivalent means between the duplicate repeats and precision within acceptable limits for both diamond core and RC samples
Pulp re-assay			Only pulp re-assays > 10xs the detection level (0.1ppm Au) considered 1,615 riffle split 1m RC drill chip assays
Results			Results show equivalent means between the duplicate repeats and precision within acceptable limits.
Laboratory standards and blanks	Four 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 16 Certified standards and one blank have been included in sample batches sent to TWL, ALS and SGS. A total of 11,507 assays were available for analysis.	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
Results	A total of 938 standards and blanks assays are available for analysis. Results show a relative low bias of up to -2.09%.	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.	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
Field Blanks	1163 blanks included	2802 blanks included	1567 blanks included
Results	Results show equivalent means and acceptable precision for both RC and diamond core samples	Results show equivalent means and acceptable precision for both RC and diamond core samples	Results show equivalent means and acceptable precision for both RC and diamond core samples

Source ;: Coffey Mining 2011, Minxcon 2012

11. Data Verification

NI 43-101 Item 12 (a), (b), (c)

Verification of the collar positions of drillholes, drillhole survey data and checks of lithological logging of the drillhole intersections was undertaken by Coffey Mining in 2011.

Reviews of the drilling, sampling, QA/QC databases were undertaken both by Coffey Mining 20011 and Minxcon 2012. The Mineral Resource estimation was based on the available exploration drillhole database which was validated by Minxcon prior to commencing the 2012 resource estimation study. Data included samples from extensive trenching, but only the RC and diamond drilling sample data were included for use in the modelling process. A total of 1,187 drillholes and 233,503 composite samples were used for the resource estimation. Checks made to the database prior to modelling included:-

- no overlapping intervals;
- downhole surveys at 0m depth;
- consistency of depths between different data tables; and
- checks for any gaps in the data.

The application of the surface drillhole data is adequate for the geostatistical estimation processes employed in the mineral resource estimation. The data is spatially well represented and of an adequate support level for estimating deposits of this nature (Minxcon 2012) The procedures and codes of practice employed by Asanko Gold personnel 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. Mineral Processing and Metallurgical Testwork

NI 43-101 Item 13

The 2011 PFS for the Esaase Project was based on an open pit mining methodology with an associated “whole-ore” leach processing plant. The study included initial mill throughputs of 9Mtpa reducing to 7.5Mtpa once mining of the fresh zone began. The 2011 PFS resulted in a positive NPV for the Project but additional studies indicated that better economic outcomes could be achieved with alternative process design methodologies. As a consequence, Asanko Gold completed a Concept Study in August 2012 with a revised project scope which included a flotation process in contrast to the “whole-ore” leach used in the 2011 PFS. The Concept Study indicated that an improvement of approximately USD80m could be achieved on the Project NPV. In September 2012 Asanko Gold appointed DRA to undertake the revised, update 2013 PFS.

12.1. Previous Metallurgical Testwork and Processing Studies

NI 43-101 Item 13 (a)

The Coffey Mining/Lycopodium 2011 study included four phases of testwork, including mineralogy, and testing of processes for oxide, transitional and fresh ore types. The gold appears to be generally free milling and coarse enough for mill/gravity recovery of a significant component, however there is a fine grained component included within sulphide particles which was shown to be less amenable to conventional processing.

A summary of the four phases of metallurgical testwork undertaken for the 2011 PFS, the results of the testwork and the consequential design modifications are summarised in Table 10:-

Table 10 : Summary Previous Metallurgical Testwork Phases I to IV

PHASE	TESTWORK SCOPE	SAMPLES	TESTS	RESULTS	COMMENTS
Phase I (2008 - 2009)	Diagnostic testing of oxide, transition and fresh ore not aimed at plant design criteria	Individual core sections	Cyanide leach tests at a grind of 80% passing 45µ - residence time 48hrs	Recovery oxide -91.0% Recovery transition 71% Recovery Fresh 54.-%	Low recoveries in fresh and transition due to coarse gold and indicated the need to include gravity recovery
			Cyanide leach tests at a grind of 80% passing 150µ for gravity/flotation circuit with leaching of the tailings	Increase in overall recovery to 92%	
			Cyanide leach tests at a grind of 80% passing 75µm for gravity/flotation circuit with leaching of the tailings	Increase in overall recovery to 97%	
Phase II (2009)	Develop Concept Study process flowsheet comprising comminution, gravity concentration and CIL on gravity tailings - 48hr residence time. Variability between fresh and oxide	Composite samples of remaining core from Phase I - two composite samples one of of fresh and the other of oxide	Cyanide leach tests at a grind of 80% passing 150µm - gravity concentration followed by and CIL (48hr residence time) on tailings	Recovery oxide -94.80% Recovery Fresh 95.5%	Gravity concentration did not include Mozley tabling and therefore the recoveries are considered to be overstated
			Cyanide leach tests at a grind of 80% passing 75µm - gravity concentration followed by CIL (48hr residence time) on tailings	Recovery oxide -96.9% Recovery Fresh 97.0%	
Phase III (2009 - 2010)	Detailed design parameters for fresh and oxide ore types. Process included comminution, gravity concentration, leaching of gravity concentrate and CIL on tails	Representative composites and individual PQ sized intercepts at differing strike positions and depths	Grind size optimisation studies	Optimum of 80% passing 150µ	
			CIL optimisation comparing 24hr residence time, NaCl addition to maintain 500mg/L in solution and pulp density of 50% solids		
			Whole ore CIL without gravity	Reduced recoveries due to coarse gold	
			Gravity concentration revised to upgrade Knelson concentrate using Mozley concentrator	Results presented in Table 11The reduced recoveries were attributed to un-liberated gold in the Mozley tailings	Conclusion that grinding the Mozley tails would improve recovery
			Variability testing on optimised conditions for oxide, transition and fresh	Results presented in Table 11. Focus to be on identifying process techniques that could recover the equivalent of Mozley tails	Resultant enhanced gravity separation test work phase which looked at Wilfley table testing and flotation
Phase IV	Addition to Phase 111 with 'extended gravity' to improve recovery of ultrafine sulphide associated gold. The process flow included comminution, gravity concentration and leaching of gravity concentrate, gravity concentration of mill product with spirals and CIL on thickened spiral concentrate	Remaining material from Phase III	Grind size optimisation studies	Optimum of 80% passing 106µm	DRA considers more significant benefit to recovery at a target grind size of 80% passing 75µm which was most apparent for the oxide material
			Gravity circuit that included enhanced gravity separation using a Wilfley table to replicate spirals	Enhanced gravity circuit	
			Enhanced gravity circuit tested to define the CIL parameters at 24hr residence time, NaCl addition to maintain 350mg/L in solution and pulp density of 50% solids		
			Variability testing on enhanced gravity flowsheet and optimised conditions for oxide, transition and fresh material	See Table 12	

Source : DRA 2013, Coffey/Lycopodium 2011

Table 11 : Phase 111 Variability Testwork Results for Fresh and Oxide Material

SAMPLE ID	CALCULATED HEAD GRADE (g/t)	MOZLEY Au RECOVERY (%)	KNELSON AND MOZLEY TAILININGS CIL		OVERALL EXTRACTION (%)	NaCN (kg/t)	LIME (kg/t)
			EXTRACTION (%)	RESIDUE (g/t)			
KEDD 721 Fresh	0.37	43.20	40.50	0.06	83.78	1.18	0.16
KEDD 723 Fresh	2.85	80.00	15.40	0.13	95.44	0.77	0.17
KEDD 729 Fresh	0.22	18.20	63.60	0.04	81.82	0.89	0.10
KEDD 751 Fresh	0.69	4.30	82.60	0.09	86.96	0.87	0.09
KEDD 753 Fresh	0.95	28.40	57.90	0.13	86.32	0.95	0.09
KEDD 754 Fresh	0.56	30.40	55.40	0.08	85.71	1.05	0.15
KEDD 763 Fresh	0.43	20.90	58.10	0.09	79.07	0.90	0.09
KEDD 536 Fresh	0.98	3.10	74.50	0.22	77.55	1.02	0.12
KEDD 537 Fresh	1.10	24.50	42.70	0.36	67.27	1.05	0.11
KEDD 538 Fresh	0.76	9.20	72.40	0.14	81.58	0.88	0.12
KEDD 539 Fresh	1.20	2.50	84.20	0.16	86.67	0.84	0.10
KEDD 540 Fresh	0.96	37.50	50.00	0.12	87.50	0.83	0.09
KEDD 541 Fresh	1.11	34.20	50.50	0.17	84.68	0.88	0.10
KEDD 542 Fresh	1.17	8.50	75.20	0.19	83.76	0.80	0.10
KEDD 552 Fresh	1.70	30.00	55.30	0.25	85.29	0.84	0.11
KEDD 657 Fresh	0.98	19.40	67.30	0.13	86.73	0.85	0.21
KERC 671 Fresh	1.19	2.50	76.50	0.25	78.99	0.90	0.11
KEDD 946 Fresh	0.42	2.40	69.00	0.12	71.43	0.89	0.07
KEDD 947 Fresh	1.47	2.70	91.20	0.09	93.88	0.78	0.05
KEDD 419 Fresh	0.79	1.30	74.70	0.19	75.95	0.81	0.12
KEDD 552 Fresh	0.59	32.20	57.60	0.06	89.83	0.67	0.11
KEDD 582 Fresh	0.94	31.90	52.10	0.15	84.04	0.75	0.33
KEDD 760 Fresh	1.36	32.40	59.60	0.11	91.91	0.98	0.20
KEDD 831 Fresh	0.96	5.20	86.50	0.08	91.67	0.75	0.11
KERC 221 Fresh	1.14	56.10	36.00	0.09	92.11	0.78	0.12
KERC 281 Fresh	0.97	43.30	44.30	0.12	87.63	0.87	0.15
KERC 504 Fresh	0.45	4.40	57.80	0.17	62.22	0.92	0.14
KERC 243 Fresh	1.00	39.00	46.00	0.15	85.00	0.76	0.13
KERC 244 Fresh	0.49	2.00	77.60	0.10	79.59	0.82	0.12
KERC 245 Fresh	0.86	27.90	52.30	0.17	80.23	0.70	0.42
KEDD 721 Oxide	0.76	11.80	56.60	0.24	68.42	0.90	0.35
KEDD 723 Oxide	1.52	21.10	59.90	0.29	80.92	0.87	0.30
KEDD 729 Oxide	1.70	58.20	28.80	0.22	87.06	0.92	0.16
KEDD 722 Oxide	1.91	23.60	64.90	0.22	88.48	0.83	0.21
KEDD 724 Oxide	0.73	32.90	57.50	0.07	90.41	0.85	0.32
KEDD 727 Oxide	1.20	14.20	78.30	0.09	92.50	0.83	0.19
KEDD 728 Oxide	0.71	5.60	70.40	0.17	76.06	0.62	0.16
KEDD 730 Oxide	0.92	27.20	57.60	0.14	84.78	0.92	0.19
KEDD 749 Oxide	0.99	25.30	59.60	0.15	84.85	0.85	0.37
KEDD 750 Oxide	1.15	25.20	64.30	0.12	89.57	1.12	0.20
KEDD 751 Oxide	0.72	41.70	50.00	0.06	91.67	0.93	0.17
KEDD 752 Oxide	0.85	25.90	63.50	0.09	89.41	1.36	0.35
KEDD 763 Oxide	1.66	18.70	55.40	0.43	74.10	0.85	0.20
KERC 581 Oxide	0.71	40.80	45.10	0.10	85.92	0.98	0.67
KERC 649 Oxide	1.78	23.00	61.20	0.28	84.27	1.07	0.91
KERC 660 Oxide	2.30	23.90	60.90	0.35	84.78	0.96	0.13
KERC 746 Oxide	1.13	18.60	54.90	0.30	73.45	0.88	0.13
KEDD 946 Oxide	0.93	10.80	62.40	0.25	73.12	1.07	0.27
KEDD 419 Oxide	0.35	8.60	60.00	0.11	68.57	0.95	0.61
KERC 221 Oxide	0.53	7.50	79.20	0.07	86.79	0.88	0.37
KERC 281 Oxide	0.69	7.20	76.80	0.11	84.06	0.97	0.28
KERC 504 Oxide	0.66	43.90	40.90	0.10	84.85	0.79	0.22
KERC 243 Oxide	1.89	5.30	77.20	0.33	82.54	1.04	0.86
KERC 245 Oxide	0.59	25.40	57.60	0.10	83.05	1.02	1.40
KERC 310 Oxide	0.45	0.00	73.30	0.12	73.33	1.27	1.03

Source : DRA 2013; Coffey Mining Lycopodium 2011

Table 12 : Phase 1V Metallurgical Testwork Summary Results

MATERIAL TYPE	HEAD GRADE (g/t Au)	*PLANT GOLD RECOVERY (%)	**CIL NaCN (kg/t)	LIME (kg/t)
Oxide	1.051	92.80%	0.40	0.52
Transition	1.093	92.10%	0.43	0.60
Fresh	1.084	89.70%	0.54	0.68

Source : DRA 2013; Coffey Mining Lycopodium 2011
*Includes a recovery discount for scale-up and solution gold losses in full scale plant operations
**NaCN consumption excludes cyanide required for leaching gravity and spiral concentrates

As a result of the four phases of testwork the final 2011 process was specified to include Semi Autonomous Grinding mill (SAG) and secondary ball milling, gravity gold recovery from the milling circuit and spiral concentration of the milled product, at a grind dependant on ore type. The spiral concentrate would then be reground and recombined with the spiral tail for thickening and CIL gold recovery to improve cyanide amenability of the ultrafine locked component. The metallurgical circuit comprised the following components:-

- crushing
- milling with gravity recovery;
- spiral concentrators;
- spiral concentrate re=grind;
- CIL on spiral concentrate and tailings; and
- elution and electrowinning.

12.2. Metallurgical Testwork Programme Phase V

NI 43-101 Item 13 (a)

Subsequent to the 2011 PFS, a fifth phase testwork programme was designed to quantify the metallurgical recovery that could be achieved through the combination of:-

- gravity recovery within the milling circuit;
- flotation recovery on the gravity tailings and
- a leach on the flotation concentrates.

The selection of the optimum grind size for the primary grind is crucial to any gold process flow sheet as it sets the first limitation with regards to gold recovery thereafter. During Phase III a grind optimisation trade off study (Table 10), suggested that a target grind of 80% passing (P80) 150µm, was adequate for the circuit which was revised to P80 = 106 µm during phase IV. All subsequent testwork for Phase V was performed at P80 = 75µm as the evidence supported the benefit of a finer target grind size.

Amdel Metallurgical Laboratories (Amdel) in Perth was commissioned to complete the Phase V testwork, based on remaining core from the previous Phase III and IV test work campaigns. In Phase V the following ore types were defined:-

- laterite;
- oxide;
- transitional; and
- fresh.

The 2013 PFS testwork utilised remnants of the initial testwork core samples to reinvestigate the option of gravity/float/CIL processing and to address the issue of grind size in more detail.

12.2.1. 2013 Testwork Scope for Phase V

NI 43-101 Item 13 (a)

The Phase V testwork scope was designed with the objective of completing all metallurgical testwork required to verify the flotation recovery and process design parameters in order to finalise the process flow sheet, size mechanical equipment and determine plant capital and operating costs. The programme included the following:-

- flotation testwork to evaluate plant recovery on oxide, transition and fresh material;
- comminution testwork on oxide, transition and fresh material– grindmill tests;
- gravity recovery testwork and leaching of gravity concentrates on oxide, transition and fresh material;
- CIL test work on flotation concentrates at fine grind on fresh material;
- whole-ore CIL test work on gravity tailings on oxide, transition and fresh material;
- evaluation of flotation and CIL reagent consumptions; and
- continuous cyanide destruction and arsenic removal test work.

12.2.2. Comminution Testwork Results Phase V

NI 43-101 Item 13 (a)

A comminution tests using Sag Mill Comminution test (SMC), Abrasion Index and Bond Ball Work Index (BBWi) characterised breakage function for three ore composites namely; fresh, transition and oxide material was conducted by Lycopodium in Phase IV. Previous specific rates of breakage had been benchmarked with similar deposits but grindability testwork in 2012–2013 was conducted at Amdel, using new grind mill apparatus, to obtain specific peak rates of breakage trends for three composited samples.

Normally a 12% order of magnitude spread for breakage rates is typical for BBWi tests, but the Esaase testwork variation 23% averaged 11.6±2.6kWh/t. (11.8kWh/t “S-curve” averaged) indicating a high degree of hardness variation. Population balance modelling was used to conduct simulations with the aim of establishing the performance at a full-scale plant size in order to achieve a target mesh of grind of 80% minus 75µm, as determined from evaluation of the Phase I – IV testwork. Based on this modelling, the net specific energy requirement was determined to be 16kWh/t when treating fresh material and 13.8kWh/t when treating oxide material.

12.2.3. Gravity Recovery Results Phase V

NI 43-101 Item 13 (a)

Gravity recovery testwork for each material type followed by flotation testing conducted on the gravity tailings. A summary of the gravity recovery results for each ore type and composite samples representative of run-of-mine (RoM) material are presented in Table 13. Composite samples A40, A100 and B40 were most representative of the mine blend.

12.2.4. Batch Flotation Results - 2013

NI 43-101 Item 13 (a)

Flotation tests were carried out on each of the different material types and process engineers from DRA were present to witness the flotation tests. The initial reagent suite used for the flotation tests was based on the conclusions drawn from the Phase IV test work. The results of the testwork are summarised in Table 14.

Table 13 : Gravity Concentration Results on Composite Samples

COMPOSITE	CALCULATED HEAD GRADE (g/t)	COMBINED TAILINGS ASSAY (g/t)	Au RECOVERY (%)	AVERAGE GOLD RECOVERY (%)
Laterite	0.69	0.58	13.4 - 19.7	16.54
Oxide	1.51	1.01	26.7 - 38.3	32.8
Transition	1.51	0.81	27.3-52.7	46.1
Fresh	2.09	1.26	31.2- 48.3	39.9
A40	1.54	1.05	31.90	
A100	1.05	0.93	11.80	
B40	1.92	1.65	14.20	
C40	1.56	0.95	39.20	

Source : DRA 2013

12.2.5. Bulk Flotation Results - 2013

NI 43-101 Item 13 (a)

Bulk flotation test were carried out on each of the different material types, as well as composite samples representing RoM mine blends. The results of the testwork are summarised in Table 15.

The oxide material bulk flotation tests had higher residue grades and lower flotation recovery than the batch flotation tests at a reduced mass pull of 5%. The transition material bulk flotation tests had a residue grade and recovery similar to that obtained in the batch flotation testing at a lower mass pull of 8.8%. The fresh material bulk flotation tests had poor accountability but produced a residue grade similar to that achieved in the fresh batch flotation tests.

The composite RoM bulk flotation tests had poor accountability and two of the four composites were excluded. A single test result indicated that a flotation recovery of 84.4% could be achieved; however the overall recovery for this test was lower than expected due to a low gravity recovery of only 11.8%.

12.2.6. Flotation Concentrate Leach Test Results

NI 43-101 Item 13 (a)

Carbon in Leach (CIL) tests were conducted on the fresh ore flotation concentrates at the “as floated” grind of 45µm and a target grind of 25µm. The fresh ore flotation concentrate was the only ore type tested as this would be regarded as the “worst” case leach conditions. The results of the series of tests at various conditions are presented in Table 16.

The inclusion of pre-oxidation in the tests with “diesel and carbon added at the start of the leach” (Tests 1 and 2 in Table 16) did not indicate any recovery improvement as the residue grades are very similar, namely 0.46g/t and 0.47g/t respectively.

The lowest gold extraction was achieved for Test 4 in which no carbon was added. In Test 3 in which carbon was only added after 4 hours indicated a gold extraction of 84.7% which was significantly lower than the 92.0% - 94.7% extraction achieved for the tests in which carbon was added at the start of the leach.

The CIL test conducted with carbon at t=0 and 100g/t diesel but at a target grind of 80% passing 25µm (Test 6 Table 16), indicated that a residue grade of 0.15g/t could be achieved with an equivalent gold recovery of 96.6%. This indicated a recovery benefit of 3.4% as compared to the “as floated” concentrate grind at 80% passing 45µm.

For the purpose of estimating plant recovery a CIL tails residue grade of 0.15g/t was used, this is the residue achieved at a concentrate grind of 80% passing 25µm in Test 6, for this reason the current flow sheet includes a regrind Vertimill.

Cyanide consumptions in Table 16 will not be indicative of actual plant requirements and in order to provide an estimate of plant operating costs, a cyanide consumption of 8kg/t was used as the estimated consumption after 16 hours of leaching.

Table 14 : Batch Flotation Results on Gravity Tailings

FEED MATERIAL	GRAVITY FEED				FLOTATION FEED			OVERALL RECOVERY (%)	FLOTATION MASS PULL (%)
	GRADE ASSAY (g/t)	CALCULATED GRADE (g/t)	RECOVERY (%)	GRADE TAILINGS (g/t)	CALCULATED GRADE (g/t)	GRADE TAILINGS (g/t)	RECOVERY (%)		
Oxide	1.57	1.40	27.80	1.01	0.55 - 1.14	0.18 - 0.25	75.70 - 79.60	83.70 - 86.70	10.20 - 12.90
Transition	1.06	1.40	41.80	0.81	0.78 - 0.92	0.10 - 0.16	82.00 - 89.00	90.20 - 93.60	10.10 - 17.20
Fresh	1.89	1.84	31.60	1.26	1.04 - 1.29	0.05 - 0.12	91.80 - 96.00	94.40 - 97.50	8.80 - 13.70

Source : DRA 2013

Table 15 : Bulk Flotation Results on Gravity Tailings

FEED MATERIAL	GRAVITY FEED			FLOTATION FEED			OVERALL RECOVERY (%)	FLOTATION MASS PULL (%)
	CALCULATED GRADE (g/t)	RECOVERY (%)	GRADE TAILINGS (g/t)	CALCULATED GRADE (g/t)	GRADE TAILINGS (g/t)	RECOVERY (%)		
Oxide	1.68	38.3	1.04	0.99 - 1.04	0.36 - 0.37	64.50 - 67.30	79.10 - 79.70	5.10 - 5.40
Transition	1.5	52.7	0.71	0.71	0.16	79.40	90.30	8.80
Fresh	1.26	48.3	0.65	1.37	0.15	90.00	89.10	8.80
RoM Composite	1.05 - 1.56	11.80 - 39.20	0.93 - 0.95	0.84 - 0.89	0.14 - 0.18	81.40 - 84.40	87.50 - 89.30	6.50 - 7.80

Source : DRA 2013

Table 16 : Fresh Material Concentrate Leach Test Summary

Table 16TEST	TEST CONDITIONS							RESULTS				
	HIGH SHEAR PREOXIDATION (hrs)	CIL LEACH (hrs)	CARBON (g/L)	CARBON ADDED @ hrs	LEAD NITRATE (g/t)	DIESEL (g/t)	80% PASSING (µm)	CIL CALCULATED HEAD GRADE (Au g/t)	RESIDUE Au GRADE (g/t)	CIL EXTRACTION (%)	NaCN (kg/t)	LIME (kg/t)
1	0	24	20	t=0hrs	50	100	45	6.72	0.46	93.15	12.85	0.44
2	4	24	20	t=0hrs	50	0	45	5.86	0.47	91.98	9.58	3.37
3	4	24	20	t=4hrs	50	0	45	5.22	0.8	84.68	11.72	5.79
4	4	24	0	none	50	0	45	5.32	1.18	77.8	8.39	4.19
5	4	24	20	t=0hrs	50	100	45	6.36	0.34	94.65	13.67	5.94
6	4	24	20	t=0hrs	50	100	25	4.35	0.14	96.55	15.7	5.88

Source : DRA 2013

Further testing is required in order to improve definition of the expected recovery, reagent consumption and design parameters for the concentrate CIL circuit. The following needs to be addressed in the next phase of testing:

- confirmation of CIL recovery at a target grinds of 80% passing 25µm, which the inclusion of gravity concentration on the flotation concentrate;
- cyanide optimisation as the current phase of testing indicated excess cyanide addition; and
- determination of the expected lime consumption as Phase V pH control was not optimised and this resulted in excessive lime consumption.

12.2.7. Whole-ore Leach Test Results

NI 43-101 Item 13 (a)

The whole-ore tests were aimed at determining the expected CIL recovery for each material type at a grind of 80% passing 75µm, to assess the benefits of high shear pre-oxidation on recovery and establish if preg-robbing was present as indicated in the Phase III testing. These tests were specifically aimed at comparing the overall gravity-CIL recovery to that obtained for the gravity-flotation-CIL tests. The test conditions were evaluated for a 24 hour residence time at 40% solids and a target leach pH of 10.0 - 10.5 for all tests. Tests 1- 4 were conducted without high shear pre-oxidation and tests 5- 8 were conducted with the inclusion of high shear pre-oxidation. The results for the three material type are presented in Table 17.

12.2.7.1. Oxide Material Whole-ore Leach

NI 43-101 Item 13 (a)

The oxide material CIL test results are presented in Table 17. The test results for the pre-oxidised material, Tests 5 to 8 in Table 17, showed that without carbon (Test 7) the CIL residue grade was 0.106g/t with a CIL gold recovery of 85.6%.

The oxide ore CIL tests with high shear pre-oxidation showed no increase in gold recovery, but more than doubled the sodium cyanide and lime consumption. The overall combined gravity and CIL recovery for the oxide material ranges 79.8% - 92.6%.

12.2.7.2. Transition Material Whole-ore Leach

NI 43-101 Item 13 (a)

The test results for the transition material are presented in Table 18 and overall recovery for combined gravity and CIL for the transition material was found to be in the range 86.2% - 93.0%. The CIL tests conducted with high shear pre-oxidation indicated no improved overall recovery at the expense of increased reagent consumption. The leach tests conducted with no carbon addition had the highest residue grades and lowest recovery, this is an indication of preg-robbing caused by the presence of carbonaceous or shale material. This was further indicated by the leach tests with carbon addition after 4hr having higher residue grades than the CIL tests with carbon addition at the start of the leach. This effect was noticed irrespective of diesel addition.

12.2.7.3. Fresh Material Whole-ore Leach

NI 43-101 Item 13 (a)

The test results for the fresh material are presented in Table 19 and the overall recovery for combined gravity and CIL ranges 82.1% - 90.0%. The CIL tests conducted with high shear pre-oxidation indicated no improved overall recovery at the expense of increased reagent consumption. The leach tests conducted with no carbon addition had the highest residue grades and lowest recovery, this is an indication of preg-robbing caused by the presence of carbonaceous or shale material. This was further indicated by the leach tests with carbon addition after 4h having higher residue grades that the CIL tests with carbon addition at the start of the leach. This effect was noted irrespective of diesel addition.

Table 17 : Oxide Material Whole-ore CIL Leach Test Summary Results

TEST	TEST CONDITIONS							RESULTS						
	HIGH SHEAR PREOXIDATION (hrs)	CIL LEACH (hrs)	CARBON (g/L)	CARBON ADDED @ hrs	LEAD NITRATE (g/t)	DIESEL (g/t)	80% PASSING (µm)	GRAVITY TAILINGS (g/t)	CIL CALCULATED HEAD GRADE (Au g/t)	CIL RESIDUE Au GRADE (g/t)	CIL RECOVERY (%)	TOTAL RECOVERY (%)	NaCN (kg/t)	LIME (kg/t)
1	no	24	20	t=0hrs	50	50	75	1.01	0.77	0.118	84.75	91.57	1.41	0.28
2	no	24	20	t=4hrs	50	50	75	1.01	0.80	0.104	86.95	92.57	1.45	0.30
3	no	24	none	none	50	50	75	1.01	0.87	0.143	83.51	89.78	1.27	0.32
4	no	24	20	t=0hrs	50	100	75	1.01	0.97	0.283	70.85	79.77	1.36	0.29
5	4	24	20	t=0hrs	50	50	75	0.89	0.85	0.190	77.75	84.61	2.46	1.24
6	8	20	20	t=4hrs	50	50	75	0.89	0.77	0.134	82.55	89.14	2.76	1.08
7	4	24	none	none	50	50	75	0.89	0.73	0.106	85.56	91.41	2.14	1.22
8	4	24	20	t=0hrs	50	100	75	0.89	0.75	0.097	87.05	92.14	2.54	1.26

Source : DRA 2013

Table 18 : Transition Material Whole-ore CIL Leach Test Summary Results

TEST	TEST CONDITIONS							RESULTS							
	HIGH SHEAR PREOXIDATION (hrs)	CIL LEACH (hrs)	CARBON (g/L)	CARBON ADDED @ hrs	LEAD NITRATE (g/t)	DIESEL (g/t)	80% PASSING (µm)	GRAVITY FEED GRADE (g/t)	GRAVITY TAILINGS (g/t)	CIL CALCULATED HEAD GRADE (Au g/t)	CIL RESIDUE Au GRADE (g/t)	CIL RECOVERY (%)	TOTAL RECOVERY (%)	NaCN (kg/t)	LIME (kg/t)
1	no	24	20	t=0hrs	50	50	75	1.40	0.81	0.95	0.098	89.67	92.98	1.60	0.23
2	no	24	20	t=4hrs	50	50	75	1.40	0.81	0.83	0.173	79.23	87.61	1.81	0.21
3	no	24	none	none	50	50	75	1.40	0.81	0.87	0.522	40.00	62.61	1.46	0.19
4	no	24	20	t=0hrs	50	100	75	1.40	0.81	0.84	0.127	84.95	90.90	1.84	0.15
5	4	24	20	t=0hrs	50	50	75	1.10	0.64	0.61	0.086	85.88	92.21	2.60	0.62
6	8	20	20	t=4hrs	50	50	75	1.10	0.64	0.65	0.015	76.42	86.05	3.03	1.27
7	4	24	none	none	50	50	75	1.10	0.64	0.56	0.280	49.73	74.63	2.22	0.68
8	4	24	20	t=0hrs	50	100	75	1.10	0.64	0.62	0.090	85.55	91.85	2.58	0.67

Source : DRA 2013

Table 19 : Fresh Material Whole-ore CIL Leach Test Summary Results

TEST	TEST CONDITIONS							RESULTS							
	HIGH SHEAR PREOXIDATION (hrs)	CIL LEACH (hrs)	CARBON (g/L)	CARBON ADDED @ hrs	LEAD NITRATE (g/t)	DIESEL (g/t)	80% PASSING (µm)	GRAVITY FEED GRADE (g/t)	GRAVITY TAILINGS (g/t)	CIL CALCULATED HEAD GRADE (Au g/t)	CIL RESIDUE Au GRADE (g/t)	CIL RECOVER Y (%)	TOTAL RECOVERY (%)	NaCN (kg/t)	LIME (kg/t)
1	no	24	20	t=0hrs	50	50	75	1.84	1.26	1.11	0.111	89.970	93.97	1.36	0.12
2	no	24	20	t=4hrs	50	50	75	1.84	1.26	1.13	0.184	83.760	90.00	1.44	0.19
3	no	24	none	none	50	50	75	1.84	1.26	1.09	0.659	39.650	64.19	1.44	0.19
4	no	24	20	t=0hrs	50	100	75	1.84	1.26	0.97	0.117	87.960	93.64	1.48	0.14
5	4	24	20	t=0hrs	50	50	75	1.17	0.8	0.79	0.118	85.100	89.88	2.58	0.82
6	8	20	20	t=4hrs	50	50	75	1.17	0.8	0.70	0.353	49.210	69.72	2.61	0.81
7	4	24	none	none	50	50	75	1.17	0.8	0.58	0.282	51.560	75.81	2.39	0.60
8	4	24	20	t=0hrs	50	100	75	1.17	0.8	0.72	0.128	82.120	89.02	2.44	0.88

Source : DRA 2013

12.3. Process Plant Recovery Estimate

NI 43-101 Item 13 (b)

The plant recovery estimate for Esaase Project aims to provide an estimate of recovery for full scale operations in two combinations namely:-

- gravity recovery within the milling circuit; flotation on the gravity circuit tailings; and a CIL leach on the flotation concentrate (Flotation Included);
- as compared to a conventional combination of gravity recovery within the milling circuit and conventional CIL on the gravity tailings without flotation (Flotation Excluded).

The recovery estimate for the gravity-flotation-CIL (Flotation Included) process has been based on the flotation tails and CIL tails residue grades that were achieved in the Phase V test work campaign conducted at Amdel. The recovery estimates for a conventional gravity-CIL (Flotation Excluded) process has been based on the CIL residue grades that were achieved in the Phase III and V test work campaigns at Amdel. The phase IV test work results were not used because this testing was conducted on a CIL circuit that included enhanced gravity recovery and leaching of gravity concentrates and tailings. This would be equivalent to tests in which the flotation tailings stream was also leached and for this reason, the recovery and reagent consumption achieved is not considered representative of that for a conventional CIL circuit with conventional gravity concentration in the milling circuit.

In order to provide an estimate of the expected recovery for full scale continuous plant operations, the bench scale laboratory recoveries must be discounted in order to account for process inefficiency and solution gold losses due to:-

- CIL carbon fines losses to tailings;
- CIL solution gold losses;
- scale-up of bench scale flotation to plant scale; and
- scale-up of bench scale CIL to plant scale.

The resultant recovery estimates are summarised in Table 20, for both Flotation Included and Flotation Excluded process flows, together with the applicable estimation criteria and the appropriate discount factors:-

Table 20 : Estimated LoM Plant Recoveries for Flotation Included and Flotation Excluded Process Flows

RoM MATERIAL	FLOTATION INCLUDED PROCESS		FLOTATION EXCLUDED PROCESS		
	FLOTATION RESIDUE GRADE (g/t)	RECOVERY (%)	CIL RESIDUE GRADE (g/t)	RECOVERY at 150µm (%)	RECOVERY at 75µm (%)
Laterite	estimated same as oxide	84.54	estimated same as oxide	84.80	92.80
Oxide	0.21	84.67	0.143	84.80	92.90
Transition	0.14	91.20	0.100	84.90	93.70
Fresh	0.074	94.23	0.119	88.40	91.50
Recovery discount		1.11		1.63	1.65
LoM		**90.06**		**85.30**	**90.55**
Estimation Criteria					
Mass Pull 9%					
Regrind to get 80% passing 20µm			80% Passing 150µm and 75µm		
Target CIL residue grade for all material types 0.15(g/t Au)					
Discount Factors					
CIL concentrate carbon fines losses @10g/t @ 50g/t Au			CIL concentrate carbon fines losses @10g/t @ 50g/t Au		
Solution gold losses based on 42% solids in CIL tailings and 0.01g/L Au in solution			Solution gold losses based on 42% solids in CIL tailings and 0.01g/L Au in solution		
Scale-up factor of 0.85% after commissioning and optimisation			Scale-up factor of 0.85% after commissioning and optimisation		

Source : DRA 2013

12.4. Conclusions for Mineral Processing and Metallurgical Testwork

NI 43-101 Item 13 (c), (d)

The combined historical and current metallurgical testwork results have been summarised and used to estimate LoM recoveries for two process flows, namely a combined gravity-flotation-CIL process and a conventional gravity-CIL circuit.

The Flotation Excluded gravity-CIL process increases plant capital and operating cost (see Section 20) as a result of the larger tonnages that need to be treated in the CIL and cyanide destruction circuits. Furthermore, the large CIL circuit has increased capital cost and environmental risk associated with the TSF facility.

The final process recommended by DRA comprises run-of-mine ball milling (RoMB), with primary gravity recovery from the mill circuit, flotation of the milled product, with regrind and secondary gravity recovery of the float concentrate ahead of CIL gold recovery of the reground float concentrate. Such a design has the benefit that the flotation tailings comprising approximately 90% of the feed are benign and can be disposed to a clay lined TSF, whilst the CIL and downstream plants can be downsized accordingly.

13. Mineral Resource Statement

NI 43-101 Item 14

The mineral resource estimates were compiled by Minxcon (2012) 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” and in accordance with the Rules and Policies of the National Instrument 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1 and Companion Policy 43-101CP.

The de-surveyed drillhole data was composited within Datamine™ on a 1m composite length. A total of 233,503 composites were used in the statistical analysis and resource estimation. Forty-six mineralised domains (reefs), including the waste, form part of the statistical analysis and resource estimation. Minxcon is satisfied that the Mineral Resource estimation globally reflects the deposit 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 drillhole information and geological information.

13.1. Assumptions. Parameters and Estimation Methodology

NI 43-101 Item 14 (a)

Grade estimation was undertaken using Ordinary Kriging and the estimation approach was considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style and geometry of mineralisation (Minxcon 2012) Higher grade veins occur within a lower grade background and the individual mineralisation boundaries of these high grade veins can be difficult to define. Indicator Kriging was therefore chosen to delineate the areas with continuous grades and was used later as a start model to adequately define the mineralisation.

Based on grade information and geological logging and observations, oxidation, transition and fresh zones, mineralised domain boundaries have been interpreted and formulated into wireframes to permit the resource estimation for the Esaase Project. The interpretation and wireframe models were developed using Datamine™ geological modelling software package. Minxcon determined that a 5mx5mx3m block size provided the best results for delineating the mineralised zones using the Indicator Kriging methodology and a 10mx10mx3m block size provided the best results for geo-statistical estimation and hence the estimation was conducted on a 10mx10mx3m (X, Y & Z respectively) block model size.

13.2. Geological and Mineralisation Domains

For the purpose of the mineral resource estimation, 11 main and 2 secondary mineralised domains were interpreted and were 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 located outside these main mineralised zones. The waste zone was assigned a default value of 0.005 g/t gold. The main domains are depicted in Figure 8.

To delineate the mineralisation inside the previously defined wireframes, Indicator Kriging was implemented using a gold cut-off grade of 0.3g/t Au. Samples with gold grades below 0.3g/t were assigned the Kriging Indicator of 0 (AUIND = 0) and samples with gold grades above 0.3g/t Au were assigned the Kriging Indicator of 1 (AUIND = 1). A probability of 0.3 was selected as the best representative to delineate the mineralisation.

13.3. Treatment of the Alteration Zones

Sample composites were coded according to the weathering profile which was modelled from drillhole data and comprises strongly weathered saprolite, moderately weathered saprolite, transition material and fresh units. In general, the weathering surfaces broadly parallel the topographical profile, although weathering tends to be deeper within zones of mineralisation, and tends to parallel the footwall to the mineralisation approaching the surface. In some sections, the intermixing of the weathering types can be quite complicated. An example cross-section showing the distribution of weathering types and the interpreted position of the top of fresh rock is presented in Figure 8..

13.4. Statistical Analysis of the Raw Data

Classical statistics of each of the he individual reefs was undertaken in order to establish the extent of the homogeneity within the reef, the global mean and outlier analysis. The classical statistical analysis results are presented in Table 21.

Table 21 : Classical Statistical Analysis per Reef

REEF	GRADE			VARIANCE	STANDARD DEVIATION	COEFFICIENT OF VARIATION
	Min (g/t Au)	Min (g/t Au)	Min (g/t Au)			
LR1	0.001	20	1.110	3.708	1.926	1.735
LR2	0.001	40	1.157	11.194	3.346	2.891
LR3	0.001	10	1.109	4.729	2.175	1.961
LR4	0.001	20	1.023	6.027	2.455	2.399
LR5	0.001	3	0.568	0.387	0.622	1.095
LR6	0.005	2	0.490	0.288	0.537	1.095
LR7	0.001	1.5	0.450	0.201	0.449	0.998
LR20	0.001	20	1.204	8.169	2.858	2.374
LR30	0.001	5	0.774	1.610	1.269	1.639
LR40	0.001	1	0.503	0.153	0.391	0.777
LRB	0.040	3	0.936	0.490	0.700	0.748
OR1	0.001	80	1.452	20.951	4.577	3.152
OR2	0.005	30	1.008	5.950	2.439	2.420
OR3	0.005	40	1.680	17.496	4.183	2.490
OR4	0.005	30	1.097	5.593	2.365	2.156
OR5	0.001	10	0.797	1.652	1.285	1.612
OR6	0.001	25	1.378	9.891	3.145	2.282
OR7	0.026	2	0.469	0.230	0.480	1.021
OR20	0.001	40	1.153	9.252	3.042	2.639
OR30	0.005	12	1.475	6.827	2.613	1.771
OR40	0.010	15	1.190	5.929	2.435	2.046
ORB	0.020	1	0.477	0.118	0.343	0.719
TR1	0.001	80	1.680	31.967	5.654	3.366
TR2	0.001	30	1.116	5.243	2.290	2.052
TR3	0.005	69	2.002	41.453	6.438	3.216
TR4	0.005	42	0.851	7.310	2.704	3.176
TR5	0.010	32	1.020	8.518	2.919	2.861
TR6	0.005	112	4.172	322.942	17.971	4.307
TR7	0.020	1	0.271	0.075	0.273	1.008
TR20	0.005	40	1.189	8.383	2.895	2.434
TR30	0.005	37	1.350	15.481	3.935	2.915
TR40	0.005	15	1.493	9.856	3.139	2.103
TRB	0.110	3	1.092	1.166	1.080	0.989
FR1	0.001	80	1.263	12.459	3.530	2.795
FR2	0.002	50	1.055	7.677	2.771	2.626
FR3	0.005	60	1.225	11.320	3.364	2.746
FR4	0.002	30	0.959	5.462	2.337	2.438

Source : Minxcon 2012

Outlier analysis was undertaken using histograms and probability plots and was determined for both the variography and the kriging stages. For variography the outliers were 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 coefficients of variation are moderately high but typical for these types of deposits. The kriging capping and the variogram top-cut values applied to the data are summarised in Table 22.

Table 22 : Variogram Top-cut and Kriging capping per Reef

VARIOGRAM TOP-CUT VALUES				KRIGING CAPPING VALUES			
Reef	Top Cut (g/t Au)	Reef	Top Cut (g/t Au)	Reef	Top Cut (g/t Au)	Reef	Top Cut (g/t Au)
FR1	80	OR20	40	FR1	60	OR20	20
FR2	50	OR3	40	FR2	30	OR3	20
FR20	50	OR30	12	FR20	30	OR30	10
FR3	60	OR4	30	FR3	40	OR4	20
FR30	20	OR40	15	FR30	15	OR40	5
FR4	30	OR5	10	FR4	20	OR5	6
FR40	20	OR6	25	FR40	10	OR6	10
FR5	15	OR7	2	FR5	10	OR7	1
FR6	12	ORB	1	FR6	8	ORB	1
FR7	15	TR1	80	FR7	10	TR1	60
FRB	4	TR2	30	FRB	3	TR2	20
LR1	20	TR20	15	LR1	15	TR20	10
LR2	40	TR3	40	LR2	20	TR3	20
LR20	20	TR30	20	LR20	15	TR30	15
LR3	10	TR4	20	LR3	4	TR4	10
LR30	5	TR40	10	LR30	3	TR40	4
LR4	20	TR5	15	LR4	10	TR5	5
LR40	1	TR6	20	LR40	1	TR6	10
LR5	3	TR7	1	LR5	2	TR7	1
LR6	2	TRB	3	LR6	2	TRB	3
LR7	2	W	60	LR7	1	W	40
LRB	3	WE	60	LRB	3	WE	40
OR1	80	-	-	OR1	60	-	-
OR2	30	-	-	OR2	15	-	-

Source : Minxcon 2012

13.5. Variography

Downhole variograms were constructed to permit the determination of the nugget value, as well as the vertical or across deposit search range for the kriging estimation. In general, it was established that the average vertical range for the domains and grade was 9m. Point experimental variograms were generated and modelled for each domain. The parameters of the modelled variograms for the Asanko Esaase Project are summarised in Table 12.

13.6. Estimation Methodology

Both simple and ordinary kriging estimation methodologies were undertaken. Simple kriging includes the global mean grade as a constituent of the kriging equation and was used primarily in areas which are not well informed by data. The mean grade of the population was included as part of the estimate and for this exercise ordinary kriging was used.

Table 23 : Variogram Parameters per Reef

REEF	PARAMETER	SILL	NUGGET (%)	SILL (1%)	RANGE 1 (m)	RANGE 2 (m)	RANGE 3 (m)	SILL (2%)	RANGE 1 (m)	RANGE 2 (m)	RANGE 3 (m)
LR1	AU	2.18	40.5	87.12	38.97	64.95	9	100	50.08	67.78	9
LR2	AU	2.71	34.38	85.88	37.5	62.5	9	100	38.54	64.37	9
LR3	AU	0.47	30.32	76.88	31.47	42.88	9	100	69.61	75	9
LR4	AU	2.67	31.74	55	38.81	46.94	9	100	63.1	92.17	9
LR5	AU	2.6	32.76	55	53.58	71.46	9	100	75	78.28	9
LR6	AU	1.72	37.95	55	38.95	111.44	9	100	84.64	95.53	9
LR7	AU	0.04	32.75	55	13.21	13.21	9	100	9.15	9.15	9
LR20	AU	1.94	37.23	87.83	62.22	65.19	9	100	101.86	143.56	9

REEF	PARAMETER	SILL	NUGGET (%)	SILL (1%)	RANGE 1 (m)	RANGE 2 (m)	RANGE 3 (m)	SILL (2%)	RANGE 1 (m)	RANGE 2 (m)	RANGE 3 (m)
LR30	AU	0.46	31.79	86.02	38.82	187.96	9	100	77.64	139.49	9
LR40	AU	0.06	31.26	99.9	36.06	36.06	9	100	43.05	43.05	9
LRB	AU	0.91	44.66	76.96	28.12	65.27	9	100	70.28	83.55	9
OR1	AU	2.22	40.07	79.17	56.76	68.05	9	100	29.01	77.22	9
OR2	AU	1.78	49.5	87.01	38.98	55.41	9	100	41.25	49.51	9
OR3	AU	1.81	44.58	85.65	23.94	47.78	9	100	39.54	49.25	9
OR4	AU	1.82	35.89	81.58	38.97	55.51	9	100	20.73	91.76	9
OR5	AU	1.82	39.67	55	31.13	52.41	9	100	67.77	83.73	9
OR6	AU	2.38	30.27	55	69.62	73.15	9	100	94.03	69.04	9
OR7	AU	0.08	32.73	76.56	26.99	36.99	9	100	41.48	45.1	9
OR20	AU	2.16	38.14	55	39.96	47.06	9	100	30.88	58.69	9
OR30	AU	4.24	32.45	76.88	85.1	93.27	9	100	81.92	107.57	9
OR40	AU	1.97	45.05	55	62.53	62.53	9	100	87.68	87.68	9
ORB	AU	1.55	61.67	89.54	5.6	5.6	9	100	6.4	6.4	9
TR1	AU	2.14	39.97	89.26	38.99	64.99	9	100	52.47	129.98	9
TR2	AU	2.17	37.29	76.84	47.6	43.13	9	100	32.08	79.36	9
TR3	AU	1.99	50.79	76.14	50.12	83.53	9	100	63.42	98.29	9
TR4	AU	1.45	61.67	93.88	45	51.36	9	100	90	150	9
TR5	AU	1.6	48.16	84.38	95.12	124.81	9	100	200.13	84.98	9
TR6	AU	2.72	59.23	59.82	29.53	48.65	9	100	97.21	162.01	9
TR7	AU	1.43	43.74	55	6.49	10.81	9	100	12.97	15.91	9
TR20	AU	1.92	45.86	90.81	42.76	84.55	9	100	17.96	63.3	9
TR30	AU	2.87	30.79	55	76.94	131.79	9	100	105.85	145.87	9
TR40	AU	1.85	45.71	55	77.01	101.3	9	100	154.02	193.97	9
TRB	AU	0.42	56.33	87.04	360.62	360.62	9	100	295.36	295.36	9
FR1	AU	2.22	45.8	85.94	54.87	9.49	9	100	159.44	130.5	9
FR2	AU	2.22	40.91	74.91	71.03	64.96	9	100	89.32	53.33	9
FR3	AU	6.17	32.86	80.5	59.43	31.85	9	100	86.5	76.89	9
FR4	AU	1.98	46.12	55	39.07	39.27	9	100	63.22	83.62	9
FR5	AU	1.8	31.13	75.59	59.33	64.93	9	100	126.8	129.88	9
FR6	AU	2.59	34.36	91.89	26.19	42.2	9	100	34.19	29.43	9
FR7	AU	2.29	38.94	75.92	29.58	37.66	9	100	25.13	69.79	9
FR20	AU	2.49	34.43	75.4	53.17	27.52	9	100	90.32	21.92	9
FR30	AU	3.19	49.15	55	42.19	66.03	9	100	61.03	79.27	9
FR40	AU	2.56	39.02	84.88	77	52.54	9	100	61.47	77.7	9
FRB	AU	1.83	30.75	75.25	51.35	52.82	9	100	131.75	126.16	9
W	AU	2.2	43.9	97.45	78.04	126.72	9	100	107.54	125.24	9
WE	AU	2.61	36.89	55	162.5	162.5	9	100	250.74	250.74	9

Source : Minxcon 2012

The global means for each domain and reef were determined through the analysis of the statistics of various regularised data set dimensions. Minxcon de-clustered the data and reviewed the means and average variances of each de-clustered data set in order to determine the most representative global mean for each domain as summarised in Table 24. The de-clustered block size used for the de-clustering was 80mx80mx250m.

Table 24 : Global Mean Gold Grade per Reef

REEF	GLOBAL MEAN (g/t Au)	REEF	GLOBAL MEAN (g/t Au)
FR1	1.26	OR2	1.01
FR2	1.06	OR20	1.15
FR20	1.44	OR3	1.68
FR3	1.23	OR30	1.48
FR30	1.25	OR4	1.10
FR4	0.96	OR40	1.19
FR40	1.10	OR5	0.80
FR5	0.82	OR6	1.38
FR6	1.22	OR7	0.47
FR7	0.94	ORB	0.48
FRB	0.76	TR1	1.68
LR1	1.11	TR2	1.12
LR2	1.16	TR20	1.19
LR20	1.20	TR3	2.00
LR3	1.11	TR30	1.35
LR30	0.77	TR4	0.85
LR4	1.02	TR40	1.49
LR40	0.50	TR5	1.02
LR5	0.57	TR6	4.17

REEF	GLOBAL MEAN (g/t Au)	REEF	GLOBAL MEAN (g/t Au)
LR6	0.49	TR7	0.27
LR7	0.45	TRB	1.09
LRB	0.94	W	0.12
OR1	1.45	WE	0.68

Source : Minxcon 2012

13.7. Classification Criteria

The Mineral Resource classification is a function of the confidence of the data from drilling, sampling, and analytical programmes and their contribution to the geological understanding and geostatistical relationships. The grade estimates have been classified as Measured, Indicated and Inferred in accordance with NI 43-101 guidelines based on the confidence levels of the key criteria that were considered during the resource estimation. The key criteria included both confidence in the quality of the data and geostatistical considerations. The confidence criteria for drilling, sampling and geological data are tabulated below:-

Table 25 : Confidence Levels for Key Input Data

DATA SOURCES	COMMENTS	LEVEL of 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 have been identified. Recent improvements have been 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 40mE x 40mN drill spacing with a small area drilled at 20mE x 20mN. Other areas more broadly spaced to approximately 80mN spaced lines (40mE spacing) reflecting a lower confidence.	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

Source : Minxcon 2012

The geostatistical criteria used in the Mineral Resource classification are summarised below:-

Table 26 : Mineral Resource Classification Criteria

CLASSIFICATION CRITERIA	MEASURED	INDICATED	INFERRED
Number of samples used	Measured: at least 4 drill holes within variogram range and a minimum of 16 one meter composited samples.	Indicated: at least 2 drill holes within variogram range and a minimum of 8 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%–40% from mean (80%–60% confidence).	Inferred: more than 40% (less than 60% confidence).
Kriging efficiency:	Measured: more than 40%	Indicated: 10%–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%–20%;	Inferred: more than 20%.

Source : Minxcon 2012

Wireframe models were constructed to delineate Measured, Indicated and Inferred Resources for each domain. Minxcon used the Coffey Mining mineral resource category boundaries as a template to determine the new resource category areas. The resource areas determined by Minxcon are similar to those defined by Coffey Mining with an estimated 5% difference between the Coffey 2011 resource category boundaries and the Minxcon 2012 resource boundaries.

The resultant mineral resource classification model is presented in Figure 9, and the associated grade distribution is shown in Figure 10.

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

13.8. Mineral Resource Estimate

NI 43-101 Item 14 (b)

The Mineral Resource estimate was based on two groups of resources, namely the Main Zone which refers all the material inside the wire frames, and the Secondary Zone which refers to the economic mineralisation material outside the wire frames. A summary of the estimated NI 43-101 compliant Mineral Resources for the Esaase Project at various cut-off grades is provided in Table 27. The estimate includes all the main mineralised geological domains including satellite deposits as shown in Figure 9 and the gold grade distribution in plan and cross-section is presented in Figure 10.

Inferred Mineral Resources have a significant degree of uncertainty as to whether they can be mined economically and it cannot be assumed that all or any part of the Inferred Resource will be upgraded to a higher confidence category. In compliance with NI 43-101 Section 3.4(e) it is noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. However, the Esaase Mineral Resources have undergone valid modification in this 2013 PFS and Mineral Reserves that do have demonstrated economic viability have been estimated (see Section 14).

13.9. Previous Mineral Resource Reconciliation

A comparison of the Coffey Mining 2011 and Minxcon 2012 Mineral Resource estimates is presented in Table 28.

The difference in tonnage and contained gold between the Coffey Mining 2011 and Minxcon 2012 estimates as a whole, is the result of Minxcon incorporating mineralised material outside the 2011 wireframes, which has been referred to by Minxcon as the ‘secondary material’.

The 2011 estimate was based in total on 1,187 drillholes with 213,905 samples which increased to 233,503 samples in the Minxcon 2012 estimate. In addition the changes in classification to the Measured category are a result of the classification criteria used by Minxcon which not only took cognisance of the drillhole spacing as in the 2011 estimate, but also such factors as also geological relationships, number of samples used for a block estimate, kriging efficiency, lower confidence limit, regression slope and variogram ranges which represent grade continuity. These parameters are all well within the confidence required for a Measured Mineral Resource category. Although there is a marked increase in the measured resource category, the tonnes in this category represent only 19% of the total tonnes.
The tonnage for Inferred is lower for the Minxcon 2012 estimate as some of this material was allocated to the Indicated category, due to the amount of new data acquired which improved delineation of the economic areas.

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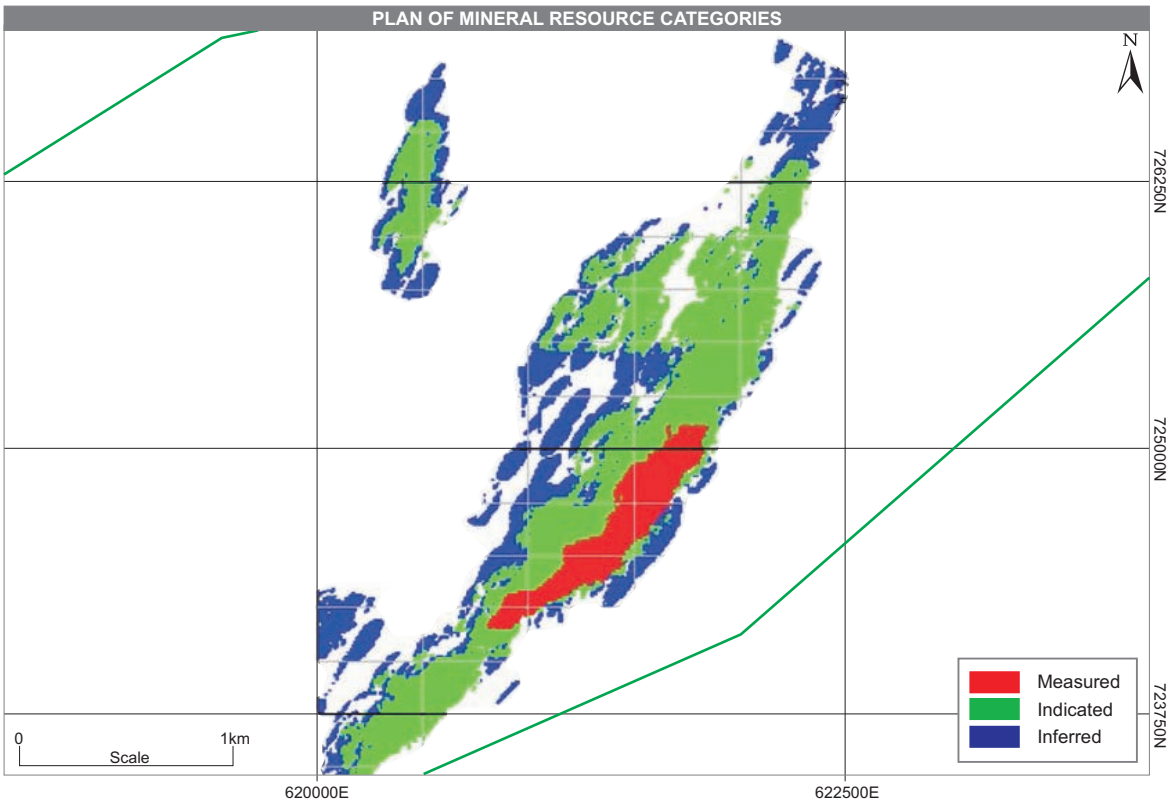
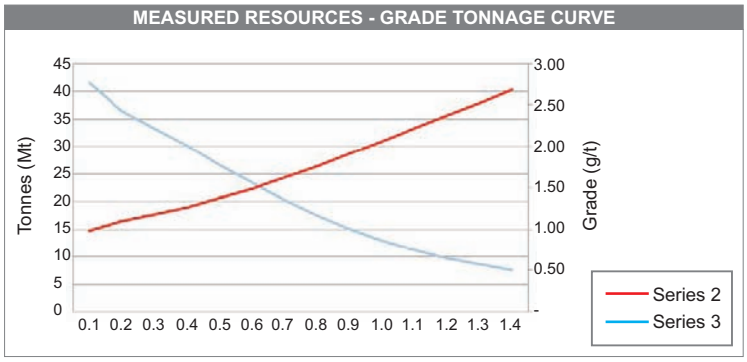
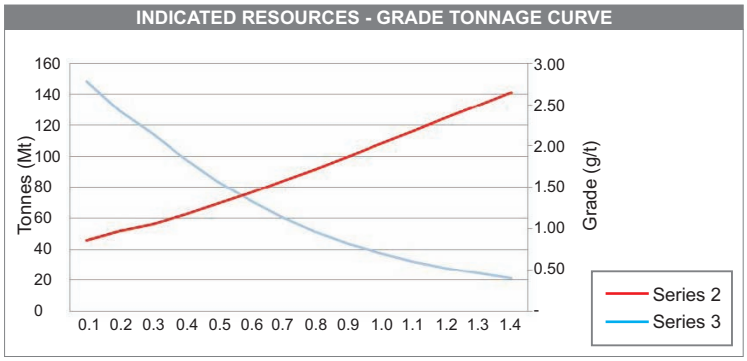
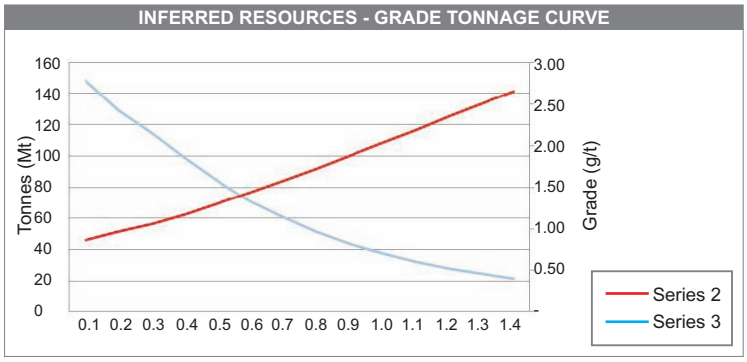
MINERAL RESOURCE CLASSIFICATION AND GRADE TONNAGE CURVES









Source: Minxcon 2012

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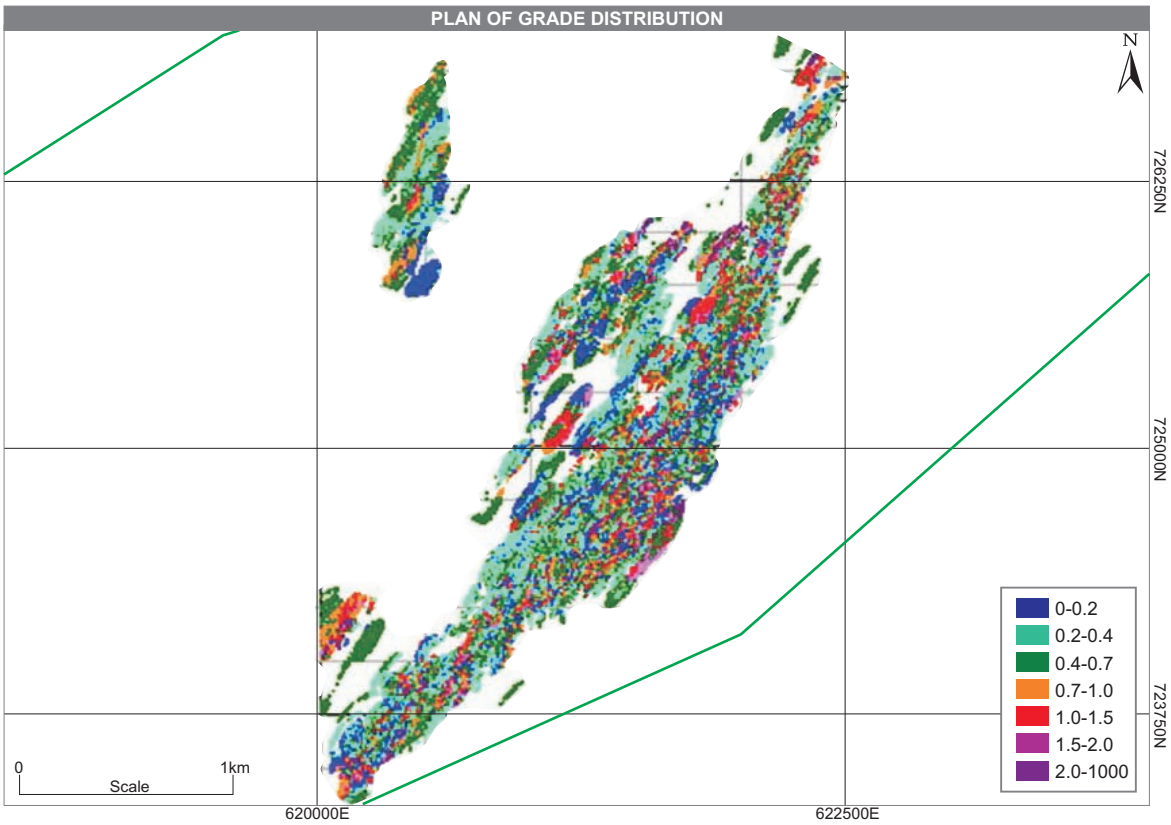
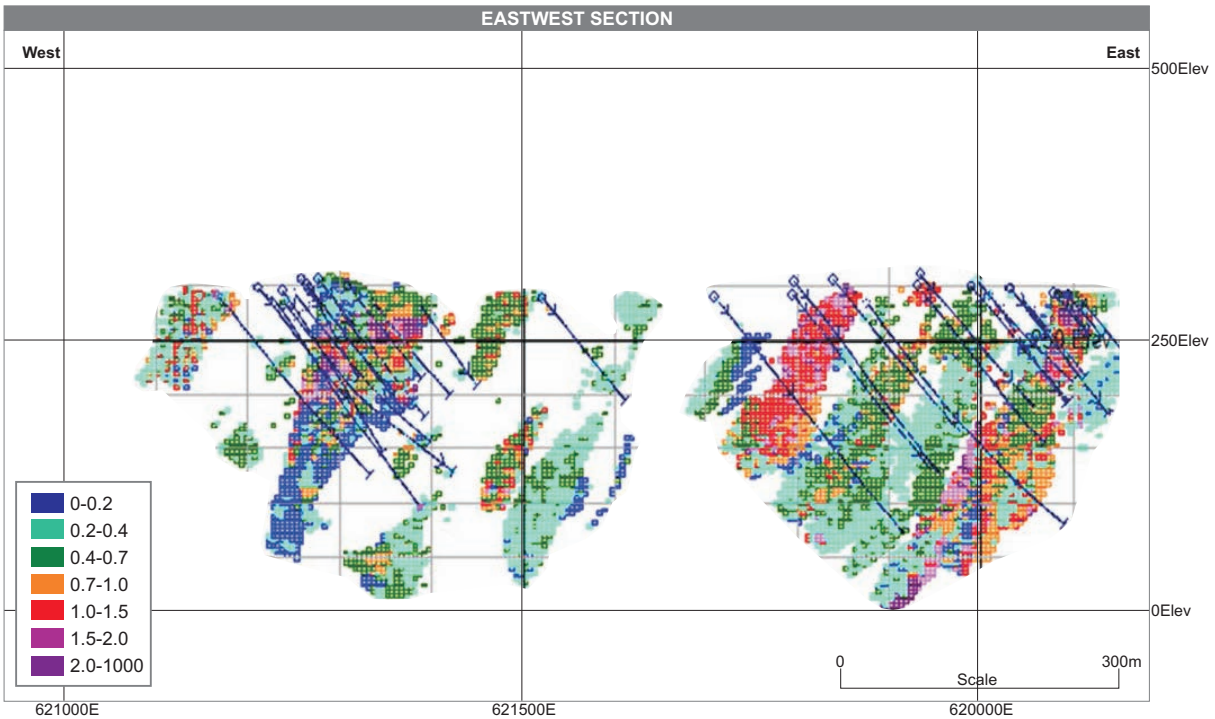
Asanko Gold

GRADE DISTRIBUTION





Source: Minxcon 2012

Table 27 : NI 43-101 Mineral Resource Estimate at Various Cut-off Grades - Sept 2012

CATEGORY	ZONES	TONNAGE (Mt)	GRADE (g/t Au)	CONTAINED GOLD (oz)
Cut-off Grade of 0.6g/t Au				
Measured (M)	Main	21.86	1.52	1.07
	Secondary	1.52	1.11	0,05
	Total Measured	23.38	1.49	1.12
Indicated (I)	Main	57.30	1.48	2.72
	Secondary	13.95	1.26	0.56
	Total Indicated	71.25	1.44	3.28
Total Measured + Indicated		**94.63**	**1.45**	**4.40**
Inferred	Main	13.83	1.46	0.65
	Secondary	19.76	1.35	0.86
	Total inferred	33.59	1.4	1.51
Cut-off Grade of 0.8g/t Au				
Measured	Main	16.50	1.78	0.95
	Secondary	1.02	1.32	0.04
	Total Measured	17.52	1.75	0.99
Indicated	Main	42.21	1.76	2.39
	Secondary	9.19	1.55	0.46
	Total Indicated	51.40	1.72	2.85
Total Measured + Indicated		**68.92**	**1.73**	**3.84**
Inferred	Main	9.90	1.76	0.56
	Secondary	12.33	1.75	0.69
	Total inferred	22.23	1.75	1.25
Cut-off Grade of 1.0g/t Au				
Measured	Main	12.32	2.08	0.83
	Secondary	0.64	1.57	0.03
	Total Measured	12.96	2.05	0.86
Indicated	Main	31.20	2.06	2.07
	Secondary	6.40	1.83	0.38
	Total Indicated	37.60	2.02	2.45
Total Measured + Indicated		**50.56**	**2.03**	**3.31**
Inferred	Main	7.07	2.11	0.48
	Secondary	8.93	2.08	0.60
	Total Inferred	16.00	2.04	1.08

Source : Minxcon 2012
Mineral Resources reported including Mineral Reserves
Mineral Resources are in situ tonnes
Inferred Mineral Resource has a significant degree of uncertainty as to whether it can be mined economically. It cannot be assumed that all or any part of the Inferred Resource will be upgraded to a higher confidence category.
Apparent computational inconsistencies due to rounding
All figures are in metric tonnes
Resources for which environmental, permitting, legal, socio-political, issues were expected to materially affect the estimate have been excluded
Tonnages and contents are stated as 100%
90% attributable to Asanko at production stage

Table 28 : Mineral Resource Estimate Reconciliation

MINERAL RESOURCE CLASSIFICATION	CUT-OFF GRADE (g/t Au)	COFFEY MINING 2011	MINXCON 2012	DIFFERENCE (%)
Tonnage (Mt)				
Measured	0.8	3.26	17.52	437
Indicated	0.8	52.51	51.40	-7
Inferred	0.8	24.79	22.23	-18
Au g/t				
Measured	0.8	1.50	1.76	17
Indicated	0.8	1.60	1.72	7
Inferred	0.8	1.50	1.76	17
Au Moz				
Measured	0.8	0.16	0.99	519
Indicated	0.8	2.78	2.84	2
Inferred	0.8	1.30	1.26	-3

Source : Minxcon 2012

The higher grades in the Minxcon 2012 model are a result of a more selective mining methodology and better definition of higher grades which is appropriate for this type of mineralisation. Coffey Mining in 2011 used larger block estimates resulting in a smooth grade profile, especially at higher grade cut-offs. Previous work by Coffey Mining with a different approach, shows similar or even higher grades at specific cut-offs.

The improved modelled grades within the same estimated volume, at the same cut-off, have also resulted in an increase in metal content.

14. Mineral Reserve Estimate

NI 43-101 Item 15 (a), (b), (d)

The Mineral Reserve estimate has been prepared by DRAM 2013 and is based on the 2013 PFS independent mining study (Section15) and the independent Minxcon 2012 Mineral Resource estimate (Section 13). The modifying factors applied are summarised in Table 29 and the Mineral Reserve estimate is presented in Table 30:-

Table 29 : Modifying Factors

FACTOR	UNIT	VALUE
No Inferred Resources included		
Mining Recovery	%	97
Mining Dilution	%	10
Contract mining costs	USD/t mined	3.23
Total processing costs	USD/t	16.64
Gold Price	USD/oz	1,400
Specific gravity		2.5
Resource Block model cell size	m	10x10x3
Geological Losses	%	none
Metallurgical recovery LoM	%	90.06
Cut-off Grade	g/t Au	0.6
Stripping Ratio	waste:ore	4.28

Source : DRAM 2013

The Esaase Project gold resources which occur in satellite pits and which are currently deemed uneconomic to extract, were excluded from Mineral Reserve estimate. Furthermore, Mineral Resources which are affected by permitting, environmental, logistic and socio-political issues such as proximity to villages or forest reserves, were also excluded from the Mineral Reserve estimate. The eliminated tonnes and gold ounces are approximately 7.20Mt with a contained metal content of 0.48Moz Au (DRAM 2013).
The additional modifying factors applied in the conversion to Mineral Reserves relating more specifically to mining recovery, mining dilution, cut-off grade and metallurgical recovery, are summarised in Table 29. The cut-off grade was determined by a series of simulations and selection was based on Asanko Gold's required grade and LoM as supported by the grade-tonnage curve presented in Figure 9.

Table 30 : NI 43-101 Compliant Mineral Reserve Estimate for the Esaase Project June 2013

PROVEN MINERAL RESERVES			PROBABLE MINERAL RESERVES			TOTAL		
TONNAGE (Mt)	GRADE (g/t Au)	CONTAINED GOLD (Moz)	TONNAGE (Mt)	GRADE (g/t Au)	CONTAINED GOLD (Moz)	TONNAGE (Mt)	GRADE (g/t Au)	CONTAINED GOLD (Moz)
22.85	1.43	1.05	29.49	1.4	1.32	**52.34**	**1.41**	**2.37**

Source : DRAM 2013
Cut-off Grade 0.6g/t Au
Inferred Mineral Resources excluded from the Mineral Reserve estimate
Apparent computational inconsistencies due to rounding
All figures are in metric tonnes
Resources for which environmental, permitting, legal, socio-political issues were expected to materially affect the estimate have been excluded
Tonnages and contents are stated as 100%
90% attributable to Asanko at production stage

A 'Mineral Reserve' is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. It includes diluting materials and allowances for losses that may occur when the material is mined. DRA is of the opinion that the classification of Mineral Reserves as reported in Table 30 meets the definitions of Proven and Probable Mineral Reserves as stated by the CIM Definition Standards (2005).

15. Mining Methods

NI43-101 Item 16

DRA Mining Proprietary Limited (DRAM) undertook the mining study component of the 2013 PFS, with the assistance of CAE Mining Proprietary Limited (CAE Mining). The scope of the study included mine design, pit optimisation, mine production and scheduling and costing. The study base case was predicated on the Measured and Indicated Mineral Resources, as supplied by Minxcon 2012.

15.1. Mining Methodology

NI43-101 Item 16 (a)

The appropriate mining methodology for the Esaase Project comprises conventional open pit drill and blast mining, followed by load and haul to various stockpiles. The drilling and blasting would be performed on benches between 10m and 20m in height. The total material movement at the Esaase Project is estimated between 30Mtpa and 35Mtpa and a mining fleet of 290t excavators and 90t capacity dump trucks would be appropriate for the scale of mining envisaged. Ore and waste boundaries would be delineated on each flitch with tape and paint markers based on grade control and geological modelling and interpretation. Bench and face mapping, for grade control as well as for geotechnical reasons, will be a routine task in finalising the ore and waste boundaries to be marked out for excavation.

A strategic decision to undertake owner mining versus contract mining depends upon the risk associated with either methodology, corporate issues involving best use and return on capital and local operational conditions. For the purpose of the PFS, Asanko Gold opted for the mining study to be conducted on a contract mining basis.

15.1.1. Geotechnical Review

NI43-101 Item 16 (a)

The geotechnical information for the mining study was provided by Coffey Mining and independently reviewed by the geotechnical consultant from Open House Management Solutions Proprietary Limited (OHMS). For the purpose of the pit optimisations, the following information was approved by OHMS as being appropriate and suitable for use in pit optimisation:-

- pit slope applied to fresh material – 52°;
- pit slope applied to transitional material – 45°; and
- pit slope applied to weathered material - 35° .

15.2. Pit Optimisation

NI43-101 Item 16 (a)

The Esaase Project pit optimisation study was undertaken on Whittle Four-X optimisation software on the mineral resource models provided by Minxcon 2012. Two separate pit optimisation scenarios were compared namely:-

- optimisation based on the “total mineral resource”, including Inferred Mineral Resources, for the purpose of resource drilling and infrastructure placement; and
- optimisation based “Measured and Indicated Mineral Resources” only, for the Mineral Reserve estimate.

Whittle Four-X software calculates, for a given mineral resource block model, cost, recovery and slope data; a series of incremental pit shells in which each shell is an optimum for a slightly higher commodity price.

The sequence of the pit shell increments is sorted from the economically best (the inner smallest shell viable for the lowest commodity price) to the economically worst (the outer largest pit shell viable for the highest commodity price).

Whittle Four-X provides indicative discounted cash flows (DCF) for two mining sequences called “best case” and “worst case” scenarios, both using time discounting of cash flows. In the best case, the optimum pit shells are mined bench by bench in increments from inner to the outer shell, resulting in a higher NPV due to lower stripping ratios and/or higher grades in the early years of mine life. The worst case scenario is based on mining the whole pit outline bench by bench as a single pit, hence resulting in a lower NPV as a result of high stripping requirements in the early years of the operation.

After the selection of the ultimate pit, several practical mining stages are designed and sequenced when developing a final production schedule, which provides an NPV between the worst and best case scenarios. The average NPVs are calculated for each pit shell (mean of the worst and best cases) in order to emulate a practical mining sequence. The cash flows are exclusive of any capital expenditure or project start- up costs, and should be used for pit optimisation comparison purposes only as they are not indicative of the Project NPV.

15.2.1. Pit Optimisation Input Assumptions and Parameters

NI43-101 Item 16 (a), (b)

Prior to the pit optimisation process, all cost and technical parameters were designed to emulate the required mining parameters based on the information available to DRAM prior to the outcome of the 2013 PFS. The mineral resource block model was re-blocked to coincide with the bench height of 10mx10m (along strike) and the input parameters and assumptions applied are summarised in Table 31. The initial 2013 PFS optimisations were undertaken at the “PFS Values” noted in Table 31 but post PFS new input values were obtained through various other optimisations and the “New Values” column shows those inputs that were modified and included in mine design that will form the basis of the DFS. The new modified inputs related to optimisation of the mining and processing costs.

Asanko Gold selected to conduct the PFS on a base case utilising contract mining as opposed to owner operations. DRA obtained four independent contract mining company quotations for the 2011 PFS which were escalated for inflation to 2013 and normalised for comparative purposes. The average mining cost obtained from the quotations was USD3.55/t and ranged USD3.39/t to USD3.70/t. A Project conceptual mining opex figure of USD3.23/t for the mine design was obtained by combining DRA’s database of costs, for similar operations and average costs from the quotations obtained. Mobilisation, site establishment, demobilisation, clearing and topsoil stripping costs formed part of capital expenditure and thus, not part of operational expenditure. As the PFS progressed and more detail became available, the opex was optimised and refined but these optimisations will be included in the DFS.

15.2.2. Pit Optimisation Results

NI43-101 Item 16 (b)

The initial optimisation produced a main pit with various satellite pits in accordance with the plan view of the Mineral Resources shown in Figure 9. Only the Main Pit and South Pit were included in the mining study and the North Pit, the northern rim of the Main Pit and Zones B and D were excluded as they contain design inefficiencies and are subject to mine concession restriction and the shell perimeters encroach the surrounding villages.

The optimum pit shell for the “total resource” scenario at PFS input values (Table 31) is Pit 31 which reached a depth of 40mamsl, or approximately 230m below the Bonte River valley surface. Pit shell 31 contains 53 Mt of mill feed at 1.42 g/t Au for 2,189koz of recovered gold (Table 32). A total of 198Mt of waste are contained within the pit, equating to a waste to ore stripping ratio of 3.7:1.

Table 31 : Pit Optimisation Input Parameters and Assumptions

PIT DESIGN ASPECT	UNIT	PFS VALUE	NEW VALUE*
RoM production	Mtpa	5	
Real Discount rate	%	10	
Mining recovery	%	97	
Mining dilution (assumed)	%	10	8
Contract mining cost (based on quotations)	USD/t	3.23	2.56
Total Processing costs (based on DRA estimates)	USD/t	20.54	
*Processing variable costs	USD/t	14.18	9.74
*Processing fixed cost allocation	USD/t	2.46	0.73
*Ore incremental	USD/t	1.00	1.8
*Processing general administration costs	USD/t	2.90	2.6
Gold price	USD/oz	1,400	
Royalty	%	5.5	
Oxide material pit slope	Degrees	35	
Transition material pit slope		45	
Fresh material pit slope		52	

Source : DRAM 2013
*Variations between the Whittle inputs and press release figures are due to evolvement of the Project post completion of the Whittle optimisation

The optimum pit shell for the "Measured and Indicated Resource" scenario at "PFS input values" (Table 28) at a cut-off grade of 0.6g/t Au was Pit 31 but Asanko Gold selected Pit 34 as it offered a longer LoM and an increase in gold recovered (Table 32). Pit shell 34 reached a depth of -19mamsl, or approximately 290m below the Bonte river valley surface. Pit shell 34 contains 59.5Mt of mill feed at 1.4g/t Au for 2,680koz of gold. A total of 244Mt of waste are contained within the pit, equating to a waste to ore stripping ratio of 4.1:1. Pit 34 was selected as the basis for the detailed mine design and the results of the final pit design are presented in Table 32.

The optimum pit shell for the "Measured and Indicated Resource" at the optimised new input values for the future DFS is presented in Table 32.

15.2.3. Pit Selection and Mine design

NI43-101 Item 16 (b), (c)

The 2013 PFS detailed pit design was based on the" Measured and Indicated Resources" scenario for Pit 34 at the PFS input values shown in Table 32, at a mining cost of USD3.20/t (Table 31) which reflects a contractor mining scenario for which the mine does not purchase the equipment. The design included only the Main and South Pits whilst the North and satellite were excluded (see Section 15.2.2) (Figure 11, Figure 15).

Table 32 : Pit Optimisation Results

ITEM	UNIT	TOTAL RESOURCES INCLUDING INFERRED- PFS INPUTS		MEASURED AND INDICATED RESOURCES-PFS INPUTS		MEASURED AND INDICATED RESOURCES – OPTIMISED INPUTS	
		PIT 31		Based on PIT 34		PIT 34	
Ore	Mt	52.99		52.34		74.12	
Waste	Mt	198.25		224.25		356.23	
Total	Mt	251.24		276.59		430.35	
Stripping Ratio		3.74		4.28		4.81	
Product		Input	Recovered	Input	Recovered	Input	Recovered
In situ gold	Mg	75.37	68.10	73.75		102.27	92.61
In situ gold	Moz	2.42	2.19	2.37	2.14		
Input grade	g/t	1.42		1.41		1.38	
Pit utility	%	0.74		0.73		73.30	
NPV	USD	596.70				1,005.71	
Life of Mine		10.64		12		15.9	

Source : DRAM 2013

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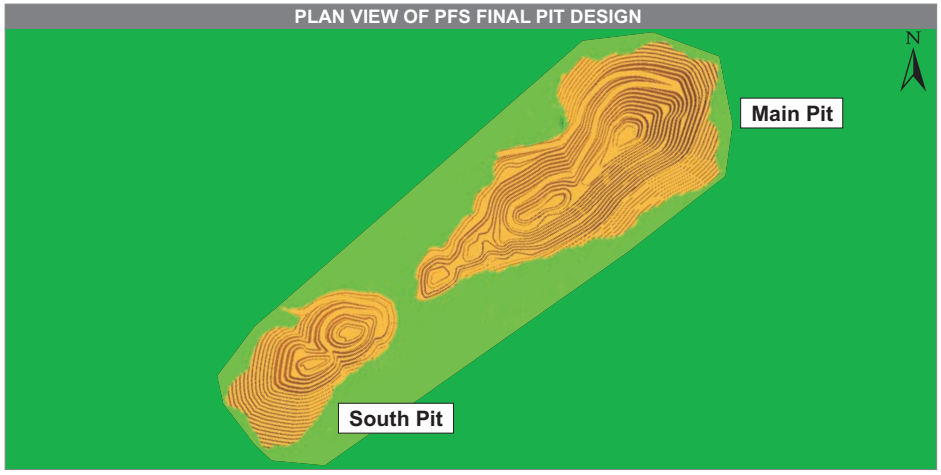
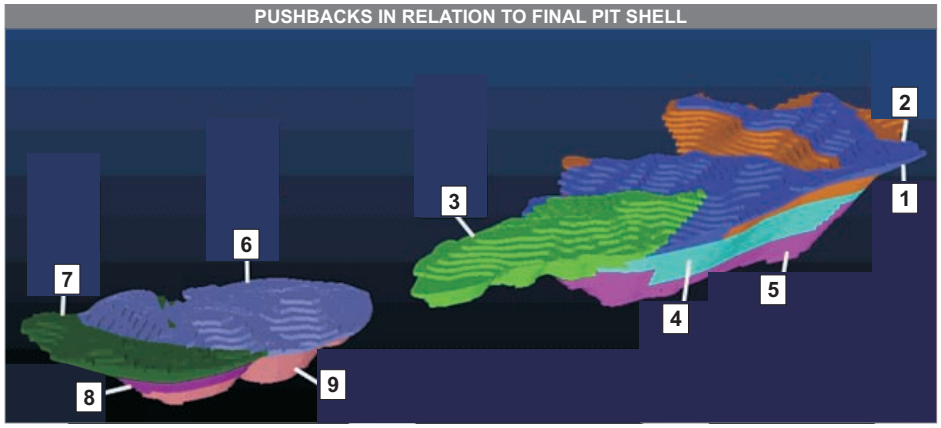
ESAASE PROJECT MINE DESIGN

PLAN VIEW OF PFS FINAL PIT DESIGN

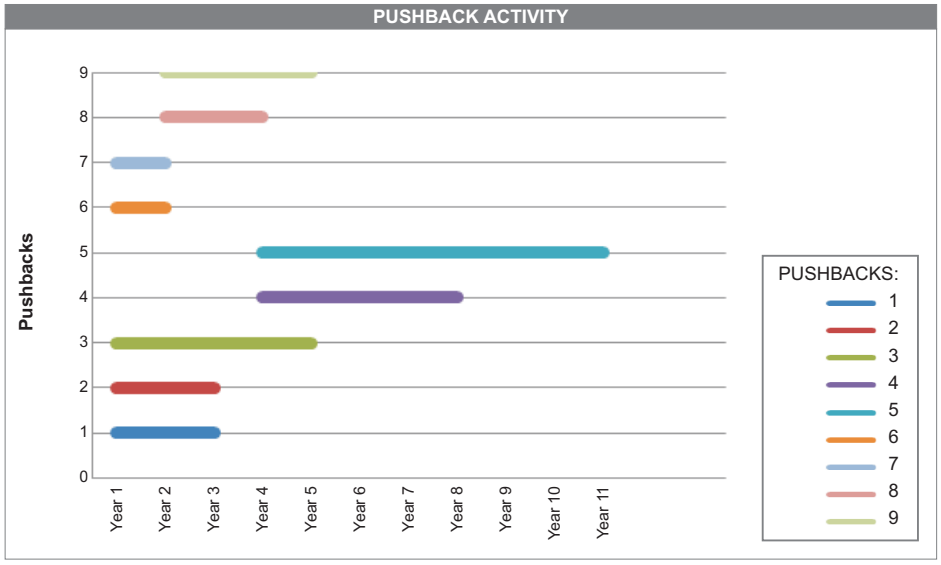


PUSHBACKS IN RELATION TO FINAL PIT SHELL



PIT	PUSHBACK	TONNAGE (Mt)	WASTE (Mt)	STRIPPING RATIO (waste:ore)	MILL FEED (Mt)	GRADE (g/t)	CONTAINED GOLD (Moz)
Main Pit	1	25.90	18.80	2.60	7.10	1.20	0.265
	2	26.00	21.60	4.90	4.40	1.40	0.205
	3	54.70	48.60	7.90	6.20	1.50	0.296
	4	56.00	40.80	2.70	15.20	1.40	0.678
	5	65.80	53.70	4.40	12.10	1.40	0.542
South Pit	6	11.60	10.10	6.70	1.50	1.50	0.071
	7	13.70	12.50	10.50	1.20	1.50	0.059
	8	10.20	8.20	4.10	2.00	1.60	0.103
	9	12.70	10.00	3.80	2.70	1.80	0.153
TOTAL		276.60	224.20	4.30	52.30	1.40	2.37

PUSHBACK ACTIVITY



Source: Minxcon 2012

The mine design is based on a minimum pit base width of 30m and a minimum cutback width of 50m. The haul roads have gradient of 10% and dual/single lane widths of 25m and 15m respectively.

The Main and South Pits were developed in 9 pushbacks, with the South Pit being mined within four years in order to supply backfill material (Figure 11).

15.2.4. Waste Dump Design and Site Plan

NI43-101 Item 16 (c)

A base case mine layout site plan was initially developed for the 2013 PFS with elements based on the 2011 layout. Optimisations in the course of the 2013 PFS indicated that a revised site layout would prove beneficial and the two plans are presented in Figure 14 and Figure 15 (see Section 0). A quantity of approximately 224Mt of mine waste has to be disposed of during the Project LoM and based on an average waste bulk density of 2.55t/bcm and a swell factor of 30%, after dump compaction, a dump volume of approximately 114Mm³ will be required to store the mine waste.

The issues that must to be considered in evaluating the best option for a waste storage areas include the following:-

- storage capacity;
- limiting visual impact;
- haul distance and disposal costs;
- site drainage; and
- site access and preparation

The waste dumps for the purpose of the 2013 PFS entailed no detailed design. The dump volumes were constructed using the 1:4 slope recommended by Epoch, to determine overall foot print estimated to a swell factor of 30% after compaction, dump overall slope of 14° and an angle of repose of 37°. The EPA approved practice in Ghana of permitting dump slopes of between 20º and 30º would provide potential for optimisation and this will be investigated in the DFS.

Waste material will be dumped in three different dump locations (Figure 15) namely west of the main pit close to the RoM pad, to the south east of the Main Pit between the sacred forest and Esaase village and on southern half of the South Pit (Figure 15). Some of the waste material will be used for road terracing (1Mbcm), TSFs (18Mbcm) and backfilling of the South Pit (19Mbcm).
The required combined capacity for the waste dumps is reduced as a result of the waste requirement for other mine infrastructure items and by backfilling the South Pit. The waste dumps reach a maximum height of 550mamsl, cover 362ha combined with a combined capacity of 111Mm³.

The waste dumps will be built by tipping from ground level upwards so as to lessen the vertical climb by the dump trucks early on to reduce costs. The dumps will be progressively rehabilitated with topsoil, where possible. The surfaces of dumps will be contoured to minimise batter scour and ripped at 1.5m centres to a depth of 400mm, where practicable. All such rehabilitation work will be carried out progressively. Rock-lined drains will be constructed, where required, to ensure excess run-off is controlled and directed down to sediment traps. The waste dump design will incorporate features to minimise the effect of leaching of contaminants.

15.3. Mine Production Schedule

NI43-101 Item 16 (b)

The mine production schedule is based on an annual processing rate of 5.0Mtpa with a mining limit set to 35Mtpa annum for waste and ore combined giving an average LoM stripping ratio of 4.28:1. The mining limit did not restrict the ability to supply the plant with the full capacity of 5Mtpa.

The pre-production is defined production until the process plant is commissioned. The pre-production includes establishment of the haul roads from the pit to the RoM pad and waste dumps; generation of a minimum of three weeks mill feed ready for plant commission and the generation of waste for infrastructure construction. The operating ramps and initial benches that will supply mill feed will also be prepared during the pre-production stage. The first three months of the mine schedule (Figure 12) comprise this necessary pre-production.

Mining will commence during the pre-production in both Main and South Pits and as the South Pit is located closer to the starter tailings embankment it will provide waste material for the starter dam. A portion of the waste mined from the Main Pit during pre-production will be utilised to build the RoM pad.

After the pre-production phase has been completed, the Main Pit remains the principal source of RoM, with South Pit contributing, on average, a third (±31%) of the overall tonnes mined for the first four years until it's mined out and ready to be backfilled (Figure 12).

The Main Pit is the primary source of mill feed with over 85% of the total mill feed arising from this pit. The pushback schedule is summarised in Figure 11. The South Pit can be mined independently from the Main Pit with a saddle separating the two pits. The South Pit is initiated during pre-production and is essentially completed at the beginning of Year 5.

The mine production schedule was developed for the PFS on a monthly basis for the first year and quarterly thereafter and is based on bench by bench mining of the quantities calculated within the individual pit stages and pushbacks. The mine production schedule is presented in Figure 12.

15.4. Mining Fleet and Labour Requirements

NI43-101 Item 16 (d)

The contractor mining fleet will consist of four hydraulic excavators (Cat.6030) with a bucket capacity of between 20t and 30t, and 28 off highway dump trucks (Cat 777D) with capacities of 90t. Rigid frame diesel trucks have been used in the mining of large open pits for many years and are appropriate for the Esaase Project operation. In addition 6 Dozers and two graders would be required, together with service trucks, drill rigs and wheel loaders.

15.4.1. Labour Requirements

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. The labour requirements for the mining operation including owner's team and contractor's staff are summarised in Table 34.

Table 33 : Esaase Project Mining Labour Requirements

	MANAGEMENT	TECHNICAL SERVICES		PRODUCTION STAFF		MAINTENANCE	TOTAL
	ADMIN	GEOLOGY	MINING/SURVEY	SUPERVISION	OPERATORS		
Owners team	2	14	12	3	0	0	**31**
Contractor team	18	0	0	62	266	70	**416**

Source : DRAM 2013

15.5. Drill and Blast Parameters

Rock fragmentation will be undertaken by drilling and blasting and it was estimated that 50% of the weathered material and 100% of the fresh material will require blasting. The pit configuration bench height and waste material type anticipated at the Project suit drill rigs capable of drilling drillholes with a diameter of between 115mm and 165mm. Drill burden, spacing and sub-drill design will be functions of the varying material types of the deposit.

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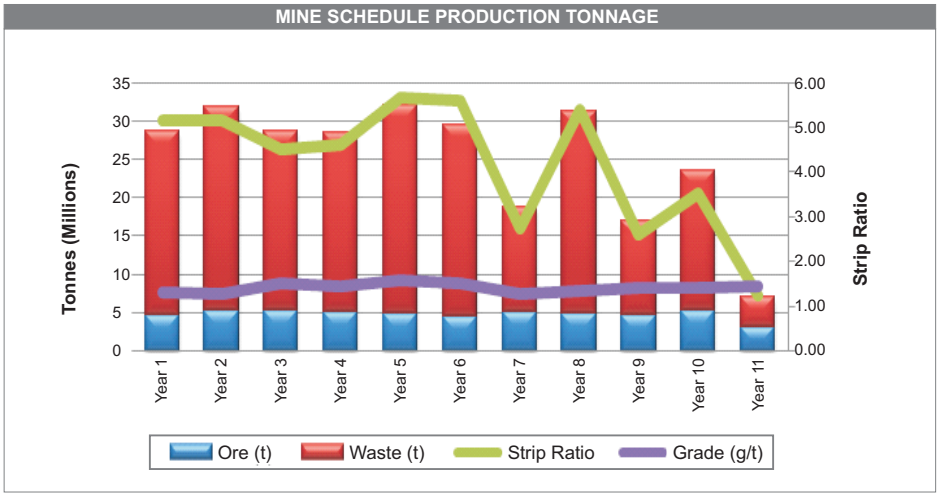
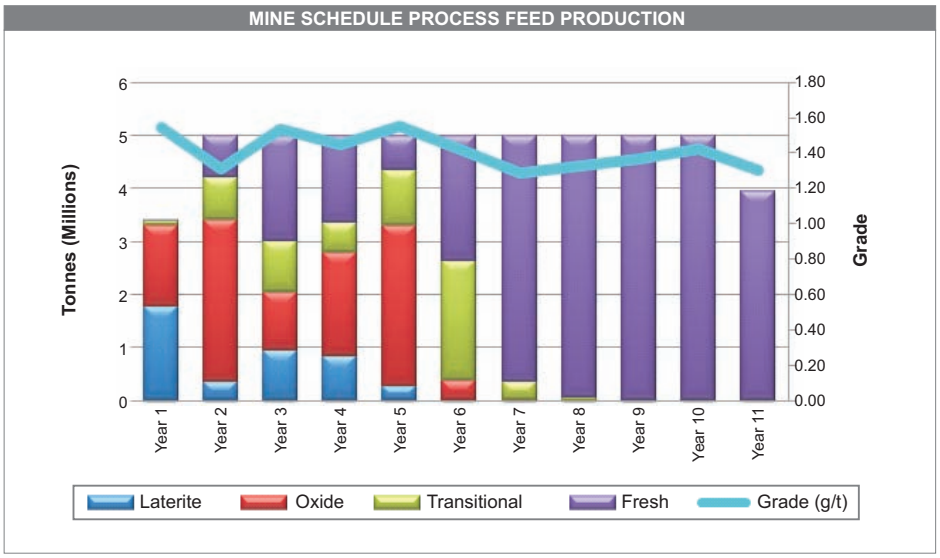
ESAASE PROJECT MINE PRODUCTION SCHEDULE

	PRE-PRODUCTION (part of Year 1)	YEAR										
		1	2	3	4	5	6	7	8	9	10	11
Ore tonnage (t)	731,887	3,925,313	5,196,416	5,210,690	5,057,399	4,795,111	4,473,701	5,004,134	4,900,778	4,698,270	5,180,125	3,166,386
Waste Tonnage (t)	3,146,524	20,790,359	26,699,673	23,488,559	23,330,205	27,185,220	24,995,567	13,727,579	26,417,424	12,191,143	18,302,866	3,970,687
Total tonnage (t)	3,878,411	24,715,672	31,896,089	28,699,250	28,387,604	31,980,330	29,469,269	18,731,713	31,318,202	16,889,413	23,482,990	7,137,073
Metal (g)	928,365	5,241,750	6,721,602	7,816,682	7,292,311	7,593,258	6,726,949	6,423,094	6,584,615	6,617,131	7,237,095	4,571,648
Grade (g/t)Au	1.30	1.34	1.29	1.49	1.43	1.58	1.50	1.28	1.33	1.41	1.36	1.45
Total ore cost (USD)	3,708,504	18,660,141	23,575,472	24,641,754	23,886,377	22,655,311	19,741,393	22,328,159	22,338,818	21,654,723	24,477,125	15,426,683
Total waste cost (USD)	10,116,457	63,218,568	74,495,737	71,281,186	70,152,018	79,761,890	66,118,317	52,142,399	35,800,971	65,223,416	40,715,878	12,136,726
Total capex (USD)	13,748,959											





Source: Minxcon 2012

An emulsion based blasting product with water resistant characteristics and high 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 UCS (Unconfined Compressive Strength) measurement results obtained from the preliminary excavation characterisation work. For weathered material the UCS range is between 8MPa and 12MPa, which suggests a very weak rock. For fresh material the UCS range is between 28MPa and 80MPa, which suggests a weak to moderately strong rock.

The Project is located in the vicinity of four communities. Definition of risks associated with drill and blast activities, and procedures to mitigate these risks will be addressed in detail during the DFS. A detailed Environmental Blast Design Report was generated for the 2013 PFS by Terrock 2013.

15.6. Haulage Roads and Pit De-watering

All haul-roads, dumps and stockpiles that will be required for the LoM will be constructed during pre-production. The beneficial results of improving haul road design are efficiency of haulage by reduction in cycle time, reduced fuel usage and reduced truck component wear. The recommended haul road width within the mining area is 20m wide, excluding drainage and shoulders.

The pit water management will primarily consist of run-off control and sumps. As the LoM pit will be operating at depths greater than 200m below surface, specialist high lift pumps will be required. Pontoon mounted pumps will be used to draw from sumps to ensure the pumps are not submerged as sump water levels rise rapidly in response to a rainfall event. The key operational requirements will be to:-

- minimise water flow 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; and
- installation of settling ponds for the removal of solids prior to discharge offsite.

15.7. Grade Control

Crusher feed quality control for the Project will be a critical component of the success of the operation. Production control relies on different levels of mine planning in daily blending operations. Plans are developed with different levels of accuracy for different time periods, including daily, weekly, monthly, yearly and LoM. Reconciliations of production and RoM quality against these plans will form part of mine planning procedures.

Bench and face mapping, for grade control as well as for geotechnical reasons, will be a routine task in finalising the ore and waste blocks to be marked out for excavation. Blast hole sampling is recommended as a test to identify the suitability of utilising the vertical blast holes for grade control purposes. Sampling of every blast hole has been allowed for in the lab under the processing costs.

16. Recovery Methods

NI 43-101 Item 17

The process design for the Esaase Project was undertaken by DRA and independently reviewed by Venmyn Deloitte (2013) in a document entitled “Asanko Gold – Esaase Project Metallurgical Report” by R Heins (May 2013). Five phases of metallurgical testwork have been completed for the Esaase Project, the results of which are presented in Section 12. The metallurgical testwork results indicate that the Esaase Project gold mineralisation is amenable to the proposed process design and the final design selection will be a function of recovery optimisation and financial considerations.

A high-level financial trade-off between the “Flotation Included” and “Flotation Excluded” routes was undertaken as the basis of selection for the most viable processing route for Esaase Project. The plant recoveries used in the financial analysis are presented in Table 20 and the estimated capital and operating expenditures (capex and opex) for the two process routes are as follows:-

Table 34 : Process Trade-off Capital and Operating Costs

	FLOTATION INCLUDED	FLOTATION EXCLUDED
Process Plant Operating Cost (USD/t)	10.37	15.34
Process Plant Capital Cost (USDm)	93.20	109.00

Source : DRA 2013
Includes USD10.3m EPCM

The financial trade-off indicated that the processing route which comprised gravity recovery within the milling circuit, flotation on gravity tailings, with a leach on the flotation concentrates (Flotation Included) resulted in reduced process plant capital and operating cost with a marginal reduction in LoM recovery of 0.52%, reduced capital cost and environmental risk associated with the L-TSF, which in combination resulted in increased project NPV.

The basis for the design aspect selection and final criteria are summarised in Table 35:-

Table 35 : Process Design Selection and Criteria

DESIGN ASPECT	TESTWORK BASIS FOR SELECTION	FINAL CRITERIA
Grind size	Amdel showed 30% Au>106µm suggesting finer grind would be beneficial.	DRA review of results targets 80% passing 75µm
Mill	DRA in-house simulations	Ball mill size 7.16mx10.8m with total installed power of 13.2MW)
Gravity recovery	Modelling and simulation in previous phases	Primary Knelson concentrators positioned on cyclone underflow in milling circuit. Also secondary gravity concentrator in regrind milling circuit
Flotation	Previous flow sheet used spirals but arsenic would be difficult to remove.	Flotation instead of spirals would handle fine free gold and gold with sulphides
Kelsall parameters	Kelsall rate constants used to predict recoveries and mass-pull at different residence times	Laboratory flotation residence time of 16.5mins selected for mass-pull of 9% on fresh material, scaled up by a factor 0f 2.5 to allow 41.2min residence time for full-scale plant operation
Concentrate regrind	Phase V flotation concentrate leach results showed 3.4% improvement in recovery at a 25µm grind size as opposed to 25µm.	Vertimill has been included in the design for regrinding the flotation concentrate
CIL	Phase V testwork concluded addition of carbon with no pre-leaching for F_{80} of 25µm gave a CIL residue grade of 0.15g/t Au	Scope to reduce CIL residence time in next study phase
Cyanide and arsenic detoxification	SO_2/air system for destruction of WAD cyanide to below 50ppm. The design is for <50ppm CHwad and 1.0ppm As in process plant tailings. Laboratory tests achieved CNwad levels of <25ppm	SO_2/air system for destruction of WAD cyanide and inclusion of arsenic precipitation from the tailings solution using ferric chloride at pH 8
TSF	Two TSF required to accommodate flotation tailings and cyanide tailings	The design permits a lined TSF for the cyanide tailings at 9% of previous size estimate with capital savings

Source : DRA 2013

16.1. Process Description

NI 43-101 Item17 (a), (b)

The recommended process flow sheet for the gravity-flotation-CIL process plant is presented in Figure 13 and described in summary in Table 36.

Table 36 : Recommended Process Flow for Esaase Project

PLANT COMPONENT	SYSTEM DESCRIPTION	PRODUCT STREAM	COMMENTS
Ore Receiving	Rom treated in open circuit primary crushing system		
Primary crushing system	RoM bin and gyratory crusher	195mm	Water spray dust suppression
Secondary crusher	Belt feeder and conveyor from primary crusher. Secondary cone crusher	90mm	Hammer sampling station for grade control on RoM stockpile
Oxide material by-pass	Oxide from primary crusher directly to mill by-passing secondary crusher and stockpile		
Mill feed stockpile	Belt feeder and conveyor from secondary crusher.		
Mill hopper	Ball milling circuit fed 635t/h dry RoM stockpile feed. Water added to controlled density 70% solids.		
Mill discharge sump screen	Screen underflow to closed circuit cyclones		
Mill cyclone	Overflow gravity fed to flotation circuit	35% solids	
Primary gravity concentration	Portion of cyclone underflow to two Knelson XD-48 concentrators each treating 400t/h pre-screened material. Diluted to 60% solids by mass. Screen oversize to mill feed conveyor. Screen underflow gravitate to gravity concentrators	3t/d high grade concentrate	Fenced for security
Intensive leach reactor (ILR)	Gravity concentrates collected in dewatering cone with an underflow of 60% solids which is sent on a batch basis to the reactor. Batch solution of cyanide and caustic soda leach.	98% Au extraction in pregnant leach solution to electrowinning cell feed tank	Spillage collected and pumped to CIL sump. Safety shower,
Flotation	Cyclone overflow from milling circuit gravity feed. Single bank of seven 130m^3 forced air rougher flotation cells.	Flotation concentrate collected in agitated concentrate tank	Area serviced by two spillage pumps. Safety shower installed.
	Tailings sampled and collected in tailings tank and pumped to TSF		
Regrind mill	Flotation concentrate to de-sliming cyclones on regrind circuit. Cyclone overflow to pre-leach thickener. Underflow to Vertimill to achieve size reduction to 20µm. Regrind product gravitates to sump where density controlled by dilution water added from mill outlet, elution and gold room.	Underflow at 70% solids regrind to 20µm.	
Secondary gravity concentration	Sump contents to secondary gravity concentration unit.	Concentrate to intensive dissolution reactor. Tailings to flotation concentrate sump	
Pre-leach thickener	Increase density of cyclone overflow in regrind circuit	45% solids to be pumped to CIL circuit	
CIL	Seven 330m3 tanks in series linked by Kemix inter-tank pumping screens. Lime for pH control and cyanide added to the first tank	Au grade 0.15g/t in tailings and solution tenor of <0.01ppm	Hydrogen cyanide and ammonia gas detectors installed. Safety showers.
	Design carbon loading of 2,500g/t and a loaded carbon batch size of 5t/day		
	Loaded carbon pumped to vibrating screen	Carbon to elution circuit and slurry underflow back to CIL tank	
Cyanide destruction	CIL tailing pumped to detox circuit of three agitated and air sparged 150m^3 tanks in series. SO_2/air process in two tanks	Tailing <50ppm CN_{WAD}	Hydrogen cyanide gas and WAD cyanide detection installed at TSF
Arsenic precipitation	Third tank in above series will have ferric chloride added the precipitate As. Tailings disposal comprises conical agitated tank and pumps.	Tailing solution <1.00ppm As	
	TSF return water to process water tank		
Elution	CIL carbon treated in elution circuit comprising acid wash and 5t elution column with heater. CIL carbon treated in 5t batches once every 24hrs.		
	Carbon acid washed in 3% HCl solution then loaded by gravity into the elution column. Carbon pre-treated with solution containing 3% cyanide and 3% caustic soda. Heated to 125ºC	Eluted carbon removed and sent to carbon regeneration kiln. Carbon fines to the tailings and oversize to CIL circuit	Spillage collected and pumped to CIL circuit
Electrowinning	Pregnant solution from Acacia reactor through electrowinning cell	Solution re-circulated for 18hrs or until barren solution attained. Gold sludge collected	
	Pregnant solution from CIL circuit circulated through four cells in parallel		
	Barren solution tested and pumped to CIL circuit for disposal		
	Loaded cathodes removed from cells and gold sludge washed off with high pressure. Gold sludge dried, mixed with fluxes and smelted in induction furnace	Gold bullion bars and slag. Slag crushed and returned to ball mill circuit	Hydrogen cyanide gas and ammonia gas detection installed . Various security systems installed

Source : DRA 2013

16.2. Process Infrastructure and Reagents

NI 43-101 Item 17 (c)

The Esaase processing plant will be supplied by borehole stored in a 500m^3 raw water tank. Raw water, process make up water and feed to a potable water plant will be withdrawn from the raw water tank. Fire water allowance will be made within the raw water tank, in accordance with IMIU fire water requirements.

Process water will be stored in a process water dam along with pre-leach thickener overflow excess and tailings return water. Process water requirements will be withdrawn from the process water dam which will have a capacity of 8,000m^3.

Compressed process and instrument air will be supplied via dedicated compressors, while oxygen requirements for CIL will be supplied by a dedicated 10t/day PSA plant. Instrument and plant air requirements for the entire plant will be supplied by two dedicated compressors, one running and one on standby with air filters and driers. Air requirements for general use in the plant and workshops will be tapped-off before the instrument air driers.

Low pressure blower air for the flotation and detoxification circuits will be supplied by three blower units and two dedicated blower units respectively, the latter of which will be a standby unit.

The reagents to be used in the process are summarised in Table 37:-

Table 37 : Process Plant Reagents

REAGENT	SPECIFICATION	QUANTITY
Frother	Aero 9967, MX gold	Liquid 1,000L bulk containers
HCl	33% strength containers	1,000L bulk containers
Flocculant		25kg bags
Activator	Copper sulphate made up to 15% solids solution	1,250kg bags
Sodium Metabisulphate	SMBS made up to 20% solids solution	1,200kg bags
Diesel	Stored in sealed tank for delivery to elution heaters, carbon regeneration kiln and furnace	
Caustic Soda	Solution made up to 20% solids	
Sodium cyanide	Supplied as briquettes mad into 20% solids solution	
Hydrated lime	CaO 85% to 90%. Solution made up to 20% solution strength	1,000 bulk bags
Collector	PAX collector pellets solution 15% concentration	25kg bags
Ferric chloride	20% solution strength	Delivered as a powder
Lead nitrate	20% solution strength	

Source : DRA 2013

17. Project Infrastructure

NI 43-101 Item 18

The designs of the supporting infrastructure required for the proposed Esaase mine were developed by DRA in consultation with the Asanko Gold owner's team and other independent specialist consultants, principally Epoch for the TSF designs, Coffey Mining Geotechnics for the water balance and hydrogeology and Knight Piesold for the geotechnical investigations. Numerous sampling, drilling and excavation programmes were undertaken to provide samples and technical information for the various specialist studies to be undertaken, as summarised in Table 39.-

17.1. Geotechnical Investigation

The independent geotechnical investigation was undertaken by Knight Piésold Consulting at the proposed location of key infrastructure including process plant site, TSF sites, RoM pad, mine services area, waste rock dump sites and haul roads. The investigations undertaken are summarised in Table 39 and the bulk and undisturbed samples taken provided:-

- natural moisture content to determine the in situ moisture regime;
- Atterberg limits and particle size distribution tests to determine the basic engineering properties of the in situ soils and for classification purposes;

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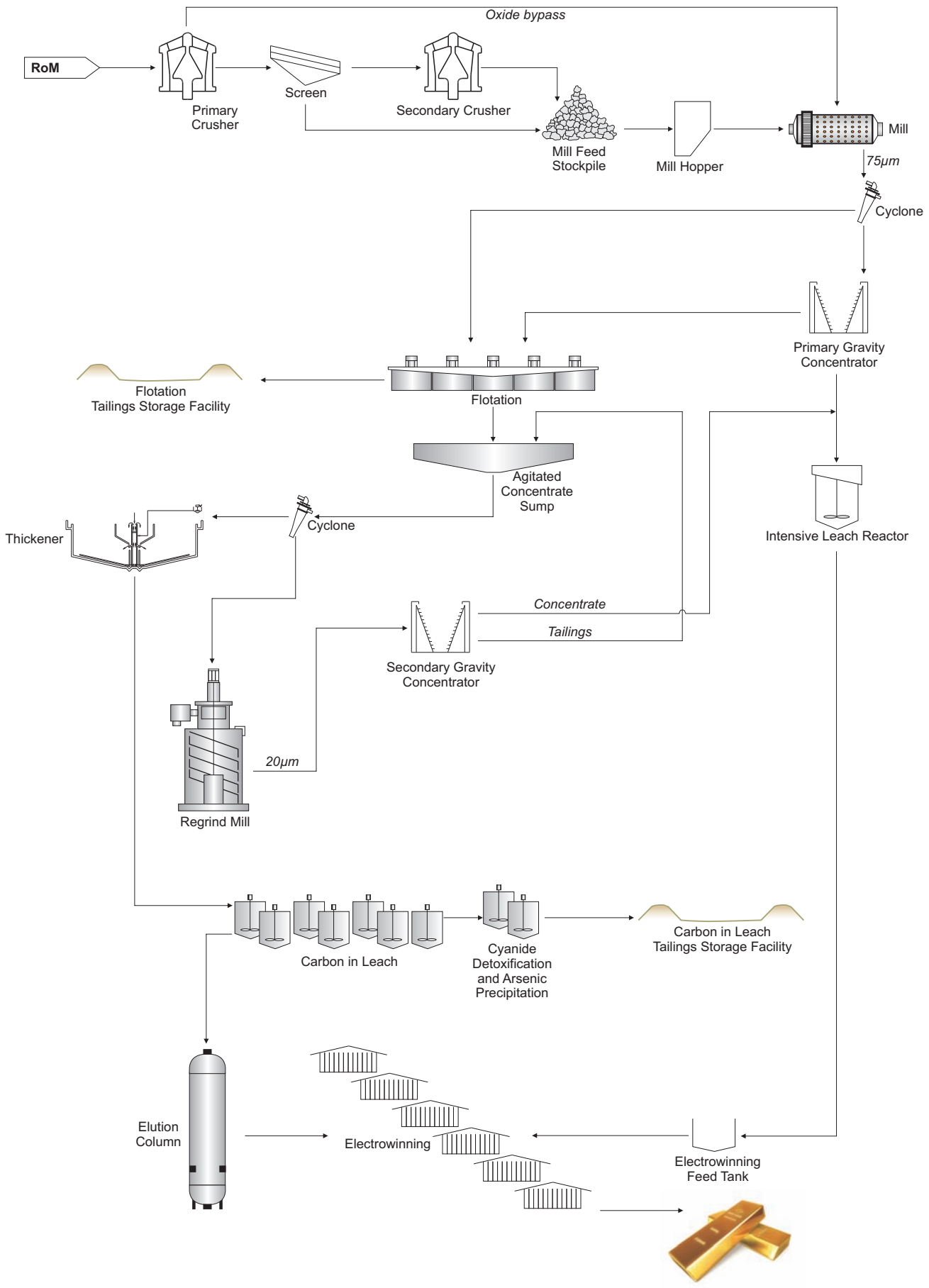
RECOMMENDED PROCESS FLOW SHEET



Source: Minxcon 2012

- crumb dispersion tests to determine the separation of soils into single particles;
- dry density / moisture content relationship;
- consolidated un-drained triaxial compression test on re-moulded samples to determine the strength and stress-strain relationship of the soil samples;
- consolidation test to determine the settlement characteristics of the soils;
- falling head permeability test on re-moulded and some undisturbed samples, and
- pH and sulphate content (SO_3).

17.2. Esaase Mine Site Layout

The proposed Esaase mine site is characterised by the low gradient Bonte river valley which is surrounded by short, steep contributing catchments with pronounced changes in elevation. The topographic elements of the area have a material influence on the size, positioning and operation of facilities and infrastructure associated with the Project as shown in Figure 15.

Two alternative site layout plans were developed for the 2013 PFS as shown in Figure 14 and Figure 15. Both site plans included the following elements:-

- 5Mtpa RoM processed through a gravity-flotation-CIL plant;
- plant location in the valley between the villages of Tetekaaso and Tetrem west of Main Pit;
- reduced pit configuration of the Main Pit and exclusion of the North Pit;
- plant tailings are processed into two streams, namely a CIL tailings stream to be sent to the lined TSF (L-TSF) accommodating approximately 10% of the tailings and a flotation plant tailings stream to be sent to a clay lined TSF (F-TSF) accommodating 90% of the tailings; and
- a buffer water storage dam configured as a return water dam collecting decant from the F-TSF and sized to store make-up water storage requirements under dry rainfall conditions and to prevent discharge of contaminated water exceeding discharge standard under wet conditions.

The key differences between the base case and improved case plant site configurations are summarised in Table 38:-

Table 38 : Site Layout Option Comparison

INFRASTRUCTURE	BASE CASE SITE LAYOUT (Figure 14)	IMPROVED SITE LAYOUT (Figure 15)
Waste Rock Dump (WRD)	Single WRD located approximately 3-4 km to west of main pit	Multiple smaller WRDs located within pit blast radius
Flotation circuit tailings TSF (F-TSF)	Located approximately 5 km to north east of main pit	Located in position of Base Case WRD
Buffer storage dam	Located south of F-TSF approximately 4km north west of main pit	Located downstream of Mpatoam on Esaase concession boundary

Source : DRAM 2013

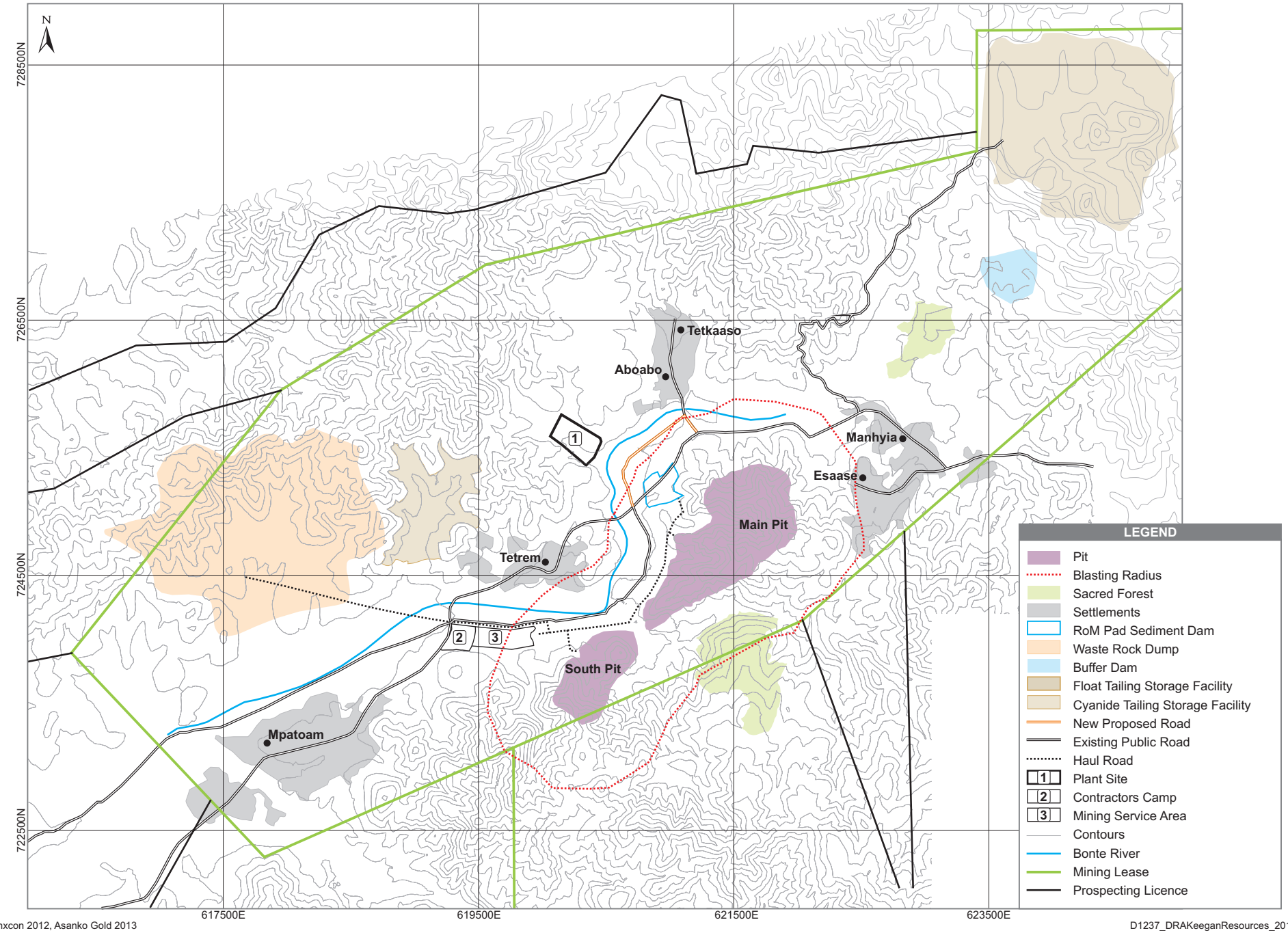
The study provided preliminary information that was utilised in the positioning of the plant, contractors terrace, RoM pad and conveyor belt. Further geotechnical investigations will be undertaken in support of the foundation design of the various plant structures. The bulk earthwork requirements were determined and can be supplied from the various residual soils, altered clay rich material and more incompressible weathered schist available within the Project area.

Table 39 : Specialist Studies undertaken for the Infrastructure Design

7.5	SPECIALIST CONSULTANT	DATE	SCOPE OF STUDY	INVESTIGATION	RESULTS
Geotechnical	Knight Piesold Consulting and drilling by independent PMI Limited	2012	Geotechnical investigation of process plant site, TSF site, RoM pad, mine services area, waste rock dump and haul roads	7 drillholes by i PMI Limited between 14m and 20m. Test pits- 26 dug to 4m	Bulk samples and undisturbed samples for laboratory testing. Geological and geotechnical conditions for each site characterised and defined. Soils affected by tropical weathering laterite, residual soil, completely weathered rock and highly weathered rock. Moderately strong basement rock which is incompressible.
Geological					Valley floor alluvial/colluvial soils higher capacity for bearing failure. Higher up the valley residual soils <2m deep and foundations to be dug to weathered schist. Conveyor positioned on a raised embankment of engineered fill. RoM pad on thin residual soil and completely weathered rock-foundations for crushers must extend to the schist.
Earthworks					Topsoil stripped to 300mm and stockpiled. Plant key components located in a cut-section to 2m to 3m deep to ensure bearing pressures down to weathered schist. Remainder plant and contractor terrace requires 275,000m^3 fill. RoM pad and contractors camp requires 615,000m^3 fill. Buffer dam requires 250,000m^3 fill and will be lined with in situ compacted clay.
Site Water management	HR Wallingford	2011	Baseline meterology, hyrological and hydraulic studies, Offin River rating curve, Mpatoam weir rating assessment		
Meterological	HR Wallingford				Aver annual rainfall 1,382mm. Aver annual evapo-transpiration - 1,340mm. Wettest month - June 214mm. Maximum 534.5mm, 168mm in one day. Wettest year 2,428mm. Lowest rainfall recorded over 5months <90mm.
Hydrology					Project located in headwaters of the Bonte river catchment. Bonte is a tributary of the Offin river. Bonte river flow east to west across northern extent of Main Pit. 3km downstream is the Mpatoam weir with 32km^2 catchment area. Weir data shows that the Bonte river in the Project area is dependent on rainfall with negligible dry season base flow. ground water not a significant contributor to river flow.
Hydrogeological	Coffey Geotechnics Limited	2009	Hydrogeological investigation to understand groundwater conditions on site	Three phase drilling programme. 28 drillholes to a maximum depth of 157m	Investigated groundwater contours across site and high yield areas such as the saddle between the Main and South Pits. Yields vary between 0L/s to .30L/s. Standing water levels at July 2011 - vary between 272mamsl and 356mamsl. Max water yield depths vary between 25m and 90m.
Community monitoring boreholes	Asanko Gold	2012	Provide baseline data for groundwater at existing villages	10 borehole drilling programme	
Geochemistry	Environmental Geochemistry International Limited	2012 to 2013	Hydrogeochemical modelling of the mine site and kinetic leach tests on arsenic release	Extensive sampling from village hand pumps, community monitoring boreholes Esaase pit boreholes	Results presented in Table 44. Arsenic present in groundwater as primary contaminant, exceeding GWC guidelines of 0.01mg/L soluble arsenic.
Water balance	Coffey Geotechnics Limited	2012	Update since previous DFS as site configuration completely refigured.		Three tier approach. **Level 1**:- closed process and TSF circuit with make-up water from pit dewatering. **Level 2**: site water capture and pit dewatering. **Level 3**:- rainfall within mine catchment released to the Bonte river
Power	Gridco - state utility company	2012	Feasibility study for34MW power supply		

Source : DRA 2013

ORIGINAL 2013 PFS MINE SITE LAYOUT PLAN



Source: Minxcon 2012, Asanko Gold 2013

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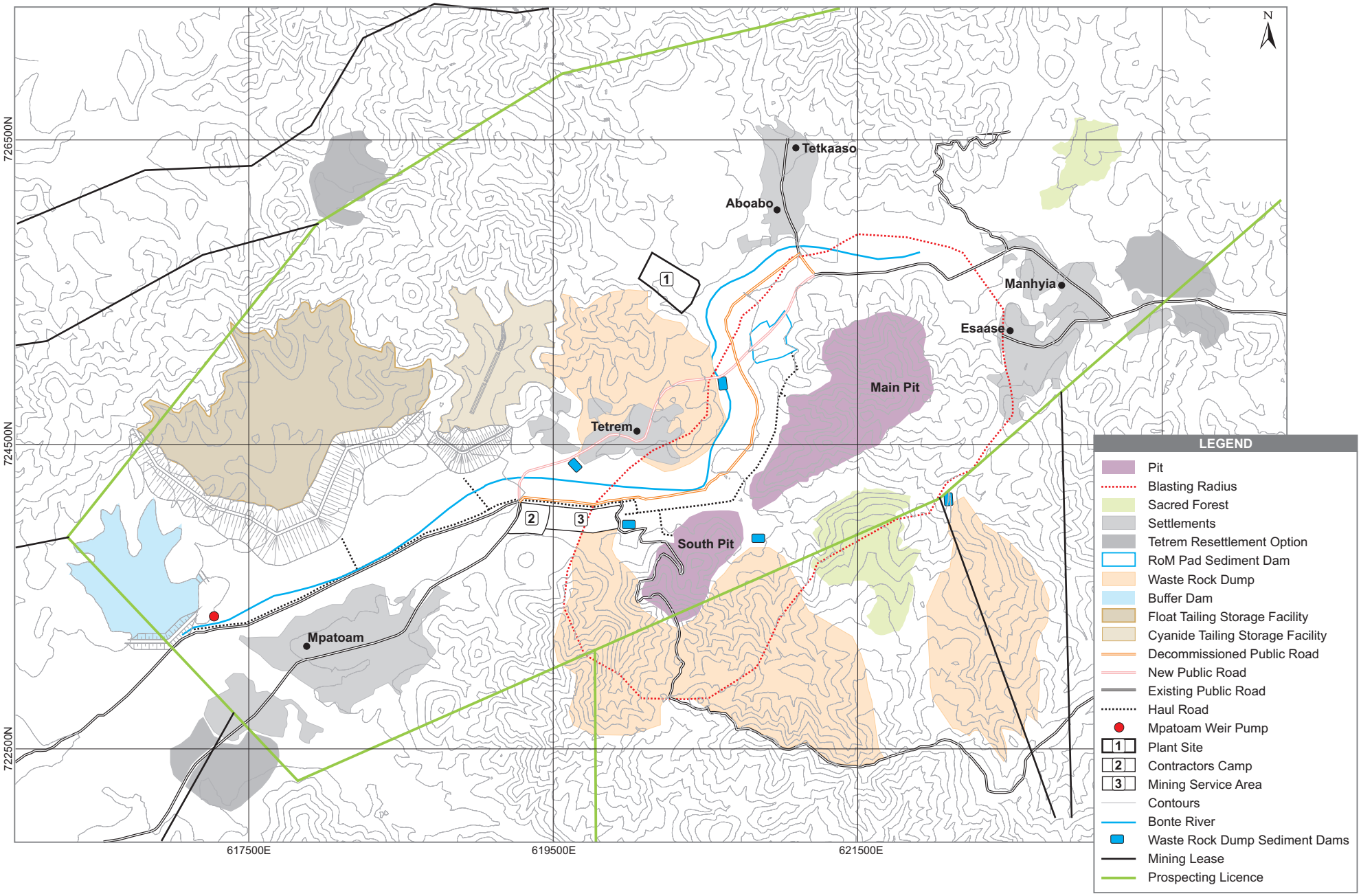
Figure 14

OVERALL OPTIMISED ESAASE PROJECT SITE PLAN FOR THE 2013 PFS



Source: Minxcon 2012, Asanko Gold 2013

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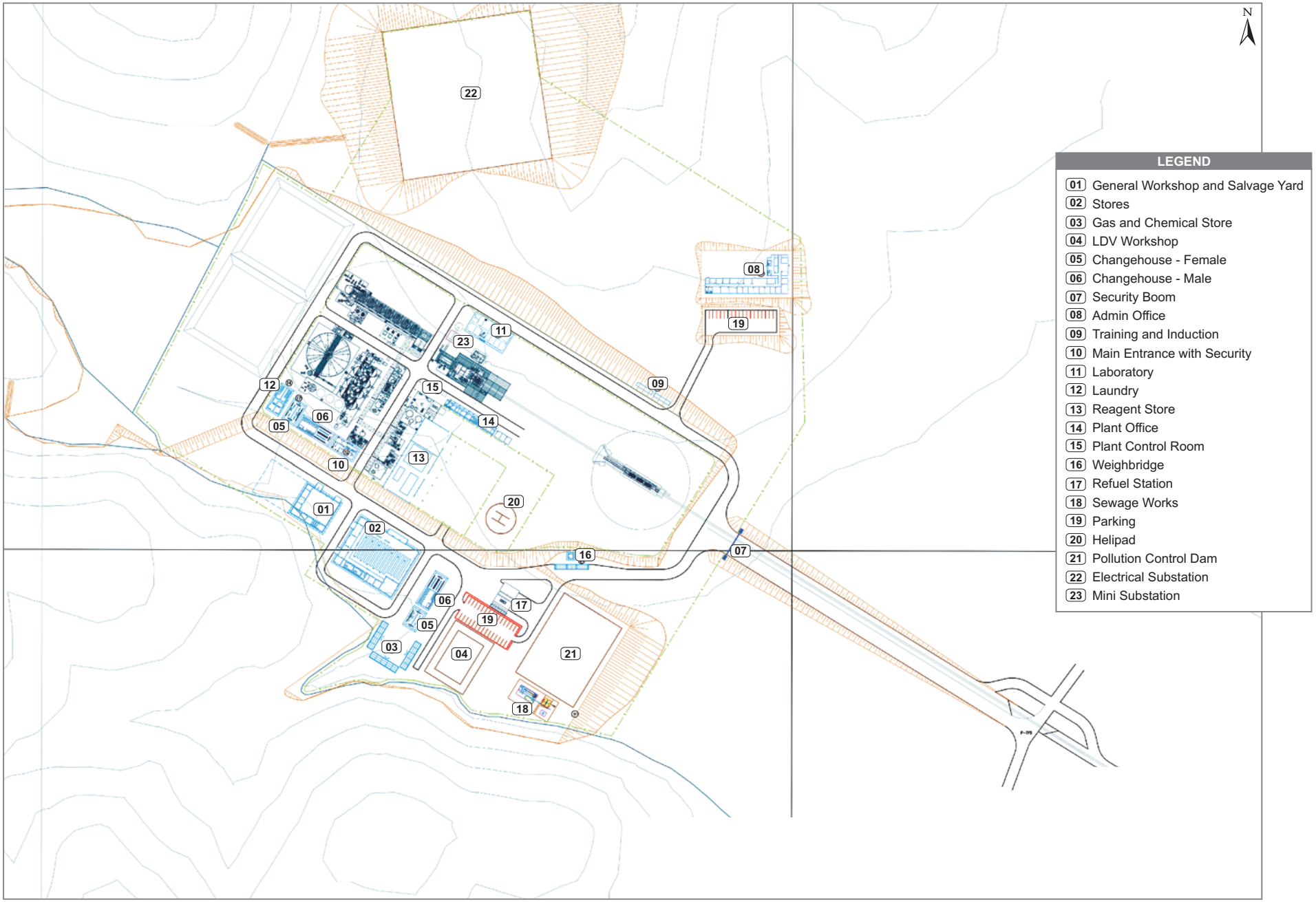
Asanko Gold

Figure 15

ESAASE PROCESS PLANT LAYOUT PLAN



Source: Minxcon 2012, Asanko Gold 2013

D1237_DRAKeeganResources_2013

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Asanko Gold

Figure 16

17.3. Esaase Project Power Supply

Two options exist for the power supply to the proposed Esaase mine, namely an off-grid connection to the national state utility company Gridco or IPP supply. The 2013 PFS was initially based on the national power supply option, however the capital cost and time to construct the GridCo connection was considered excessive, in addition to concerns regarding the reliability of supply and constrained capacity in Ghana. Asanko Gold is undertaking a study on alternative sources of electrical power with the purpose of completing a trade-off study.

17.3.1. National Power Grid Supply

The Ghanaian utility company, GridCo, prepared a feasibility study in January 2012 for a 34MW point of supply, based on a 7.5Mtpa production, which would include a compensator to help stabilise the supply at a total cost of USD32m. The cost may be significantly reduced if a reduced maximum demand is applied and by redesign of the connection by placing a new 161 kV substation on the main line which would utilise a single 11km 161 kV overhead line to feed the mine. The new 161kV substation would be located between Kenyase and Obuasi sub-stations and would fall under the auspices of GridCo and the battery limit between GridCo and Esaase would be the 11kV terminating structure from GridCo.

The mine power supply study is based on the reticulation of power across the mine site at 11kV to be from the grid via the mine consumer sub-station, located adjacent to the plant. Based on a mine production level of 5Mtpa, the total load presented to the grid is estimated to be:-

- connected: 34.1MVA (28.0 MW); and
- running: 23.6MVA (19.3 MW) – no power factor correction (19.7MVA (19.3 MW) – power factor corrected to 0.98.

The conceptual designs for the medium voltage power supply, 11kV switchgear, 6.6kV switchgear, sub-station, fire detection systems, transformers, motor control centres, earthing, control and instrumentation and lighting have all been undertaken for the PFS and are reported in detail in the DRA PFS study document. The beneficiation plant will be fully automated by use of Programmable Logic Controllers located in the Motor Control Centre via Supervisory Control and Data Acquisition (SCADA) system.

17.3.2. Independent Power Producer

The costs of the Gridco option for power supply to the proposed Esaase mine are high and Asanko Gold is investigating various alternative options. American based company USP&E Global, which supplies heavy fuel oil (HFO) power generators, diesel and natural gas turbines and renewable power plant installations, has provided a quotation for the supply, installation and commissioning of an on-site power plant.

The proposed power plant will comprise a 14MW reciprocating engine powered by either heavy fuel oil (HFO) or diesel and an 8.7MW heavy fuel oil power plant. The estimated capital expenditure is USD22.5m including shipping, installation and commissioning, with an operating cost of USD0.15/kWh. The unit is complete with all operating and support auxiliaries required for normal operation, including cooling radiators and a climate controlled control room with paralleling switchgear, power station controls, and unit automation systems.

17.4. Esaase Project Water Supply and Management

The meteorological and hydrological studies undertaken for the 2013 PFS water management, supply and water balance designs are presented in Table 39. The water management system is divided into four categories namely, process water, potable water, sewage and storm water run-off.

The water balance model is based on a three tier approach whereby:-

- Level 1 is a closed circuit for beneficiation process water and the TSFs, with make-up water supply by pit dewatering. Both TSFs have limited water storage capacity. The maximum F-TSF storage capacity is 300,000m^3 and the L-TSF storage capacity is 400,000m^3 at year 12;

- Level 2: on site water capture and storage; and
- Level 3: rainfall within the mine catchment that can released to the Bonte river.

The lined TSF for the CIL tailings stream is to be used for temporary start up storage and a process water buffer dam have been designed to capture F-TSF decant to avoid discharge of high arsenic concentration process water to the environment under worst-case wet scenario conditions. Should water treatment ever be necessary, this would allow the treatment to be done on a reduced, but sustained process. If the open pit discharge water and run-off waters from waste rock dumps and other facilities, are less than 0.1 mg/l As these waters will only be introduced into the process circuit if there is a water shortfall. Sediment control and other controls will be undertaken outside the TSF's and outside the process circuit. For water balance modelling purposes, the overall catchment area of the L-TSF and buffer storage catchments is assumed to be $2km^2$.

Make-up water sources to supplement for the losses in the water circuit include:-

- raw water from the Bonte river at the Mpatoam weir when the Bonte river is flowing sufficiently to support it. The net affect will be a prolonged period of comparatively low flow into the Bonte river downstream of Mpatoam weir;
- dewatering boreholes located in low lying areas adjacent to the Main and South Pits. Approximately 80,000 m^3/m is anticipated to be delivered to the plant raw water tank for use as lime free water increasing to 140,000m^3/m as the pit develops; and
- collected run-off from mine affected areas including pit dewatering, waste rock dumps, RoM and plant terraces and other "dirty" areas pumped to the plant process water dam to be routed through sediment control facilities and only introduced into the process water circuit if there is a shortfall.

The water balance includes the following:-

- 1.4Mm^3 start-up storage for commissioning in a drought, stored in the lined L-TSF and supplied from the Mpatoam weir and Bonte river in the wet season;
- 300,000m^3 storage capacity retained within the F-TSF;
- construction water will be supplied from the existing "Main" Dam situated on the edge of the proposed Main Pit;
- the process circuit will be a "no release" facility and under the wettest sequence of years modelled, the peak storage of the TFSs and buffer dam is approximately 3,0Mm^3. The clay lined buffer dam is sized to contain 3,0Mm^3 at full supply level and is configured as a return water dam collecting decant from the F-TSF and sized to address the driest/wettest conditions as a final point of storage to prevent discharge of contaminated process water exceeding the discharge standard. A water treatment plant may be installed to remove salts prior to discharge to the Bonte river;
- at steady state production the total water in slurry discharged to the mine residue disposal facilities will be 920m^3/hr of which 60% to 90% is returned to the plant process water circuit;
- it is expected that the operations will recover make-up water from discharge from the pit, and from drainage through Mpatoam weir;
- at an arsenic threshold of 0.1mg/l, Esaase mine can release water from most mining areas (with the exception of drainage from the RoM pad) without having to deal with metals levels. Arsenic content of waters from mine affected areas are anticipated to rise from 0.04mg/l to 0.06 g/l over the LoM ;
- the mine will eventually be a net water positive situation. The average release rate over the LoM is 25l/s, but there is no release for nominally 6 months of the year, and peak release rates are as high as 800,000m^3/month, or 300l/s over a month); and
- no mine closure conditions are modelled in the current water balance.

Potable water demand during construction and steady state production will reach a maximum of 168m^3 per day to be supplied by two boreholes supplying 3l/s each. The groundwater will be stored in a potable water storage facility which will include a chlorination plant.

Sewage will be treated in vendor package treatment plants at the process plant and accommodation camp. Waste water loading will be a maximum of 160m^3 per day during construction and steady state production.

17.4.1. Surface Water Management

The surface water management system for the Project will consist of two separate systems namely, a clean water diversion system to control the uncontaminated run-off from the higher lying natural environment, and a dirty storm water system to capture the contaminated storm water from plant, operational and processing areas. Water accumulated within the clean water system will discharge towards the natural watercourses. The dirty water system will be captured in process water ponds for potential reuse in the process water system.

The surface water management system will consist of open concrete lined and/or pipe systems, linked to lined and unlined dams and sediment control ponds. The system will also include diversion channels, whose function is to minimise risk to infrastructure and personnel by diverting flows away from critical points. The design elements include capacity to accommodate 1:50 24hr flood event. Road crossing for the plant access road is sized for the 1:100 year 24hour flood event whilst other road river crossings will be designed to avoid overtopping in the 1:50 year 24hour flood event. Storm water channels draining roads will be sized for a 1:10 year 24hour event and river diversions and cuttings will be designed to convey the 1:100 year 24hour event away from the open pit.

17.5. Tailings Storage Facility Design

The design of the mine residue disposal facilities for the 2013 PFS was undertaken by independent design specialists Epoch Resources. The design incorporated two facilities namely a storage facility for the tailings arising from the gravity-flotation circuit of the beneficiation plant (F-TSF) and the facility design to accommodate the tailings from the CIL circuits of the process plant (L-TSF). The study was undertaken to an accuracy of approximately +20%/-5% for the capital costs and an accuracy of approximately 35% for the operating costs.

The F-TSF comprises an in situ clay compacted, fully-contained, valley dam, ring-dyke facility catering for a depositional tonnage rate of 4.5Mtpa over an 11 year LoM located on the west of the Bonte river valley as illustrated in Figure 15. The costings for the F-TSF were optimised by relocation of the facility from the original location to a valley directly adjacent to the L-TSF which in the base case site plan was originally a waste rock dump site. The optimised design comprises an embankment to be constructed out of waste rock in four, phased, downstream wall raises to a maximum height of 65.0m. The F-TSF has a total footprint area of 196.2ha and a maximum elevation of 315.0mamsl. Removal of tailings supernatant water and storm water accumulating on the F-TSF is to occur through a penstock decant system to a drainage collector sump from where the water is pumped back to the plant for re-use as process water or discharged into the buffer dam.

The tailings facility for the CIL circuit comprises a high density polyethylene (HDPE) lined, fully-contained, valley dam L-TSF catering for a depositional tonnage rate of 0.5Mtpa over an 11 year LoM. The embankment is to be constructed out of waste rock in two, phased, downstream wall raises to a maximum height of 42.4m. Furthermore, a 5.0m wide compacted clay skin is to be constructed on the upstream face of the waste rock wall in order to provide a protective bedding layer for the HDPE liner. The L-TSF has a total footprint area of 49.0ha and a maximum elevation of 300.0mamsl. Since the L-TSF is to operate as a zero discharge facility, removal of tailings supernatant water and storm water accumulating on the L-TSF is to occur through a penstock decant system with all water returned back to the beneficiation plant for use as process water.

17.6. Roads and Buildings

The Project site is accessed by existing public roads from two directions, namely via the tarred Kumasi-Sunyani road to the north-east and the tarred Kumasi-Obuasi road to the south. Roads from both directions become gravel topped (fair to poor condition) for the last 20km. Within the project area, the existing public road is the principal transport and logistics route for light mine vehicles. Access to the plant area, the accommodation camp and construction camp is from this public road.

The plant area has a 250m long, purpose built multi-service access road corridor which crosses the Bonte river. Along with vehicular traffic, this corridor includes both RoM conveyor and pipeline servitudes. The existing main road is to be realigned along the old Bonte mining road and suitably upgraded to ensure safety and usability for locals and light mine vehicles (Figure 15). The heavy vehicle mine haul road and public road will be in close proximity with a vegetative screen to limit dust between the two roads. The re-alignment of the public road will require the upgrading of approximately 3,420m of existing Bonte mining road with a cross-section width of 7m. 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. The roads will be constructed from selected waste rock material.

The site buildings have been designed for the 2013 PFS and include prefabricated modular buildings for the main administrative buildings, clinic and training centre and accommodation.

18. Market Studies and Contracts

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18.1. Market Studies

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The only commodity anticipated to be produced at the proposed Esaase mine is gold, which is widely and freely traded on the international market with known and instantly accessible pricing information. The basis gold price for the 2013 PFS was USD1,400/oz which is an important factor in estimating the project Net Present Value (NPV) and Internal Rate of Return (IRR).

The USD1,400/oz price was selected during the mine design phase of the PFS when the spot gold price was in excess of USD1,600/oz with a three year trailing average of USD1,570/oz.

The forecast gold price as per the “CIBC 25 broker consensus” dated June 2013 for first two years of Esaase mine operations (2015 – 2016) are USD1,568/oz and USD1,529/oz respectively.

18.2. Material Contracts

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The Ghanaian government has a standard 10% free carried interest in all permits within the country but this interest only comes into effect once exploitation and production commences. The Minerals and Mining Act 2006 also makes provision for a royalty on mining revenue which was increased in 2010 to a maximum of 5%. In addition, the mining leases contain provision for a 0.5% royalty payment to the Bonte Liquidation Committee.

At the current level of study no formal contracts with contractor mining companies, power supply manufacturers, equipment suppliers or civil engineering contractors have been

19. Environmental Studies

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An Environmental and Social Pre Feasibility Study (Environmental and Social Report-ESR) for the Esaase Project was undertaken independently of Asanko Gold by Epoch Resources Proprietary Limited (Epoch). The results of the study were reported in a document entitled “Environmental and Social Pre-Feasibility, Esaase Gold Project, Ghana – Asanko Gold” (April 2013 Ref no. 127-038). The study was independently reviewed by Venmyn Deloitte for the NI 43-101 ITR.

The Venmyn Deloitte review examined the ESR results in the context of the specifications of Ghanaian legislative requirements, and internationally accepted standards of practice, including the Equator Principles and the International Finance Corporation (IFC) Performance Standards. The study provides the current status of environmental studies for the Project, but it is important to note that at this stage not all studies and licenses will have been completed or obtained. The environmental specialist has detailed recommendations as to specialist studies and licensing activities that will be undertaken in the near future, in accordance with the environmental licensing process in Ghana.

Environmental Due Diligence necessitates the examination of both international and in-country processes and requirements. The current environmental landscape in Ghana requires that prior to the Project advancing to development stage authorisations in terms of National Ghanaian legislation are required (see Section 19.1). In support of these authorisations, a scoping report was submitted to the Ghana Environmental Protection Agency (EPA) and upon approval, it is anticipated that the Environmental Impact Statement (EIS) report will be submitted to the EPA for approval of the scoping report in July 2013.

In addition to complying with the relevant Ghanaian laws and regulations, Asanko Gold will be guided and informed by a number of international guidelines, including the Equator Principles and the IFC Performance Standards.

19.1. Legislative Requirements

Ghanaian environmental, social and mining related legislation is applicable to the proposed Project, with the legislation and authorisations applicable to the Project listed below:

- an environmental permit through the approval of an EIS in terms of the Environmental Assessment Regulations 1999 (L.I. 1652), in compliance with the Environmental Protection Agency (EPA) Act (1994);
- a mining permit in terms of the Minerals and Mining Act (2006); and
- a number of water use permits in terms of the Water Resources Commission Act (1996).

19.1.1. Equator Principles

Banks in the project finance sector and the IFC have developed a common and coherent set of environmental and social policies and guidelines called the Equator Principles that are applied globally and across all industry sectors. The Equator Principles (WBG, 2006) apply to all new project financings with total project capital costs of USD10m or more. There are currently a total of 72 Equator Principles Financial Institutions (EPFI), that will only provide loans to projects where the borrower complies with these principles. Many other banks, while not signatories, use these Principles in assessing projects. The ten equator principles are as follows:-

- Principle 1: Review and Categorisation - Projects are classified according to social and environmental impacts, in Category A (significant impacts), Category B (limited impacts) and Category C (minimal or no impacts);
- Principle 2: Social and Environmental Assessment - For Category A and B projects, sponsors complete an Environmental Assessment;
- Principle 3: Applicable Social and Environmental Standards;
- Principle 4: Action Plan and Management System;
- Principle 5: Consultation and Disclosure;
- Principle 6: Grievance Mechanism;
- Principle 7: Independent Review;
- Principle 8: Covenants;
- Principle 9: Independent Monitoring and Reporting; and
- Principle 10: Equator Principle Finance Institutes (EPFI) Reporting.

19.1.2. IFC Performance Standards

The Esaase EIS will make reference to, and aim to comply with the applicable IFC Performance Standards and the applicable general and industry specific Environmental Health and Safety (EHS) Guidelines. The relevant IFC Performance Standards are:

- Performance Standard 1: Social and Environmental Assessment and Management System;

- Performance Standard 2: Labour and Working Conditions;
- Performance Standard 3: Pollution Prevention and Abatement;
- Performance Standard 4: Community Health, Safety and Security;
- Performance Standard 5: Land Acquisition and Involuntary Resettlement;
- Performance Standard 6: Biodiversity Conservation and Sustainable Natural Resource Management;
- Performance Standard 8: Cultural Heritage; and
- Performance Standard 7, (Indigenous Peoples) is not applicable to the project, as no indigenous people, as per the IFC's definition, occur within, or in close proximity to the project.

The latest versions of the World Bank Group Environmental, Health and Safety Guidelines (EHS Guidelines) have been compiled by the IFC and are applicable from 1 January 2012. The EHS Guidelines are technical reference documents with general and industry-specific examples of Good International Industry Practice (GIIP). According to IFC requirements, where Ghanaian regulations differ from the levels and measures presented in the EHS guidelines, the Esaase EIS is required to apply, whichever is more stringent. The relevant IFC Industry sector guideline relevant to Esaase Project is the Environmental Health and Safety guidelines for mining.

19.1.3. Project Categorisation

As part of the review of a project's expected environmental and social impacts, Equator Principle Finance Institutes use a system of social and environmental categorisation, based on the IFC's environmental and social screening criteria, to reflect the magnitude of environmental impacts understood as a result of assessment. These categories are:

- Category A – Projects with potentially significant adverse social, or environmental impacts that are diverse, irreversible, or unprecedented;
- Category B – Projects with potential limited adverse social, or environmental impacts that are few in number, generally site specific, largely reversible and readily addressed through mitigation measures; and
- Category C – Projects with minimal or no social or environmental impacts.

Due to the anticipated irreversible impacts on the existing biophysical and social environment, the Esaase Project is classified as a Category A project.

19.2. Stakeholder Consultation

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Formal consultation and stakeholder engagement regarding the Project have been on-going since 2007. Consultation and disclosure of information involved the relevant national, regional and local authorities, registered interested parties as well as directly affected individuals:-

- contact with various government ministries, departments and agencies;
- ongoing efforts by Asanko Gold to engage with and educate affected people in the Project area; and
- the formal ESIA Process, (including Public Hearings).

The Esaase Project implemented various engagement and consultation mechanisms through the development and implementation of a Stakeholder Engagement Action Plan, the creation of Community Liaison Committees in a number of the surrounding villages, as well as ongoing monthly meetings with surrounding communities.

19.3. Baseline Environment Studies

The pre-mining physical, biophysical and social environment has been characterised to, in conjunction with public consultation, identify possible environmental and social impacts that may arise due to the proposed Esaase mining project. Baseline and specialist studies have focused on the following project aspects:-

- **climate and air quality**; the Project area is located within the wet semi-equatorial climatic zone of Ghana, characterised by an annual double-maxima rainfall pattern. Air quality is expected to meet Ghana EPA Guidelines with the exception of particulate (dust) during the Harmattan period.

  Motor vehicles are the primary source of dust and gaseous pollutants in the study area, but the generally low numbers of vehicles would indicate these are also a minor source of gaseous pollutants. During certain times of the year, crop field burning is a source of gaseous pollutants;

- **topography and visual**; the area is undulating with an elevation of 260mamsl in the lowlands. The Project site is located in the west-central portion of the Pra river basin, specifically in the headwaters of the Bonte river drainage, which flows into the Gyeni river and then the Offin river The Project area is bounded on both sides by steep hills that reach elevations of approximately 450masl;

- **noise and vibration**; noise generated by existing activities within the Project area exceed the daytime residential requirement of 55 dBA. At night, only the average data for Tetrem (52 dBA) exceeded the night time 48 dBA requirement;

- **biodiversity (fauna, flora, and aquatic environments**); the Project encompasses approximately 17,547ha of forest reserves, remnant patches of forest maintained as sacred groves, land disturbed by past mining, wetlands and surface water features, villages and hamlets (286ha), and land used as cropland. Apart from the remnant forest patches, biodiversity within the Project area has been altered by crop production, timber removal, and mining. No flora species of conservation concern were recorded within the proposed mining area but seven fauna species are protected in Ghana under the First Schedule, Ghana Wildlife Conservation Regulations (1971). Species of conservation concern identified on the IUCN Red List include: tree pangolin (Near Threatened), giant pangolin (Near Threatened), black duiker, Maxwell's duiker, bay duiker, royal antelope, and red river hog (Lower Risk/Near Threatened); lesser spot-nosed monkey, and Lowe's mona monkey (Lower Risk); olive colobus monkey (Near Threatened); and Geoffroy's pied colobus (Vulnerable);

- Approximately 229 bird species recorded of which 120 species are "biome restricted" to the Guinea-Congo Forest of West and Central Africa and 26 "wholly protected" in Ghana under Schedule I of the Ghana Wildlife Conservation Regulations of 1971. Wholly protected species include the crowned eagle and Cassin's hawk-eagle, recorded in the Gyemera and Gyeni River forest reserves; the long-tailed hawk, in the Okyem Kwaye; and the African hobby in the Prako Kwaye and surrounding area.

- **hydrology and hydrogeology**; the results of the surface and groundwater monitoring and sampling programme are summarised in Table 40. The EPA's standard for arsenic concentration for the discharge of effluent into the environment is 0.1 mg/l.

- **heritage, cultural and archaeology**; 21 sites of archaeological or heritage importance were identified within the Esaase concessions area;

- **socio-economic**: population within the Project area ranges from 594 residents in Manhyia to 4,793 residents in Mpatoam. The population of the entire study area is projected to be 18,711 people; including just over 14,000 in the Project area and 4,645 in the larger area of influence. Approximately 49% of the studied population is male and with 60% of the surveyed population is under the age of 26.

- The majority of household heads (89%) identified their ethnicity as Ashanti. Approximately 53 % of adults were reported to be literate

The main land uses in the area include subsistence farming, settlements, small scale and non-permitted mining, infrastructure and community graveyards. The main economic activity in the Study Area is cocoa production followed by galamsey and trading. Two-thirds of all households use electricity as their main source of lighting, while one-third use kerosene or battery-operated/rechargeable lanterns. Access to drinking water from an improved source (i.e., private or public tap/borehole) was available for all households surveyed.

Table 40 : Hydrological and Hydrogeological Study Results

PARAMETER	UNIT	VALUE
Surface Water		
Electricity conductivity	milliSiemens per cm (mS/cm)	0.16
Mean Total dissolved solids-Bonte River	TDS mg/L	200
Mean Total dissolved solids-Offin River	TDS mg/L	145
Mean pH		7.3
Mean alkalinity	mg/L	60
Mean sulphate	mg/L	5
Range sulphate	mg/L	1 to 70
Mean chloride	mg/L	13
Turbidity	TSS total suspended sediment mg/L	1 to 30,000
Turbidity	NTU nephelometric turbidity units	2 to >999
Median turbidity	TSS total suspended sediment mg/L	25
Arsenic-Bonte River	mg/L	<0.001 to 0.007
Arsenic-Offin River	mg/L	<0.001 to 0.002
Ground Water		
Ground water follows topography		
Total coliforms and ecoli	CFU/100mL	2,420 and 360
Coliforms and ecoli (see legend below)	present in X of XX boreholes	in 12 of 63 MW; 39 of 197 HP; 18 of 21 CBh
Electricity conductivity	milliSiemens per cm (mS/cm)	0.147 in MWs, 0.429 in CBhs and 0.213 in HPs
Mean pH		7.0 for MW and CBh ; 6.0 for HB
Total dissolved solids	TDS mg/L	254 in MW; 297 in CBhs; 196 in HP
Mean alkalinity	mg/L	184 in MW; 160 in CBhs; 90 in HP
Mean sulphate	mg/L	10 to 18
Mean chloride	mg/L	14 to 26
Mean arsenic	mg/L	0.68 to 0.80 for MW; 0.10 to 0.010 for CBhs; 0.004 HP
Max arsenic	mg/L	0.598 for MW; 0.043 for CBhs; 0.041 for HP

Source : Epoch 2013
MW – monitoring wells
CBh – community borehole
HP – hand-pump wells

19.4. Key Environmental Risks Identified and Mitigation

Various potential positive and negative environmental impacts have been identified that will need to be assessed during the Environmental Impact Assessment (EIA) process. The following key risks have been identified:-

- the potential impacts on both the quality and quantity of surface and ground water due to construction, operational and post mining activities;
- community health and safety due to an anticipated increase in traffic on the existing road network, blasting activities and the proximity of various communities to the proposed mining area;
- noise and lighting impacts on nearby communities due to construction and operational mining operations;
- the impact of increased levels of dust on communities due to opencast mining, haul roads, blasting and an increase in traffic;
- community health and safety, air quality and noise related impacts associated with the construction of deflection berms associated with the mine residue disposal facilities;

- negative impacts of a potential influx of people from outside the Project area on the health and safety of the current social environment, as well as the additional pressure on existing social and service infrastructure within the Project area; and
- livelihood compensation and restoration due to various fields and crops being located within the proposed mining area.

19.5. Mine Closure

A conceptual Rehabilitation and Closure Plan for the proposed Esaase mine and its associated infrastructure was undertaken by DRAM for the 2013 PFS. An estimate of the required financial provision to ensure that the plan can be implemented was required and the information used in the formulation of the conceptual closure and costs was sourced from the ESR and also the PFS design of the mine and associated infrastructure.

In the planning and implementation stages of a mining project the focus of rehabilitation and closure planning is to ensure that:-

- the proposed post closure land use(s) for the site are defined and agreed with the regulatory authorities;
- the nature, scale and cost of the works required to return the site to a condition consistent with the requirements of the post closure land use(s) are defined and understood;
- the necessary financial provisions are made for closure of the mine and that these are included in the assessment of the project viability;
- a plan is developed for the implementation of the rehabilitation and closure works so as to ensure that the process proceeds concurrently with mining operations as far as possible; and
- the build-up in rehabilitation and closure liabilities is limited so as to mitigate as far as possible the impacts of premature, or unplanned closure.

The proposed Esaase mine will comprise a conventional open pit operation with a LoM of approximately 11 years. The objective of the rehabilitation closure process is to restore as much as possible of the area disturbed during the operation of the mine to a land use as close as possible to that previously practiced before mining operations. Natural soil covers and vegetation will as far as possible be re-established over these areas but access by humans and/or livestock will be discouraged due to potential health and safety concerns. Examples of possible exclusion zones would include the high wall of the final void of the open pit and the final void itself. Rehabilitation and closure of areas disturbed in mining and related operations will be considered to be complete when:-

- all structures, equipment and infrastructure not consistent with the post closure land use have been decommissioned, demolished and removed from site;
- ownership of all remaining infrastructure and services required to support the proposed post closure land use have been formally transferred to the local authority responsible for the administration of the area;
- the area has been made safe for all post closure land users and livestock;
- all surface disturbances and remaining landforms are structurally and ecologically stable and have sustainable soil and vegetation covers;
- surface water management structures are in place and are free of damage due to erosion; and
- all surface and groundwater discharges from the site satisfy agreed target water quality objectives.

The conceptual rehabilitation and closure plan and associated estimate of closure costs for the mine has been formulated to ensure that the completion criteria as defined above are achieved.

The estimated costs of rehabilitation and closure presented in Table 41 have been structured so as to distinguish between concurrent rehabilitation of the mine and the works required at closure. The cost of rehabilitation for the North Pit, has been excluded.

Table 41 : Mine Closure and Rehabilitation Estimated Costs

PROJECT COMPONENT	PROGRESSIVE REHABILITATION	DECOMMISSIONING	REHABILITATION	AFTERCARE AND MAINTENANCE	TOTAL (USD)
CIL plant		719,177	415,467	27,698	1,162,342
Main Pit	664,176	350,000	167,015	11,264	1,192,455
South Pit	1,046,262	300,000	253,433	14,090	1,613,785
Waste Dump	3,913,093		1,272,311	91,553	5,276,957
Waste dump	1,841,642		621,122	45,259	2,508,023
Flotation TSF	1,038,960	517,901	4,041,102	253,682	5,851,645
L-TSF	205,864	228,699	1,397,059	87,414	1,919,036
Sub-total	8,709,997	2,115,777	8,167,509	530,960	19,524,243
Contingency 10%					1,952,424
Contractors preliminary costs 30%					6,443,000
Permitting and Management 7.5%					2,093,975
TOTAL					**29,57,000***

Source : Epoch 2013

*Computation inconsistency due to rounding

20. Capital and Operating Costs

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20.1. Capital Expenditure

The Esaase Project capital expenditure (capex) estimation is derived from the studies on mining, processing, mine infrastructure, TSF designs, dam construction, electrical supply, owner's costs and indirect costs. The capital costs for the mining operation, process plant and TSFs are provided in Sections 20.1.2, Section 20.1.3 and Section 20.1.3 respectively. The summary capital cost estimates provided in Table 42 apply to initial capital requirements only and the following are excluded (Table 42):-

- escalation;
- operating costs;
- replacement capital;
- stay in business capital;
- closure costs; and
- taxes, tariffs and duties.

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 3, 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 preliminary and general (P&G) estimates for earthworks, dam construction, mining, and steel construction have been allocated between 0% and 40%. 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.

Table 42 : Initial Esaase Project Capital Expenditure Estimates as at March 2013

PROJECT COMPONENT	DETAILED COMPONENT	COST (USDm)	COMMENT
Plant	Capital costs	82.90	
	EPCM	10.30	
	Sub-total	93.20	
Infrastructure	Mining	13.70	Mobilisation top soil stripping Year 1 only
	Alternative Power Supply	22.50	Heavy Fuel Oil (HFO) power plant
	MV Reticulation	8.70	Excluding plant
	Float TSF	14.10	1st lift
	Cyanide TSF	12.60	1st lift incl. HDPE liner
	Contractors Camp	9.20	600 people including contractor camp/terrace
	General	15.40	Earthworks, buildings, roads, store, water
	P & G's	14.80	
	Sub-total	111.00	
Indirect costs	Allowance for potential crop compensation	17.00	
	Provision for community disruption	12.00	
	Social responsibility	1.00	
	Owners costs	9.50	
	Consumables	3.00	
	Balance of EPCM	10.30	
	General	3.50	Spares, Vehicles
	Sub-total	56.30	
Sub-total		**260.50**	
Contingency		26.00	
TOTAL		**286.50**	

Source : DRA 2013 and numerous specialist studies

An estimate of the additional capex required over the LoM is USD95.2m and includes additional stage lifts to the mine residue disposal facilities, buffer dam construction, top soil stripping, secondary crusher and mine rehabilitation and closure. The total Project capex over the LoM is estimated to be USD381,7m.

20.1.1. Mining Capital Expenditure Estimate

The DRAM 2013 mining study provided the capital estimates for the mining operation at the proposed Esaase mine. The estimate includes mobilisation and site establishment, topsoil stripping and site clearing and demobilisation. The pre-production including mobilisation and establishment, and topsoil stripping is estimated at USD13.70m. In Year 12 a demobilisation cost of USD0.125m is allocated to the mining capex.

20.1.2. Process Plant Capital Expenditure

A summary of the capital expenditure on the process plant is presented in Table 43:-

Table 43 : Process Plant Capex

PLANT CONSTRUCTION	COST (USD)
Earthworks and general	4.80
Civils	5.90
Structural steel and platework	14.40
Mechanicals	35.80
Piping and valves	5.20
Electrical and instrumentation	12.80
Transportation	4.00
Sub-total	82.90
EPCM	10.30
TOTAL	**93.20**

Source : DRA 2013 and numerous specialist studies

20.1.3. Mine Residue Disposal Facilities Capital Expenditure

The initial optimised capital costs for the mine residue disposal facilities are estimated at USD34.63m. In addition the on-going capital costs are USD44.23, resulting in a total LoM capital expenditure of USD78.86m. The pre-optimisation total capex was USD86.98m and the optimised operating costs are USD1.17m/a.

20.2. Operational Expenditure

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, tailings storage, water treatment, general and administrative and refining costs. Opex estimates for the mining operation and process plant are presented in Sections 20.2.1 and Section 20.2.2 respectively.The average LoM cash operating cost is estimated at USD736/oz Au based on the treatment of 5Mtpa producing an average of 200,000oz/a.

Table 44 : Operating Cost Estimates Esaase Project as at June 2013

PROJECT COMPONENT DESCRIPTION	OPEX (USD/oz Au)	OPEX (USD/tonne milled)
Waste mining	300.26	12.24*
Ore mining	113.84	4.64
Processing	254.19	10.37
General and administrative	62.14	2.53
Refining	5.35	0.22
Total cash operating costs	735.78	30.00
Royalties	77	3.14
Total cash costs	812.78	33.14
Sustaining and deferred capex	30.36	1.24
Pre-tax sustaining cash cost	843.14	34.38
Taxation	146.71	5.99
Post tax inclusive sustaining cash cost	**989.85**	**40.37**

Source : DRA 2013

*Applying the LoM stripping ratio of 4.28

20.2.1. Mining Operational Expenditure

Asanko Gold selected to conduct the PFS on a base case utilising contract mining as opposed to owner operations. DRA obtained four independent contract mining company quotations for the 2011 PFS which were escalated for inflation to 2013 and normalised for comparative purposes. The average mining cost obtained from the quotations was USD3.55/t and ranged USD3.39/t to USD3.70/t.

An initial conceptual Project mining opex figure of USD3.23/t was obtained by combining DRA's database of costs, for similar operations and average costs from the quotations obtained. Mobilisation, site establishment, demobilisation, clearing and topsoil stripping costs formed part of capital expenditure and thus, not part of operational expenditure. As the PFS progressed and more detail became available, the opex was optimised and refined but these optimisations will be included in the DFS. The optimisations included repositioning of the RoM pad and waste dumps, and refined reference bench costs.

The final mining opex over the LoM is summarised in Table 45:-

Table 45 : LoM Mining Operational Expenditure

	ORE	WASTE
Tonnage (Mt)	52.34	224.25
Total costs (USDm)	243.09	641.16
LoM costs (USD/t)	2.86	4.64

Source : DRA 2013

20.2.2. Process Operating Costs

The opex for the processing plant is summarised in Table 46:-

Table 46 : Process Plant Operating Costs

	FIXED COSTS (LoM Blend)		VARIABLE COSTS (LoM Blend)		TOTAL PROCESSING COST	
	(USD)	(USD/t)	(USD)	(USD/t)	(USD)	(USD/t)
Labour	31,548,297	0.60	0.00	0.00	31,548,297	0.60
Laboratory	5,606,201	0.11	1,889,334	0.04	7,495,535	0.14
Consumables			272,834,578	5.21	272,834,578	5.21
Power			199,087,726	3.80	199,087,726	3.80
Maintenance			22,662,541	0.43	22,662,541	0.43
Tailings			9,900,000	0.19	9,900,000	0.19
TOTAL	**37,154,498**	**0.71**	**506,374,179**	**9.67**	**543,528,677**	**10.37**

Source : DRA 2013

21. Economic Analysis

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Venmyn Deloitte constructed a Discounted Cash Flow (DCF) model for the purposes of the economic analysis of the Esaase Project. The DCF model was constructed in Excel and was based on input assumptions generated from the 2013 PFS mining schedule, processing schedule, operating costs and capital expenditure estimates. Venmyn Deloitte received input from DRA and Asanko Gold on the timing of the various inputs, including working capital requirements. The DCF model assesses the post-tax real cash flows for the Project.

21.1. DCF Model In-put Parameters

The main economic, production and processing assumptions made in the construction of the DCF model, are summarised in Table 47 and are discussed in more detail in Section 20. The DCF model assumes that both revenue and expenses, as well as royalty and taxes, are incurred in USD, therefore, no exchange rate assumptions are necessary. The 2013 PFS was undertaken at a gold price of USD1,400/oz and Venmyn Deloitte assumed this flat rate for all future revenue streams. The selected USD1,400/oz gold price is below the current 3 year average gold price of USD1,551/oz.

For the purposes of the economic analysis, Venmyn Deloitte used a discount rate of 5%. However, in the sensitivity analysis, the results are reported over a range of discount rates (Table 48).

Table 47 : DCF Economic In-put Parameters (Source : Venmyn Deloitte 2013)

DCF MODEL IN-PUT PARAMETER	UNIT	VALUE
Economic		
Corporate Tax Rate	%	35.00
Royalties Rate	%	5.50
Real Discount Rate	%	5.00
Gold Price	(USD/oz)	1,400
Realisation Cost	(USD/oz)	5.35
Payback Period (from production to capital payback)	(Years)	3
Production		
Total Ore Tonnes Mined	(Mt)	52.34
Stripping ratio	waste:ore	4.28:1
Average Grade	(g/t)	1.41
Life of Mine	(Years)	11
Gold Recoveries	(%)	90
Total Gold Recovered	(Moz)	2.14
Opex		
Ore Mining Cost	(USD/tonne mined)	4.64
Waste Mining Cost	(USD/tonne mined)	2.86
Processing Cost	(USD/tonne milled)	10.37
General and Administration	(USD/tonne milled)	2.53
Capex		
Initial Capex	(USDm)	260.50
Sustaining Capex	(USDm)	51.90
Deferred Capex	(USDm)	12.95
Closure Capex	(USDm)	29,60
Capex Contingency	(%)	10

21.2. DCF Model Results and Sensitivities

At a discount rate of 5%, the economic analysis indicates a positive Net Present Value (NPV) of the Esaase Project of USD354.7m. A range of project NPVs at various gold prices and discount rates is presented in Table 48:-

Table 48 : Economic Analysis of the Esaase Project at Various Discount Rates and Gold Prices

PRICE (USD/oz)	PROJECT NPV (USDm) at various discount rates (%)					IRR (%)
	3%	5%	6%	7%	8%	
1,100	95.07	50.75	31.24	13.29	-3.27	7.8
1,200	209.73	155.14	131.03	108.77	88.20	13.4
1,300	321.08	255.51	226.50	199.70	174.91	18.5
1,400	431.50	354.71	320.71	289.27	260.17	23.2
1,500	541.65	453.58	414.55	378.45	345.02	27.6
1,600	651.61	552.19	508.10	467.31	429.53	31.7
1,700	761.53	650.77	601.62	556.15	514.01	35.8

Venmyn Deloitte investigated the sensitivity of the project NPV to different input parameters (Figure 17). The NPV generated from the DCF model proved to be most sensitive to changes in parameters affecting revenue. A 10% change in revenue changes the NPV by 39.1%. The NPV is less sensitive to changes in operating expenditure (opex) with a 10% change in opex translating into a 21.9% change in the NPV. A 10% change in capital expenditure changes the NPV by 17.4% as illustrated in Figure 17.

At a 5% discount rate and all else being equal, the critical gold price which generates an NPV of 0 is USD1,049/oz. The gold price which generates an NPV of 0 at differential discount rates is shown in Table 49:-

Table 49 : Critical Gold Price at which NPV=0

DISCOUNT RATE (%)	GOLD PRICE AT WHICH NPV=0 (USD/oz Au)
5.00%	1,048.66
7.50%	1,091.13
10.00%	1,134.78
12.50%	1,179.64

Source : Venmyn Deloitte 2013

21.3. Conclusions from the Economic Analysis

Based on the forgoing economic analysis, Venmyn Deloitte concludes the following:-

- based on a 5% discount rate, the NPV of the Project is USD354.7m; and
- the Project is sensitive to changes in revenue with a 10% change in operating income resulting in a 39.1% change in the NPV.

22. Adjacent Properties

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The Esaase Project is located within the northern Asankrangwa belt, approximately 20km north-northwest of PMI Gold Corporation's Abore North deposit, located on the southern strike direction of the Kaniago West- Esaase Shear Zone. A series of parallel shears host numerous gold exploration projects in the region.

The Esaase Project is bounded in the north by forest reserve and in the west by African Gold Group Incorporated's Assuowunu exploration project. The exploration of the Assuowunu project has comprised soil sampling and airborne geophysical surveys. The initial results defined four anomalous zones in the silt sampling programme and the soil sampling program defined five gold-in-soil anomalous zones. A public domain map of the project (www.africangold.com) suggests a drillhole intersection of 1.2g/t over 8m width was obtained west of the defined Esaase Mineral Resource area.

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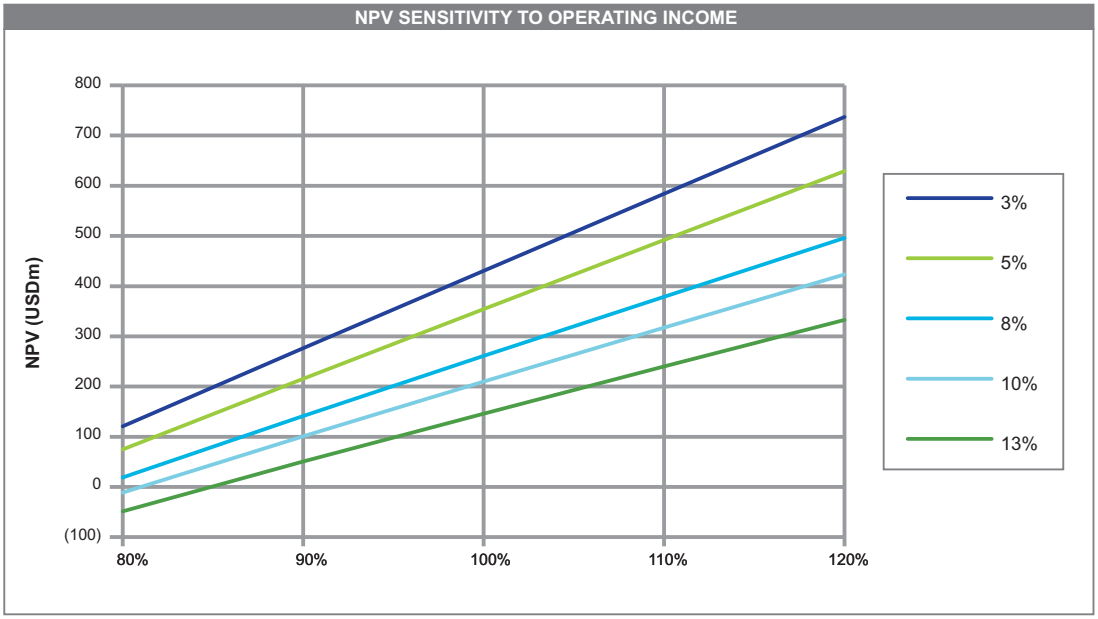
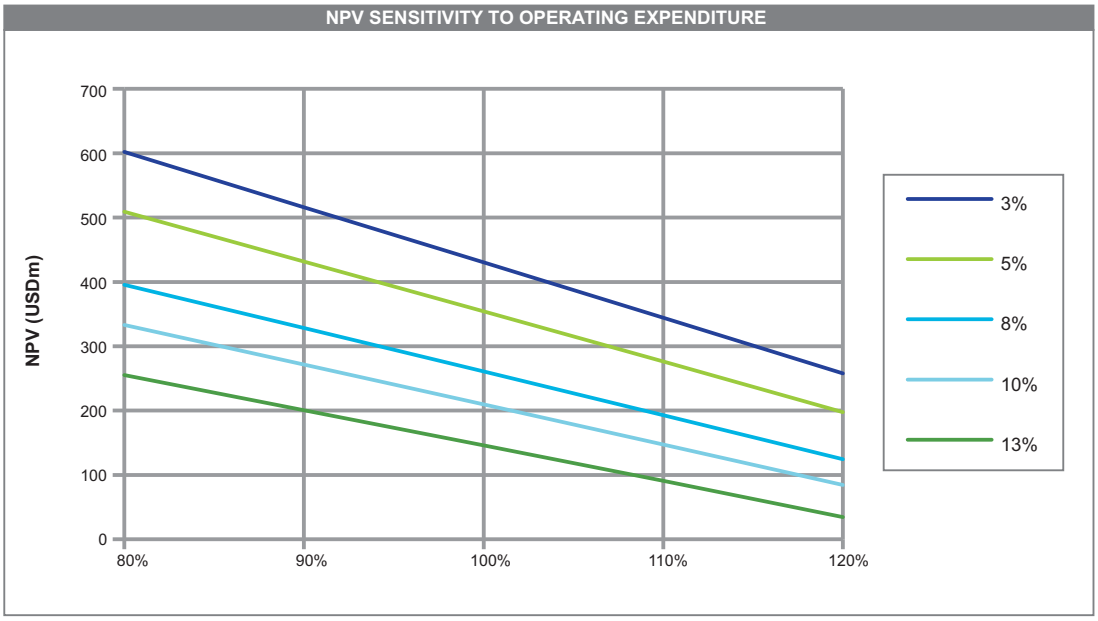
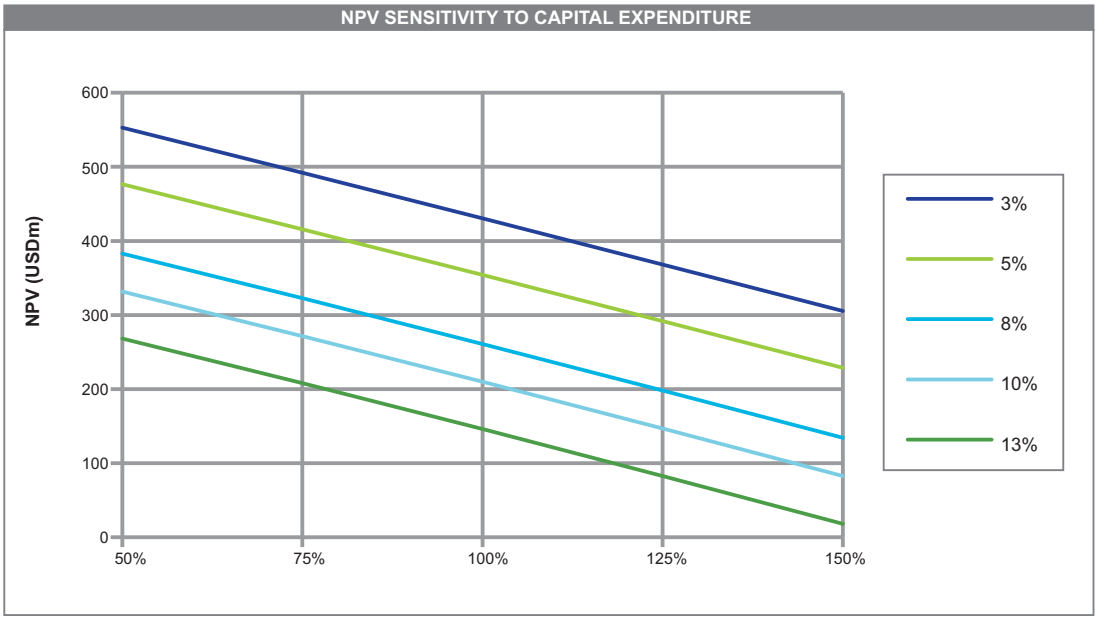
ESAASE PROJECT NPV SENSITIVITY ANALYSIS







Source: Venmyn Deloitte 2013

The southern boundary of the Esaase Project coincides with the Abore North exploration project which forms part of PMI Gold Corporations Asanko Gold exploration project (www.PMIgoldcorp.com). Drilling has been undertaken but not reported in the public domain. Almost 20km directly south of Esaase Project on the Kampeses-Nkran-Asuadai Shear Zone is the PMI Gold 100,000oz Asuadai deposit and 3.52Moz Nkran deposit. The Nkran deposit comprises three vertical, parallel mineralised zones below a previously mined open pit. The deposit is 500m in strike length with mineralisation over a 100m width to at least 250m depths. A Mineral Reserve of 34.2Mt at a grade of 2.2g/t for 2.4Moz contained gold was announced in 2012. Mine construction is due to commence in 2013 with first production in 2014.

Further southwest, on the Kaniago West- Esaase Shear Zone, the Midlands Minerals Corporation (www.midlandsminerals.com) holds exploration licences over the Mmooho and Kaniago West deposits.

The information supplied in this section has been obtained from the public domain and therefore has not been verified. The information is not necessarily indicative of the mineralisation on the Esaase Project.

23. Other Relevant Information

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23.1. Risk Assessment

The Esaase Project risk assessment was undertaken by DRA, all the contributing specialist consultants and engineers and Asanko Gold. The risk assessment was based on the DRA Risk Management Framework and Criteria which is based on the International Risk Management Standard ISO 31000, as well as the Australian Standard AS/NZS 4360. The objective was to involve all project contributors to identify risks appropriate to the Project at this stage, to determine a risk profile for the Project moving into the following stage and assist in making key decisions, identify any risk that could significantly impact on the successful delivery of the project and develop a risk control strategy and plan to manage the identified risks.

The risk analysis criteria used are presented in Table 50 and the results of the analysis identified 83 risks, of which 2 are extremely high, 44 high, 31 moderate and 6 low to very low. At the current project study stage these risks are without mitigating controls but with such controls identified in the DFS stage, the risks are likely to be reduced. The two extremely high risks relate to the haul roads cross over public roads on level crossings where there is a serious risk of accidents and increased project traffic on the public roads during construction. These risks will require appropriate controls in order to decrease them to appropriate levels.

23.2. Project Execution Plan

The 2013 PFS included a full project execution plan which will be undertaken and managed by DRA. The execution plan in included the following aspects at PFS level of accuracy; procurement strategy; assessment of required contracts; organisation and staffing requirements; engineering and design; construction management; health and safety; quality controls and assurance; project accounting; software and hardware systems; change and risk management; commissioning; testing and handover.

The Project Execution Plan entails orders for long-lead-time item and early contracts to be placed in September 2013 contingent on receipt of necessary permits and authorisations. Front-end engineering and early work on site is scheduled for the start of the dry-season in November 2013 and main construction is planned to begin in March 2014. Commissioning is planned for May 2015.

Table 50 : Risk Evaluation Criteria

PROBABILITY-LIKELIHOOD			RISK CLASSIFICATION (as per Legend below)				
More than once a day	Inevitable	6	6 – Moderate	12 – High	18 – High	24 – Extremely High	30 – Extremely High
Every day	Almost certain	5	5 – Low	10 – Moderate	15 – High	20 – High	25 – Extremely High
Once a month	Likely	4	4 – Low	8 – Moderate	12 – High	16 – High	20 – High
Once a quarter	Possible	3	3 – Low	6 – Moderate	9 – Moderate	12 – High	15 – High
Once a year	Unlikely	2	2 – Very Low	4 – Low	6 – Moderate	8 – Moderate	10 – Moderate
Once every ten years	Rare	1	1 – Very Low	2 – Very Low	3 – Low	4 – Low	5 – Low
Impact/Consequences							
Description			**1**	**2**	**3**	**4**	**5**
			Insignificant	**Minor**	**Moderate**	**Major**	**Catastrophic**
Impact on the Health and Safety of Personnel			First-Aid case / Exposure to minor health risk	Medical Treatment case / Exposure to major health risk	Lost time injury / Reversible impact on health	Single fatality or loss of quality of life / Irreversible impact on health	Multiple fatalities / Impact on health ultimately fatal
Environmental Impact			Minimal environmental harm – Remediable within same shift	Material environmental harm - remediable over short term < 3 months	Serious environmental harm - remediable within LoM	Major environmental harm - remediable post LoM	Extreme environmental harm - Incident irreversible
Financial Impact/ Property Damage			Less than 1% impact on the budget of the project	May result in overall project budget overrun equal to or more than 1% and less than 5%	May result in overall project budget overrun of equal to or more than 5% and less than 20%	May result in overall project budget overrun of equal to or more than 20% and less than 50%	May result in overall project budget overrun of 50% or more
Business Interruption			No disruption to operation less than 1% impact on overall project timeline	Brief disruption on operation between 1 and 5% impact on overall project timeline	Partial shutdown between 5 and 20% impact on overall project timeline	Partial Loss of operation Between 20 and 50% impact on overall project timeline	Substantial or total loss of operation over 50% impact on overall project timeline
Legal & Regulatory			Low level legal issues	Minor legal issues; non-compliance and breaches of law	Serious breach of law; investigation/report to authority; prosecution and/or moderate penalty possible	Major breach of the law; considerable prosecution and penalties	Very considerable penalties & prosecutions. Multiple law suits & jail terms.
Impact on Reputation / Social / Community			Slight impact - public awareness may exist but no public concern	Limited impact - local public concern	Considerable impact - regional public concern	National impact - national public concern	International impact - international public attention

Extremely High	Cease activity immediately until the residual risk is reduced to an acceptable level
High	Eliminate, avoid, implement specific action plans/procedures to manage & monitor daily
Moderate	Proactively manage & monitor weekly
Low	Actively manage & monitor monthly
Very Low	Continue and monitor identified risk & monitor monthly

24. Summary, Interpretations and Conclusions

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The results of specialist studies conducted for the 2013 PFS on the Esaase Project have been reviewed by Venmyn Deloitte and incorporated into an economic analysis of the Project. The following summary, conclusions and interpretations were highlighted in the review:-

- the Esaase Project is located in southwest Ghana in West Africa within the Amansie West District of the Ashanti Region, approximately 35km southwest of the regional capital Kumasi;
- the Esaase Project comprises two contiguous mining leases and four prospecting licences that cumulatively cover an area of approximately 99.10km^2 in the prospective gold mining Ashanti region. The Esaase and Jeni concessions contain alluvial gold deposits in the Bonte river valley which have been exploited historically by artisanal miners. The Esaase Project however, is focussed on the primary host Birimian aged meta-sediments and not on evaluation of the alluvial deposits;
- the Jeni River and Esaase mining leases were originally granted in March 1990 to the Jeni River Development Company Limited and to Bonte Gold Mines Limited respectively. Both of these companies filed for bankruptcy in 2002 and the Esaase mining lease was acquired from the Bonte Liquidation Committee by Sametro Company Limited, a private Ghanaian company. Asanko Gold entered into an agreement with Sametro in May 2006 to earn 100% of the mineral right and transfer of the Esaase mining lease to Asanko Gold was completed in June 2007;
- the mining leases are valid until 2020 and can be renewed for an additional thirty year period. The mining leases permit Asanko Gold to undertake mining activities provided certain conditions and fee payments are maintained with the Ministry of Lands, Forestry and Mines;
- the Esaase Project is located in the Archaean (Birimian 2.17Ga) aged Asankrangwa Belt within the Kumasi Basin. World renowned gold mines such as AngloGold Ashanti Limited's Obuasi deposit, which began production in 1898, are located in the neighbouring Ashanti and Sefwi greenstone belts. Similar vein deposits in the region, such as Prestea and Konongo have robust production histories. The gold quartz vein mineralisation in these deposits is exclusively structurally controlled along faults and shear zones resulting from compressional tectonic episodes, particularly along the margins of greenstone belts such as the Ashanti and Sefwi Belts;
- the target Esaase Project mineralisation is classified as a mesothermal orogenic quartz vein deposit. Such mesothermal orogenic deposits are associated with Precambrian and Phanerozoic convergent plate boundaries and are hosted in sequences generally metamorphosed to greenschist facies.

  The Esaase Project area comprises a sequence of intensely folded and faulted meta-sediments and the mineralisation comprises a system of gold-bearing quartz veins hosted by the tightly folded meta-sediments. The mineralisation is considered to have been produced by a series of fluid influxes which were channelled along lithological, rheological and structural boundaries. The target gold mineralisation occurs in association with quartz-carbonate-sulphide veins hosted within parallel, northeast trending, moderately to steeply, west dipping units of extremely deformed siltstone/shale;
- the Esaase Project mineralisation is structurally controlled by Eburnean age compressional tectonic events which produced fold-thrust patterning followed by a late stage strike-slip deformation event. The deformational intensity increases systematically from the northwest to the southeast across the Project area. The regional structural synthesis suggests that most of the structural elements are compatible with a single, extended and progressive phase of regional deformation involving substantial northwest-southeast compression. Towards the close of the tectonic event, the stress direction changed from northeast-southwest to north-south. The change in stress direction caused left lateral strike-slip movement along pre-existing faults which created a northsouth dilational opening permitting the emplacement of late stage non-mineralised northsouth oriented veins;
- the Esaase Project mineralisation is characterised by free-gold and very fine gold in association with sulphides;

- the weathering profile on the Esaase Property is strongly influenced by topography and influences the proposed mine site layout and mining methodology. The weathered profile comprises surface laterite and ferruginous duricrust followed by saprolite, which gradationally merges with oxidised bedrock. In regions of higher elevation at Esaase Project, the laterite and underlying saprolite are removed by erosional processes and only the oxidised bedrock remains;
- since mid-2006, Keegan Resources has undertaken an extensive exploration programme combining soil geochemistry and IP and VTEM geophysical surveys, followed by diamond and reverse circulation exploration and resource drilling. The geophysical surveys delineated the resistivity of the host and bedrock at Esaase Project and the >6,100 soil sample geochemistry programme clearly supports the mineralisation model of parallel northwest trending mineralised structures;
- the drilling programme conducted at the Esaase Project focused mainly on the northwest striking gold bearing structures and a total of 1,496 RC and diamond drillholes were completed on the Project area. A geotechnical drilling programme of 28 diamond drillholes was also completed. Sample recovery was good except in the moderate to highly weathered saprolite and highly fractured and brecciated zones where poor recoveries were experienced. Recovery factors are unlikely to materially affect the accuracy and reliability of the sampling results. Quality control, twin drillhole verification showed no negative bias in the diamond drilling due to the use of water. Reviews of the drilling and sampling protocols by independent Qualified Persons from Coffey Mining and Minxcon provide confidence that the data is spatially well represented and that the methodologies employed are within international standards and the resultant information is suitable for Mineral resource estimation purposes;
- suitable specific gravity measurements were undertaken to support the Mineral Resource estimation process;
- sample preparation and analysis was undertaken in three independent internationally accredited laboratories and two umpire laboratories. Field Quality Assurance/Quality Control measures included insertion of CRM standards to monitor accuracy, blank samples to identify contamination, duplicate samples to check precision and submission to umpire laboratory. Each laboratory included in-house reference standards duplicates and blanks and the sample preparation and analytical techniques employed for the Esaase sampling campaign are appropriate for the style of mineralisation. Detailed analysis of the quality control data was undertaken by Coffey Mining and Minxcon with the concluding opinion that the Esaase Project quality control and assurance is adequate but suggests a relative high bias in Transworld Laboratories data and a lack of precision between laboratories which should be investigated;
- the 2011 Coffey Mining PFS for the Esaase Project was based on an open pit mining methodology with an associated “whole-ore” leach processing plant. The study included initial mill throughputs of 9Mtpa reducing to 7.5Mtpa once mining of the fresh zone began. The PFS resulted in a positive NPV for the Project but additional studies indicated that better economic outcomes could be achieved with alternative process design methodologies and consequently the 2013 PFS was initiated with a revised scope which included investigation of a conventional crushing, milling, gravity recovery plant followed by flotation, with the flotation concentrate being reground and then leached in a standard CIL circuit using AARL elution technology;
- under the supervision of Lycopodium in Australia four extensive phases of metallurgical testwork were completed for the 2011 PFS and DRA undertook a Phase V testwork programme in 2012 to support the new process design. The final process recommended by DRA comprises SAG milling, with primary gravity recovery from the mill circuit, flotation of the milled product, with regrind and secondary gravity recovery of the flotation concentrate ahead of CIL gold recovery of the reground flotation concentrate. Testwork results indicated an optimal grind size of 75µm with a LoM recovery of 90.06% which includes a 1.09% recovery discount to allow for practical processing limitations in a full-scale operating plant environment. The final design has the benefit that the flotation tailings, comprising approximately 85% to 90% of the feed, are benign and can be disposed to a non-HDPE lined waste TSF, whilst the CIL and downstream plants can be downsized accordingly;

- the metallurgical testwork and process design criteria were independently reviewed by Venmyn Deloitte and Venmyn Deloitte is of the opinion that the metallurgical testwork is acceptable for the type and style of mineralisation and that the process design is reasonable with potential for additional optimisation for later study stages;
- Minxcon conducted the 2013 PFS Mineral Resource estimate on a refined block model utilising existing data with an additional 19,598 assay results to more precisely delineate the mineralised zones within the resource area. Grade estimation was undertaken using Ordinary Kriging, with Indicator Kriging chosen to delineate the areas with continuous grade. Mineralised domains were defined and modelled on an approximate lower cut-off grade of 0.3g/t. A total of 46 zones, including waste, were identified and included in the statistical analysis and resource estimation. Outlier analysis and variogram determinations were undertaken and the final Mineral Resource estimate at a 0.6g/t cut-off grade is:-
    - Measured Mineral Resources of 23.4Mt at a grade of 1.49g/t Au for 1.12Moz contained gold;
    - Indicated Mineral Resources of 71.3Mt at a grade of 1.44g/t Au for 3.29Moz contained gold;
    - Total Measured plus Indicated Mineral Resources of 94.6Mt at a grade of 1.45g/t Au for 4.41Moz contained gold; and
    - Inferred Mineral Resources of 33.6Mt at a grade of 1.40g/t Au for 1.51Moz contained gold.
- the Mineral Reserve estimate was based on the 2013 PFS mining study at a cut-off grade of 0.6g/t Au, a USD1,400/oz gold price and a mining recovery of 97%. Gold resources which occur in satellite pits and which are currently deemed uneconomic to extract were excluded from the Mineral Reserve estimate, together with Mineral Resources which are affected by permitting, environmental, logistic and socio-political issues such as proximity to villages or forest reserves. The 2013 PFS Mineral Reserve estimate is
    - Proven Mineral Reserves of 22.9Mt at a grade of 1.43g/t Au for 1.05Moz contained gold; and
    - Probable Mineral Reserves of 29.5Mt at 1.40g/t Au for 1.32Moz contained gold.
    - Total Proven and Probable Reserves of 52.34Mt at a grade of 1.41g/t Au for 2.37Moz contained gold.
- DRA Mining undertook the mining study for the 2013 PFS, which included mine design on Indicated and Measured Mineral Resources only, pit optimisation, mine production and scheduling and costing. The appropriate mining methodology for the Esaase Project comprises conventional open pit drill and blast mining, followed by load and haul to various stockpiles. The drilling and blasting would be performed on benches between 10m and 20m in height and total material movement over the 11 year LoM is estimated between 30Mtpa and 35Mtpa. For the purpose of the 2013 PFS, Asanko Gold opted for the mining study to be conducted on a contract mining basis. Geotechnical studies confirmed slope angles of 52° in fresh material and 35º in oxide material. Pit optimisation produced a main pit with various satellite pits but only the Main Pit and South Pit were included in the mining study. The selected pit shell reaches a depth of -19mamsl, or approximately 290m below the Bonte river valley surface and contains 52.34Mt of mill feed at 1.4g/t Au for 2,37Moz of in situ gold. A total of 224.3Mt of waste are contained within the pit, equating to a waste to ore stripping ratio of 4.28. A pit dewatering programme has been designed and grade control planning is essential. Bench and face mapping, for grade control as well as for geotechnical reasons, will be a routine task in finalising the ore and waste blocks to be marked out for excavation;
- the mine plan includes pre-production stripping and waste provision for starter dams. Thereafter, the Main Pit remains the principal source of RoM, with South Pit contributing, on average, a third (±31%) of the overall tonnes mined for the first four years until it's mined out and ready to be backfilled;

- the mine site plan was optimised during the 2013 PFS and the final plan includes various waste dumps, two TSFs located in a valley to the west of the Main Pit, a plant site west of the Main Pit, contractor miner camp near the Bonte River and an upgraded and relocated public road;
- two options exist for the power supply to the proposed Esaase mine, namely an off-grid connection to the national state utility company Gridco or IPP supply. The Ghanaian utility company, GridCo can supply 34MW at a total cost of USD32m, which maybe significantly reduced due to a redesign of the connection by placing a new 161kV substation on the main line and utilising a single 11km 161 kV overhead line to feed the mine. The costs of the Gridco option for power supply to the proposed Esaase mine are high and Asanko Gold is investigating various alternative options. A quotation from an American based company USP&E Global for a heavy fuel oil power generation installation has been obtained which suggests a capital expenditure of USD22.5m will be required with an operating cost of USD0.15/kWh;
- the initial project start–up water requirement of 1.4Mm^3 will be sourced from the Mpatoam wier and Bonte river in the wet season and stored initially in the L-TSF. Potable water demand during construction and steady state production will reach a maximum of 168m^3 per day to be supplied by ten boreholes. The water balance model is based on a closed circuit for beneficiation process water and the TSFs, with make-up water supply from pit dewatering. A process water buffer dam has been designed to capture low arsenic process water. The Esaase mine will eventually be a net water positive situation and at steady state production, the total water in slurry discharged to the mine residue disposal facilities will be 920m^3/hr of which 60% to 90% is returned to the plant process water circuit. The surface water management system for the Project will consist of two separate systems namely, a clean water diversion system to control the uncontaminated run-off from the higher lying natural environment, and a dirty storm water system to capture the contaminated storm water from plant, operational and processing areas;
- the design of the mine residue disposal facilities for the 2013 PFS was undertaken by independent design specialists Epoch Resources and included two facilities namely a clay lined storage facility for the tailings arising from the gravity-flotation circuit of the beneficiation plant and an HDPE lined facility designed to accommodate the tailings from the CIL circuits of the process plant. The cost of the mine residue disposal facilities is a significant factor in the Project economics and approval of the designs by the Ghanaian authorities remains to be granted;
- the environmental and social Pre-Feasibility study indicates that in the context of the specifications of Ghanaian legislative requirements, and internationally accepted standards of practice, including the IFC Performance Standards and Equator Principles, the environmental aspects of the Project PFS have been acceptable at this stage of project development. The Equator Principle Finance Institutes would categorise the Esaase Project as a Category A project. Formal consultation and stakeholder engagement regarding the Project have been on-going since 2007 and extensive baseline environment studies have been completed.

  Key areas of potential environmental impact have been identified and measures to mitigate these risks are to be assessed in the DFS. Key authorisations that are required include various water use licences in terms of the Water Resources Commission and an approved EIS to be submitted in July 2013;
- a conceptual rehabilitation and mine closure plan for the proposed Esaase mine and its associated infrastructure was undertaken. An estimate of the required financial provision is USD29.57m including on-going re-habilitation and aftercare;
- the Esaase Project pre-production capital expenditure estimate including mining, processing, mine infrastructure, TSFs, power supply, owners' costs and indirect costs totals USD260.50m before contingency and USD286.40m including a 10% contingency allowance. The main capital items included in the pre-production capital cost estimate are the processing plant (USD82.9m), the establishment of the initial L-TSF (USD34.4m) and mining pre-stripping and infrastructure (USD13.7m);
- in addition to pre-production capital costs, sustaining capital expenditures will be required throughout the LoM mine totalling USD51.9m which includes raises to the TSFs. Certain capital expenditures estimated at USD12.95m were deferred until production commences;

- the operating cost 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 average LoM cash operating cost is estimated at USD736/oz Au based on the treatment of 5Mtpa producing an average of 200,000oz/a. The RoM opex is for ore is USD4.64/t with a waste mining opex of USD2.86/t. The process opex is USD10.37/t;
- Asanko Gold elected to conduct the PFS on a base case utilising contract mining as opposed to owner operations. The conceptual average mining opex figure for the pit optimisation of USD3.2/t was obtained by combining DRA's database of costs, for similar operations and average costs from the escalated 2011 mining contractor quotations. The processing plant opex was USD10.37/t milled resulting in a total cash operating cost (excluding royalties) of USD30.00/t milled;
- Venmyn Deloitte constructed a Discounted Cash Flow (DCF) model for the purposes of the economic analysis of the Esaase Project at a gold price of USD1,400/oz Au. The DCF model was based on input assumptions generated from the 2013 PFS mining schedule, processing schedule, operating costs and capital expenditure estimates. Venmyn Deloitte received input from DRA and Asanko Gold on the timing of the various inputs, including working capital requirements. The DCF model assesses the post-tax real cash flows for the Project at a 5% real discount rate. The economic analysis indicates a positive Net Present Value (NPV) of the Esaase Project of USD354.7m with a post-tax IRR of 23.2%;
- the NPV generated from the DCF model proved to be most sensitive to changes in parameters affecting revenue. A 10% change in revenue changes the NPV by 39.1%. The NPV is less sensitive to changes in operating expenditure with a 10% change in opex translating into a 21.9% change in the NPV. A 10% change in capital expenditure changes the NPV by 17.4%. At a 5% discount rate, the critical gold price which generates an NPV of 0 is USD1,049/oz;
- risk assessment of the Esaase Project identified 83 risks, of which 2 are extremely high, 44 high, 31 moderate and 6 low to very low. At the current project study stage these risks are without mitigating controls and with such controls identified in the DFS stage, the risks are likely to be reduced. The two extremely high risks relate to the haul roads cross over public roads on level crossings;
- the 2013 PFS included a full project execution plan which will be undertaken and managed by DRA. Front-end engineering and early work on site is scheduled for the start of the dry-season in November 2013 and main construction is planned to begin in March 2014. The construction schedule is 18 months with first gold in H1 2015 and steady state production in H2 2015;
- Venmyn Deloitte concludes that the 2013 PFS has fulfilled its scope of optimising the process flow design, improving project economics and has succeeded in providing a robust basis for the DFS going forward. The Mineral Resource and Reserve basis is founded in international standard exploration and analytical results and the process flow design is based on reasonable and appropriate metallurgical testwork.

  The mine design has already identified optimisation opportunities for the DFS which provides positive upside potential to the Project. No environmental or risk factors that cannot be mitigated have been identified and the Project site layout plan adequately accommodates all Project components without impacting areas of environmental sensitivity. Some permits and authorisations remain outstanding and may pose a Project delay risk. The economic analysis shows that the Esaase Project is robust, with an attractive positive NPV even at gold prices less than the current June gold spot price of USD1,379/oz.

25. Recommendations

NI 43-101 Item 24

The Venmyn Deloitte review of the 2013 PFS for the Esaase Project has highlighted that the Project is robust and economically viable and Venmyn Deloitte concurs with Asanko Gold's decision to progress the Project to the DFS level of investigation. Several opportunities exist to improve upon the economic results of the 2013 PFS which should be investigated in the DFS, namely:-

- a metallurgical testwork program focusing on the optimisation of the flotation reagent suite which could result in the reduction of the concentrate mass pull through the float plant to further optimise the process flow sheet and associated costs;
- a detailed mine design using the PFS developed modifying factors to improve the conversion of Mineral Resources to Mineral Reserves;
- a detailed mine design reflecting higher mining and processing rates whilst treating softer oxide RoM in the early years of the Project LoM; and
- further geotechnical engineering studies to determine if steeper pit slope angles can be introduced into the design of the open pit.

The metallurgical testwork can commence immediately on the remaining material from the previous metallurgical testwork programme. The detailed mine design will include the deeper sections of the orebody, classified as Measured and Indicated Resources, which would extend the LoM and thereby improve the Project economics further. The proposed costs for the DFS are presented in the table below:-

Table 51 : Proposed Expenditure for the Esaase Project DFS

DFS PROJECT COMPONENT	ESTIMATED COSTS (USD)
EPCM	1,210,239
Mine Residue storage facilities and water storage dams	31,595
Environmental Studies	115,006
Metallurgical testwork	119,518
Geotechnical studies	269,549
Land survey	63,189
IPP	324,977
DFS implementation and management	541,629
Contingency	273,636
Additional	60,662
TOTAL	**3,010,000**

The Venmyn Deloitte review highlighted the following areas which can be addressed or optimised in the future studies;-

- investigation of the low bias in analytical results for SGS Laboratories, Tarkwa; and
- investigation of a relative CRM high bias for Transworld Laboratories as supported by the umpire analysis programme.

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Report Date : 27 June 2013
Effective Date: 14 May 2013

26. References

NI 43-101 Item 26

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Mayes J.	2013	Environmental and Social Pre feasibility Report	Epoch Resources (Pty) Limited 127-038

27. Glossary and Abbreviations

^{0}C	Degrees celcius
%	Percentage
+	Plus
±	Approximately
º	Degrees
µ	Microns
<	Less than
>	Greater than
/	Per
amsl	Above mean sea level
AusIMM	Australian Institute of Mining and Metallurgy
Au	Chemical Symbol for gold
bn	billion
BBWi	Bond ball work index
B.Sc. (Geol)	Bachelor of Science Degree in Geology
B.Sc. Hons	Bachelor of Science degree with Honours
CIL	Carbon in leach
cmg/t	centimetre grams per tonne
CRM	Certified reference materail
L-TSF	CIL tailings storage facility
DCF	Discounted cash flow
DFS	Definitive Feasibility Study
DTM	Digital terrain model
EIA	Environmental Impact Assessment
ESIA	Environmental and social impact assessemtn
EIS	Environmental Impact Statement
EP	Equator Priniclpes
EPCM	Engineering procurement construction and management
F-TSF	Flotation tailings storage facility
g/t	grams per tonne
GDP	Gross domestic product
GPS	Geographical Positioning System
ha	hectares
HDPE	High density polyethylene
JORC	Joint Ore Reserves Committee
IFC	International Finance Corporation
IP	Induced polarisation
IPP	Independent Power Producer
IRR	Internal rate of Return
km	Kilometres
kV	Kilo volts
kWh	Kilo watt hour
kt	Kilo tonnes
LoM	Life of Mine
m	metres
mg/L	Milligram per litre
MW	Megawatt
mRL	Metres relative level
Mt	Million tonnes
mamsl	Metres above mean sea level
mbs	Metres below surface
Moz	Million troy ounce
Mtpa	Million tonnes per annum
my	million years
LEI	Layered earth inversions
NPV	Net present value
pa	per annum
P&G	Preliminary and general
oz	troy ounce
PFS	Pre Feasibility Study
SAG	Semi autonomous grinding
SMC	SAG mill comminution test
SCADA	Supervisory control and data acquistion
t	tonnage

tph	Tonnes per hour
tpm	Tonnes per month
TSF	Tailings storage facility
TMC	The Minerals Corporation
TSS	Total suspended solids
RC	Reverse circulation
RoM	Run-of-mine
UCS	Unconfined compressional strength
USD	United States of America Dollar
2D	Two dimensional
3D	Three dimensional
VTEM	Versatile time domain electromagnetic survey

Assay	A chemical test performed on a sample of ores or minerals to determine the amount of valuable metals contained.
Assay map	Plan view of an area indicating assay values and locations of all samples taken on the property.
Arsenopyrite	It is the principal ore of arsenic and a common mineral with lead and tin ores in ore veins, and in pegmatites, probably having been deposited by action of both hydrothermal solutions and vapours
Block Model	Technique for modelling which divides the resources into mineable blocks.
Breccias	Coarse grained clastic rock composed of broken, angular rock fragments enclosed in a fine-grained matrix or held together by a mineral content. Fault breccias are composed of fragments produced by rock fragments produced by rock fracturing during faulting and other crustal deformation.
Borehole	A hole drilled from surface or underground, in which core of the rock is cut by diamond drill bit as the cutting edge.
Bulk sample	A large sample of mineralised rock, frequently hundreds of tonnes, selected in such a manner as to be representative of the potential orebody being sampled. Used to determine metallurgical characteristics, Large sample which is processed through a small-scale plant, not a laboratory.
Carbonates	A mineral type containing the carbonate radical $(CO_3)^{2-}$
Carbon-in-leach	The recovery process in which Au is leached from Au ore pulp by cyanide and simultaneously adsorbed onto activated carbon granules in the same vessel. The loaded carbon is then separated from the pulp for subsequent Au removal by elution. The process is typically employed where there is a naturally occurring Au adsorbent in the ore.
Carbon-in-pulp	A method of recovering Au and silver from pregnant cyanide solutions by adsorbing the precious metals to granules of activated carbon, which are typically ground up coconut shells.
Chalcopyrite	A brassy or golden-yellow tetragonal mineral $CuFeS_2$, that is an important ore of copper.
Coffey Mining	Coffey Mining Proprietary Limited
Cross section	A diagram or drawing that shows features transected by a vertical plane drawn at right angles to the longer axis of a geologic feature.
DRA	DRA Minerals Projects Limited
Density	Measure of the relative "heaviness" of objects with a constant volume, density = mass/volume
Deposit	Any sort of earth material that has accumulated through the action of wind, water, ice or other agents.
Development property	A mineral property that is being prepared for mineral production and for which economic viability has been demonstrated.
Diamond drilling	A drilling method, where the rock is cut with a diamond bit, to extract cores.
Dip	The angle that a structural surface, i.e. a bedding or fault plane, makes with the horizontal measured perpendicular to the strike of the structure.
Epoch	Epoch Resources Proporietary Limited
Estimation	The quantitative judgement of a variable.
Exploration	Prospecting, sampling, mapping, diamond drilling and other work involved in the search for mineralisation.
Exploration Property	A Mineral Asset which is being actively explored for Mineral deposits or petroleum fields, but for which economic viability has not been demonstrated.
Feasibility study	A definitive engineering estimate of all costs, revenues, equipment requirements and production levels likely to be achieved if a mine is developed. The study is used to define the economic viability of a project and to support the search for project financing.
Grade	The relative quantity or percentage of gold within the rock mass. Measured as grams per tonnes in this report.
Hanging wall	The overlying unit of a stratigraphic horizon, fault ore body or stope
In situ	In its original place, most often used to refer to the location of the mineral resources.
Indicated Mineral Resource	That part of a mineral resource for which tonnage, densities, shape, physical characteristics, grade and average mineral content can be estimated with a reasonable level of confidence. It is based on exploration sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed and sufficient minerals have been recovered to allow a confident estimate of average mineral value.
Inferred Mineral Resource	That part of a mineral resource for which tonnage, grade and average mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified by geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that may be limited or of uncertain quality and reliability.
Keegan Resources	Keegan Resources Ghana Limited
Life-of-Mine/LoM	Expected duration of time that it will take to extract accessible material.
Liberation	Release of Au from the host rock through processing.
Lithologies	The description of the characteristics of rocks, as seen in hand-specimens and outcrops on the basis of colour, grain size and composition.
Lycopodium	Lycopodium Minerals Proprietary Limited
Mineral Asset(s)	Any right to explore and / or mine which has been granted ("property"), or entity holding such property or the securities of such an entity, including but not limited to all corporeal and incorporeal property, mineral rights, mining titles, mining leases, intellectual property, personal property (including plant equipment and infrastructure), mining and exploration tenures and titles or any other right held or acquired in connection with the finding and removing of minerals and petroleum located in, on or near the earth's crust. Mineral Assets can be classified as Dormant Properties, Exploration Properties, Development Properties, Mining Properties or Defunct Properties.
Mineral Reserve	The economically mineable material derived from a Measured and/or Indicated Mineral Resource. It is inclusive of diluting materials and allows for losses that may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, including consideration of and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction is reasonably justified. Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proved Mineral Reserve.

Mineral Resource	A concentration of material of economic interest in or on Earth's crust in such form, quality and quantity that there are reasonable and realistic prospects for eventual economic extraction. The location, quantity, grade, continuity geological characteristics of a Mineral Resource are known, estimated from specific geological evidence and knowledge, or interpreted from a well constrained and portrayed geological model. Mineral Resources are subdivided, in order of increasing confidence in respect of geoscientific evidence, into Inferred, Indicated and Measured categories.
	A deposit is a concentration of material of possible economic interest in, on or near the Earth's crust. Portions of a deposit that do not have reasonable and realistic prospects for eventual economic extraction must not be included in a Mineral resource.
Measured Mineral Resource	That part of a mineral resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes. The locations are spaced closely enough to confirm geological and grade continuity.
Mineralisation	The presence of a target mineral in a mass of host rock.
Mining Property	a Mineral Asset which is in production.
National instrument 43-101	Canadian National Instrument on the reporting of exploration, mineral resources and mineral reserves for the TSX.
Opencast / Open pit	Surface mining in which the ore is extracted from a pit. The geometry of the pit may vary with the characteristics of the ore body.
Optimisation	Creating the best mining scenario while taking into account the economic parameters of the deposit.
Orebody	A continuous well defined mass of material of sufficient ore content to make extraction economically feasible.
Overburden	The alluvium and rock that must be removed in order to expose an ore deposit.
Probable reserves	Is the economically mineable material derived from a Measured and/or Indicated Mineral Resource. It is estimated with a lower level of confidence than a Proved Reserve. It is inclusive of diluting materials and allows for losses that may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction is reasonably justified.
Reef	Mineralised lode.
Rehabilitation	The process of restoring mined land to a condition approximating to a greater or lesser degree its original state. Reclamation standards are determined by the Russia Federation Department of Mineral and Energy Affairs and address ground and surface water, topsoil, final slope gradients, waste handling and re-vegetation issues.
Sample	The removal of a small amount of rock pertaining to the deposit which is used to estimate the grade of the deposit and other geological parameters.
Sampling	Taking small pieces of rock at intervals along exposed mineralisation for assay (to determine the mineral content).
Sedimentary	Formed by the deposition of solid fragmental or chemical material that originates from weathering of rocks and is transported from a source to a site of deposition.
Specific gravity/S.G.	Measure of quantity of mass per unit of volume, density.
Stockpile	A store of unprocessed ore or marginal grade material.
Stripping	Removal of waste overburden covering the mineral deposit.
Stripping ratio	Ratio of ore rock to waste rock.
Subduction	The movement of one crustal plate (lithospheric plate) under another so that the descending plate is "consumed.
Tailings	The waste products of the processing circuit. These may still contain very small quantities of the economic mineral.
Tailings dam	Dams or dumps created from waste material from processed ore after the economically recoverable metal or mineral has been extracted.
Tonnage	Quantities where the tonne is an appropriate unit of measure. Typically used to measure reserves of metal-bearing material in-situ or quantities of ore and waste material mined, transported or milled.
Veins	A tabular or sheet like body of one or more minerals deposited in openings of fissures, joints or faults, frequently with associated replacement of the host rock.

28. Certificates of Qualified Persons'

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CERTIFICATE OF THE AUTHOR OF 'NATIONAL INSTRUMENT 43-101 INDEPENDENT TECHNICAL REPORT ON ASANKO GOLD INCORPORATED'S PRE-FEASIBILITY STUDY ON THE ESAASE PROJECT IN GHANA'

I, Andrew Clay, do hereby certify that I am a Managing Director of Venmyn Deloitte (Pty) Limited:-

2. I am a graduate in Geology and a Bachelor of Science from University College Cardiff in 1976
3 I am a member/fellow of the following professional associations. I have extensive experience in gold deposits as summarised below.

CLASS	PROFESSIONAL SOCIETY	YEAR OF REGISTRATION
Member	Canadian Institute of Mining, Metallurgy and Petroleum	2006
Advisor	JSE Limited Listings Advisory Committee	2005
Associate Member	American Association of Petroleum Geologists	2005
Member	South African Institute of Directors	2004
Fellow	Geological Society of South Africa	2003
Member	American Institute of Mineral Appraisers	2002
Member	South African Institute of Mining and Metallurgy	1998
Fellow	Australasian Institute of Mining and Metallurgy	1994
Member	Natural Scientist Institute of South Africa	1988
Member	Investment Analysts Society of South Africa	1990

4 I have practiced my profession continuously since graduation;
5 I have not visited the Esaase Project;
6 I have read the definition of 'Qualified Person' and 'Qualified Valuator' as set out in NI43-101 and CIMVAL and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a 'Qualified Person' and 'Qualified Valuator' for the purposes of NI43-101 and CIMVAL;
7 I have had no prior involvement with the property that is the subject of the Technical Report;
8 I have read NI43-101, Form 43-101F1 and CIMVAL Standards and Guidelines, and the Technical Report and Valuation Report have been prepared in compliance with these instruments and form;
9 I am responsible for Section 1, 2, 18, 21, 22, 24, 25, 26, 27 of the Technical Report entitled 'National Instrument 43-101 Independent Technical Report on Asanko Gold Incorporated's Pre-feasibility Study on the Esaase Project in Ghana'
10 At the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
11 I am independent of the issuer applying all of the tests in Section 1.5 of NI43-101; and
12 I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 27 June 2013 at Johannesburg, South Africa.

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Managing Director
Andrew Neil Clay – Gold Experience

YEAR	CLIENT	COMMODITY	DOCUMENTATION
2013	Eureka	Gold	Technical Statement
2013	Gold One Tulo Gold	Gold	Valuation
2013	Eureka Delta Gold	Gold	Technical Statement
2012	Banro Lugushwa	Gold	Technical Review
2012	Central Rand Gold	Gold	CPR
2012	Loncor Ngayu	Gold	Mineral Resource Valuation
2012	Loncor Makapela/Mangajuripa	Gold	Mineral Resource Valuation
2012	Stonebridge Hanieal Mozambique	Gold	Corporate Advice and Project setup
2012	Stonebridge Zim Gold	Gold	Corporate Advice and Project setup
2012	Terra Nova Manica Investment	Gold	Technical and Corporate Valuation
2012	Virgil Mining	Gold	Technical Report
2012	Sikhuliso Harmony Dumps	Gold	Corporate Transaction Advice
2012	Wits Gold	Gold	CPR and Valuation
2012	Pan African Resources	Gold	CPR and Valuation
2012	Banro	Gold	Technical Report and Valuation
2012	Harmony Evander	Gold	Full CPR and Valuation
2012	NMIC	Gold	Technical Report and Valuation
2011	SSC Mandarin	Gold	Independent Corporate and Technical Advisor
2011	Harmony	Gold	CPR
2011	Banro	Gold	Independent Technical Statement
2011	Xceed Capital	Coal	Independent Valuation Statement
2011	Taung	Gold	Hong Kong Listing
2011	Axmin	Gold	Technical and Economic Documentation
2011	AMRT	Copper/Gold	Scoping Study
2011	Jindal Mining	Coal	Techno-Economic Statement on the Mbili Coal Project
2011	Kibo Mining	Gold/Various	Tanzanian Assets
2010	AMRT	Gold	Independent Sampling and Mass Balance Report
2010	White Water Resources	Gold	Independent Competent Persons' Report
2010	White Water Resources	Gold	Independent Technical Statement
2010	West Wits Mining	Gold	Independent Prospectivity Review
2010	SSC Mandarin	Gold	Independent Corporate and Technical Review
2010	Taung	Gold	Independent Technical Review
2010	Taung	Gold	Independent Valuation Statement
2010	Mzuri Capital	Gold	Independent AIM Compliant CPR
2010	Loncor	Gold	Independent Techno-Economic Valuation Report
2010	Nyota Minerals	Gold	Independent Inferred Resource Estimate
2010	White Water Resources	Gold	Short-Form Valuation Statements
2010	Central African Gold	Gold	NI 43 – 101 Technical Report
2009	Metorex	Gold	Independent Fairness Opinion
2009	Taung Gold	Gold	Independent Competent Person's Report
2009	Ernst & Young Jordan	Gold	Independent Valuation Report on mineral assets of a Gold Mining Concession in Ethiopia
2009	Dwyka Resources	Gold	Independent Technical Statement on Tulu Kapi Gold Project
2009	Central African Gold	Gold	Information Memorandum in the form of NI 43-101 Compliant Technical Statement
2009	New Dawn	Gold	Independent Technical Statement

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CERTIFICATE OF THE AUTHOR OF 'NATIONAL INSTRUMENT 43-101 INDEPENDENT TECHNICAL REPORT ON ASANKO GOLD INCORPORATED'S PRE-FEASIBILITY STUDY ON THE ESAASE PROJECT IN GHANA"

I, Douglas Heher, Pr. Eng. (Reg. No. 990333) do hereby certify that:-

1. I am a Project Manager of DRA Mineral Projects (Pty) Ltd
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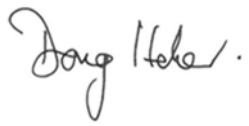
   South Africa
2. I graduated with a B.Sc. Engineering (Mechanical) degree from the University of KwaZulu Natal - Durban in 1992;
3. I am a member of the following professional associations:-
   Registered Professional Engineer with the Engineering Council of South Africa (ECSA – Reg. No. 990333)
   I have completed gold and other studies as summarised below.
4. I have practiced my profession from 1997 to now
5. I have visited the Esaase Project;
6. I have read the definition of 'Qualified Person' as set out in NI43-101 and certify that by reason of my education and affiliation with a professional association (as defined in NI43-101), I fulfil the requirements to be a 'Qualified Person' for the purposes of NI43-101;
7. I have had prior involvement with the property that is the subject of the Technical Report – through previous concept and trade off work;
8. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;
9. I am responsible for Sections 17 (except 17.5), 20, 23.1, 23.2 of the Technical Report entitled 'National Instrument 43-101 Independent Technical Report on Asanko Gold Incorporated's Pre-feasibility Study on the Esaase Project in Ghana';
10. At the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
11. I am independent of the issuer applying all of the tests in Section 1.5 of NI43-101; and
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 27 June 2013 at Johannesburg, South Africa.



D. Heher
B.Sc. (Mechanical Engineering.)
Pr. Eng;
Project Manager

Project Management Experience D. Heher

YEAR	CLIENT	COMMODITY	TYPE OF STUDY	PROJECT DESCRIPTION
2008	Red Island Minerals	Coal	PFS	PFS on Sakoa deposit in Madagascar incl. infrastructure
2008/9	Lonmin / Anglo JV	Platinum	DFS	Full process study incl. test work on the Pandora mine
2010	Confidential	Iron Ore / Coal	Concept	Bulk port on Yangtze river in China
2011	ENRC	Copper	Concept	3.6 Mtpa Copper Concentrator in the DRC
2011	ENRC	Copper	FS	3.6 Mtpa Copper Concentrator in the DRC
2012	ENRC	Copper	Execution	3.6 Mtpa Copper Concentrator in the DRC
2012	Frontier	Rare Earths	PFS	Zandkopsdrift study in Northern Cape (still in progress)
2012	CIC Energy	Coal	Concept	Capex estimate for complete coal complex in Botswana
2012	Keegan	Gold	PFS review	Review of 3rd party PFS for Esaase mine in Ghana
2012	Keegan	Gold	Concept	Trade Off Study for the Esaase mine in Ghana
2013	Asanko	Gold	PFS	Full PFS from mining to economics – Esaase Ghana

Charles Muller

Suite 5 Coldstream Office Park,
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Johannesburg, South Africa

CERTIFICATE OF THE AUTHOR OF 'NATIONAL INSTRUMENT 43-101 INDEPENDENT TECHNICAL REPORT ON ASANKO GOLD INCORPORATED'S PRE-FEASIBILITY STUDY ON THE ESAASE PROJECT IN GHANA"

1. I, Charles Muller do hereby certify that I am a I am Director of Minxcon (Pty) Ltd

   Suite 5 Coldstream Office Park,
   Cnr Hendrik Potgieter and Van Staden Roads,
   Little Falls,
   Johannesburg, South Africa

2. I graduated with a B.Sc. (Geology) degree from the Rand Afrikaans University in 1988. In addition, I have obtained a B.Sc. Hons (Geology) from the Rand Afrikaans University in 1994 and attended courses in geostatistics and advanced Datamine modelling and geostatistical evaluation through the University of the Witwatersrand.
3. I am a member/fellow of the following professional associations.

Class	Professional Society	Year of Registration
Member	Geostatistical Association of Southern Africa	2008
Member	South African Council for Natural Scientific Professions (Pr. Sci. Nat. Reg. No. 400201/04)	2004

4 I have practiced my profession from 1988;
5 I have visited the Esaase Project between 29 September and 2 October 2012;
6. I have read the definition of 'Qualified Person' as set out in NI43-101 and certify that by reason of my education and affiliation with a professional association (as defined in NI43-101), I fulfil the requirements to be a 'Qualified Person' for the purposes of NI43-101;
7. I undertook the 2012 Mineral Resource estimate for the Esaase Project;
8. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;
9. I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 13 of the Technical Report entitled 'National Instrument 43-101 Independent Technical Report on Asanko Gold Incorporated's Pre-feasibility Study on the Esaase Project in Ghana';
10. At the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
11. I am independent of the issuer applying all of the tests in Section 1.5 of NI43-101; and
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 27 June 2013 at Johannesburg, South Africa.

C. Muller
B.Sc.Hons (Geol.)
Pr Sci Nat
Director Minxcon

Fanie Coetzee
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CERTIFICATE OF THE AUTHOR OF 'NATIONAL INSTRUMENT 43-101 INDEPENDENT TECHNICAL REPORT ON ASANKO GOLD INCORPORATED'S PRE-FEASIBILITY STUDY ON THE ESAASE PROJECT IN GHANA"

I, Fanie Coetzee, Pr. Sci. Nat (400017/08) do hereby certify that:-

1. I am a Director at Epoch Resources (Pty) Ltd
   Building 22A
   The Woodlands
   South Africa
2. I graduated with a B.Sc.Hons (Environmental Management) degree from the Potchefstroom University in 1996;
3. I am a member/fellow of the following professional associations:-
   International Association for Impact Assessors
   Member South African Council for Natural Scientific Professions (400440/04)

   I have completed numerous Technical Reports and due diligences for gold projects (see attached table).
4. I have practiced my profession from 1997;
5. I have visited the Esaase Project;
6. I have read the definition of 'Qualified Person' as set out in NI43-101 and certify that by reason of my education and affiliation with a professional association (as defined in NI43-101), I fulfil the requirements to be a 'Qualified Person' for the purposes of NI43-101;
7. I have had no prior involvement with the property that is the subject of the Technical Report;
8. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;
9. I am responsible for Section 19 of the Technical Report entitled 'National Instrument 43-101 Independent Technical Report on Asanko Gold Incorporated's Pre-feasibility Study on the Esaase Project in Ghana';
10. At the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
11. I am independent of the issuer applying all of the tests in Section 1.5 of NI43-101; and
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 27 June 2013 at Johannesburg, South Africa.

F. Coetzee
B.Sc.Hons (Env Management.)
Pr Sci Nat (400440/04)
Director

Relevant Gold Project Experience for F. Coetzee

YEAR	CLIENT	COMMODITY	TYPE OF STUDY	PROJECT DESCRIPTION
2009	Development Bank of SA	Coal	Due Diligence	Due diligence study of the Hwange Colliery as part of loan application, SA
2006	Anglo Coal	Coal	Closure Review	Liability assessment associated with mine closure planning, SA
2009	Snowden Resources	Coal	Environmental and closure review	Legal compliance and Environmental and closure review for the Medupi Coal Mine expansion project, SA
2010	GeoPro Mining	Copper	Environmental audit	Legal compliance of the Agrarak Copper Mine, Armenia
2012	Mantra Tanzania Limited	Uranium	Environmental review and ESIA	ESIA amendment for authorisation, Tanzania
2012	Falcon Gold Resources	Gold	Environmental review and ESIA	ESIA amendment for authorisation of the Mkuju Project, Tanzania
2013	ENRC	Coal	ESIA	ESIA for authorisation for the Estima Coal Mine, Mozambique
2012	Todal Mining	Platinum	ESIA	ESIA for authorisation for the Bokai Mine, Zimbabwe
2011	Blackthorn Resources	Zinc	ESIA	ESIA addendum for authorisation for the Perkoa Zinc Mine, Burkina Faso
2010	ARM Platinum	Platinum	Feasibility study, environmental audit and performance assessment	Feasibility study for the Two Rivers Platinum Mine, SA
2011	Iron Ore Producer	Iron	Due diligence	Environmental due diligence and legal compliance of activities, SA
2013	Papillion Resources	Gold	ESIA	ESIA for the authorisation of the Fekola Gold Mine, Mali

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CERTIFICATE OF THE AUTHOR OF 'NATIONAL INSTRUMENT 43-101 INDEPENDENT TECHNICAL REPORT ON ASANKO GOLD INCORPORATED'S PRE-FEASIBILITY STUDY ON THE ESAASE PROJECT IN GHANA"

I, Guy John Wiid Pr.Eng (940269) do hereby certify that:-

1. I am a practicing Professional Engineer and Director of Epoch Resources (Pty) Ltd specialising in the design and management of Mine Residue Disposal Facilities and in Mine Closure Planning
   First Floor, Building 22A, The Woodlands
   Woodlands Drive
   Woodmead
   2080
   South Africa
2. I graduated with BSc Eng (Civil) (1988) and an MSc Eng (Civil) (1995) degrees from the University of the Witwatersrand;
3. I am a member/fellow of the following professional associations:-
   Member : Engineering Council of South Africa (ECSA)
   Associate Member : South African Institute of Mining and Metallurgy (SAIMM)
4. I have completed numerous Technical Reports and due diligences for gold projects as summarised below;
5. I have practiced my profession from 1990 to 2013;
6. I have not visited the Esaase Project;
7. I have read the definition of 'Qualified Person' as set out in NI43-101 and certify that by reason of my education and affiliation with a professional association (as defined in NI43-101), I fulfil the requirements to be a 'Qualified Person' for the purposes of NI43-101;
8. I have had no prior involvement with the property that is the subject of the Technical Report;
9. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;
10. I am responsible for Sections 17.5, 19.5 and 20.1.3 of the Technical Report entitled 'National Instrument 43-101 Independent Technical Report on Asanko Gold Incorporated's Pre-feasibility Study on the Esaase Project in Ghana';
11. At the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
12. I am independent of the issuer applying all of the tests in Section 1.5 of NI43-101; and
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 27 June 2013 at Johannesburg, South Africa.

GJ Wiid
MSc Eng (Civil)
Pr Eng; MSAIMM
Director, Epoch Resources (Pty) Ltd

GJ Wiid - Selection of Recent Gold Project Experience

YEAR	CLIENT	COMMODITY	TYPE OF STUDY	PROJECT DESCRIPTION
2013	London Mining Company / Tenova Bateman	Iron Ore	Bankable Feasibility Study	Design of Tailings Storage Facility, Sierra Leone
2013	Sephaku Fluorspar	Fluorspar	Pre-Feasibility Study	Design of Tailings Storage Facility, South Africa
2012/3	ENRC	Coal	Feasibility Study	Estimate of rehabilitation and closure liabilities for inclusion in EIA / EMP submissions, Mozambique
2012/3	Sedex Minerals	Rare Earths	Preliminary Economic Assessment, Pre-Feasibility Study	Design of Tailings Storage Facility and Waste Rock Dumps, South Africa
2012/3	Exxaro / Aurecon	Coal	Bankable Feasibility Study	Design of Coal Discard Facility, South Africa
2012/3	ENRC	Copper	Review and Ongoing Technical Assistance	Review of Tailings Storage Facility and ongoing technical assistance in re-commissioning and operations for a Tailings Storage Facility, DRC
2012	AngloGold Ashanti	Gold	Pre-Feasibility Study	Definitive Feasibility Design Tailings Storage Facility, Namibia
2012	Nyota Minerals / Golder Associates	Gold	Definitive Feasibility Study	Definitive Feasibility Design Tailings Storage Facility, Ethiopia
2011	Metallon Gold	Gold	Technical and Operational Review of Tailings Operations	Technical review of the operations of 5 Tailings Storage Facilities, Zimbabwe
2011	Platmin	Platinum	Pre-Feasibility Study	Design of Platinum Tailings Storage Facility, Waste Rock Dumps and Surface Water Management Works, North Wast, South Africa
2010	Samancor	Manganese	Pre-Feasibility and Feasibility Studies	Pre-Feasibility and Feasibility Design of Manganese Tailings Storage Facility, Gabon
2009	Sephaku Fluorspar	Fluorspar	Definitive Feasibility Study	Design of Tailings Storage Facility, South Africa
2009	Anglo Coal / Snowdens	Coal	Review and Update of Rehabilitation and Closure Liabilities	Review and update of rehabilitation and closure liabilities for 10 collieries, South Africa
2008/9	Boynton Investments	Platinum	Bankable Feasibility Study	Design of Tailings Storage Facility, Limpopo, South Africa
2008/9	Boynton Investments	Platinum	Bankable Feasibility Study	Design of Tailings Storage Facility, Mpumulanga, South Africa
2006 - 8	Tati Nickel	Nickel	Review and Update of Rehabilitation and Closure Liabilities	Annual review and update of financial provisions for closure
2006	Ferro Nickeli	Nickel	Review and Estimate of Rehabilitation and Closure Liabilities	Review and estimate of closure liabilities for a group of nickel mines, smelter and slag dumps, Macedonia, Kosovo, Albania
2006				
2005	Uranium One	Uranium / Gold	Feasibility Study	Status review and of existing TSF and feasibility design of proposed expansion, South Africa
2005 - Present	Platmin	Platinum	Feasibility and Detailed Design, Ongoing Technical Assistance	Feasibility and Detailed Design of Platinum Tailings Storage Impoundment, North West, South Africa

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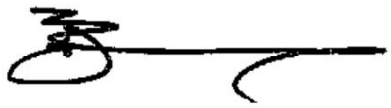
CERTIFICATE OF THE AUTHOR OF 'NATIONAL INSTRUMENT 43-101 INDEPENDENT TECHNICAL REPORT ON ASANKO GOLD INCORPORATED'S PRE-FEASIBILITY STUDY ON THE ESAASE PROJECT IN GHANA"

I, Glenn Bezuidenhout, FSAIMM (Member No. 705704) do hereby certify that:-

1. I am a Process Director (Pty) Ltd
   DRA Campus
   3 Inyanga Close
   Sunninghill
   2157
   South Africa
2. I graduated with a National Diploma from the Witwatersrand Technicon - Johannesburg in 1979;
3. I am a member of the following professional associations:-
   Fellow of the South African Institute of Mining and Metallurgy (Membership No.705704)
   I have experience in gold operations, studies and projects as summarised below.
4. I have practiced my profession from 1979 to now
5. I have not visited the Esaase Project;
6. I have read the definition of 'Qualified Person' as set out in NI43-101 and certify that by reason of my education and affiliation with a professional association (as defined in NI43-101), I fulfil the requirements to be a 'Qualified Person' for the purposes of NI43-101;
7. I have had prior involvement with the property that is the subject of the Technical Report – through previous concept and trade off work;
8. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;
9. I am responsible for Sections 12 and 16 of the Technical Report entitled 'National Instrument 43-101 Independent Technical Report on Asanko Gold Incorporated's Pre-feasibility Study on the Esaase Project in Ghana';
10. At the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
11. I am independent of the issuer applying all of the tests in Section 1.5 of NI43-101; and
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 27 June 2013 at Johannesburg, South Africa.



G. Bezuidenhout
N.D.T. Ex. Met.
FSIAMM
Process Director

YEAR	CLIENT	COMMODITY	TYPE OF STUDY	PROJECT DESCRIPTION
1979 to 1982	Eastern Transvaal Consolidated Mines	Gold	Gold Plant Operations	Plant Superintendent
1982 to 1985	Village Main Reef	Gold	Gold Plant Operations	Plant Superintendent
1985 to 1987	Consolidated Modderfontein Gold Mines	Gold	Gold Plant Operations	Plant Superintendent
2009/10	West Wits	Gold	FS	3.0Mtpa Dump Retreatment – Gold Fields
2010/11	Perseus	Gold	Execution	5.5Mtpa Edikan Gold Plant in Ghana
2012	Aureus	Gold	FS	1.1Mtpa New Liberty Gold Plant in Liberia
2012/13	Pappillon	Gold	PFS	4.0Mtpa Fekola Gold Plant in Mali
2013	Newmont	Gold	Execution	6.0Mtpa Ahafo Gold Plant in Ghana
2012	Keegan	Gold	PFS review	Review of 3rd party PFS for Esaase mine in Ghana
2012	Keegan	Gold	Concept	Trade Off Study for the Esaase mine in Ghana
2013	Asanko	Gold	PFS	Full PFS from mining to economics – Esaase Ghana

Thomas Obiri-Yeboah
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CERTIFICATE OF THE AUTHOR OF 'NATIONAL INSTRUMENT 43-101 INDEPENDENT TECHNICAL REPORT ON ASANKO GOLD INCORPORATED'S PRE-FEASIBILITY STUDY ON THE ESAASE PROJECT IN GHANA"

I, Thomas Obiri-Yeboah, do hereby certify that I am a Senior Mining Engineer of:-

DRA Mining (Pty) Ltd
Bldh H, DRA Mineral Park
3 Inyanga Close, Sunninghill 2157
P.O Box 3567, Rivonia 2128
Johannesburg, South Africa
Telephone: +27 11 517 0638
Fax: +27 11 517 0744

2. I am a graduate in Mining Engineering - a Bachelor of Science and Post Graduate Diploma in Mining Engineering from University of Mines and Technology, Tarkwa Ghana in 1991 and 1992 respectively;
3. I am a member/fellow of the following professional associations:-
   Member South African Institute of Mining Metallurgy 2009
   Member Engineering Council of South Africa 2010
4. I have practiced my profession continuously since graduation;
5. I have not visited the Esaase Project;
6. I have read the definition of 'Qualified Person' as set out in NI43-101 and certify that by reason of my education and affiliation with a professional association (as defined in NI43-101), I fulfil the requirements to be a 'Qualified Person' for the purposes of NI43-101;
7. I have had no prior involvement with the property that is the subject of the Technical Report;
8. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;
9. I am responsible for Sections 14 and 15 of the Technical Report entitled 'National Instrument 43-101 Independent Technical Report on Asanko Gold Incorporated's Pre-feasibility Study on the Esaase Project in Ghana';
10. At the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
11. I am independent of the issuer applying all of the tests in Section 1.5 of NI43-101; and
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 27 June 2013 at Johannesburg, South Africa.

THOMAS K. OBIRI-YEBOAH
BSc./Post Graduate Diploma Mining Engineering
Pr ECSA, MSAIMM
Senior Mining Engineer-DRA Mining

Summary Recent Gold Project Experience T Yeboah

YEAR	CLIENT	COMMODITY	TYPE OF STUDY	PROJECT DESCRIPTION
1999-2001	Ashanti GoldFields	Gold	Prefeasibility/Bankable Feasibility Study	Geita Mine-Tanzania
2002	Ashanti GoldFields	Gold	Due Diligent Study	Teberebie Project - Ghana
2002	Ashanti GoldFields	Gold	Due Diligent Study	Abore Project - Ghana
2002	Ashanti GoldFields	Gold	Due Diligent Study	Abronye Project - Ghana
2002	Ashanti GoldFields	Gold	Due Diligent Study	Akokerri Project-Ghana
2002	Ashanti GoldFields	Gold	Economic Viability	Bibiani Mine Main Pit Cutback - Ghana
2002	Ashanti GoldFields	Gold	Due Diligent Study	Chirano Project-Ghana
2002	Ashanti GoldFields	Gold	Due Diligent Study	Mampong Project-Ghana
2003	Ashanti GoldFields	Gold	Due Diligent Study	Akim Project(now for Newmont)-Ghana
2003	Ashanti GoldFields	Gold	Due Diligent Study	Ntotoroso Project-Ghana
2003	Ashanti GoldFields	Gold	Due Diligent Study	Wassaw Mpohor(now for BogosoMine)-Ghana
2003	Ashanti GoldFields	Gold	Due Diligent Study	Ajopa Project-Ghana
2003	Ashanti GoldFields	Gold	Feasibility Study	Youga Mine Project-Burkina Faso
2003	Ashanti GoldFields	Gold	Due Diligent Study	Essanka Project-Burkina Faso
2003	Ashanti GoldFields	Gold	Due Diligent Study	Ity Mine Project-Cote D,ivoie
2003	Ashanti GoldFields	Gold	Due Diligent Study	Siguiri Gold Mine Project-Guinea
2003	Ashanti GoldFields	Gold	Due Diligent Study	Ran and Phoenix Projects-Freda Rebecca Mines-Zimbabwe
2004	Anglogold Ashanti	Gold	Due Diligent Study	Loulu Project-Mali
2004	Anglogold Ashanti	Gold	Due Diligent Study	Tabakoto Project-Mali
2006	Anglogold Ashanti	Gold	Yearly 'Mine Planning Wheel'	Sadiola/Yatela Mine - Mali
2007	Glencore-Mutanda	Copper/Cobalt	Feasibility Study	Mutanda Mines-Kolwezi, Lubumbashi-DRC
2008	Glencore-KatangaMines	Copper/Cobalt	Mining Engineering	Katanga Mines-Kolwezi, Lubumbashi-DRC
2009	Platinum Australia	Gold	Feasibility Study	Kalplats Project-South Africa
2010	Mantra	Uranium	Feasibility Study	Mkuju River Project-Tanzania
2011	Mantra	Uranium	Definitive Feasibility Study	Mkuju River Project-Tanzania
2012	Mantra	Uranium	Definitive Feasibility Study-2	Mkuju River Project-Tanzania
2013	Uranium One(Russia)	Uranium	Feed Phase	Mkuju River Project-Tanzania