Technical Report on the Feasibility Study of the Massawa Gold Project, Senegal

Report for NI 43-101

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Cautionary Statement on Forward-Looking Information

Certain information contained or incorporated by reference in this technical report, including any information as to our strategy, projects, plans, or future financial or operating performance, constitutes “forward-looking statements”. All statements, other than statements of historical fact, are forward-looking statements. The words “estimate”, “forecast”, “plan”, “evaluate”, “schedule”, “project”, “potential”, “develop”, “expect”, “possible”, “propose”, “following”, “strategy”, “continues”, “target”, “recommend”, “develop”, “suggest”, “indicate”, “appear”, “update”, “design”, “budget”, “can”, “will”, “would” and similar expressions identify forward-looking statements. In particular, this technical report contains forward-looking statements related to Barrick Gold Corporation’s (“Barrick” or the “Company”) Massawa Project (“Massawa” or the “Project”), including, without limitation, with respect to: (i) forecasts of cash flow and net present value, economic assessments, projected capital and operating expenditure, mine life and production rates; (ii) metal or mineral recoveries; (iii) expected grades; (iv) estimates of mineral resources and mineral reserves; (v) benefits of proposed mine design and layout; (vi) projected costs; (vii) proposed infrastructure and future infrastructure requirements; (viii) future exploration and the ability to further add to Massawa’s project inventory; (ix) potential environmental and social impacts related to the execution of the Project; (x) associated royalties payable; (xi) the ability of Barrick to realize the full potential of Massawa in cooperation with the government of Senegal; and (vii) planned rehabilitation for the Project following the completion of mining.

Forward-looking statements are necessarily based upon a number of estimates and assumptions including material estimates and assumptions related to the factors set forth below that, while considered reasonable by the Company as at the date of this technical report in light of management’s experience and perception of current conditions and expected developments, are inherently subject to significant business, economic, and competitive uncertainties and contingencies. Known and unknown factors could cause actual results to differ materially from those projected in the forward-looking statements, and undue reliance should not be placed on such statements and information. Such factors include, but are not limited to: fluctuations in the spot and forward price of gold, copper, or certain other commodities (such as silver, diesel fuel, natural gas, and electricity); the speculative nature of mineral exploration and development; changes in mineral production performance, exploitation, and exploration successes; risks associated with projects in the early stages of evaluation, and for which additional engineering and other analysis is required; disruption of supply routes which may cause delays in construction and mining activities; diminishing quantities or grades of reserves; increased costs, delays, suspensions and technical challenges associated with the development and construction of capital projects; operating or technical difficulties in connection with mining or development activities, including geotechnical challenges and disruptions in the maintenance or provision of required infrastructure and information technology systems; failure to comply with environmental and health and safety laws and regulations; timing of receipt of, or failure to comply with, necessary permits and approvals; uncertainty whether some or all of targeted investments and projects will meet the Company’s capital allocation objectives and internal hurdle rate; the impact of global liquidity and credit availability on the timing of cash flows and the values of assets and liabilities based on projected future cash flows; the impact of inflation; fluctuations in the currency markets; changes in national and local government legislation, taxation, controls or regulations and/ or changes in the administration of laws, policies and practices, expropriation or nationalization of property and political or economic developments in Canada, the United States and Senegal; damage to the Company’s reputation due to the actual or perceived occurrence of any number of events, including negative publicity with respect to the Company’s handling of environmental matters or dealings with community groups, whether true or not; lack of certainty with respect to foreign legal systems, corruption and other factors that are inconsistent with the rule of law; the possibility that future exploration results will not be consistent with the Company’s expectations; risks that exploration data may be incomplete and considerable additional work

may be required to complete further evaluation, including but not limited to drilling, engineering and socioeconomic studies and investment; risk of loss due to acts of war, terrorism, sabotage and civil disturbances; litigation and legal and administrative proceedings; contests over title to properties, particularly title to undeveloped properties, or over access to water, power and other required infrastructure; risks associated with illegal and artisanal mining; the risks of operating in jurisdictions where infectious diseases present major health care issues; disruption of supply routes which may cause delays in construction and mining activities; risks associated with working with partners in jointly controlled assets; employee relations including loss of key employees; increased costs and physical risks, including extreme weather events and resource shortages, related to climate change; and availability and increased costs associated with mining inputs and labor. In addition, there are risks and hazards associated with the business of mineral exploration, development and mining, including environmental hazards, industrial accidents, unusual or unexpected formations, pressures, cave-ins, flooding and gold bullion, copper cathode or gold or copper concentrate losses (and the risk of inadequate insurance, or inability to obtain insurance, to cover these risks).

Many of these uncertainties and contingencies can affect our actual results and could cause actual results to differ materially from those expressed or implied in any forward-looking statements made by, or on behalf of, us. Readers are cautioned that forward-looking statements are not guarantees of future performance. All of the forward-looking statements made in this technical report are qualified by these cautionary statements. Specific reference is made to the most recent Form 40-F/Annual Information Form on file with the SEC and Canadian provincial securities regulatory authorities for a more detailed discussion of some of the factors underlying forward-looking statements and the risks that may affect Barrick's ability to achieve the expectations set forth in the forward-looking statements contained in this technical report.

Barrick disclaims any intention or obligation to update or revise any forward-looking statements whether as a result of new information, future events or otherwise, except as required by applicable law.

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1. Executive Summary

The Massawa Gold Project (Massawa or the Project), located in Senegal, was discovered by Randgold Resources Limited (Randgold), an exploration and mining company that explored, developed, and operated mines in West and East Africa.

On 24th September 2018, Barrick Gold Corporation (Barrick) agreed to merge with Randgold in a zero premium, share-for-share deal. The merger was completed on the 1st January 2019 and resulted in Randgold becoming a wholly-owned subsidiary of Barrick and delisting from the London Stock Exchange and NASDAQ. Randgold is now part of the Barrick group which is listed on the Toronto and New York stock exchanges, and is one of the largest gold mining and exploration companies in the world. Randgold continues to be the owner of the Massawa Permit. On 22nd January 2019, Randgold changed its name to Barrick Gold (Holdings) Limited.

This Technical Report conforms to National Instrument 43-101 Standards for Disclosure of Mineral Projects (NI 43-101) and has been prepared to support the 2018 year-end Mineral Resource and Ore Reserve estimation at Massawa based on feasibility level studies. Recent targeted project development work programmes, including resource drilling and metallurgical testwork in 2017 and 2018, have made a material impact on the understanding of the mineralised bodies and potential of the Project to provide significant economic returns at a range of gold prices. All currency in this report is US dollars ($) unless otherwise noted.

The Project feasibility study includes the open pit material from four pits, namely, Massawa Central Zone (CZ) and North Zone (NZ), Sofia, and Delya. The deposits consist of free milling ore from the oxide contribution of the pits and the fresh material of Sofia and the bulk of the CZ pit. Refractory fresh material is sourced from the northern part of the CZ pit as well as the NZ and Delya pits. The refractory ores have proven to be highly recoverable through a bio-oxidation process. Subsequent to the completion of the feasibility study, an application has been lodged with the Senegalese government to convert the Kanoumba Permit into a mining licence under the 2003 Mining Code of Senegal.

1.1. Location

The Project is located approximately 700 km SE of the Senegalese capital city, Dakar. Dakar is serviced by commercial daily flights from European cities. The proposed main transport route for capital equipment would be by road from the port of Dakar.

The Millennium highway from Dakar to Bamako runs to the south of the permit. The laterite road from Mako to Sabodala, currently being upgraded, provides access to site from the SW. Alternative access from the south is via a laterite road from Bembou. Charter flights are available from Dakar to Kedougou, the regional centre, located 71 km from Massawa, between Bembou and Mako along the Millennium highway.

Massawa can also be accessed by road from Barrick's Loulo Mine in Mali which is located 90 km to the east of Massawa, with an average road journey time of five hours including the international border crossing.

1.2. Ownership

The government of Senegal presently has a non-contributory 10% share of any mine developed on the property. The government's interest is activated at the Exploitation Stage, when a local mining subsidiary must be created. Compagnie Senegalaise de Transports Transatlantiques Afrique de l'Ouest (CSSTAO), a local partner of Randgold in Senegal, has a 6.75% holding of the remaining 90% equity, resulting in Randgold having an effective 83.25% ownership of the Project. Conventions under the 1988 Mining Code (Law No 88-06 of 26th August 1988) covered the original Kanoumering (2002) and Kounemba Permits (2003). Subsequent to the 1988 Mining Code, an updated mining code was passed in November 2003 (Law No 2003-36 of 24th November 2003).

Randgold Senegal had the right under the 2003 Mining Code to accept the new code should it negotiate a new single convention that merged the two permits over the Project. This option was duly exercised, and on 14th April 2010 the two permits were combined into a single Kanoumba Permit of 621 km^2 covered by a single convention based on the 2003 Mining Code. The arrêté (government legal document signed by the Minister of Mines) was approved on 21st May 2010 (No. 04638 MMITPME/DMG) and is valid until May 2019. One three-year extension is allowed after May 2019. On completion of a final feasibility study (FS) and if Barrick takes the decision to construct a mine, a mining permit will be sought for the Kanoumba Permit.

1.3. Geology and Mineralisation

Regionally, Massawa is located on the over 150 km long NE trending Main Transcurrent Shear Zone (MTZ), which is a significant transcrustal dislocation between the Mako Belt (basaltic flow rocks, minor intercalated volcaniclastics, and ultramafic sub-volcanic intrusions) and the Diale-Dalema Basin (volcano-sedimentary to sedimentary rocks) within the Paleoproterozoic (Birimian) Kedougou-Kenieba inlier. Mineralisation is present within various lithologies but is structurally controlled within anastomosing shears which converge to the north.

The Massawa deposit is differentiated into two zones, CZ and NZ, based on differing host rock geology, mineralogy, geometallurgical response and structural controls. Both zones are further divided into two sub-zones. CZ marks the southerly extent of the deposit and is hosted mainly in volcaniclastics, felsic porphyries, and gabbroic intrusions. In the CZ, multiple thin, NE trending, sub-vertical mineralized shears have been identified, each with average thicknesses between 3 m and 10 m and which anastomose horizontally and vertically along strike. Mineralisation in the individual lodes is domained into:

- A distal zone of lower-grade, broad halo of disseminated pyrite and minor arsenopyrite.
- A proximal zone of quartz-stibnite veins and disseminated arsenopyrite and minor pyrite.

The proximal mineralisation contains 80% of the grade in the CZ with coarse visible Au within the veins and Au inclusions within the arsenopyrite and arsenian pyrite disseminations along the silicified contacts of the intrusive rocks.

The northern part of the CZ pit represents a transition zone between CZ and NZ, with structures merging into one main mineralised shear which continues towards the north. NZ has a strike length of 2.5 km and consists of a main NNE trending mineralised structure with discontinuous footwall (FW) and hanging wall (HW) lodes. NZ is further sub-divided into two zones based on structure (NZ1 and NZ2). The southern 1.1 km of the NZ (NZ1) hosts discontinuous, weaker gold mineralisation (average grade of 1 g/t Au to 1.5 g/t Au). The higher-grade but narrow mineralisation is focussed at the margins of a medium-grained greywacke and lithological contacts with contrasting grain size.

NZ2 represents the northern and highest-grade (above 4 g/t Au) portion of the deposit. Mineralisation is predominately confined to a single, continuous, narrow zone (10 m to 15 m average width), which is sub-vertical to steeply dipping (>70°) to the WNW. The mineralisation consists of fine disseminated arsenopyrite and pyrite associated with carbonate and sericite alteration. Gold mineralisation in the NZ is largely refractory in nature, locked up in the crystal lattice of arsenopyrite.

Sofia is located approximately 8 km to the west of Massawa, along the Sofia-Sabodala Shear Zone which hosts the Sabodala gold deposit approximately 20 km to the north. The deposit has been differentiated into two zones based on different structural trends along the tectono-stratigraphic boundary. At Sofia Main the primary structures strike 040°, whereas at Sofia North mineralized structures strike 010°.

Mineralisation at Sofia Main is defined by a linear altered shear zone, where the mineralisation is thought to be structurally controlled. The majority of the high-grade mineralisation (above 2 g/t Au) is situated within a plunging tabular dilation zone, which is modelled to terminate on the FW structure. There are also a number of small discontinuous FW splays of mineralisation.

Sofia North consists of one main NNE trending mineralized structure and a discontinuous FW lode. The main mineralised structure that controls the alteration is developed at the major eastern contact of the Western Mafics that turns in strike from 040° to 010° and has been delineated over more than 2 km by trenching and reverse circulation (RC) drilling.

Delya consists of three parallel zones of mineralisation, over a one-kilometre strike length. The main zone of mineralisation is hosted at the lower margin of the gabbro within highly sheared and silicified and sericitised schist. The mineralised zone varies in thickness from 3 m to 10 m (average of 5 m), contains higher grades (up to 5 g/t Au), and dips to the east at 85°. The other branches are located to the west and have an average dip of 84° to the west. Mineralisation has been drill tested to a vertical depth of 150 m below the surface. The mineralisation consists of strong disseminated fine arsenopyrite and pyrite. Gold is largely refractory in nature and locked up in the crystal lattice of arsenopyrite.

Two satellite deposits with Inferred Mineral Resources occur in a 15 km radius around Massawa (Tina and Bambaraya).

Tina target is located along the Bakan Corridor, where mineralisation is controlled by the Kossanto Shear. Two mineralized branches have been identified; a main western branch, which is steeply dipping, occurs on two sub-parallel sets and is hosted by a felsic intrusion and a set of eastern lower grade mineralised branches that are related to shears within the tuffs and gossanous rocks. Gold mineralisation is primarily associated with disseminated pyrite sulphide assemblages, which are non-refractory in nature.

Bambaraya is located in the NW corner of the Kanoumba Permit along the Sabodala Shear Corridor, approximately 18 km to the north of Sofia. The NE trending shear zone marks the brecciated contact between pillowed basalts and massive and foliated andesites. These volcanic rocks have been intruded by gabbro, dolerite, and felsic plutonic rocks in the vicinity of the deposit.

1.4. Exploration Concepts

Randgold's extensive integrated exploration programmes resulted in the greenfield discoveries of the of the Massawa, Sofia, and Delya deposits.

The Massawa, Sofia, and Delya soil anomalies were initially discovered during the regional Kounemba soil grid sampling programme in 2004 by Randgold Senegal. This was followed up by detailed soil, regolith, and lithological mapping. Field validation of soil anomalies returned mineralised rock chip samples and the first trench identified an intersection of 10.9 m at 2.03 g/t Au. This was followed up by rotary air blast (RAB) drilling and then diamond drilling. Initial wide spaced diamond drill lines, spaced approximately 400 m apart, were used to determine the strike length of mineralisation. Subsequent infill drilling programmes have been completed to build a geological model on the controls of mineralisation and to generate mineral resources.

1.5. Status of Exploration

Recent exploration has been heavily focussed on better understanding the geological controls and the grade distribution along the Massawa CZ deposit and particularly how these relate to the mineability and metallurgy. Drilling of orientation grade control (GC) grids has led to an increased understanding of the structural and lithological controls on the quartz-stibnite veins and surrounding mineralisation. A resource drilling programme at 15 m by 10 m has allowed for modelling the continuity of the coarse gold lodes within the pit.

Follow-up trenching in Massawa NZ and RC drilling over a 150 m strike (5 m by 10 m hole spacing) has confirmed the continuity and high-grade nature of mineralisation in the NZ, with the remainder of the pit being drilled at 40 m by 40 m.

Infill surface trenching, RC, and diamond drilling to a nominal 40 m by 40 m spacing have been completed over the Sofia Main, Sofia North, and Delya deposits. This has confirmed the geological continuity and enabled the estimation of Indicated and Inferred Mineral Resources for these satellite pits.

1.6. Status of Development and Operations

The Project has been competed to a feasibility level as of the 31st December 2018. Indicated Mineral Resources and Probable Ore Reserves have been estimated at Massawa CZ and NZ, Sofia Main, Sofia North, and Delya.

1.7. Mineral Resource Estimate

The Massawa Mineral Resources consist of Massawa CZ, Massawa NZ, Sofia, Tina, Delya, and Bambaraya.

The CZ, NZ, Sofia, and Delya Mineral Resources were re-estimated during 2018 using updated geological interpretations generated from both resource definition and advanced GC (AdvGC) drilling completed as part of the pre-feasibility study (PFS). Additionally, modifying factors for reserve definition have been updated and refined, including updated mining costs, selectivity, and recoveries, which are incorporated as part of the cut-off grade criteria. All Mineral Resource pit optimisations and associated underground reporting areas have been updated accordingly.

The 2018 AdvGC drill programmes in CZ, Sofia, and Delya were designed to determine the optimal drill spacing required to outline an accurate local estimation, which could be used during the mining process.

Outside of the three principal deposits (Massawa, Sofia, and Delya), Inferred Mineral Resources exist for two satellite projects (Bambaraya and Tina). No exploration has taken place on Bambaraya and Tina since the last Mineral Resource estimate in 2013. The satellite deposits have been delineated at 50 m to 100 m spaced drilling and thus estimated using inverse distance squared and as such have been classified as Inferred Mineral Resources. These satellite Inferred Mineral Resources represent less than 28% of the total declared tonnes and less than 7% of the contained gold ounces of the Project. Accordingly, they do not form part of the declared Ore Reserve and have no impact on the current economic evaluation of the Project. Further exploration programmes are planned to be completed on these satellites to evaluate their prospectively with additional resource development.

As at 31st December 2018, the open pit (OP) Indicated Mineral Resource is estimated to be 23 Mt at an average grade of 4.00 g/t Au containing 2.97 Moz of gold and the OP Inferred Mineral Resource is estimated to be 3.7 Mt at an average grade of 2.2 g/t Au for 0.26 Moz of gold. An underground Inferred Mineral Resource, situated below the NZ1 and NZ2 open pit solid, is estimated to be 2.6 Mt at an average grade of 4.1 g/t Au containing 0.35 Moz gold (Table 1-1).

Table 1-1 Massawa Project Mineral Resource Statement as at 31st December 2018

Mineral Resource	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Attributable Contained Gold (Moz)*
OP Measured	-	-	-	-
OP Indicated	23	4.00	3.0	2.5
Total Measured + Indicated	**23**	**4.00**	**3.0**	**2.5**
OP Inferred	3.7	2.2	0.26	0.22
UG Inferred	2.6	4.1	0.35	0.29
Total Inferred	**6.3**	**3.0**	**0.61**	**0.51**

* Attributable gold (Moz) refers to the quantity attributable to Barrick based on Barrick's 83.25% interest in the Massawa Project.
Open pit Mineral Resources are reported as the insitu mineral resources falling within the $1,500/oz pit shell reported at an average cut-off grade of 0.8 g/t Au.
Underground Mineral Resources are those insitu mineral resources below the $1,500/oz pit shell of the North Zone 2 deposit reported at a 2.5 g/t Au cut-off grade.
Mineral Resources are reported inclusive of Ore Reserves.
Mineral Resources for Massawa were generated by Simon Bottoms, MGeol, FGS CGeol, FAusIMM, an employee of the company and Qualified Person.

Barrick is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

The Mineral Resource estimate has been prepared according to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves standards and guidelines published and maintained by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (the JORC (2012) Code). Barrick has reconciled the Mineral Resources to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources, Mineral Reserves dated 10th May 2014 as incorporated into NI 43-101, and there are no material differences.

1.8. Ore Reserve Estimate

The Massawa Ore Reserves have been estimated in accordance with the JORC (2012) Code. Barrick has reconciled the Ore Reserve to CIM Definition Standards for Mineral Resources and Mineral Reserves dated 10th May 2014 as incorporated in NI 43-101, and there are no material differences.

Only Measured and Indicated Mineral Resources have been used for the conversion to Ore Reserves and thus the Massawa, Sofia, and Delya deposits have been incorporated into reserves. Tina, Bambaraya, and Massawa North Zone Underground only contain Inferred Resources and, therefore, have not been converted to Ore Reserves.

The 2018 Ore Reserve estimate includes an OP Probable Ore Reserve of 7.1 Mt at 4.69 g/t Au for 1.1 Moz Au from the Massawa CZ; 4.6 Mt at 4.9 g/t Au for 0.72 Moz Au from the Massawa NZ; 5.7 Mt at 2.9 g/t Au for 0.54 Moz Au for Sofia; and 0.7 Mt at 4.4 g/t Au for 0.92 Moz for Delya. The OP Ore Reserves are those reserves occurring within a $1,000/oz pit design.

Total Massawa, Sofia, and Delya Ore Reserve estimates, as of 31st December 2018, are presented in Table 1-2.

Table 1-2 Massawa, Sofia, and Delya Ore Reserves as at 31st December 2018

Ore Reserve	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Attributable Gold (Moz)*
CZ Probable	7.1	4.69	1.1	0.89
NZ Probable	4.6	4.89	0.72	0.60
Sofia Probable	5.7	2.91	0.54	0.45
Delya Probable	0.66	4.40	0.092	0.077
Total OP Probable	**18**	**4.17**	**2.4**	**2.0**

*Attributable gold (Moz) refers to the quantity attributable to Barrick based on Barrick's 83.25% interest in the Massawa Project.
Open pit Ore Reserves are reported at a gold price of $1,000/oz and include dilution and ore loss factors.
Open pit Ore Reserves were generated by Shaun Gillespie, an employee of the company, under the supervision of Rodney Quick, MSc, Pr Sci Nat. an officer of the company and Qualified Person.

Barrick is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Ore Reserve estimate.

1.9. Technical and Financial Study

A technical and financial study has been conducted by Barrick to support the disclosure of updated Ore Reserves. The study is based on an open pit mining project whereby the ore is mined and fed through an on-site metallurgical plant to produce gold in doré.

1.9.1. Dilution and Ore Loss

Mining dilution of 10% and ore loss of 3% were applied to the Massawa NZ, Sofia, and Delya deposits, based on knowledge of similar ore bodies at the Loulo-Gounkoto complex.

The CZ dilution of 36% and ore loss of 8% was applied based on a grade control optimisation dilution study carried out by Maptek taking into account the very narrow nature in areas of the CZ ore body.

1.9.2. Hydrogeology

The hydrogeological assessment for the Massawa mining feasibility evaluation found naturally shallow groundwater levels across the site (typically 10 m to 30 m below surface), and therefore groundwater inflow to the excavations will occur throughout mining. The groundwater inflow horizons are associated with the near surface weathered horizons, and the NNE-SSW fracture zones and lithological contacts at depth. In general, the sedimentary units are more permeable than the igneous units.

The inflows depend on the hydraulic characteristics of the units, the pit size, and the mining advance rate. Due to the accelerated mining rate of each pit (up to 40 m per year), high groundwater drawdown rates need to be achieved to keep pace with the operation. The likely range of pumping rates per pit during their Life of Mine (LOM) are as follows:

- Massawa Central Zone Pit: average 70 L/s (range: 19 – 96 L/s).
- Massawa North Zone 1 Pit: average 43 L/s (range: 32 – 72 L/s).
- Massawa North Zone 2 Pit: average 18 L/s (range: 7 – 21 L/s.

- Sofia Main Pit: average 10 L/s (range: 1 – 14 L/s).
- Sofia Main Satellite Pit: average 3 L/s (range: 2 – 4 L/s).
- Sofia North Pit: average 17 L/s (range: 12 – 22 L/s).
- Delya North Pit: average 3 L/s (range 1 – 3 L/s).
- Delya South Pit: <1 L/s.

According to the geophysical surveys, the monitoring data, the insitu hydraulic testing, and the calibrated numerical forecast models, the majority of the groundwater can be intercepted by perimeter and in-pit dewatering wells (approximately 80% of groundwater inflow). The remaining water will be drained using sub-horizontal drains drilled into the pit wall. This groundwater seepage will collect in the in-pit sumps.

The geotechnical slope stability analysis can therefore be based on ‘drained’ or ‘dry’ pit slopes. In addition to seepage water, the in-pit sumps’ storage volume and pumping capacity are designed to retain and evacuate a 1:50 year return period storm event (183 mm/24 hours). The size of each sump varies between <1,000 m^3 and 40,000 m^3 and can be accommodated along the pit floor.

The capital and recurring capital investment for the dewatering system is estimated to amount to a minimum of $16.5 million for all pits combined. This investment is spread across the LOM in accordance with the mine advance rates. The operating costs are estimated to be in the order of $3.7 million. An additional contingency should be built into the final costs calculations as adjustments and modifications to mine dewatering plans commonly occur.

1.9.3. Geotechnical Slope Design Parameters

The slope design is defined in terms of three material types:

- Soils: free digging, shovel dressed benches with face angle at 60°; dewatering measures.
- Broken Rock: loosening blasts with vertical buffer wall row and shovel dressed nominal bench face angles.
- Rock: Twin benching with wall control blasting using inclined pre-split, buffer and trim shots; wall drilling angle and berm width varies by domain, mid-level operating offset 2 m.

The standard operating lift (single bench height) has been taken across all material types and domains.

The soils slope design applies across the area and does not depend upon the orientation of the pit walls. The soils slopes, which are dependent on the vertical wall height, are differentiated into two categories:

- Slopes down to <40 m depth (top four benches): generally 44° inter-ramp (I-R) or as locally adjustment for vulnerable slopes
- Slopes where the saprolite/saprock are deeper than or equal to 40 m depth, or zones of vulnerability: 40° I-R

The Broken Rock slopes occur as determined by the base-of-soft and top of fresh planning surfaces. The zone may be absent, or in the greatest extent encountered, 40 m thick. Design geometry reflects the rock slope design, but the wall drilling is vertical and the berm widths are wider to accommodate more loose rock.

The Rock slopes are divided into two domains according the position of the wall with respect to the mineralised zones:

- HW
- FW

The bench face drilling angles and the berm width vary, giving different I-R angles, for each domain and for each of the properties, according to the specific stability conditions indicated by the probabilistic discontinuity-controlled failure analysis.

1.9.4. Mining Method

The mining method proposed for this study is conventional drill and blast followed by truck and excavator. Due to the narrow, multi lode and sub-vertical nature of the ore body with a likely strip ratio of 7.6:1, an open pit mining method is proposed using 2.0 m flitches in ore blasted in 10 m benches, with movement limiting methods such as choke blasting being suggested to minimise dilution.

Using a processing rate of 1.2 million tonnes per annum (Mtpa) per pit as the current target, a mine life of nine years is expected. Based on the average LOM strip ratio of approximately 7:6 (based upon year by year strip ratio), this requires an average digging rate of approximately 6 Mtpa per excavator.

Mine scheduling of the OP reserve is dictated by the different metallurgical ore domains present. The plan focusses on the mining of the oxide and oxidised transition material from all pits first, followed by Sofia fresh, then CZ fresh, and finally Massawa NZ and Delya refractory ore.

Randgold has used various mining contractors over the last 20 years in West Africa and has a firm idea of realistic and achievable rates, productivities, and costs in the region. Mining costs have been based on contract tenders for $3.55/t over the life of the mining operation. The mining cost covers drill, blast, load, and haul from the open pit to the Run-of-Mine (ROM) pad, as well as the re-handle of the mined material into the primary crusher. In addition, there are owner's costs that cover water management, grade control, survey, and mine management. The open pit mining schedule is presented in Table 1-3.

Table 1-3 Open Pit Mining Schedule

Schedule		Year										
		1	2	3	4	5	6	7	8	9	10	LOM
Ore Tonnes Mined - Oxide	kt	2,060	814	375	446	283	255	78	0	0	0	4,311
Ore Tonnes Mined - Transitional	kt	274	942	639	155	208	225	189	0	0	0	2,632
Ore Tonnes Mined - Fresh	kt	29	994	1,805	1,841	1,071	1,415	2,178	1,021	703	0	11,056
Total Ore Grade (g/t Au)		**2.87**	**3.23**	**3.84**	**5.31**	**4.08**	**4.57**	**4.56**	**4.71**	**6.86**	**0.00**	**4.18**
Total Waste Mined	kt	**20,176**	**20,175**	**19,331**	**16,814**	**17,901**	**16,419**	**15,253**	**7,358**	**4,477**	**0**	**137,904**
Total tonnes Mined	kt	**22,538**	**22,925**	**22,149**	**19,256**	**19,463**	**18,315**	**17,698**	**8,379**	**5,180**	**0**	**155,903**
Strip Ratio (Waste/Ore)		8.54	7.34	6.86	6.89	11.46	8.66	6.24	7.21	6.37	0.00	7.66

The grade profile increases as the depth increases due to the plunging nature of the high grade in the NZ. Initial years are dominated by oxide material followed by significant hard rock mining from year 3 onwards (Figure 1-1 and Figure 1-2).

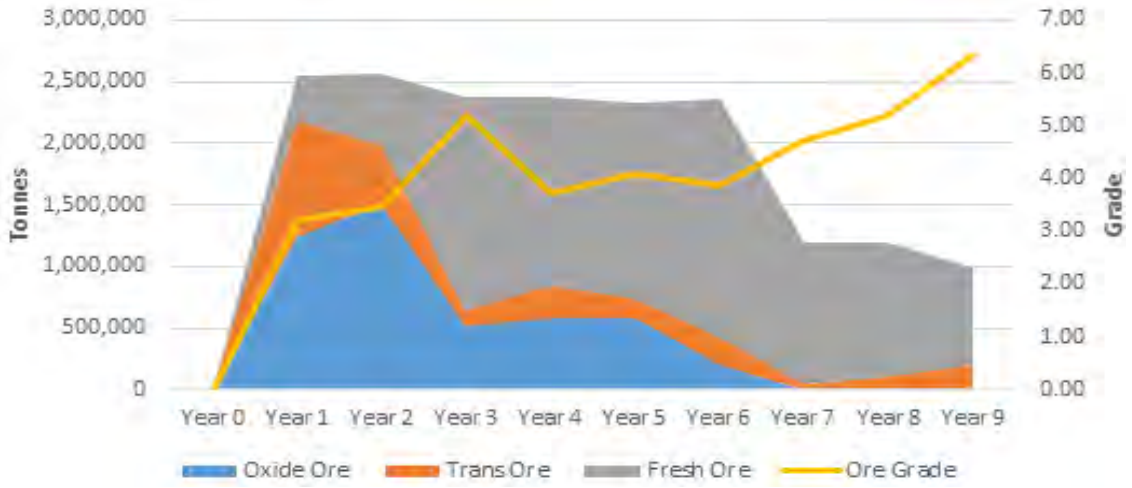


Figure 1-1 Graph of Ore Mining Schedule and Material Type

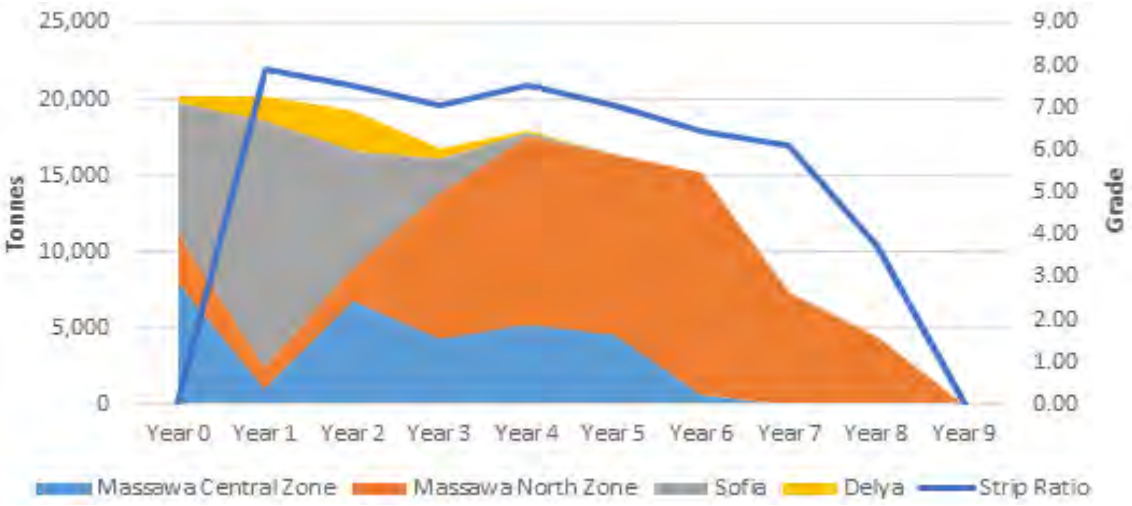


Figure 1-2 Graph of Waste Mining Schedule by Pit

1.9.5. Mining Operating Costs

Mining operating costs are based on contractor mining as shown in Table 1-4, Table 1-5, and Table 1-6. Mining costs average of $3.55/t mined over the LOM.

Table 1-4 LOM Material Movement

Item	Units	Total
Ore	kt	17,999
Waste	kt	137,904
Total	kt	155,903

Table 1-5 Breakdown of Contactor Operating Costs

Item Description	Total LOM Cost ($ '000)	Unit Cost ($ /t mined)
Establishment of Contractor's Facilities	5,775	0.04
Mobilisation of Contractor's Equipment	3,360	0.02
Demobilisation of Contractor's Equipment	3,175	0.02
Monthly Management Fee	103,874	0.64
Preparatory Works	1,086	0.01
Load and Haul	281,574	1.72
Drilling and Blasting	134,028	0.82
RC Grade Control Drilling	5,301	0.03
Ore Re-handle	18,377	0.11
Dewatering	3,623	0.02
Owners Costs	20,000	0.12
Total	**580,173**	**3.55**

Table 1-6 Breakdown of Fixed and Variable Operating Costs

Mining Activity	Unit	Cost
Variable	$ '000	463,990
Fixed	$ '000	116,184
Total	**$ '000**	**580,173**
Variable	$/t	2.84
Fixed	$/t	0.71
Total	**$/t**	**3.55**

A total of 3.55 Mt of ore will be hauled from the Sofia pit to the ROM pad at Massawa. The hauling cost has been calculated at $3.20/t hauled. This amounts to a total of $11.35 million over the LOM based upon existing contracts from other Barrick operating mines.

1.9.6. Processing and Metallurgy

The Massawa deposits consist of free-milling ore in Sofia and most of CZ, and a refractory portion in the northern part of CZ as well as NZ and Delya fresh. The refractory ores have been proven to be highly recoverable through bacterial oxidation (BIOX) as an oxidative step.

The CZ and the refractory ore contain deleterious elements such as arsenic (As) and antimony (Sb). Where the CZ is not processed through BIOX, testwork has proven that the As and Sb can be removed successfully by a precipitation plant designed by Multotec. During the refractory processing phase, the neutralisation section of the BIOX plant will remove the precipitate and these elements. As an additional mitigation, the Multotec precipitation plant will process any liquor that is released from the tailings storage facility (TSF) to the environment.

There have been a number of testwork programmes completed on Massawa ores. Some testwork was performed to help validate the results of historic studies. During the PFS and FS phases, several trade-offs were completed. Process specific trade-offs were generated and used as part of process development, and ultimately the design of the plant.

The extensive metallurgical testwork campaigns demonstrate two distinct behavioural patterns where some ore sources, in particular the oxides, and some fresh rock sulphide sources from Sofia Main, Sofia North, and most of CZ, exhibit free-milling characteristics suitable for gold extraction by a conventional carbon-in-leach (CIL) process. These ores also contain gravity recoverable gold (GRG). In CZ, the GRG can be as high as 50% of the available gold.

Other ore sources in the north of CZ, the NZ pit, and Delya exhibit very low GRG and are highly refractory. Flotation and BIOX is combined to achieve recoveries in excess of 85% overall. The flotation tailings grades achieved in the testwork were below 0.4 g/t Au and exhibited very low CIL recoveries.

The Massawa plant design has to cater to these observations through two distinct processing circuits which will run sequentially:

- For the first seven years, the free-milling ore sources will run through a conventional gravity and CIL circuit, which includes an arsenic and antimony precipitation processing plant before TSF release to environment.
- From year 7 onwards, the refractory ore sources are processed through a flotation circuit with a concentrate fine grinding step to P80 = 45 µm, followed by a BIOX step and CIL.

Sampling for testwork, was completed by the geologists in conjunction with the modelled and simulated pits and mine plans. The samples were selected to ensure spatial as well as geological representivity. This approach ensured that the testwork completed is representative and that the optimal results can be incorporated with high confidence into the design criteria, and ultimately the plant design and sizing.

1.9.7. Recovery

The gold recovery achieved from the free-milling ores is highly dependent on the grade processed through the circuit.

A best fit linear curve was drawn to give predicted tails grade from head grade values, and using this relationship, predicted tails were calculated for each head grade and the corresponding gold dissolution were determined. The predicted gold dissolution for the given head assays provides a method for predicting the recovery of a reserve, given its feed grade.

CZ was a special case exhibiting a wide range of recoveries along strike with the result that a specific geometallurgical recovery curve was generated for this ore body based largely on grade but also the position of the lodes.

Table 1-7 provides a summary of the recoveries per ore type and weathering.

Table 1-7 Recoveries Summary per Ore Type and Weathering

WOL 2.7, 2.4 and BIOX 1.2 Mtpa		Central Zone			North Zone		Sofia Main	Sofia North	Delya		
		WOL	BIOX - 100% CZ	BIOX - 25% CZ / 75% NZ	WOL 100% NZ	BIOX 100% NZ	WOL	WOL	WOL 100% Delya	BIOX 100% Delya	BIOX 20% Delya 80% NZ
Recoveries	Oxide	94%	N/A	N/A	90%	N/A	92%	91%	94%	N/A	N/A
	Oxide Trans	94%	N/A	N/A	89%	N/A	92%	91%	90%	N/A	N/A
	Reduced Trans	Geomet Model	57%	57%	51%	57%	90%	89%	52%	57%	57%
	Fresh	Geomet Model	87%	88%	13%	88%	89%	85%	9%	92%	89%

The proposed whole ore leach (WOL) and refractory process plant design is based on well-known and established gravity/CIL technology, which consists of crushing, milling, and gravity recovery of free gold followed by leaching/adsorption of gravity tailings, elution and gold smelting, and tailings disposal. The refractory process which includes sulphide flotation, regrind, and BIOX process, is also a well-known technology which will be supplied by Outotec at a later stage. Services for the plant will include reagent mixing, storage and distribution, water, and air services.

1.9.8. Processing Operating Cost

Operational expenditure is derived from first principles and reagent consumptions directly related to testwork. Benchmarking and the SENET in-house database was used for any assumptions when data was not available.

Plant costs average of $21.2/t ore but include a range of costs dependent on ore feed and process route. The process operating costs were generated based on the various process routes possibly required. All of the deposits were considered including Massawa CZ, Massawa MZ, Sofia Main, Sofia North, and Delya, thus the operating cost estimate was split by the various ore types as shown in Table 1-8.

Table 1-8 Process Operating Cost Estimate Summary

($/t milled)	Central Zone			North Zone		Sofia Main	Sofia North	Delya		
	WOL	BIOX – 100% CZ	BIOX – 25% CZ/ 75% NZ	WOL 100% NZ	BIOX 100% NZ	WOL	WOL	WOL 100% Delya	BIOX 100% Delya	BIOX 20% Delya 80% NZ
Oxide	12.59	N/A	N/A	12.90	N/A	12.46	13.39	13.31	N/A	N/A
Oxide Trans	16.44	N/A	N/A	15.50	N/A	14.97	14.86	15.88	N/A	N/A
Reduced Trans	17.95	57.05	41.52	16.71	40.37	16.41	15.75	16.17	62.41	44.25
Fresh	19.58	57.05	41.52	17.93	40.37	17.84	16.48	17.81	62.41	44.25

1.9.9. Deleterious Elements

Mitigation measures are described below and the associated costs have been captured within the unit process operating costs.

Cyanide

Massawa chooses to abide by the guidelines of the International Cyanide Code. The TSF design has been lined with a high-density polyethylene (HDPE) liner. Protocols call for limited threshold

discharges to the TSF and cyanide discharge concentrations are controlled through use of an on-line cyanide analyser and controller. Further mitigation is included at the discharge end of the TSF, where a polishing cyanide destruction step can be included to ensure legal requirements are met before release to the environment. During the refractory ore treatment, an additional ASTER cyanide destruction facility, also situated at the TSF, is added to destroy the thiocyanate to below the 7 ppm level.

Arsenic, Antimony and Copper

The CZ, NZ, and Delya ores have a presence of arsenic, copper, and antimony. The NZ and Delya material is processed through a neutralisation step within the BIOX circuit of the main plant. Arsenic and other deleterious elements are, to a large extent, eradicated before any tailings are pumped to the TSF.

Testwork has proven that all three deleterious elements can be precipitated and meet the stringent World Health Organisation (WHO) and International Finance Corporation (IFC) standards through mitigation measures via a precipitation plant through which any water released from the TSF to the diversion dam will be processed. The plant is of a modular construction capable of treating in increments up to a maximum of 300 m^3/h.

In-pit dewatering and other waste dump run-off also indicated the presence of arsenic and antimony. These are at much lower levels and a simple greensand filtration system has been included in the design, which is highly effective at simple absorption at these lower concentrations. The greensand is manufactured by coating the mineral glauconite with manganese dioxide, while pyrolusite is a naturally mined ore composed of solid manganese dioxide. Greensand media is commercially available and has been shown to be capable of removing up to 80% of arsenic by oxidation/adsorption. The filtration sands are then replaced every two years and the plant can treat up to 600 m^3/h.

1.9.10. Logistics and Transport

Transport to site is not expected to pose any issues. A central store at Massawa will hold all supplies for the Project and the same transport route as presently used for the Loulo Mine will be used at Massawa. Access from Dakar to Massawa will be via the new sealed Millennium highway linking Dakar to Bamako. An all-weather laterite service road will run between the sealed road and the mine.

The supply chain will be outsourced to a specialist service provider. Capital goods and services will be sourced from the most cost-effective source. Bulk specialized capital items will be sourced locally whenever feasible; where not available locally, capital goods and consumables will be sourced from the rest of the world, given the favourable costs and freight rates for shipment to Senegal.

1.9.11. Social and Environmental

Independent consultants Digby Wells Environmental (Jersey) Limited (Digby Wells), in conjunction with Tropica Environmental Consultants (Tropica) of Senegal, were contracted to

conduct the environmental and social impact assessment of the proposed Massawa Mine. As is common with mining projects in West Africa, several potential positive and potential negative impacts have been identified. An environmental and social management plan (ESMP) has been completed to ensure these impacts are systematically and correctly managed.

Potential positive impacts include:

- Employment opportunities.
- Local procurement of goods.
- Skills development.
- Increase in local trade.
- Increased contribution to the national economy.
- Community Development Plan (Education and training, local economic development and infrastructural improvements for local communities including roads, schools, clinics, water provision etc).
- No physical resettlement required.

Potential negative impacts include:

- Dust and noise generation.
- Visual impact due to the development of rock dumps.
- Migration of job seekers to the area.
- Removal of artisanal and small-scale miners out of the mining permit.
- Direct and indirect impacts to critical biodiversity and habitat within the Niokolo-Koba National Park catchment.
- Disturbance of populations of Critically Endangered Western Chimpanzee populations confirmed within the planned mining area.

No impacts which could present a fatal flaw to the successful execution of the Project have been identified to date. The ESMP includes a number of recommended mitigation measures that, if implemented effectively, will enhance the positive impacts of the Project, and minimise the negative effects. With this effective implementation of the ESMP, none of the negative impacts are believed to be sufficiently significant to prevent the development of the proposed Project. The potential positive impacts associated with the local and national Senegalese economies are expected to be significant.

1.9.12. Human Resources

Barrick's well tested and proven human resources policies and procedures including mine safety, amended where necessary, will be applied on the Project. Barrick's policy is to promote nationals of the host country to manage the Project. Where locally qualified and experienced staff are not available, recruitment from elsewhere is undertaken, with the clear understanding that local personnel are given the training and experience required to allow them to replace the expatriates as soon as possible.

The Project will comprise the following phases:

1. Pre-construction phase.
2. Construction phase.
3. Operational phase throughout the life of mine.
4. Closure and rehabilitation phase.

Community consultation, liaison, and development will be a key focus throughout each of the above-mentioned phases, and the Project will employ experts on community development to ensure that effective and constructive relationships with the communities surrounding the mine are maintained.

The Project will comply with the guidelines set out in the Senegalese Labour Code

The Project will comply with all labour provisions as enacted in 'The Employment Act' – Law N°97-17 of 1st December 1997 and 'The Industrial and Labour Relations Act' Paragraph 2 of the law of 15th December 1952 as well as the provisions from the National Collective Inter-Professional Convention of 27th May 2082 and the Mining Code Law n°2003-36 of 24th November 2003. The above conventions will be used when recruiting employees and will base recruitment decisions on qualifications and experience required to do the job. Preference will be given throughout all phases of the Project to job applicants with the required competencies or qualifications in the following order:

1. Applicants from the local villages surrounding the Project.
2. Candidates who reside in the administrative area, then the region, then from other parts of Senegal.
3. Candidates from other West African countries.
4. If qualified candidates are not available from the previously mentioned areas, qualified candidates from other countries will be sourced.

It is expected that at the peak of the construction phase of the Project approximately 2,000 construction employees would be employed on the Project.

During the operational phase of the mine, it is envisaged that 1,500 employees would be employed as Massawa Mine employees, including contractors (Table 1-9). During the operational phase, the majority would be provided by the mining contractor. Other contractors would include security. Where possible national suppliers will be used, which will have a multiplier effect on the national economy.

Table 1-9 Massawa Manpower

Massawa Department	Total
Mining	23
Mining Contractor	626
Blast Contractor	13
Mineral Resources	67
Process Plant	211
Plant Engineering	129
Administration	10
Finance	11
Human Resources	5
Community	4
S.H.E Manager	1
Safety	3
Health	10
Environment	6
Security	398
Warehouse & Logistics	31
Information Technology	8
Total	**1,556**

1.9.13. Infrastructure

The selected Massawa site is a greenfield site without any existing infrastructure although some laterite roads do exist, providing access to local villages scattered around the area.

The on-site infrastructure required will be related to the processing plant and the supporting facilities as follows:

- In-plant access roads.
- Plant buildings.
- Plant reagents and consumables stores.
- Process plant site drainage.
- Sewerage disposal.
- Security.
- Water supply.
- Communications.
- Power supply.

The proposed infrastructure will support the mining and plant operations. The main off-site infrastructure required for the development of the Project will be the following:

- Airstrip.
- Road network.
- Office complex.
- Accommodation facilities.
- Rail infrastructure.
- Diesel fuel storage and refuelling facility.
- Heavy Fuel Oil (HFO) / Light Fuel Oil (LFO) storage facility.

- Power plant.
- Water supply system.
- Waste management.
- Sewerage disposal site.

1.9.14. General and Administration

General and administration (G&A) costs cover all operational costs outside of mining and processing. They include administrative, finance, medical, environmental, and social costs, together with outside engineering which covers all engineering costs outside of the processing plant. The G&A cost equates to $8.60/t of ore processed over the LOM. This correlates well on a gross level to Barrick's Tongon Mine, which is also a West African coastal country operation.

1.9.15. Capital Costs

The capital programme is broken into a number of phases with each phase employing additional processing units and technology to optimally recover the gold content.

- Phase 1: Initial capital to construct the mine and treat oxide and oxidised transition is expended over the first two years of the Project.
- Phase 2: Processing of fresh ore from Sofia and CZ deposits commences. The processing of Sofia fresh ore will continue as per the Phase 1 leach and detoxification process. CZ ore will be processed through a gravity circuit prior to CIL.
- Phase 3: Construction of the BIOX circuit takes place in year six which will be designed to process the sulphide material in the northern part of the CZ, NZ, and Delya pits.

Table 1-10 lists the capital cost estimate by phase.

Table 1-10 Capital Cost Estimate per Phase

Description	Phase 1	Phase 2	Phase 3	Total
Direct Field Cost	91,278,343	10,933,099	47,559,136	149,770,578
Indirect Field Costs	21,023,557	4,253,992	12,309,577	187,357,703
Home Office Costs	12,488,244	1,456,895	5,538,306	19,483,445
Total Plant Cost	**124,790,144**	**16,643,986**	**65,407,019**	**206,841,148**
Other Costs	**147,803,128**	**181,009**	**7,541,511**	**155,525,648**
Mining Costs	**37,102,894**	**6,525,273**	**6,565,398**	**50,193,565**
Total Construction Capital	**309,696,166**	**23,350,267**	**79,513,928**	**412,560,361**

The estimated LOM capital expenditure is detailed in Table 1-11.

Table 1-11 LOM Estimated Capital Expenditure

Item	Total
Construction & Project Capital	412,560,361
On-going Capital	12,362,500
Pre-Production Capitalised	16,933,000
Rehabilitation Asset	23,000,000
Total	**464,855,861**

1.10. Economic Analysis

1.10.1. Basis of the Economic Analysis

An economic assessment to confirm the reserve status of the Massawa, Sofia, and Delya pits was carried out based on the key parameters summarised below:

- Total ore mined from Massawa, Sofia Main, and Delya pits of 18 Mt of ore at an average grade of 4.2 g/t Au containing 2.4 Moz of gold.
- Strip ratio of 7.6:1 to give total tonnes mined of 156 Mt.
- Mining costs average of $3.55/t mined over the LOM.
- Haulage cost average of $1.20/t of ore milled over the LOM ($0.18/t km hauled).
- Plant cost average of $21.33/t ore but include a range of costs dependant on ore feed and process route.
- G&A costs of $8.76/t ore milled over LOM, including outside engineering costs.
- Pre-production related capital amounts $16.9 million. $80 million will be spent on pre-production mining and absorbed in Total Cash Cost as the ore is fed.
- Capital construction cost of $413 million.
- On-going capital of $12 million over the LOM.
- Rehabilitation cost of $23 million at the end of the LOM.

The financial model is based on annual cash flow projections, with technical and economic parameters stated above using constant money terms. No escalation or de-escalation has been applied. In generating the financial model for the operations at the Project, the following principles were adopted:

- Financial implication on the methods of funding was not evaluated, since it has been assumed that the Project will be financed by Barrick.
- Annual figures are based on financial years 1st January to 31st December.
- Real term annual cash flows were used to calculate the internal rate of return (IRR), net present values (NPV), and simple and discounted payback periods in real after-tax terms.
- Costs up to start of construction are considered as sunk costs.
- No salvage value for plant and equipment on cessation of operations was included.
- Calculations are based in US dollars ($).

1.10.2. Production and Cash Flow Forecast

The estimated production and cash flow forecast are summarised in Table 1-12.

Table 1-12 Year 1 to 10 Production and Cash Flow Forecast

Item	Year											
	-1	0	1	2	3	4	5	6	7	8	9	Total
Production (koz)	-	-	226	257	295	201	212	211	159	173	158	1,893
Cash Flow ($ Million)	(106)	(247)	(3)	109	154	62	80	20	31	79	79	258

1.10.3. Financial Analysis

A financial model was run using a $1,000/oz gold price feeding the reserve mining schedule, together with a 3% royalty on revenue, seven-year tax holiday (two-year construction, five years for operation), followed by corporate tax at a 25% rate, which produced a total net cash flow after tax of $258 million, and IRR of 12%. Payback is five years from start of production. A sensitivity table on NPV, IRR, and payback against gold price is supplied in Table 1-13. The Project is profitable at $1,000/oz and thus justified to be reported as an Ore Reserve at $1,000/oz gold price.

Table 1-13 Project Financial Analysis

Discount Rate	Gold Price ($/oz)		
	1,000	1,200	1,400
0%	$258 million	$591 million	$925 million
5%	$114 million	$361 million	$608 million
10%	$24 million	$212 million	$400 million
IRR	12%	25%	37%
Payback	5 years	2.8 years	2.4 years

1.10.4. Government Revenue

The government revenue earnings from the Project are sourced from the following assumptions incorporated into the financial model:

- 3% royalty on revenue.
- Tax rate of 25%. Payments are made throughout the year after the seven-year tax holiday from issuance of the mining permit.
- Dividends from 10% free carry share of the Project, which are payable after capital has been redeemed.
- Other taxes which include withholding taxes on dividends and salaries.

Under the current Kounemba Convention, Barrick will be exempt from all taxes, levies, and duties for a period of seven years from issuance of the mining title.

A sensitivity of expected government revenue based on the cash flow is detailed in Table 1-14

Table 1-14 Government Revenue Sensitivity

Revenue	Unit	Gold Price ($/oz)	
		1,000	1,250
Royalty	$ million	57	71
Company Tax	$ million	14	56
Dividends	$ million	21	59
Other Tax ($M)*	$ million	62	105
Total	**$ million**	**154**	**291**

*Other taxes include withholding taxes on dividends and salary taxes.

1.10.5. Sensitivity Analysis

The proposed Project is profitable at current gold prices, but sensitive to gold price fluctuations and therefore becomes a marginal project at gold prices lower than $1,000/oz, but very attractive at current or higher gold prices (Table 22-4Table 1-15).

Table 1-15 NPV Sensitivity at Different Gold Prices and Discount Rates

$ millions	Gold Price ($/oz)						
Discount	**900**	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
0%	84	258	425	591	758	925	1,091
5%	(14)	114	238	361	484	608	731
10%	(73)	24	118	212	306	400	493
15%	(108)	(33)	40	113	186	259	332
20%	(129)	(70)	(12)	46	104	162	220
25%	(141)	(93)	(46)	0	47	94	140

The Project is fairly resilient to grade changes and at current gold prices would be able to absorb a 10% reduction in grade (Table 1-16). The Project is sensitive to gold price fluctuations with a $1,000/oz and a 20% reduction in grade making the Project very marginal. Conversely, at higher gold prices than current, the Project is very resilient to a grade reduction.

Table 1-16 NPV at 20% Grade Variation at Different Gold Prices

Grade	$ millions	Gold Price ($/oz)						
		900	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
3.35	-20%	(246)	(100)	47	189	324	457	591
3.77	-10%	(81)	84	241	391	541	691	841
4.19	**0%**	**84**	**258**	**425**	**591**	**758**	**925**	**1,091**
4.60	10%	242	425	608	792	975	1,158	1,341
5.02	20%	392	592	792	992	1,192	1,392	1,592

At current gold prices, the Project can absorb a 20% increase in operating costs and still remains largely profitable (Table 1-17). The Project will require a gold price of less than $1,000/oz and an increase of 20% in operating costs to become a marginal project.

Table 1-17 NPV at 20% Change in Operating Costs

Operating Cost		Gold Price ($/oz)						
Operating Cost ($/t)	**$ millions**	**900**	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
49.57	-20%	280	447	613	780	946	1,113	1,280
55.77	-10%	184	352	519	686	852	1,019	1,185
61.97	**0%**	**84**	**258**	**425**	**591**	**758**	**925**	**1,091**
68.16	10%	(19)	162	330	497	664	830	997
74.36	20%	(123)	61	235	403	569	736	903

The Project is sensitive to increases in capital cost (Table 1-18). At a $1,000/oz gold prices, an increase of 10% on capital reduces the NPV by $42 million, or 16% of the NPV.

Table 1-18 LOM Capital Cost Sensitivity

Capital $ millions	Gold Price ($/oz)						
	900	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
-15%	150	318	485	652	818	985	1,151
-10%	129	298	465	631	798	965	1,131
-5%	107	278	445	611	778	945	1,111
0%	**84**	**258**	**425**	**591**	**758**	**925**	**1,091**
10%	38	217	385	551	718	884	1,051
25%	-30	153	324	491	658	824	991
50%	-143	40	221	391	557	724	890

1.11. Implementation

The FS capital cost estimate has been compiled on the basis that Barrick will adopt an Engineering, Procurement and Construction Management (EPCM) approach to implementing and executing the Project.

Upon the onset of the Project and following a formal EPCM Contractor tender, selection, and award process to be conducted by Barrick, some 1st Phase early project execution activities may take place such as front-end engineering and design (FEED) of critical deliverables, which can commence in the first quarter of 2019 as Project approval is expected from both the Senegal government and the Barrick Board before June 2019.

The implementation strategy to be adopted is generally structured into four broad stages undertaken by the successful EPCM Contractor under the direct auspices of Barrick:

- Detailed design of the process plant and infrastructure whereby the FS capital cost estimate is further developed into a Barrick approved and EPCM controlled budget estimate for the Project going forward during execution.
- Procurement comprising formal tender, adjudication, award and thereafter fabrication and expediting with logistics undertaken by the Barrick nominated logistics supplier.
- Construction management and cost control.
- Commissioning management and handover to operations.

1.12. Alternate Case Ore Reserves and Economics at $1,200/oz Gold Price

Historically Barrick estimates its Ore Reserves using a gold price assumption of $1,200/oz. In order to investigate the effect on the Massawa Gold Project of using a $1,200/oz gold price, an alternate case has been prepared which results in the following upsides to the Project:

- A reduction in the cut-off grade results in an increase in the total Ore Reserves at a slightly lower grade, thus extending the mine life by 1.5 years.
- The size of the Massawa, Sofia, and Delya pits is increased by 6%.

- The increase in the total Ore Reserves results in a more efficient use of capital as the capital construction cost is expended over more tonnes.

For the alternate case the Ore Reserve was rerun at a gold price of $1,200/oz. No changes have been made to the mining dilution, ore loss, or gold recovery between the base case and the alternate case. For the alternate case, the cut-off grade for each of the deposits was re-estimated based on the higher gold price. A life of mine plan for the alternate case was determined based on the $1,200/oz Ore Reserve.

The 2018 Ore Reserve estimate for the alternate case at a $1,200/oz gold price includes an open pit (OP) Probable Ore Reserve of 7.8 Mt at 4.6 g/t Au for 1.2 Moz Au from the Massawa CZ; 5.2 Mt at 4.7 g/t Au for 0.8 Moz Au from the Massawa NZ; 7.1 Mt at 2.7 g/t Au for 0.6 Moz Au for Sofia; and 0.81 Mt at 4.21 g/t Au for 0.11 Moz for Delya. The OP Ore Reserves are those reserves occurring within a $1,200/oz pit design.

Total Massawa, Sofia, and Delya Ore Reserve estimates, as of 31st December 2018 within a $1,200/oz OP, are presented in Table 1-19.

Table 1-19 Massawa, Sofia, and Delya Ore Reserves as at 31st December 2018 at $1,200/oz Gold Price

Ore Reserve	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Attributable Gold (Moz)*
CZ Probable	7.8	4.59	1.15	0.96
NZ Probable	5.2	4.67	0.79	0.65
Sofia Probable	7.1	2.66	0.61	0.51
Delya Probable	0.81	4.21	0.11	0.091
Total OP Probable	**20.9**	**3.94**	**2.6**	**2.2**

*Attributable gold (Moz) refers to the quantity attributable to Barrick based on Barrick's 83.25% interest in the Massawa Project. Open pit Ore Reserves are reported at a gold price of $1,200/oz and include dilution and ore loss factors. Open pit Ore Reserves were generated by Shaun Gillespie, an employee of the company, under the supervision of Rodney Quick, MSc, Pr Sci Nat, an officer of the company and Qualified Person.

A financial model was run on the alternate case using a $1,200/oz gold price feeding the alternative case reserve mining schedule, together with a 3% royalty on revenue, seven-year tax holiday (two year construction, five years for operation), followed by corporate tax at a 25% rate, which produced a total net cash flow after tax of $696 million, and IRR of 28%. Payback is 2.5 years from start of production. Table 1-20 compares the base case financial model run at a $1,000/oz gold price versus the alternate case financial model run at a $1,200/oz gold price.

Table 1-20 $1,000/oz Gold Price Base Case Results versus Alternate Case $1,200/oz Gold Price

Item	Units	Base Case	Alternate Case
Gold Price	US$/oz	1,000	1,200
Total Ore Mined	Mt	18	21
Grade	g/t Au	4.19	3.92
Contained Gold	Moz	2.4	2.6
Total tonnes Mined	Mt	156	166
Strip Ratio	w:o	7.6	6.9
Mining Cost	$/t mined	3.55	3.55
Haulage Cost Average	$/t milled	1.20	1.25
Plant Costs	$/t milled	21.20	18.02
G&A Costs	$/t milled	8.76	8.60
Pre-production Capital	$ million	16.9	0
Capital Construction Cost	$ million	413	413
On-Going Capital	$ million	12	16
Rehabilitation Cost	$ million	23	23
Net After-Tax Cash Flow	$ million	258	696
$NPV_{5\%}$	$ million	114	421
$NPV_{10\%}$	$ million	24	251
IRR	%	12	28
Payback	Years	5	2.5

1.13. Interpretation and Conclusions

Barrick has documented standard procedures for the drilling, logging, and sampling processes, which meet industry standards. The mineralisation wireframe parameters at Massawa, Sofia, and Delya are based on visibly identifiable geological contacts, which ensure that a geologically robust interpretation can be developed. Randgold's procedures ensure a reliable database of exploration information, but the implementation of a digital system is an opportunity for improving the systems and checks in place.

Massawa Mineral Resources are estimated using industry accepted methods. Portions of the Massawa mineralisation have been recognised to have significantly higher grades than the remainder of the mineralisation, and the top cutting, domaining, and estimation approach taken by Randgold to limit the effects of the high grades is considered to be appropriate. The Qualified Persons consider the Mineral Resource estimates at the Project to be appropriately estimated and classified.

The Qualified Persons concur with the parameters used in the Mineral Resource to Ore Reserve conversion process.

The strategic focus for Massawa has been to prioritise the Sofia and CZ ore over the refractory ores of the Massawa NZ. Consequently, increasing the reserves of non-refractory material will further benefit the Project and active exploration work is continually underway targeting additional areas on the permit.

The open pit mining operations at Massawa, Sofia, and Delya will consist of multiple open pits, i.e. CZ, NZ, Sofia Main, Sofia North, and Delya. The open pits are being planned to be mined by a mining contractor and a down-the-hole blasting service will be provided by an appropriate blasting contractor. The proposed mining method of conventional 90 t truck and excavator open pit mining is appropriate for the ore body and suitable dilution and ore loss factors have been applied. Randgold has significant experience in other mining operations in the region on similar

ore bodies to Sofia, Delya, and NZ and has compared production, modification factors, and costs against these operations to ensure they are suitable. The CZ ore body has execution risk in that the bulk of the gold is hosted within thin lodes containing a large coarse gold component. Higher dilution and ore loss factors have been applied to this ore body to compensate. Any misallocation or misinterpretation of the CZ ore body will result in a loss of value. As such, a detailed GC programme will be an important requirement to successfully mine the CZ ore body.

Significant testwork has already been undertaken on the various ore types from oxide and free leaching ores of Sofia, to the high gravity and partially refractory ores of the CZ and the highly refractory ores of the Massawa NZ and Delya. Based on testwork completed, the overall recoveries of 78% for the Project are realistic.

The processing feed plan extends over a nine-year period where the WOL plant has a nameplate capacity of 2.4 Mtpa of fresh ore which can increase to 3.0 Mtpa when treating the softer oxides. The plant is divided into two streams, i.e. two parallel grinding ball mills form the hub of the processing plant. Each subsequent process route is implemented sequentially from a single stream. This means that the flotation circuit is erected well into the mine life in conjunction with the BIOX circuit, which matches the phase capital schedule.

The completed Environmental and Social Impact Assessment (ESIA) analysis includes all required specialist studies. The Project is within the Niokolo-Koba headwaters and within a relatively environmentally pristine area. Significant effort has been made to generate an Environmental Management Plan that is practical and effective in minimising the impact of the Project on the environment. An inclusive transparent approach that utilises the concept of environmental offsets to benefit the regions biodiversity preservation will be implemented.

1.14. Recommendations

It is recommended that additional exploration be conducted to expand the non-refractory ore to extend the life of mine and improve the economics and pay back of the Project. It is recommended that the entire Massawa, Sofia, and Delya Mineral Resources be drilled to an AdvGC spacing suitable to the variography for each of the pit's ore types prior to commencement of mining.

Process operating costs have been estimated based on specialist studies on the variable ore types and it is recommended that optimisation studies be undertaken on the CZ and BIOX amenable ore types to further optimise the process recoveries and costs. Mining costs have been developed from the first round of mining tender and it is recommended that the final mine plan be submitted to the short list of tender contractors to obtain the most efficient cost, and a trade-off be done against an owner mining option.

A full updated ESIA has been completed for the Project, and Environmental and Social Management workshops will be required including all affected parties to find practical and effective management measures to leverage the benefits of the Project to the region and the Senegalese economy, but minimise the negative impact on communities and environment.

It is recommended that Randgold, which owns the permit, submit an application to the Senegalese government to convert the Kanoumba Permit into a mining licence under the Senegalese 2003 Mining Code.

It is further recommended that, upon approval of the mining permit by the Senegalese government, a new entity company be registered in Senegal into which the mining permit will be transferred, and at this stage the State will receive its participation interest in that company. It is the intention that the newly registered company will have a new name and that ownership of the Kanoumba Permit will be transferred into the newly formed company.

As previously indicated, Randgold is now a wholly owned subsidiary of Barrick following the merger transaction which was completed on 1st January 2019.

2. Introduction

The Massawa Gold Project (Massawa or the Project), located in Senegal, was discovered by Randgold Resources Limited (Randgold), an exploration and mining company which explored, developed, and operated mines in West and East Africa.

On 24th September 2018, Barrick Gold Corporation (Barrick) agreed to acquire Randgold in a share-for-share deal. The merger was completed on 1st January 2019 and resulted in Randgold becoming a wholly-owned subsidiary of Barrick and delisting from the stock exchange as a separately listed stock on 28th December 2018. The new company is still known as Barrick but its trading symbol on the NYSE has changed to GOLD, the ticker formerly held by Randgold on NASDAQ. On the TSX, the ticker remains ABX.

This Technical Report conforms to National Instrument 43-101 Standards for Disclosure of Mineral Projects (NI 43-101) and has been prepared to support the public disclosure of the 2018 year-end Mineral Resource and Ore Reserve estimation at Massawa based on feasibility level (FS) studies. Recent targeted project development work programmes, including resource drilling and metallurgical testwork in 2017 and 2018, have made a material impact on the understanding of the mineralised bodies and potential of the Project to meet Randgold's corporate investment requirements. All currency in this report is US dollars ($) unless otherwise noted.

2.1. Effective Date

The effective date of this report is 31st December 2018.

2.2. Qualified Persons

The Qualified Persons (QP) responsible for this report are:

- Rodney Quick, MSc, Pr Sci Nat; Mineral Resource Management and Evaluation Executive, Barrick; is a Qualified Person (QP) as defined by NI 43-101; is responsible for Sections 5, 15, 16, and 18 through 22 and co-authored Sections 1 through 3 and 24 through 27. Mr Quick was assisted by Shaun Gillespie, NHD Min Eng, SAIMMM; Africa & Middle East Mine Planning Engineer; on Sections 15, 16, 18, 21, and 22.
- Simon Bottoms, MGeol, FGS CGeol, FAusIMM; Senior Vice President, Africa & Middle East Mineral Resource Manager, Barrick; is a QP as defined by NI 43-101; is responsible for Sections 6 through 12, 14, 23 and co-authored Sections 25 through 27.
- Richard Quarmby, BSc (Chemical Engineering), Pr Eng, CEng, MSAIChE, MIMMM; Africa & Middle East Capital Projects Metallurgist, Barrick; is a QP as defined by NI 43-101 and is responsible for Sections 13 and 17 and co-authored Sections 1 to 3 and 24 to 27.
- Graham E. Trusler, MSc, Pr Eng, MIChE, MSAIChE; CEO Digby Wells Environmental (Jersey) Limited (Digby Wells); is responsible for Section 20 and relevant disclosure in Sections 1 to 3 and 25 to 27.

2.3. Site Visit of Qualified Persons

Mr Quick and Mr Bottoms regularly visit the Massawa Permit site. During 2016, 2017, and 2018 several visits took place to review both the on-going exploration and resource and reserve development programme. Mr Quarmby visited the Massawa Permit several times in 2018 and Mr Trusler visited the site once in 2018.

2.4. Other Contributors

The following specialist consultancies made contributions to the FS:

Geotechnical

- Dr P Gash, of MineNet Consulting Mining Engineers, UK.

Hydrogeological

- Digby Wells and Associates (Pty) Ltd, South Africa.
- Peens and Associates Civil Engineering and Training Consultant (Pty) Ltd, South Africa.
- Artois Consulting, USA.
- Epoch Resources (Pty) Ltd, South Africa.

Metallurgical

- Advanced Mineral Technology Laboratory Ltd, Canada
- Senet Pty Ltd, South Africa.
- Hazen Research Inc, USA.
- SGS Johannesburg, South Africa.
- Maelgwyn Mineral Services Africa, South Africa.
- Vietti Slurrytec (Pty) Ltd, South Africa.
- Greentechnical (Pty) Ltd, South Africa.
- CyPlus GmbH, Germany.
- Orway Mineral Consultants, South Africa.
- Multotec, South Africa.

Mining

- Maptek Pty Ltd, UK.

Environmental and Social

- Tropica Environmental Consultants, Senegal.
- Digby Wells and Associates (Pty) Ltd, South Africa.

- Max Planck Institute for Evolutionary Anthropology, Germany.

Infrastructure

- Epoch Resources (Pty) Ltd, South Africa.

Senet (Pty) Ltd, South Africa.

2.5. List of Abbreviations

AMTEL Advanced Mineral Technology Laboratory Ltd
As Chemical symbol for Arsenic
Asp Arsenopyrite
Ag Chemical symbol for Silver
AngloGold Ashanti Limited AngloGold Ashanti
AARL Anglo American Research Laboratory
AAS Atomic Absorption Spectrometry
AdvGC Advanced Grade Control
Au Chemical symbol for Gold
BBWi Bond Work Index
BFS Bankable Feasibility Study
BRWi Bond Rod Work Index
BIOX BIOX process (Bacterial Oxidation)
BIOX BAT BIOX Batch Amenability Test
BRL Bottle Roll Leach
CBE Controlled Budget Estimate
CCD Counter-Current Decantation
CCE Capital Cost Estimate
CCTV Closed-Circuit Television
CRO Control Room Operator
CIL Carbon-in-leach
CIM Canadian Institute of Mining and Metallurgy
CMD CMD Consulting Pty Ltd
CN Cyanide
CRM Certified Reference Material
Cu Chemical symbol for Copper
Cr Chemical symbol for Chrome
CV Coefficient of Variation
CWi Crushability Work Index
CZ Central Zone of the Massawa deposit
DBs Distribution Boards
DD Diversion Dam System
DDH Diamond Drill Hole
DGPS Differential Global Positioning System
DO Dissolved Oxygen
E,C&I Electrical, Control and Instrumentation
EDS European Drilling Services
EGL Effective Grinding Length
EGRG Electrolytically Gravity Recovery Gold
EP Engineering and Procurement
EPCM Engineering, Procurement and Construction Management
ESIA Environmental and Social Impact Assessment
ESMP Environmental and Social Management Plan
EW Electrowinning
FA Fire Assay
FAT Factory Acceptance Testing
Fe Chemical symbol for Iron
FEED Front-End Engineering and Design
FF Fracture Frequency
FMG Food Marketing Group
FO Field Operator
FoS Factor of Safety
FS Feasibility Study
FTE Forages Technic-Eau Inc
FW Footwall
FWOZ Footwall Ore Zone (Mineralisation Domain)
G&A General and Administrative
GC Grade Control
GCO Grade Control Optimiser
GRG Gravity Recovered Gold
HARD Half Absolute Relative Difference
HDPE High-Density Polyethylene
HFO Heavy Fuel Oil
HPGR High Pressure Grinding Roll
HW Hanging wall
HWOZ Hanging Wall Ore Zone (Mineralisation Domain)
I&APs Interested and Affected Parties
IBC Intermediate Bulk Container
ICP-AES Inductively Coupled Plasma Atomic Emission Spectroscopy
IEC International Electrotechnical Commission
IFC International Finance Corporation
IFC PS IFC Performance Standards
IPC-MS Inductively Coupled Plasma Mass Spectrometry
ICP-OES Inductively Coupled Plasma Optical Emission Spectrometry
ID Identification
ILR InLine Leach Reactor
Inroads Inroads Consulting LLC
IP Intersection Point
IRR Internal Rate of Return
ISO International Standards Organisation
ITCZ Inter Tropical Convergence Zone
IVD Inverse Distance
JORC Joint Ore Reserves Committee
KE Kriging Efficiency
LED Light Emitting Diode
LFO Light Furnace Oil
LIDAR Light Detection and Ranging
LP Location Point
LPG Liquefied Petroleum Gas
MBCM Million Bank Cubic Metre
MCCs Motor Control Centres
MEL Mechanical Equipment List

MEO Medium Earth Orbit
MMSA Maelgwyn Mineral Services Africa
MOZ Main Ore Zone (Mineralisation Domain)
MORB Mid-Ocean Ridge Basalt
MP Mass Pull
MTZ Main Transcurrent Shear Zone
NDCS South African National Dust Control Regulation
NER Neutral Earthing Resistor
NGL Natural Ground Level
NGO Non-governmental Organisation
NI 43-101 Standards of Disclosure for Mineral Projects
No Number
NPV Net Present Value
NZ Northern Zone of Massawa deposit
NZ1 Northern Zone 1 portion of Massawa Northern Zone Deposit
NZ2 Northern Zone 2 portion of Massawa Northern Zone Deposit
ODBC Open Database Connectivity
OEM Original Equipment Manufacturer
OHLs Overhead Power Lines
OK Ordinary Kriging
OMC Orway Metallurgical Consultants
ON/AN Oil natural/Air natural
OZ1-8 Ore Zone 1 through 8 (Mineralisation Domain)
P&IDs Piping and Instrumentation Diagrams
PAX Potassium Amyl Xanthate
Pb Chemical symbol for Lead
PFDs Process Flow Diagrams
PFS Pre-Feasibility Study
PID Process and Instrumentation Diagram
PLC Programmable Logic Controller
POX Pressure Oxidation
PSA Pressure Swing Adsorption
PSD Particle Size Distribution
P-T Pressure-temperature
PTMP Point to Multipoint
PVC Polyvinyl Chloride
Py Pyrite
pXRF Portable X-Ray Fluorescence
QA/QC - Quality Assurance and Quality Control
QKNA Quantitative Kriging Neighbourhood Analysis
QFP Quartz-feldspar porphyry
QP Qualified Person
RAB Rotary Air Blast
Randgold Randgold Resources Limited
RC Reverse Circulation
RO Reverse Osmosis
ROM Run-of-Mine
RQD Rock Quality Designation
RWD Return Water Dam
S Chemical symbol for Sulphur
SABS South African Bureau of Standards
SAG Semi-autogenous Grinding
SAT Site Acceptance Testing
Sb Chemical symbol for Antimony
SCADA Supervisory Control and Data Acquisition
SCN Thiocyanate
SEC United States Securities and Exchange Commission
SENELEC La Société Nationale d'Électricité du Sénégal
SENET SENET Pty Ltd
SG Specific Gravity
SGS Société Générale de Surveillance
SHEQ Safety, Health, Environmental, and Quality
SLD Single Line Diagram
SMBS Sodium Metabisulphite
SMU Smallest Mining Unit
SOP Set Out Point
SR Slope of Regression
SSC Species of Special Concern
TCLP Toxicity Characteristic Leaching Procedure
TD Tailings Dam
Ti Chemical symbol for Titanium
TP Tangent Point
TSF Tailings Storage Facility
UFG Ultra Fine Grinding
VSDs Variable Speed Drives
WHB Wash Hand Basins
WHO World Health Organisation
WOL Whole Ore Leach
WP Work Point
WRD Waste Rock Dump
Zr Chemical Symbol for Zirconium

2.6. Units

cm	Centimetre
ekW	Generator Output Rating in kW
g	Grammes
Ga	Billion years
g/L	grammes per litre
g/t	Grammes per metric tonne – gold concentration
ha	Hectare
hr	Hour
Kbar	Kilobar of pressure
kg	Kilogram
km	Kilometre
km^2	Square kilometre
koz	Thousand ounces
kt	Thousand metric tonnes
ktpa	Thousand metric tonnes per annum
ktpm	Thousand tonnes per month
kW	Kilo Watts
m	Metre
m^2	Square meter
m^3	Cubic meter
Moz	Million fine troy ounces
Mt	Million metric tonnes
Mtpa	Million metric tonnes per annum
MW	Mega Watts
tph	Tonnes per hour

3. Reliance on Other Experts

In preparation of this Massawa Gold Project Technical Report, the QPs have had to rely on the opinions and expertise of other Barrick staff and external consultants. David Mbaye (Barrick Senegal Country Manager) was relied upon in Sections 1 and 4 for his opinion regarding permitting and Senegalese tax.

4. Property Description and Location

The Massawa gold deposit straddles the original Randgold Exploration Permit of Kanoumba in Eastern Senegal and is located approximately 700 km SE of the capital city, Dakar (Figure 4-1), and 90 km due west of the Barrick Loulo Mine in Mali. The deposit is positioned in the centre of the Kanoumba Permit (Figure 4-2).

The original Kanoumering and Kounemba permits were granted to the company by Presidential Decrees in 2002 and 2003 and renewed twice by arêtes issued by the Ministry of Energy and Mines, the last of which merged the two original permits into the single Kanoumba Permit in 2010. The concession boundaries are described by latitude and longitude within the decree. The boundaries are located using a Differential Global Positioning System (DGPS).

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Location Map

Legend:

- *National Capital*
- *Regional Capital*
- *Town, Village*
- *Major Airport*
- *International Boundary*
- *Regional Boundary*
- *Main Road*
- *Secondary Road*
- *Other Road or Track*
- *Railroad*

The boundaries and names shown and the designations used on this map do not imply official endorsement or acceptance by the United Nations.

July 2019

Source: Map No. 4174 Rev. 3, United Nations, 2004.

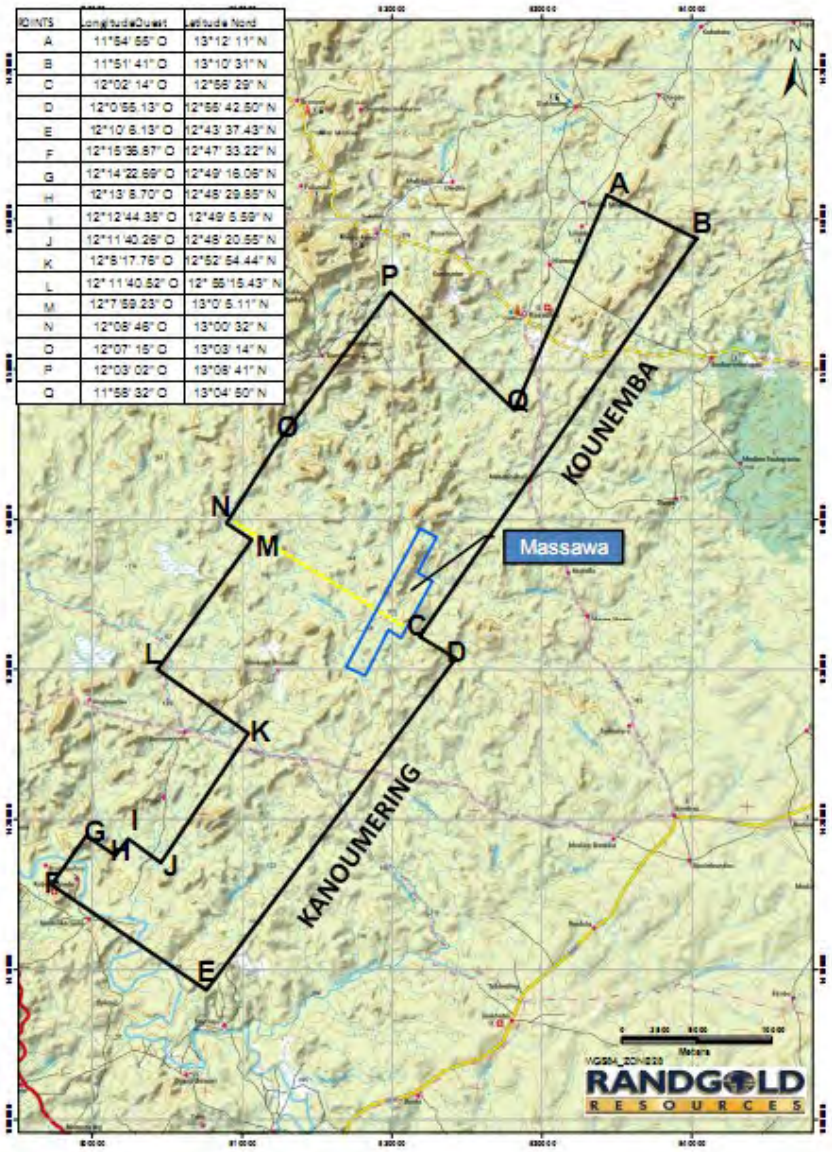


Figure 4-2 Kanoumba Permit

The original Kanoumering exploration licence of 405 km^2 was granted in October 2002 for an initial period of four years and further renewed for another three years in June 2006. As per the requirements in Senegal, this required a mandatory reduction in size of 102 km^2 to a total of 303 km^2 (Table 4-1 and Table 4-2). The original Kounemba Exploration Permit of 408 km^2 was granted in May 2003 for an initial period of four years. The first renewal occurred in August 2007 for a further three year period with a reduction in size of 103 km^2 to a total of 305 km^2 (Table 4-1 and Table 4-2). The exploration permits are exclusively and lawfully accessible for exploration and prospecting purposes and cannot be contested by any other mining company.

Table 4-1 Kanoumering and Kounemba Decrees

Permit Name	Granting Decree	Licence Type	Area (km^2)
Kanoumering	Presidential Decree N° 2002-1080 of 29 Oct 2002	Exclusive Exploration Permit (EEP)	405
Kounemba	Presidential Decree N°2003-381 of 28 May 2003	Exclusive Exploration Permit (EEP)	408

Table 4-2 Original Kanoumering and Kounemba Exploration Permits

Permit Name	Initial Renewal	Relinquished Area (km^2)	Area after First Renewal (km^2)	Second Renewal Due	End of Exploration Rights
Kanoumering	Arête ministerial N°4771 MNI/DMG of 13 June 2007	102	303	13/06/2010 for an additional 3 years	June 2013
Kounemba	Arête ministerial N°8208MNI/DMG of 20/08/2007	103	305	20/08/2010 for an additional 3 years	August 2013

The State owns all mineral rights within Senegal. The terms of reference for exploration and mining are established through a convention ('Convention de Recherché') between the company and the State. The Senegalese government maintains a 10% free carried interest in all projects. The government's interest is activated at the Exploitation Stage, and a local mining subsidiary must be created. The subsidiary company is required to finance the 10% share of the State in the capital. The State does not contribute to the expenses for exploration, feasibility studies, development, or mining. In the case of an increase in the capital during the mine life, the State will receive a 10% free carried interest in the new investment in order to keep its participation at 10%.

Conventions under the 1988 Mining Code (Law No 88-06 of 26th August 1988) covered the original Kanoumering (2002) and Kounemba (2003) Permits. Subsequent to the 1988 Mining Code, an updated mining code was passed in November 2003 (Law No 2003-36 of 24th November 2003). Randgold had the right under the new code to accept the new code should it negotiate a new single convention that merged the two permits over the Project. This option was duly exercised, and on 14th April 2010 the two permits were combined into the single Kanoumba Permit with an area of 621 km^2 covered by a single convention based on the 2003 Mining Code (Table 4-3). The arête was approved on 21st May 2010.

Table 4-3 New Kanoumba Decree - 2010

Permit Name	Approval	Actual Area (km^2)	Next Renewal Due	End of Exploration Rights
Kanoumba	Arête ministerial N°04638 MMITPME/DMG of 21 May 2010	621	21/05/2013 for an additional 3 years	May 2019

In 2016, the second renewal of an agreement with the Minister of Mine department combined parts of the Tomboronkoto, Miko, and Kanoumba Permits, with the newly merged permit covering an area of 606 km^2 (Table 4-4).

Table 4-4 New Kanoumba Decree – Second Renewal 2016

Permit Name	Approval	Actual area (km^2)	End of Exploration Rights	Possible Extension
Kanoumba	Arête ministerial N°11204 MIM/DMG of 2 August 2016	606	21 May 2019	For 3 years

This allowed for the extension of the Sofia deposit toward the Miko Permit and also the southern continuity of the Sabodala structure into the Tomboronkoto Permit. The co-ordinates of the newly merged permit are shown in Figure 4-3 and Table 4-5.

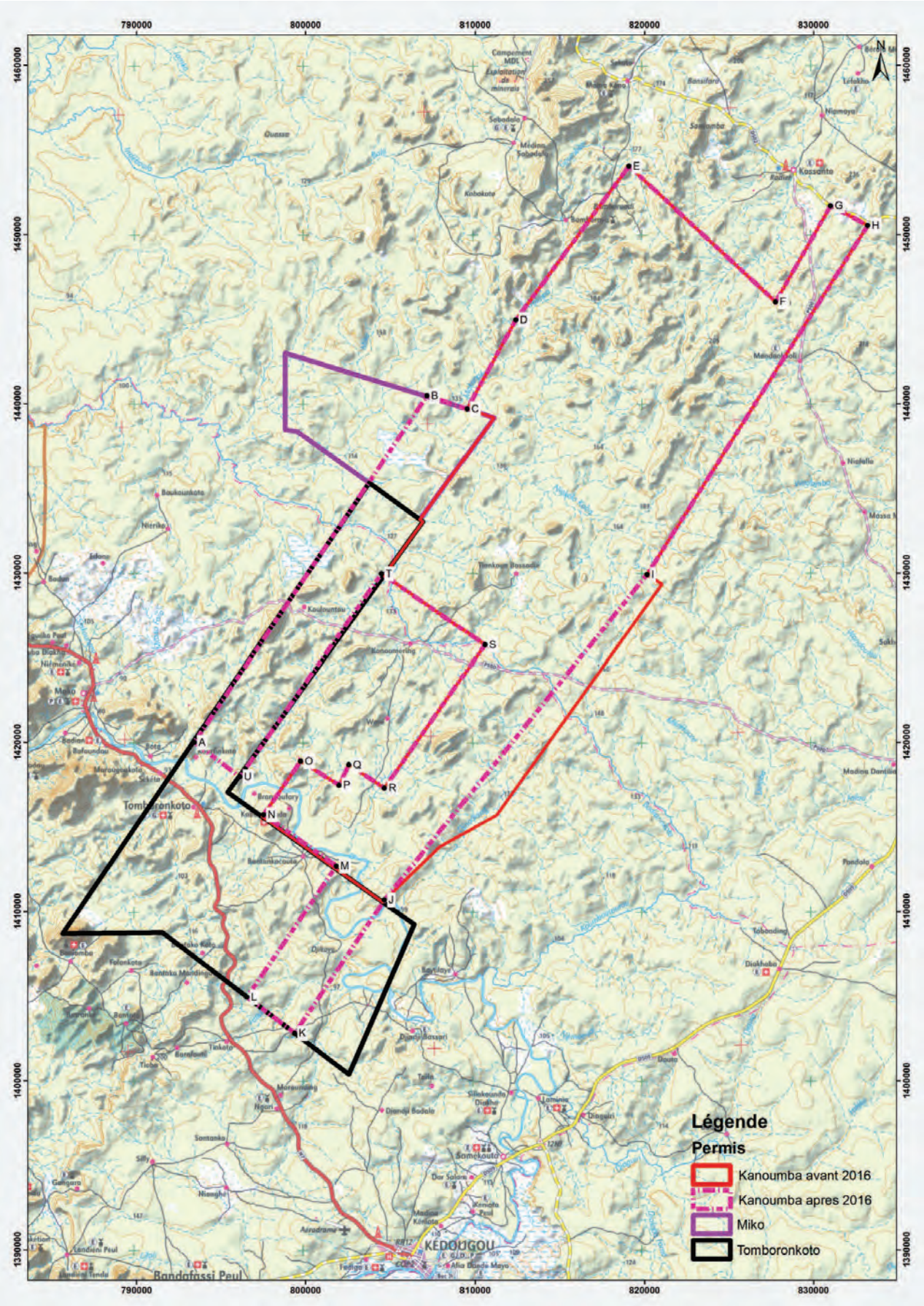


Figure 4-3 Kanoumba Exploration Permit Co-ordinates

Table 4-5 Kanoumba Exploration Permit Co-ordinates

Point	Easting	Northing
A	793416	1420000
B	807173	1440455
C	809514	1439687
D	812390	1444945
E	819077	1454030
F	827753	1446014
G	830999	1451693
H	833200	1450527
I	820147	1429890
J	804656	1410688
K	799356	1402808
L	796529	1404943
M	801811	1412696
N	797515	1415731
O	799697	1418893
P	801969	1417468
Q	802545	1418676
R	804657	1417304
S	810569	1425750
T	804486	1429944
U	796114	1417973

It is worth noting that within the new 2003 Mining Code, the State can, over and above the 10% free carried interest, negotiate for itself or the local private sector a participation in the capital of the local mining company (Article 30 of LOI N° 2003-36 du 24th November 2003 Mining Code).

Randgold's local partner Compagnie Senegalaise de Transports Transatlantiques Afrique de l'Ouest (CSSTAO) presently has 7.5% ownership of the remaining 90%, after the government 10% participation, leaving Randgold with 83.25% ownership of the Project. As previously indicated, Randgold is now a wholly owned subsidiary of Barrick following the merger transaction which was completed on 1st January 2019.

Article 19 of the 2003 Mining Code, Rights of the Holder of a Prospection Permit, states: 'Any discovery of a commercially exploitable deposit by the holder of a prospection permit gives an exclusive right of the granting of an exploitation permit or mining concession of the said discovery if requested prior to the expiry of the prospection permit.

There is a 3% royalty applicable on the Project.

There are no known environmental liabilities to which the property is subject. The Gambia River flows in a westerly direction on the southern limit of the permit.

There are no know significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to the Massawa Project is by road from Dakar in the west, or Loulo Mine via Mali from the east. The main transport route for capital equipment will be from the port of Dakar, by road to Mako, and to site via a dirt road. The Dakar - Kédougou National 7 road (RN7) cuts through the Tomboronkoto perimeter and has been recently upgraded. Upgrading of the road from Mako (see Figure 4-1)to site will be required to facilitate the transport of heavy equipment. The Millennium highway, which connects Dakar with Bamako is complete and allows very good access between the Loulo Mine in Mali and the Massawa Project. A well-maintained laterite road between Bembou and Sabodala exists and crosses the permit. Charter flights from Dakar to the Kedougou airstrip can be used. The Kedougou airstrip is a 1.5 km long sealed strip. It is approximately 85 km from site by road. Kedougou is the regional centre and accessible by off-terrain vehicles on dirt roads from site.

The climate at Massawa is strongly influenced by the north and southward movement of the Inter Tropical Convergence Zone (ITCZ) which creates distinctive wet and dry seasons. The wet season extends from June to October and access to the site can be affected by poor road conditions. The site is in the Sahelian Transition Zone between the Sahara Desert in the north and the tropical climate in the south. The low altitude of the site (170 m to 240 m above mean sea level) and the absence of any intervening mountains mean that the humidity is directly related to the wind moving from the south and west.

The local population are essentially artisanal gold miners and subsistence farmers. Bread and small quantities of vegetables may be sourced locally but most supplies are obtained from Kedougou.

Local infrastructure is limited to small rural settlements connected by gravel roads and paths. Tinkoto, an artisanal mining village consisting of between 7,000 and 10,000 people, is the nearest village to the Project and is located 7 km SW of the southern extent of the deposit. Tinkoto has developed around the artisanal mining activities in the area.

Equidistant to the north, east, and SW of the site are the Peul villages of Mandinkole and Kanoumering. These are both predominantly pastoral communities, rearing goats, sheep, and cattle. Kanoumering has an estimated population of 700 people, whilst approximately 200 people reside in Mandinkole. Cultivation of maize, millet, and peanuts also takes place in the vicinity of these villages, although agriculture is limited by poor soils and a relatively long dry season.

Up to 240 people a day are employed from these villages in the Massawa exploration camp, fulfilling roles such as samplers, aides, and general laborers. In general, however, the local population is largely uneducated and untrained, and skilled personnel will need to be sourced from Kedougou and further afield within Senegal during the subsequent phases of the Project. It is Barrick's experience in similar type projects in West Africa that, with the provision of training, it is possible to employ local people in the construction and operational phases of the projects.

Process water is planned to be supplied from an on-site storage facility which will be required for the dry periods. No electrical power is available on the permit and it is currently supplied by on-site diesel generating set for the exploration camp.

The topography of the area is generally undulating with elevations ranging from 100 m above sea level to a maximum of 300 m above sea level. The area comprises a dense network of small seasonal streams between undulating hill and ridges. A detailed LIDAR topographic survey has been conducted for areas of the permit in which mining activity is planned to take place.

6. History

Artisanal mining has long been one of the primary activities of the Malinke people in the region, and active artisanal mining areas still exist on the permit. The Massawa deposits themselves have never been exploited by artisanal mining, possibly due to their relatively low tenor surface expression.

AngloGold Ashanti Limited (AngloGold Ashanti) previously held the Kanoumering Permit and conducted exploration on the permit between December 1996 and January 2000. During this period AngloGold Ashanti undertook a multiphase research programme involving regional geochemistry, detailed box media sampling and mapping, airborne survey, and multipurpose drilling over selected targets. Results of soil sampling indicate that 37% of the samples collected returned grades greater than 500 ppb Au and most of the anomalous samples were located in the vicinity of the Tinkoto pluton. Numerous other gold anomalies were found within tuffs and andesites occasionally associated with the linear trends identified by Landsat imagery. Detailed soil sampling was carried out over 12 identified targets, of which four were selected for more focussed work where detailed soil sampling revealed some encouraging gold in soil anomalies over areas KA, KB, KC, and KD. These areas were followed up with rotary air blast (RAB) and diamond drilling. A total of 21 diamond drill (DD) holes for 3,451 m were drilled, which included 10.9 m at 1.96 g.t Au from 0 m in KB99004D, 18 m at 1.23 g/t Au from 56 m in KA98001D, and 51.4 m at 0.7 g/t Au from 40 m in KA98003D.

Randgold conducted a regional and detailed soil survey on the Kounemba Permit which was completed in 2004. The area of the Central Zone and North Zone open pits was initially a greenfields discovery by Randgold targeting a 3.5 km long gold in soil anomaly at over 20 ppb on the Kounemba Permit. In 2006, the first trench, MWTR001, returned an encouraging result of 10.9 m at 2.03 g/t Au, and subsequent positive results from RAB drilling were followed up with a number of phases of diamond drilling. This resulted in work up to 2012 confirming 4.5 km of continuous mineralisation at Massawa. The 4.5 km crosses the boundary between the Kounemba and Kanoumering Permits, and is the reason that, in 2010, the two permits were merged to form the new Kanoumba Permit.

In 2015, a revised scope of metallurgical testwork was designed to target refractory ore types, which in turn, resulted in challengingly high power and processing costs.

A strategy was implemented to target non-refractory ore types on satellite deposits on the property. The Sofia Main pit was subsequently converted to reserve in 2016 and the Sofia North pit, in 2017. The Delya pit, which contains refractory sulphides, was converted to reserve in 2017.

Following the addition of these reserves to the proposed mine schedule, in 2017 and 2018 a metallurgical plant was designed and costed along with a tailings facility and mine site infrastructure for the FS.

There have been no historic resource or reserve estimates and there has been no historic production from the Massawa deposits.

7. Geological Setting and Mineralisation

7.1. Regional Geology

The West African Craton can be divided into three main regions exposed beneath Phanerozoic cover. In the north, the Reguibat Rise extends over Mauritania and western Algeria and consists of an Archaean terrain in the west and Paleoproterozoic (Birimian) terrain in the east. The southern Leo Rise covers a large area over southern Mali, Côte d'Ivoire, Burkina Faso, Niger, Ghana, and Guinea; and is separated from the Reguibat Rise by the Late Proterozoic to Phanerozoic sedimentary Taoudeni Basin. The western Archaean portion, known as the Man Shield, is separated from the eastern Birimian Supergroup of the Baoule Mossi domain by the Sassandra fault. Two Birimian inliers, the Kayes and Kedougou-Kenieba, suggest the continuity of the Proterozoic basement underneath the Taoudeni intra-cratonic basin. The Massawa Project is located within the Kedougou-Kenieba inlier (KKI) (Figure 7-1).

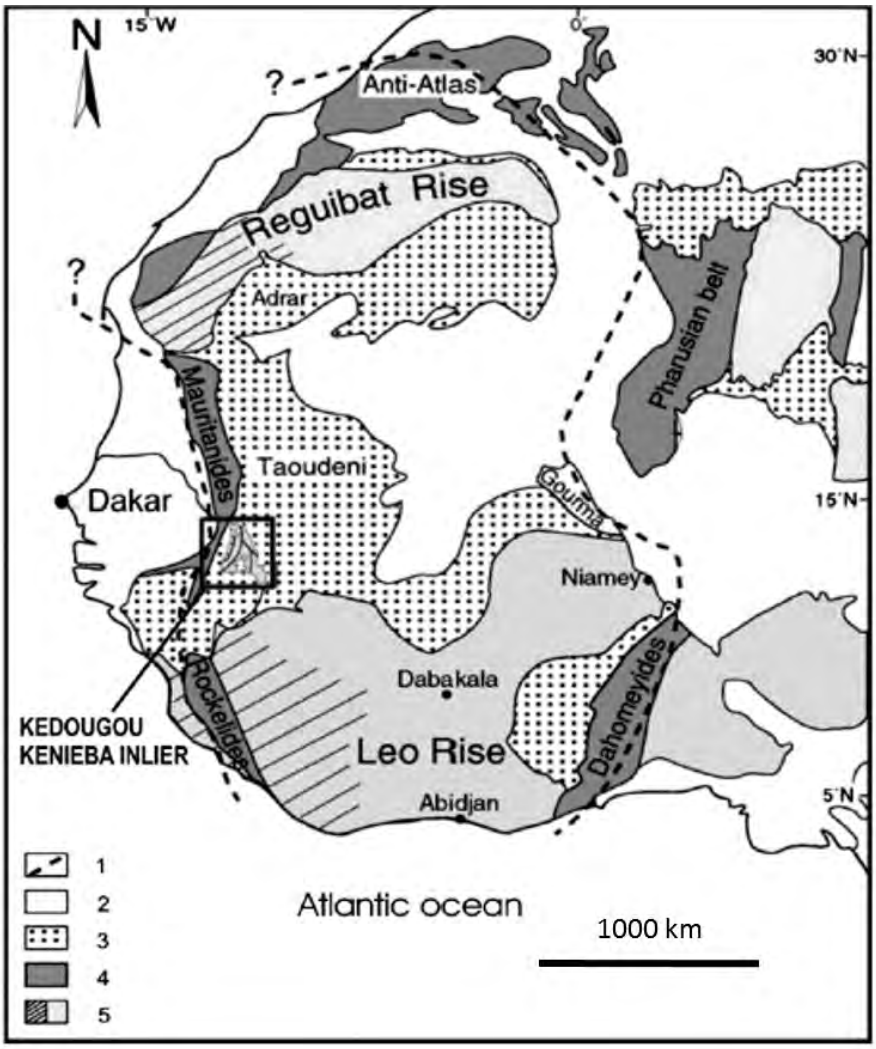


Figure 7-1 Location of the Kedougou-Kenieba-Inlier in the West African Craton

Key: 1- Limits of West African Craton; 2+3- Taoudeni Basin; 4- Pan-African Belts; 5- Archean (stripes) and Birimian terranes (Gueye et al., 2008

Volcanic and sedimentary rocks of the Birimian Supergroup form a substantial portion of the West African Craton. These rocks and associated intrusive units of the Eburnean tectonic-thermal event represent a major Paleoproterozoic juvenile crust forming event that took place during the time interval of 2.25 Ga to 2.09 Ga (Abouchami et al., 1990; Boher et al., 1992). The Birimian crust formed from large ocean closure (Leahy et al., 2005) between two Archaean cratons, involving the progressive accretion of island arcs and oceanic plateaus to a growing continental mass (Feybesse et al., 2006). Subsequent thickening associated with the intrusion of kinetically late granitoids and compressional deformation facilitated the formation of structurally controlled orogenic gold deposits.

7.2. Local Geology

The KKI is divided into the Mako volcanic series to the west, and an overlying Dialé–Daléma sedimentary basin to the east (Dia et al., 1997) (Figure 7-2). The Massawa and Sofia deposits are located along the 150 km long Mako Belt. The Mako Belt consists of greenstones and sedimentary rocks, dated between 2160 Ma and 2200 Ma, which were intruded by ultramafic to felsic plutons yielding ages of 2070 Ma to 2210 Ma (Dia et al., 1997; Hirdes and Davis, 2002; Gueye et al., 2007). All rock types, excluding post-Birimian dykes, were metamorphosed to a lower greenschist facies during the Eburnean orogeny. The belt basin margin between the Mako and Dialé–Daléma series is structurally controlled and marked by the regional-scale, NE trending, Main Transcurrent Shear Zone (MTZ) (Ledru et al., 1991; Treloar et al., 2014) which hosts Massawa. A second major first order structure is located further to the west, within the Mako Belt, and is referred to as the Sabodala Shear Zone. This structure hosts the Sofia deposit and Teranga Gold Corporation's (Teranga) Sabodola deposits, located to the north.

7.3. The Mako Volcanic Belt

The Mako Belt consists of folded metavolcanic and metavolcaniclastic rocks intruded by various generations of granitic and mafic rocks (Figure 7-3). The belt is characterised by NE trending corridors, or domains, whose shape is, at least in part, controlled by a network of anastomosing north to NE trending lineaments, shear zones, and plutonic belts that form the Badon-Kakadian batholith (Gueye et al., 2008; Diene et al., 2012). Massive basaltic and andesitic flows outcrop in the southern parts of the belt where they overlie pillowed basalts (Diallo, 1994). The flows are inter-bedded with volcanic agglomerates, breccias, and banded tuffs. The mafic Mako volcanic rocks are tholeiitic in character (Abouchami et al., 1990; Dia et al., 1997). The pillowed basalts have a normal mid-ocean ridge basalt (N-MORB) affinities, whereas the massive basalts have an enriched mid-ocean ridge basalt composition (E-NORB) (Ngom, 1995).

The Mako metavolcanic rocks are intruded by the Badon-Kakadian batholith to the west and north; and to the east by small, tonalite-trondhjemite-granodiorite (TTG) plutons. The Badon-Kakadian batholith represents a multi-phase suite of deformed ultramafic to dominantly felsic (metaluminous) intrusions (Dioh et al., 2006; Gueye et al., 2008). The Sonfara-Sandikounda intrusive rocks (the Sandikounda Layered Complex and the Sandikounda Amphibolite-Gneissic Complex) are dated between 2,158 ± 8 Ma and 2,205 ± 15 Ma (U/Pb and Pb/Pb zircon methods; Dia et al., 1997; Gueye et al., 2007). The Lamina Kaoura Plutonic Complex (granites, tonalites, and rare mafic lithologies), east of Sandikounda, is dated by Dia et al. (1997) at 2,138 ± 6 Ma

(zircon Pb evaporation age). The Badon granodiorite, in the southern parts of the batholith, yielded Pb/Pb zircon ages of 2,198 ± 2 Ma to 2,213 ± 3 Ma (Gueye et al., 2007). Calc-alkaline felsic stocks on the eastern side of the belt, such as the Tinkoto and the Mamakono plutons, are younger than rocks of the Badon-Kakadian batholith, yielding Pb/Pb ages of 2,080 Ma to 2,090 Ma (Hirdes and Davis, 2002; Gueye et al., 2007).

The Mako granitoids, along with the volcanic packages, are interpreted to have island arc affinities on the basis of their geochemical and petrological characteristics (Dia, 1988; Dia et al., 1997; Diallo, 1994; Pawlig et al., 2006).

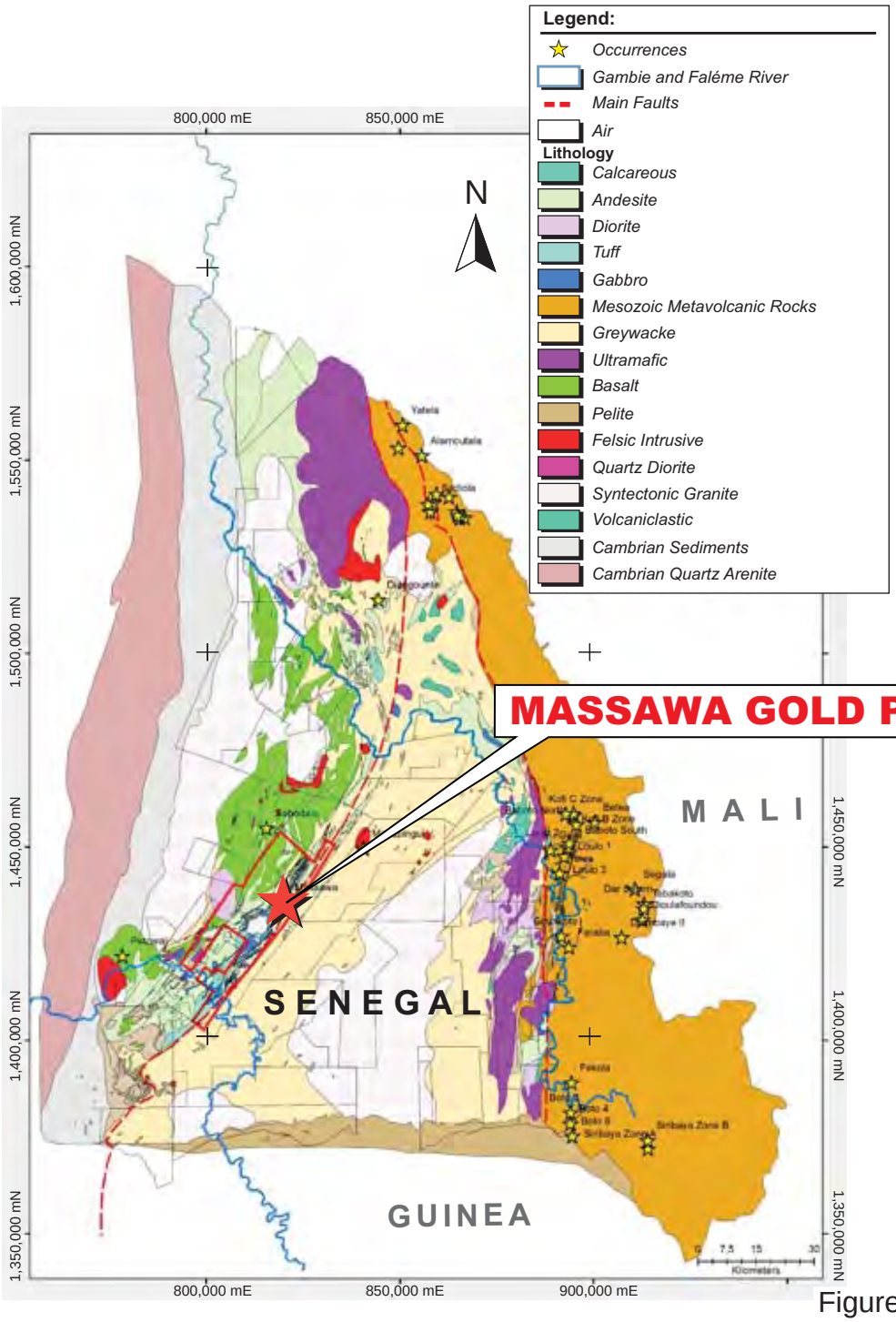


Figure 7-2

0 15 30 45 60
Kilometres

Barrick Gold Corporation

Massawa Gold Project

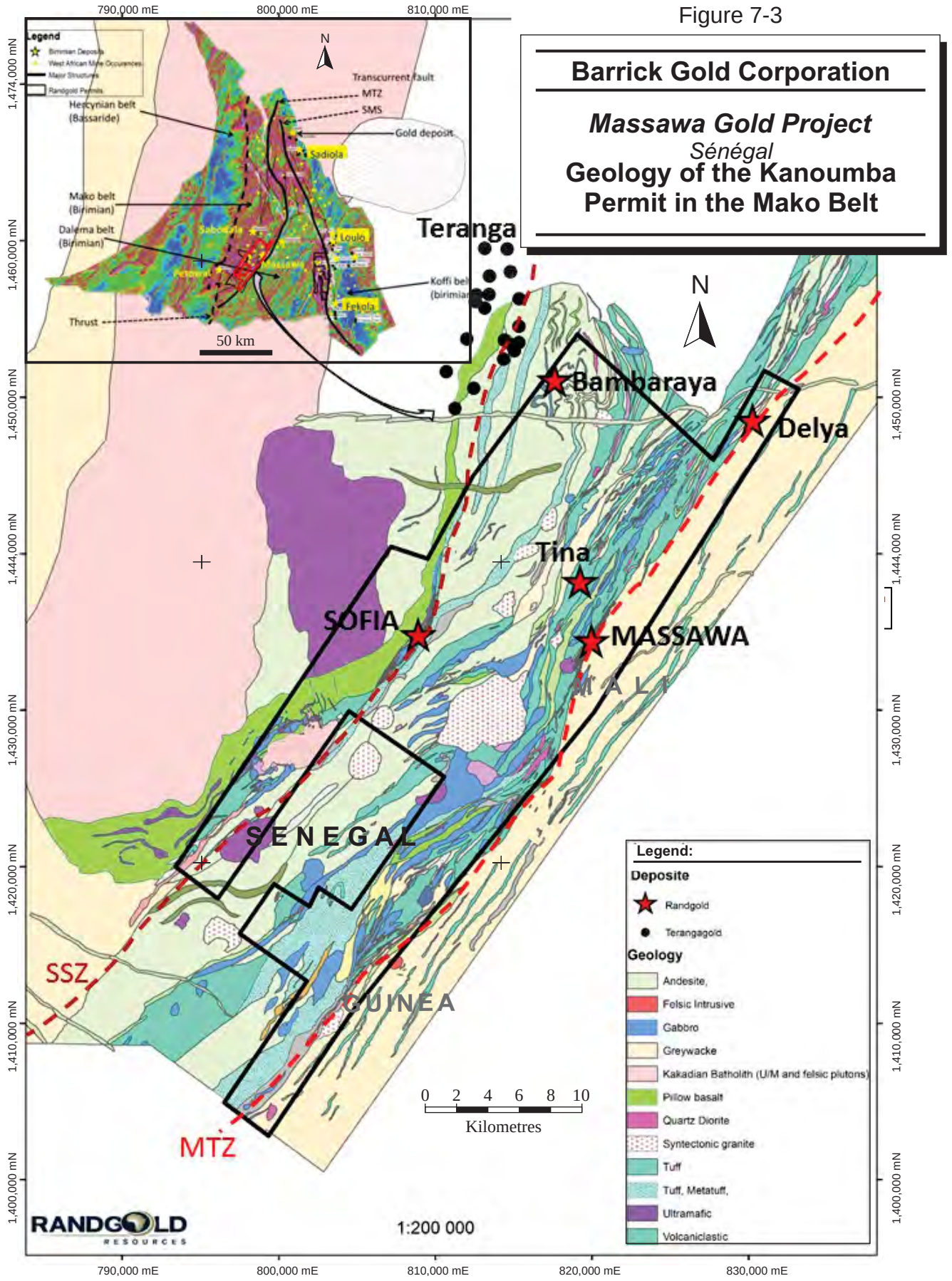
Sénégal

Regional Geology Map of the Senegalese Portion of the Kedougou-Kenieba Inlier

July 2019

Source: Randgold, 2018.

Figure 7-3



July 2019

Source: Randgold, 2018.

7.3.1. The Dialé-Daléma Sedimentary Basin

The neighbouring Dialé-Daléma basin is composed of a sequence of sandstones, siltstones, and carbonates, inter-bedded with calc-alkaline ash- and lapilli-tuffs (Bassot, 1987; Hirdes and Davis, 2002). All sedimentary rocks are isoclinally folded, with upright or slightly overturned folds to the SE. The Dialé-Daléma series is generally considered to be younger than the Mako series (Bassot, 1987; Dia, 1988; Abouchami et al., 1990). Siliciclastic rocks show detrital zircon ages ranging between 2096 ± 8 Ma and 2165 ± 1 Ma (Milési et al., 1989; Hirdes and Davis, 2002). The sedimentary basin is centrally intruded by the Saraya quartz monzonite granite (Pons et al., 1992), which yielded a U/Pb zircon age of 2079 ± 2 Ma (Hirdes and Davis, 2002), and the plutons of Balangouma and Boboti further to the east (Ndiaye et al., 1997).

7.4. Massawa Deposit Geology

7.4.1. Regolith

Gold mineralisation is rarely exposed at surface, with the majority of the Massawa deposit being covered from the south to north by three laterite levels Cp3 (Lower), Cp2 (Middle), and Cp1 (Upper). Cp3 is believed to be transported off Cp2 and is clearly multi-layered and over 7 m thick in trench exposures. Percussion RAB drilling has identified Cp2 and Cp1 to be five to seven vertical metres thick. Gossanous shears and quartz stockworks hosted by volcaniclastics and sedimentary rocks are rarely exposed in Cp1. The relatively thick transported laterites can explain the low gold in soil response encountered over the deposit.

7.4.2. Lithology

The Massawa stratigraphy is dominated by a western package of volcaniclastic rocks and an eastern package of greywackes, with bedding striking at 210° ± 10° and dipping steeply (75° to 80°) toward the west. This dominantly clastic sequence is intruded by several igneous rocks including sills of gabbro, felsic intrusions, and feldspar (and/or quartz-feldspar) porphyries. Core photographs of the main lithologies are shown in Figure 7-4.

Volcaniclastic rocks: The volcaniclastic rocks have a bimodal mineralogy with both mafic and felsic variants. They consist of a package of agglomerates, lapilli tuffs, tuffs, ash-tuffs, and fine-grained carbonaceous ash-tuffs and include both purple and green variants. The purple colouration probably indicates deposition in an arid to semi-arid terrestrial environment. Laminated, fine-grained, volcaniclastic material is often green and could represent ash fall into standing water. The coarser volcaniclastic rocks are matrix supported and include elongated and sub-rounded felsic lithic clasts (up to 5 cm in size) with glassy textures and rounded mafic clasts (up to 10 cm in size).

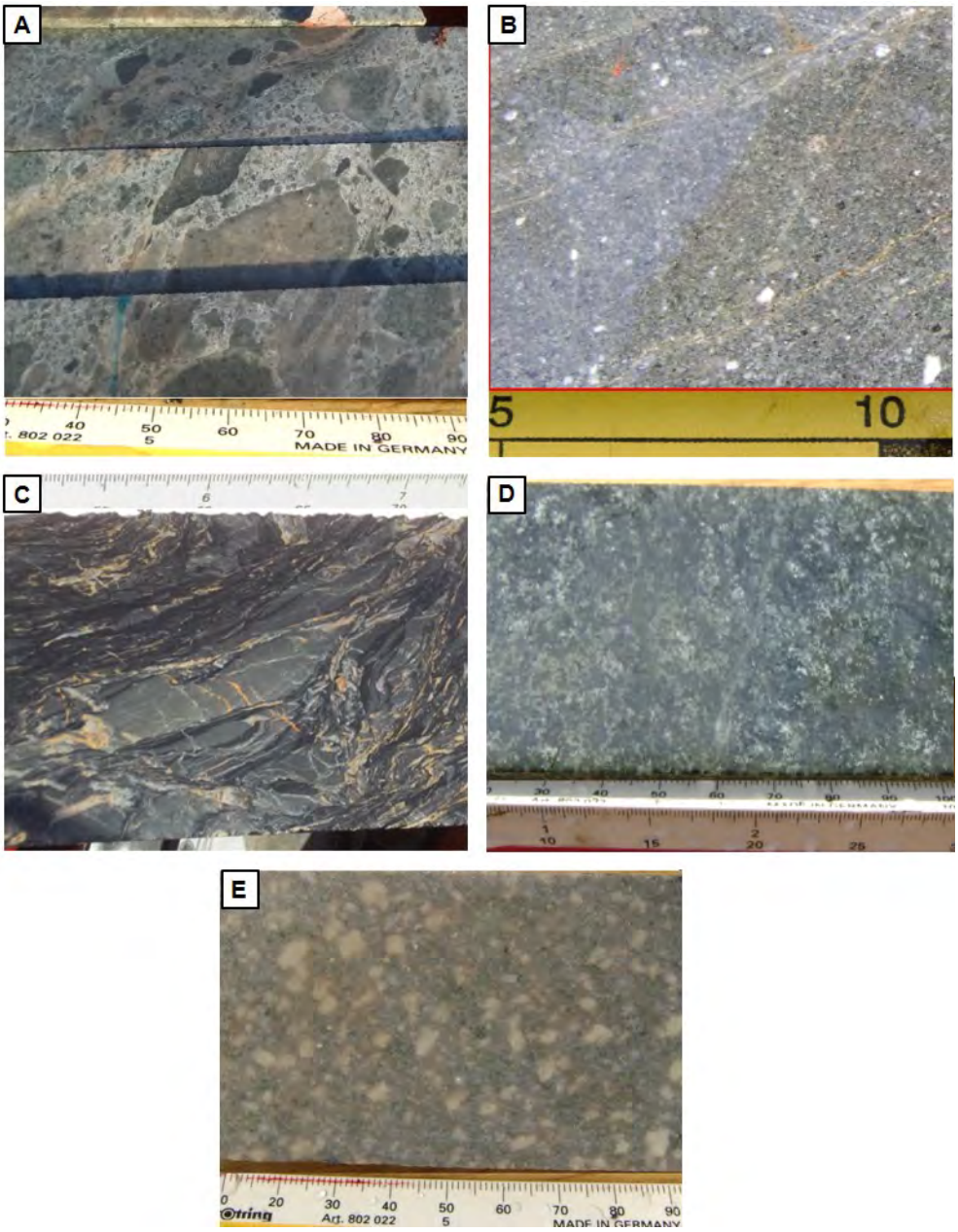


Figure 7-4 Core Photographs of Dominant Lithologies at Massawa

A- Volcaniclastic; B- Greywacke; C- Carbonaceous schist; D- Gabbro; E- Quartz-feldspar porphyry

Siliciclastic sedimentary rocks: Felsic and lithic wackes, which underlie the volcaniclastic rocks to the east, are composed of fine-grained layers at the top of the sequence with coarser units at the base. Graded bedding is common within these rocks and shows a downward younging direction implying that the steeply west dipping volcano-sedimentary package is overturned, with the volcaniclastic rocks being older than the greywackes.

At its northern end, the Massawa deposit is bounded by two prominent carbonaceous shale layers which act as the hanging wall and footwall of the mineralisation. The carbonaceous shale is very fine grained, well laminated, and is inter-bedded with the coarser-grained greywackes. Layers are variable in thickness, on a millimetre scale, and form discontinuous lenses. Graphitic bands are common where the rock is more deformed and these units are referred to as graphitic

schists. The greywacke and shale package displays soft sediment deformation fabrics typical of turbidite systems including load casts, slump and flame structures, and intra-sedimentary faults.

Intrusive rocks: Two main concordant gabbroic bodies are present at Massawa. The sills are up to 30 m in thickness; although towards the north they occur as narrow bands ranging from 0.5 m to 18 m in thickness. The gabbros predate shearing and host gold when intersected by the mineralised structures. Outside of the mineralised system, gabbro sills are massive and coarse grained.

Quartz porphyry and fine-grained felsic sills, forming sheets 2 m to 60 m thick, occur throughout the deposit but are generally thicker and more abundant in the southern portion of the deposit. The porphyries are intrusive into the volcano-sedimentary sequence and show contacts against greywacke, volcaniclastic rock, and carbonaceous shale. The quartz-feldspar porphyry has a tonalitic mineralogy, comprising phenocrysts of oligoclase/andesine (altered to sericite) and quartz (50:50 ratio), and a groundmass composed of fine-grained plagioclase, albite, and small amounts of quartz. The earliest sills display a weak foliation and weak to strong alteration. Younger intrusive units are unaltered, undeformed and cross-cut the sheared rocks, and thus the emplacement of the porphyries spans the shearing event.

Late stage (post-mineralisation) mafic dykes (less than 15 m thick) crop out in the southern part of the mineralised body, where they intrude both the volcaniclastic rocks and the greywackes.

7.4.3. Structure

Massawa lies on a NE trending (030º to 035°) sinistral structure, likely a second order splay off the neighbouring MTZ. To the south and north of Massawa, the NE trending shears are dextrally offset by discordant north-south structures, resulting in dilation and mineralisation. The mineralisation is associated with anastomosing brittle-ductile shearing commonly localised at intrusive contacts.

7.4.4. Mineralisation

The Massawa deposit occurs over a strike length of 4.5 km and is divided into two zones (Central Zone and Northern Zone) that differ in terms of host rock geology, mineralogy, structural controls, and metallurgy. The Central and Northern zones are separated by a 0.3 km gap zone, where less intense deformation is observed. Both mineralised zones are discussed separately below.

7.5. Massawa Central Zone

Mineralisation in the Central Zone (CZ) is hosted by an anastomosing brittle-ductile shear network localised by pre-existing gabbro and felsic porphyry intrusive contacts. The CZ is divided into four blocks based on metallurgy (Block 1 to Block 4) generally becoming more refractory to the north, associated with higher arsenopyrite content and a change in host lithology from volcaniclastics to sediments (greywacke) (Figure 7-5, Figure 7-6, and Figure 7-7).

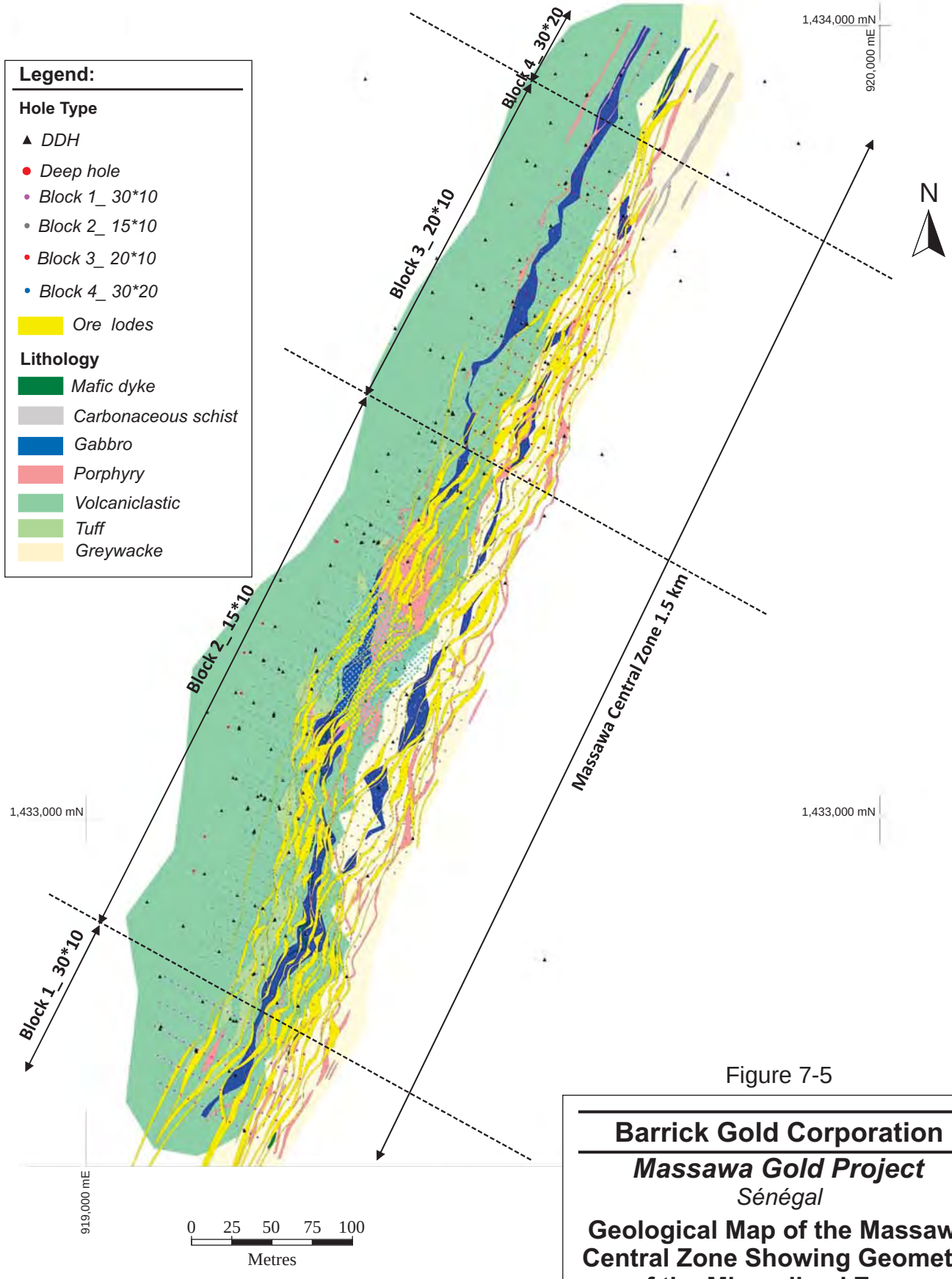


Figure 7-5

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Geological Map of the Massawa Central Zone Showing Geometry of the Mineralised Zones

July 2019

Source: Barrick, 2019.

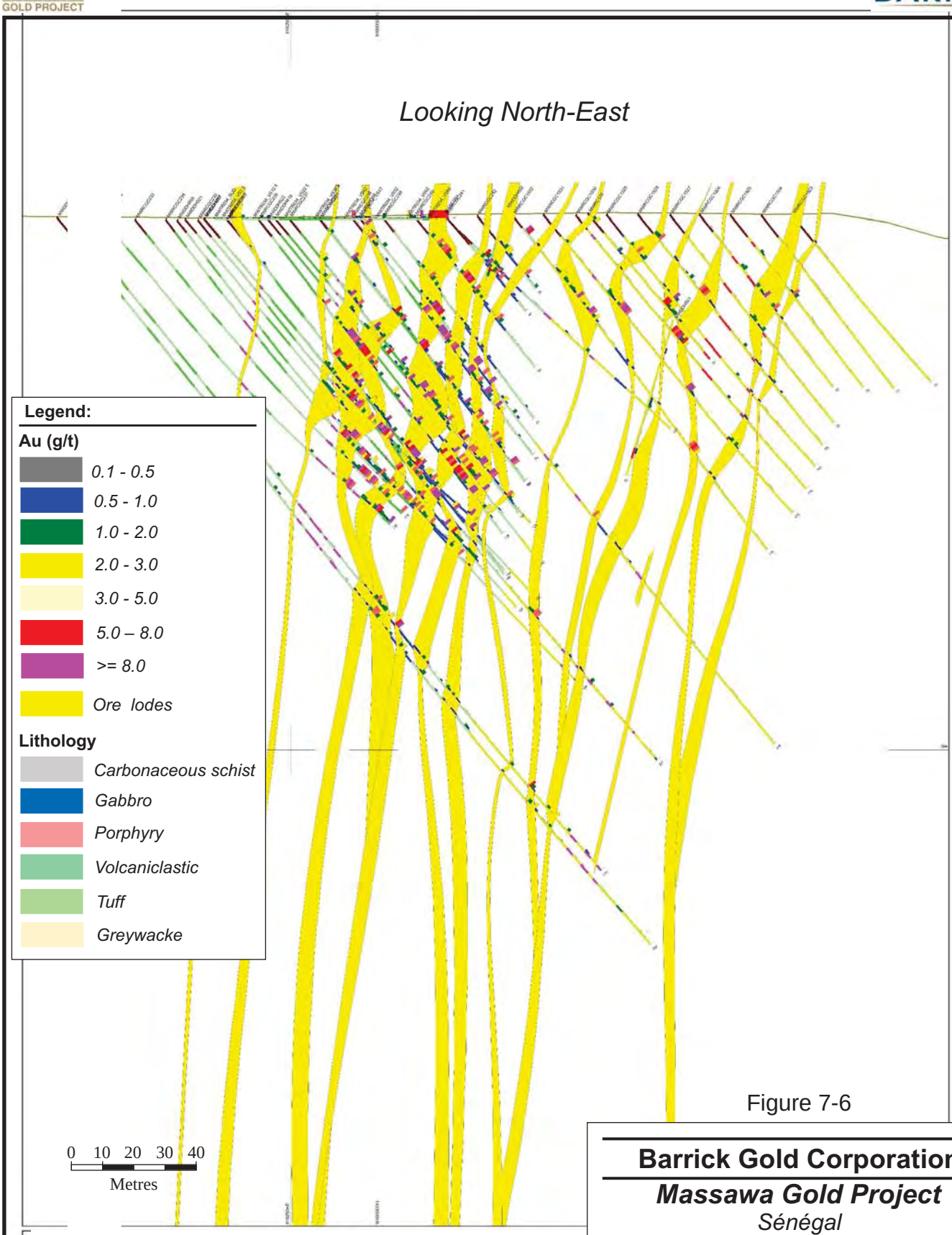


Figure 7-6

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Central Zone Typical Section of the New Mineralisation Model Showing Multiple Anastomosing Lodes

July 2019

Source: Barrick, 2019.

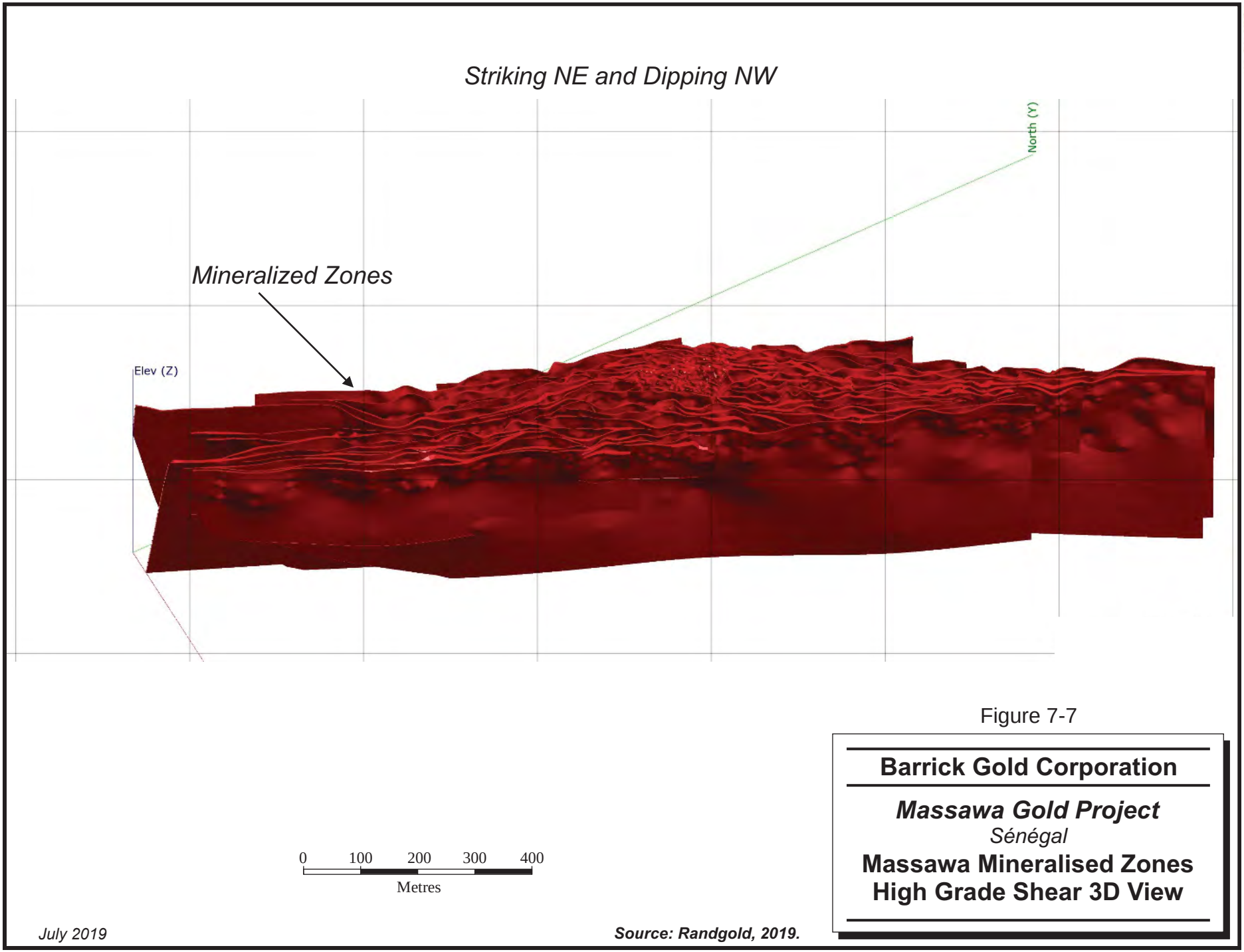


Figure 7-7

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Massawa Mineralised Zones
High Grade Shear 3D View

July 2019

Source: Randgold, 2019.

There is a deposit scale correlation between increasing brittle-ductile strain and increasing gold grade, with the grade of mineralisation variable along strike and down-dip related to the variable strain associated with the structural framework. The continuity of mineralisation is localised along gabbro and felsic porphyry intrusive contacts (Figure 7-8) with high-grade mineralisation associated with high strain and arsenopyrite (Figure 7-9).

Numerous vein opening directions and vein styles highlight high fluid overpressure, with differential stress trending toward zero. Previously, a genetic link between quartz-stibnite veins and high-grade mineralisation was made leading to a 'Phase 1 and Phase 2' nomenclature. Subsequent review with additional drilling has advanced the understanding of the geological setting and mineralisation style, with quartz-stibnite veins observed parallel to and cross-cut by brittle-ductile shearing related to mineralisation. Subsequent analysis highlighted no significant statistical correlation between gold grade and stibnite content, with a good correlation noted with arsenopyrite. The grade and continuity to mineralisation is now characterised by alteration style, deformation intensity, and intrusive contacts.

Low-grade (+1 g/t Au) mineralisation is associated with weak to moderate shearing with silica-carbonate alteration and disseminated sulphides with weak strain. Arsenopyrite is rare. High-grade (+3 g/t Au) mineralisation is associated with high strain including brecciation, extensional and shear veins, with moderate to strong silica-carbonate alteration and sulphides. Arsenopyrite is the dominant sulphide associated with gold, with arsenopyrite and pyrite also observed as vein selvedges +/- visible gold.

Veins identified by trenching and diamond drilling vary in style and include extensional, sheared, and boudinage veins. Veining associated with +1 g/t Au mineralisation is sub-parallel in orientation to primary strain (shearing) highlighting the genetic link between deformation and mineralisation.

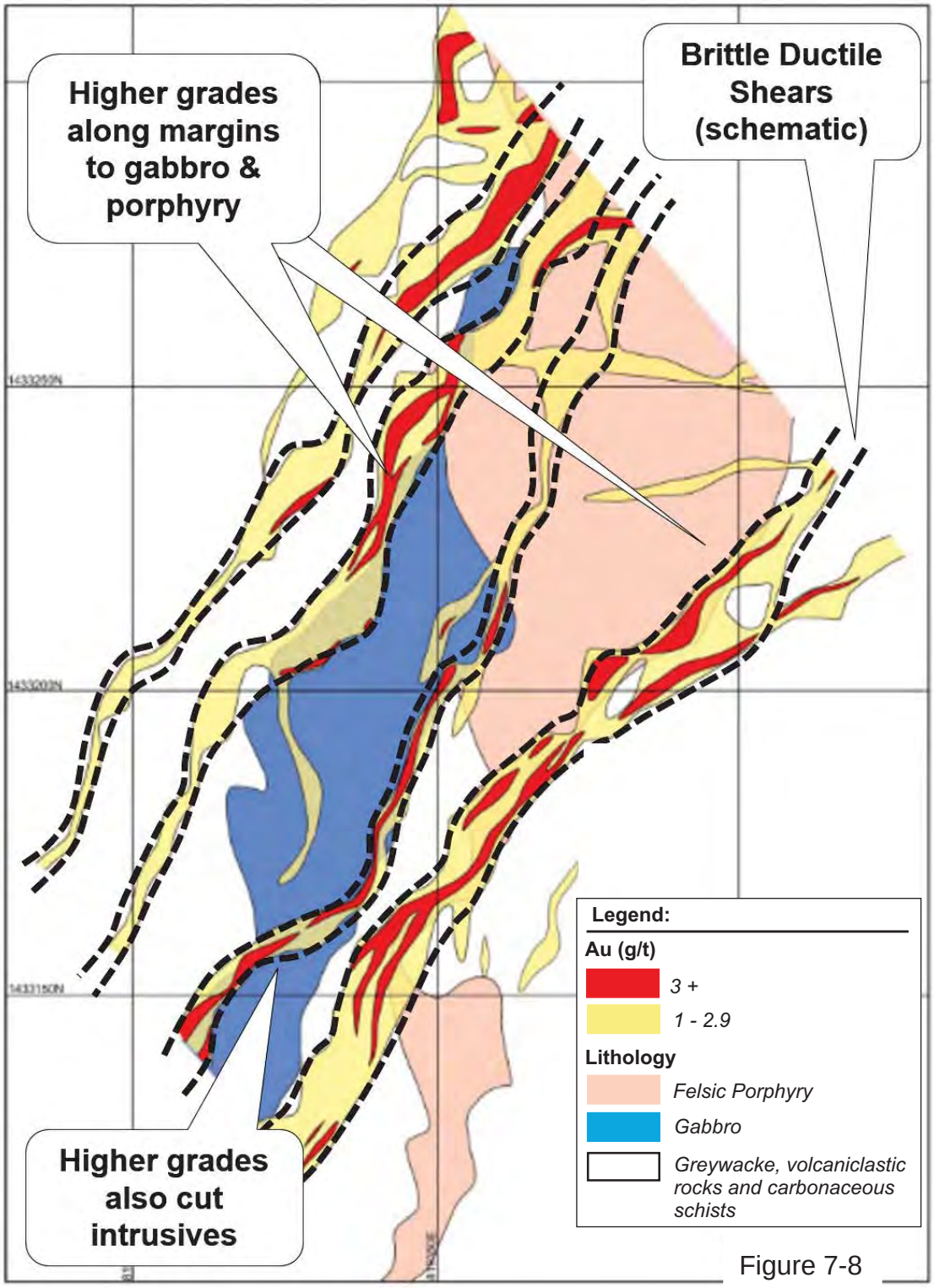


Figure 7-8

Showing the continuity and grade distribution of mineralized zones with respect to the gabbro and felsic porphyry intrusions

0 15 30 45 60
Metres

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Level Plan Through Five Metre Spaced Drill Fences in the Central Zone of Massawa at 150 m RL

July 2019

Source: Barrick, 2019.

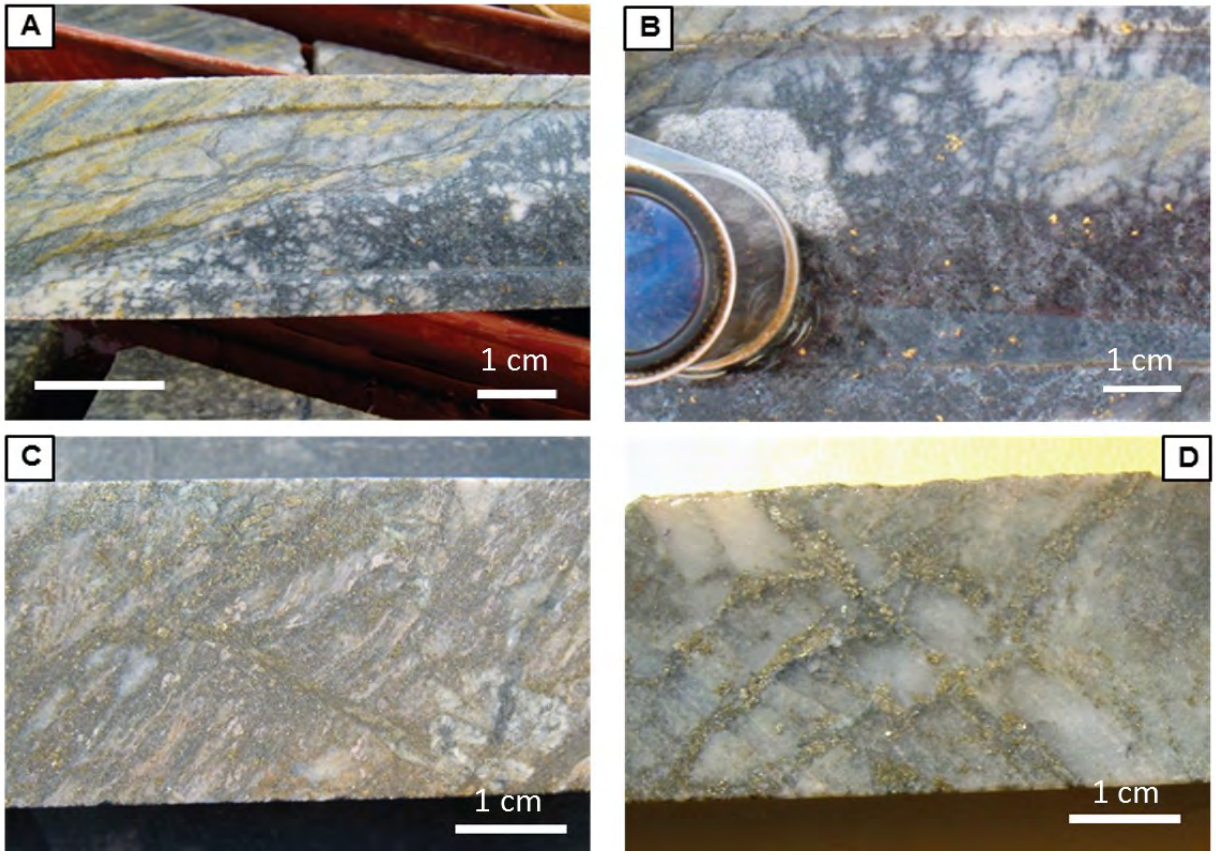


Figure 7-9 Core Photographs Showing Alteration and Mineralisation Styles Present in the Central Zone

A- Quartz-Stibnite-Au Vein with Proximal Sericite-Carbonate Alteration of Volcaniclastic Host with Disseminated Arsenopyrite and Pyrite in the Wallrock. B- Coarse Visible Gold Associated with Stibnite Veins. C and D- Silicified Rock with Fine Disseminated and Stringer Distal Pyrite and Lesser Amounts of Arsenopyrite

7.6. Massawa Northern Zone

The Northern Zone (NZ) has a strike length of 2.5 km and consists of a main NNE trending mineralised structure with discontinuous footwall (FW) and hanging wall (HW) lodes (Figure 7-10 and Figure 7-11). Mineralisation is localised in a damage zone adjacent to highly strained bands of fine- to medium-grained felsic and lithic wacke, wacke with subordinate carbonaceous shales, and gabbros. The NZ is sub-divided into two further zones based on structure (Northern Zone 1 and Northern Zone 2).

The southern 1.1 km of the Northern Zone (NZ1) hosts discontinuous, weaker gold mineralisation (average grade of 1 g/t Au to 1.5 g/t Au). The weakly silicified, brittle-ductile, mineralised shear is less than 10 m in thickness and is sub-vertical to steeply dipping to the ESE. The higher-grade but narrow mineralisation is focussed at the margins of a medium-grained greywacke and lithological contacts with contrasting grain size. Northern Zone 2 (NZ2) represents the northern and highest grade (>4 g/t Au) portion of the deposit. Mineralisation is predominately confined to a single, continuous, narrow zone (10 m to 15 m average width), which is sub-vertical to steeply dipping (>70°) to the WNW. The mineralisation is bounded by two prominent carbonaceous shale horizons within the sedimentary sequence.

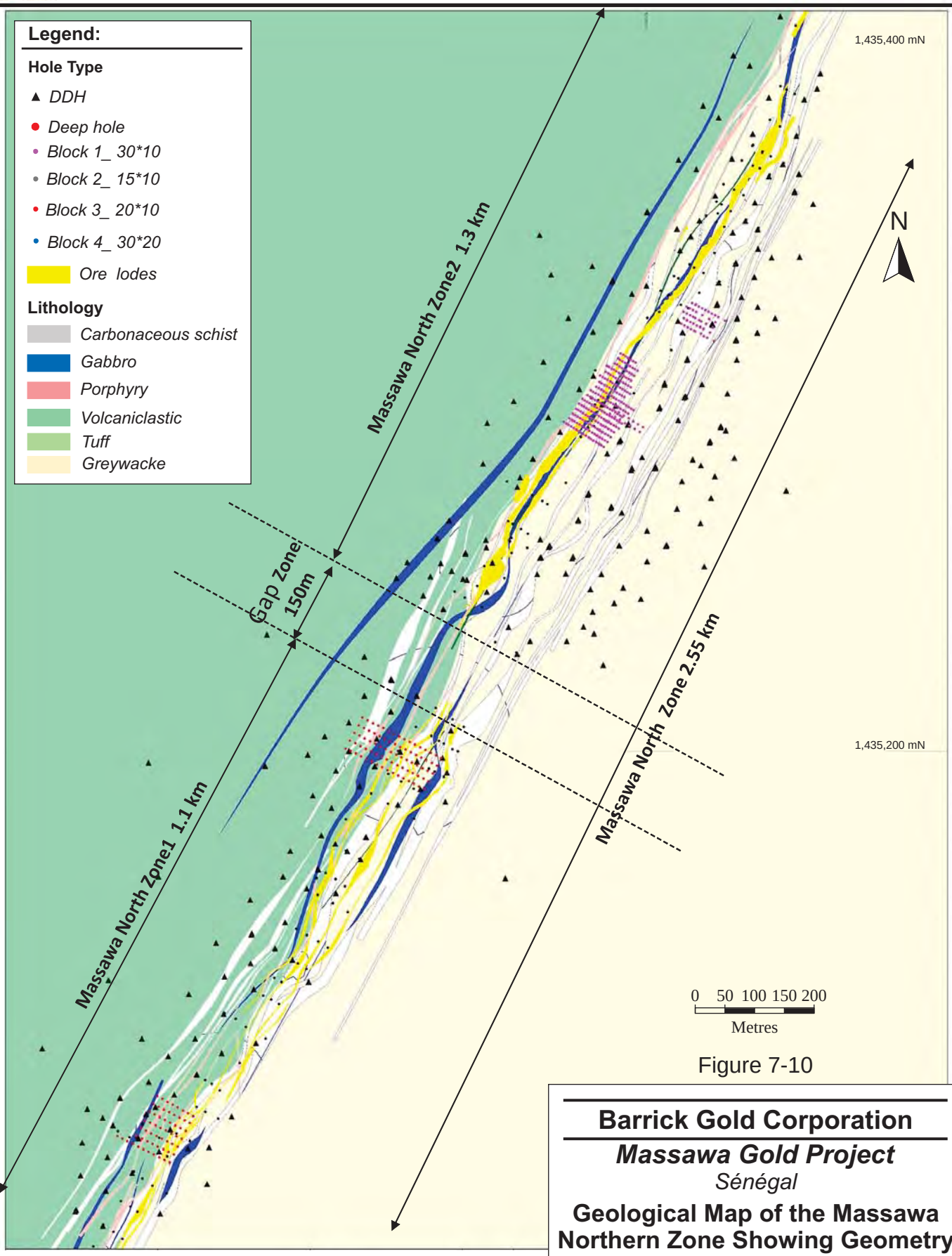


Figure 7-10

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Geological Map of the Massawa Northern Zone Showing Geometry of the Mineralised Zones

July 2019

Source: Barrick, 2019.

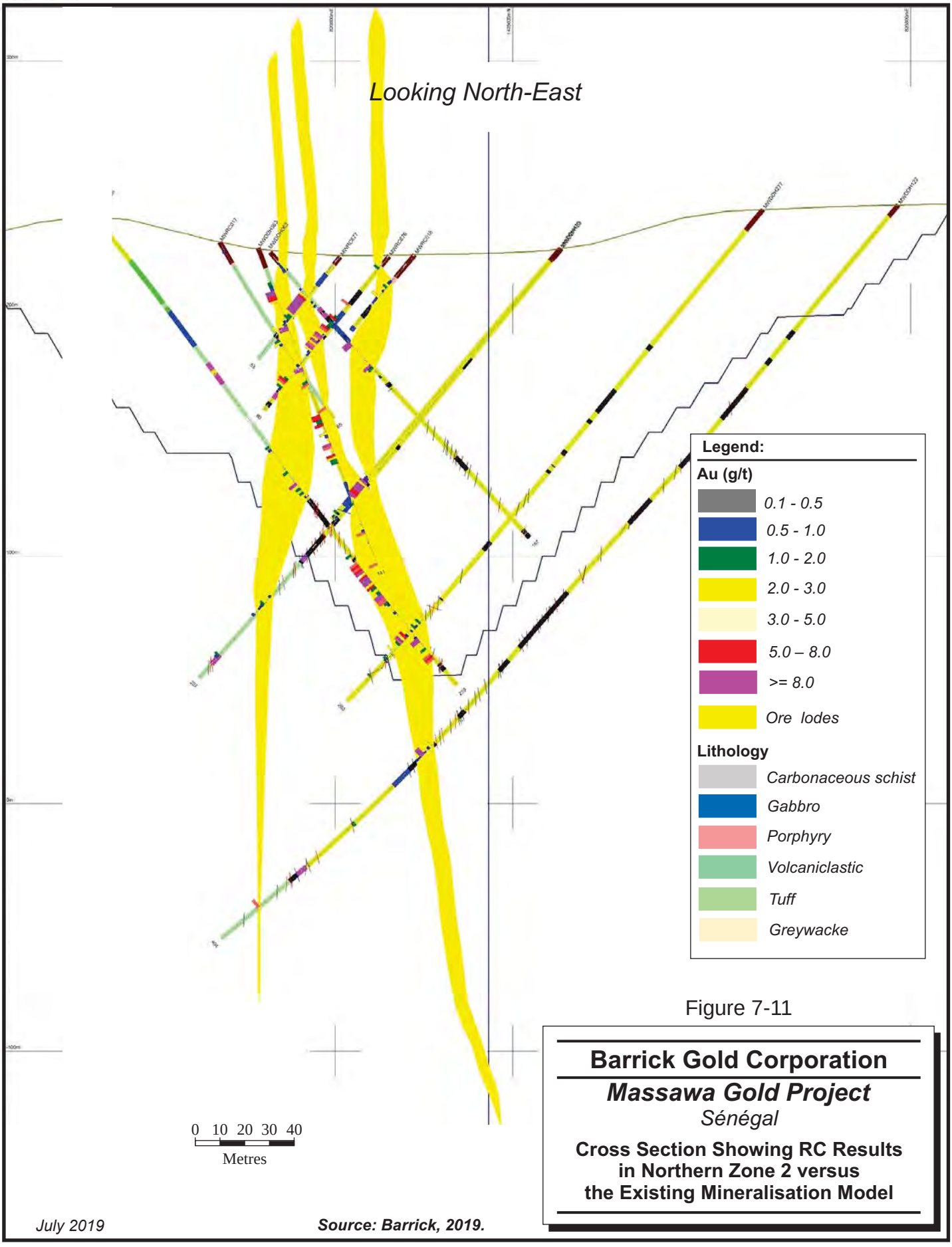


Figure 7-11

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Cross Section Showing RC Results in Northern Zone 2 versus the Existing Mineralisation Model

July 2019

Source: Barrick, 2019.

Mineralisation in the NZ is characterised by disseminated arsenopyrite and arsenian pyrite. No quartz-stibnite visible Au veins are seen in the northern parts of the deposit. High grade is associated with crack seal carbonate veining where arsenopyrite intensifies at the margins. Early silicification is less significant in the NZ, where the alteration is mainly composed of sericite, carbonate, and chlorite alteration affecting both sedimentary rocks and the gabbro unit. Primary and tectonic rock fabrics are often still clearly visible.

Two mineralised domains are modelled in the NZ:

- A higher-grade domain (average 5 g/t Au) consists of 7% to 10% disseminated sulphides (arsenopyrite>pyrite) associated with ductile shearing with extensional quartz-carbonate veins.
- A lower-grade domain (average 1.5 g/t Au) consists of 1% to 3% disseminated sulphides (pyrite>arsenopyrite) associated with brecciation and extensional quartz-carbonate veins.

Compared to the CZ, higher total sulphide contents (averaging 3% to 5%); higher average arsenopyrite-pyrite ratios (0.5-1); and higher As levels are recorded. Gold is largely refractory to direct cyanide leaching developed within the crystal lattice of arsenopyrite and, to a lesser extent, arsenian pyrite.

7.7. Sofia Deposit Geology

7.7.1. Regolith

A 3.4 km 120 ppb Au (97th percentile) soil anomaly delineates the mineralisation at Sofia. The anomaly, confirmed by trenching and drilling, identified the Sofia Main deposit which strikes NE and Sofia North with mineralisation striking north-south. Sofia Main is partially located on the upper limits of a more than 800 m wide, transported lateritic hardpan of CP2/3 (Lower) that has formed an apron believed to be transported off a steep sloped, topographic high west of the mineralisation. At Sofia North, RAB and reverse circulation (RC) drilling confirmed a thick laterite cover (>5 m) that extends for over one kilometre where it is overlain by alluvial and colluvium sediments into a major east to west trending river. This thick consolidated transported material may explain the low Au response of the soils along this area of the Sofia-Sabodala structure.

7.7.2. Lithology

Western Mafics: The NE trending Sofia–Sabodala structure defines a major terrane boundary that separates a continuous unit of predominantly mafic-ultramafic rocks (dunite, gabbro, basalts, and dacitic lavas) to the west, and andesitic lavas and tuffs to the east. The country rocks are intruded by gabbros, diorites, and narrow quartz-feldspar porphyries (QFP). The diorites are spatially related to the mineralisation and commonly altered to silica-albite. The QFP has a tonalitic mineralogy comprising of distinctive rounded quartz and plagioclase.

Footwall Mafics and Volcaniclastics: The FW geology consists of a package of chlorite altered equigranular gabbro with subordinate basaltic-andesite lavas and tuffs. The gabbro package is up to 60 m thick thinning toward the north-east ranging from 10 m to 20 m in thickness. The gabbro predates shearing and hosts gold when intersected by the mineralised structures. Here it

is commonly altered and strained with strong silica-albite-chlorite-sericite on the western margin of the package. Underlying the gabbro are lithic-wacke, lapilli and ash tuffs, and occasional carbonaceous shales and siltstones. All intrusive rocks have a weak foliation and weak to strong alteration with strain developed at the lithological contacts. Late dolerite dykes cross-cut all units described above and post-date mineralisation. Core photographs of the main lithologies are shown in Figure 7-12, Figure 7-13, and Figure 7-14.

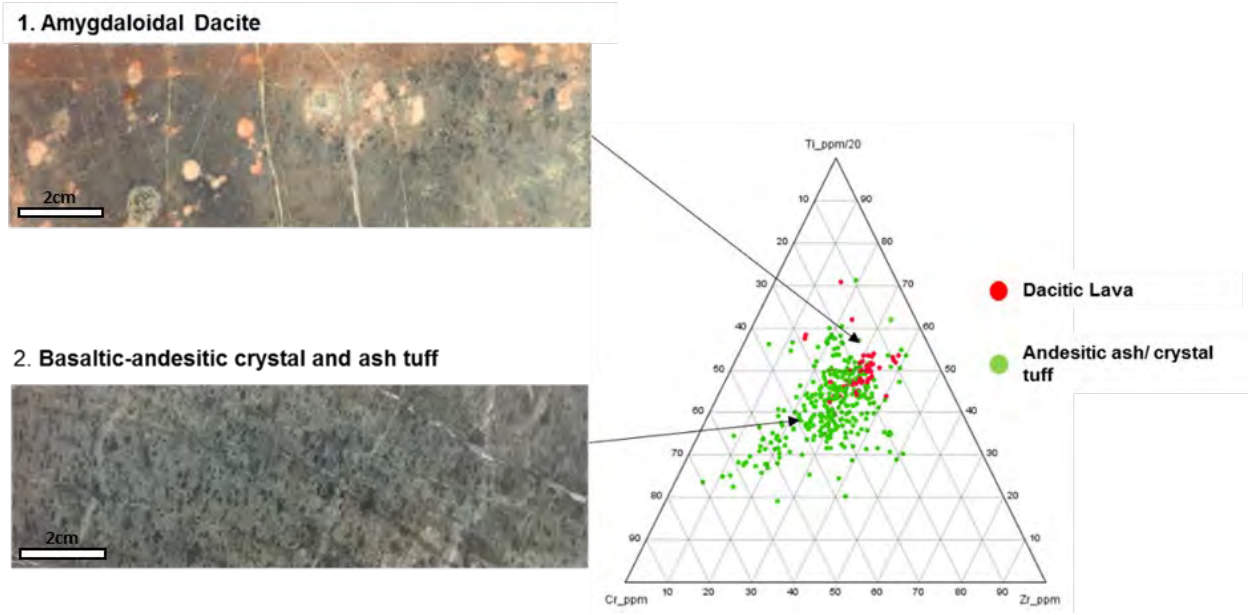


Figure 7-12 Core Photographs of the Dominant Hanging Wall Extrusive Lithologies Confirmed by Portable XRF Data at Sofia

Amygdaloidal dacitic lava; 2- Andesitic ash tuff

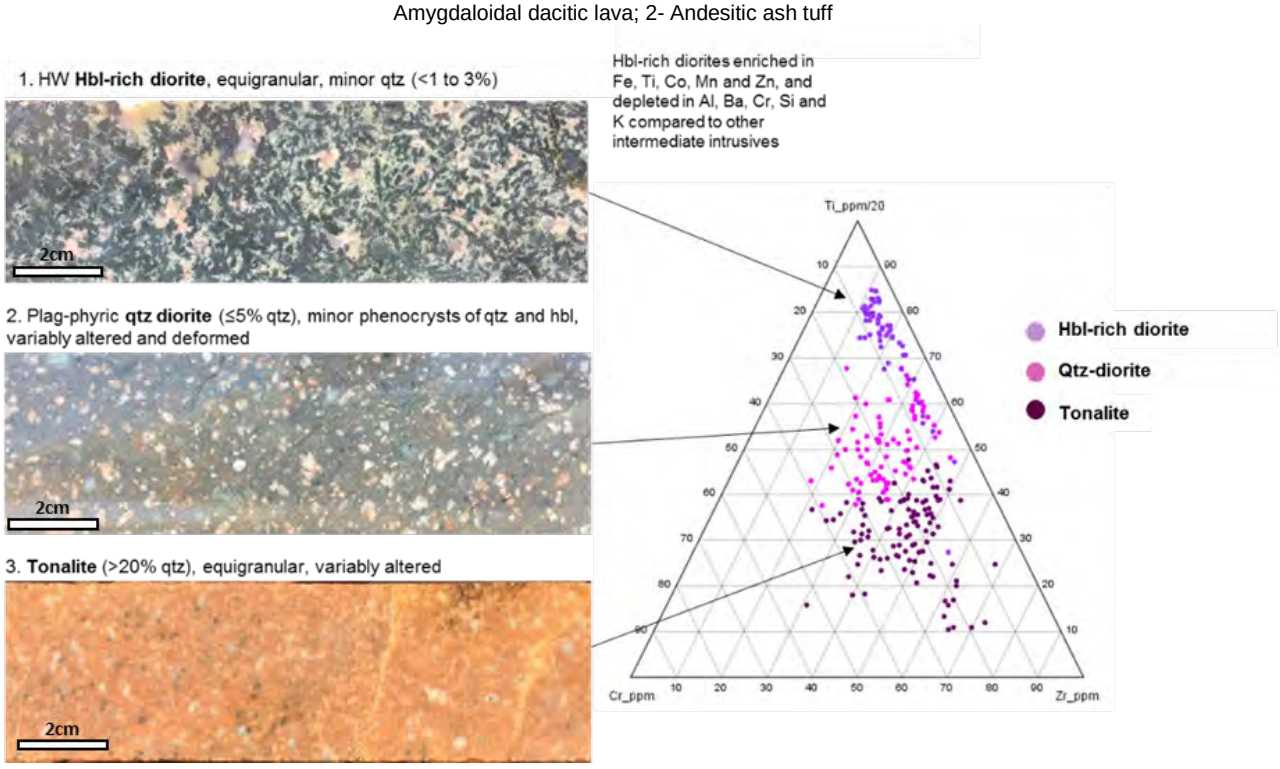


Figure 7-13 Core Photographs of the Dominant Hanging Wall Intrusive Lithologies Confirmed by P XRF Data at Sofia

1- Hornblende rich diorite; 2- Quartz diorite; 3- Tonalite

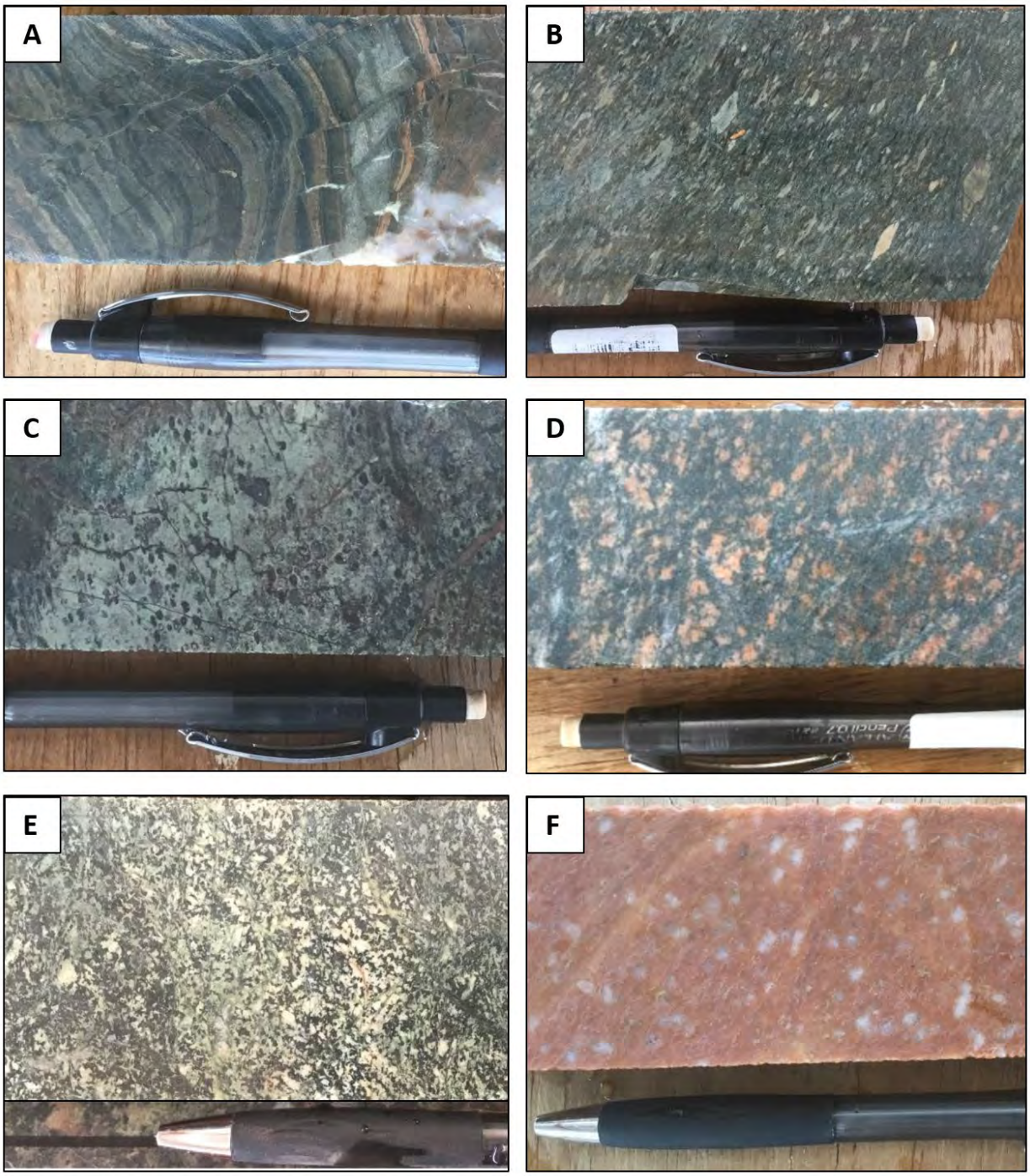


Figure 7-14 Core Photographs of the Dominant Lithologies at Sofia

A- Volcaniclastic siltstone to fine sandstone; B- Poorly sorted crystal-lithic volcaniclastic breccia; C- Amygdaloidal basalt; D- Diorite; E- Gabbro, F- Quartz-porphyry with strong albite-haematite alteration

7.7.3. Structure

Sofia lies on a NE trending (015° to 040°) structure which is likely a second or third order splay off the main Sofia-Sabodala Shear (Figure 7-15). At Sofia Main, the structures are modelled to be inter-connecting listric (20° to 80°) thrusts with zones of high strain and alteration developed in the FW. To the north of Sofia Main, the system turns to the NNE with the steep west dipping mineralisation developed at the eastern contact of the Western Mafics.

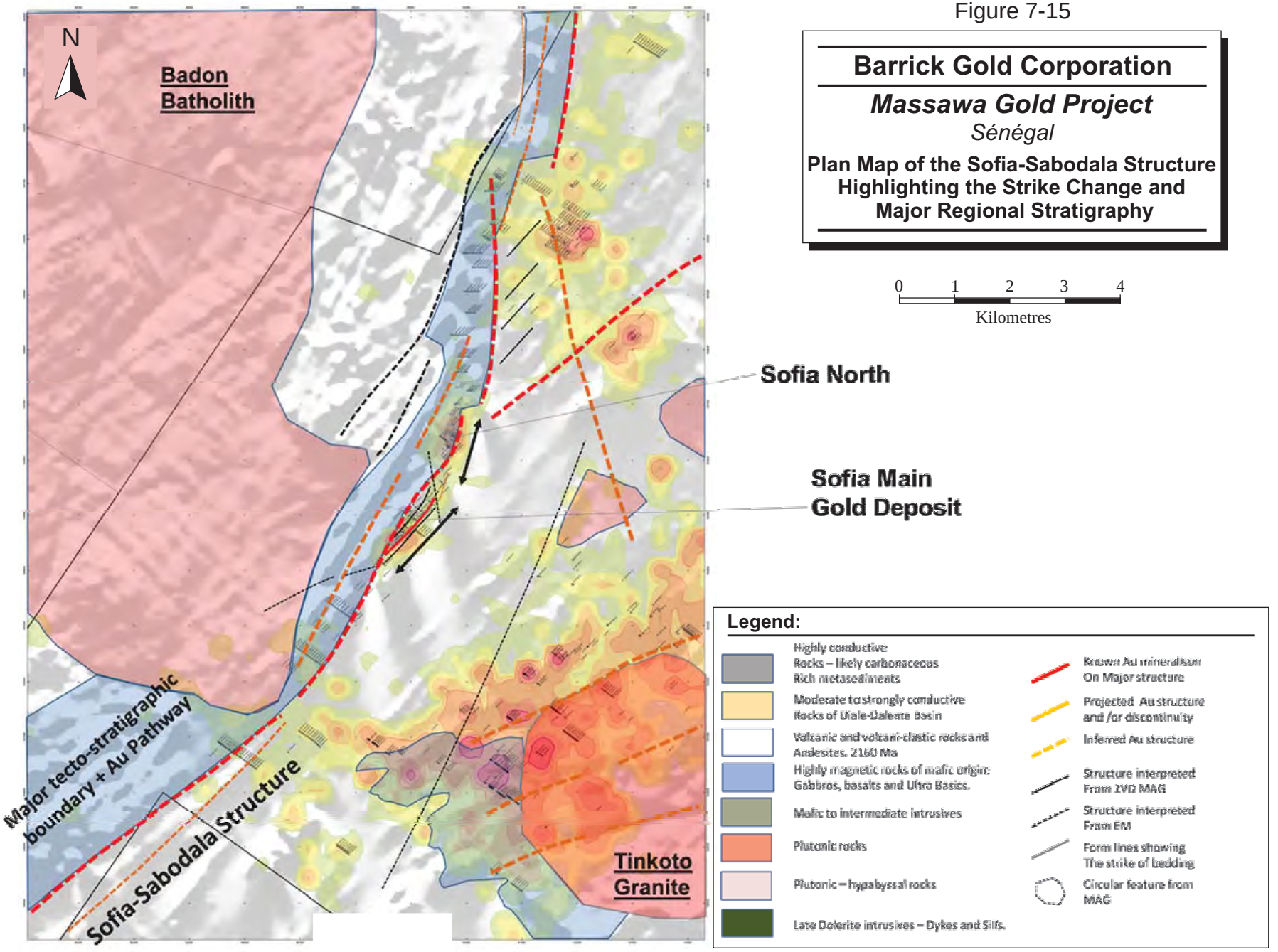
Figure 7-15

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Plan Map of the Sofia-Sabodala Structure Highlighting the Strike Change and Major Regional Stratigraphy



July 2019

Source: Randgold, 2019.

7.7.4. Mineralisation

Sofia is located approximately 10 km to the west of Massawa, along the more than 30 km long 010° Sofia-Sabodala Shear Zone which hosts the Sabodala gold deposit 27 km to the north. Gold mineralisation has been delineated over a four-kilometre strike length and is controlled by both the host lithology and geometry of strong brittle-ductile structures. The mineralised shear has been differentiated into two zones based on different structural trends along the tectono-stratigraphic boundary. At Sofia Main, brittle-ductile structures strike 040°, whereas at Sofia North mineralized structures strike 010° (Sofia North).

7.7.5. Sofia Main

Mineralisation at Sofia Main has a strike length of 1.4 km and was developed within a compressive system where changes in HW geometry from sub-vertical to steep/moderate west dipping has a structural control on the mineralisation. Hydrothermal mineralisation is localised in areas of relatively low fluid pressure associated with fault jogs. Prospective zones of localised extension can also be created within thrust zones at bends in the HW ramp (low/moderate to a high angle) (ii) and/or intersections with lower angle inter-shear thrusts and splays (Figure 7-16).

In the HW adjacent to the mineralised zone, strained tuffs and a re-crystallised and highly altered and magnetised rock of ultra-basic origin are spatially associated to the high-grade mineralisation. This dyke or ‘Fe-stone’ is continuous along strike and is parallel to the ‘hanging wall shear’. At surface, the ‘Fe-stone’ is a distinctively highly weathered, iron oxidised, and silicified rock that retains its magnetic qualities. The eastern contact of these zones can be characterised by proto-mylonitic textures with intense silica-sericite alteration. Within these zones, isolated brittle structures develop, fragmenting the rock. These are characterised by high sericite-chlorite content and associated silica-carbonate veins that can often be traced down dip. These brittle-ductile structures are distinctly developed at the HW contact of the mineralised zone where the strain is orientated sub-parallel to the core axis with intrafolial folds prominent with increased brecciation associated at the contact with the higher-grade zones. This brecciation and associated alteration represents a likely fluid pressure boundary (Figure 7-17) where the maximum relative pressure difference (high pressure vs low pressure) is directly related to the tenor of gold (extension).

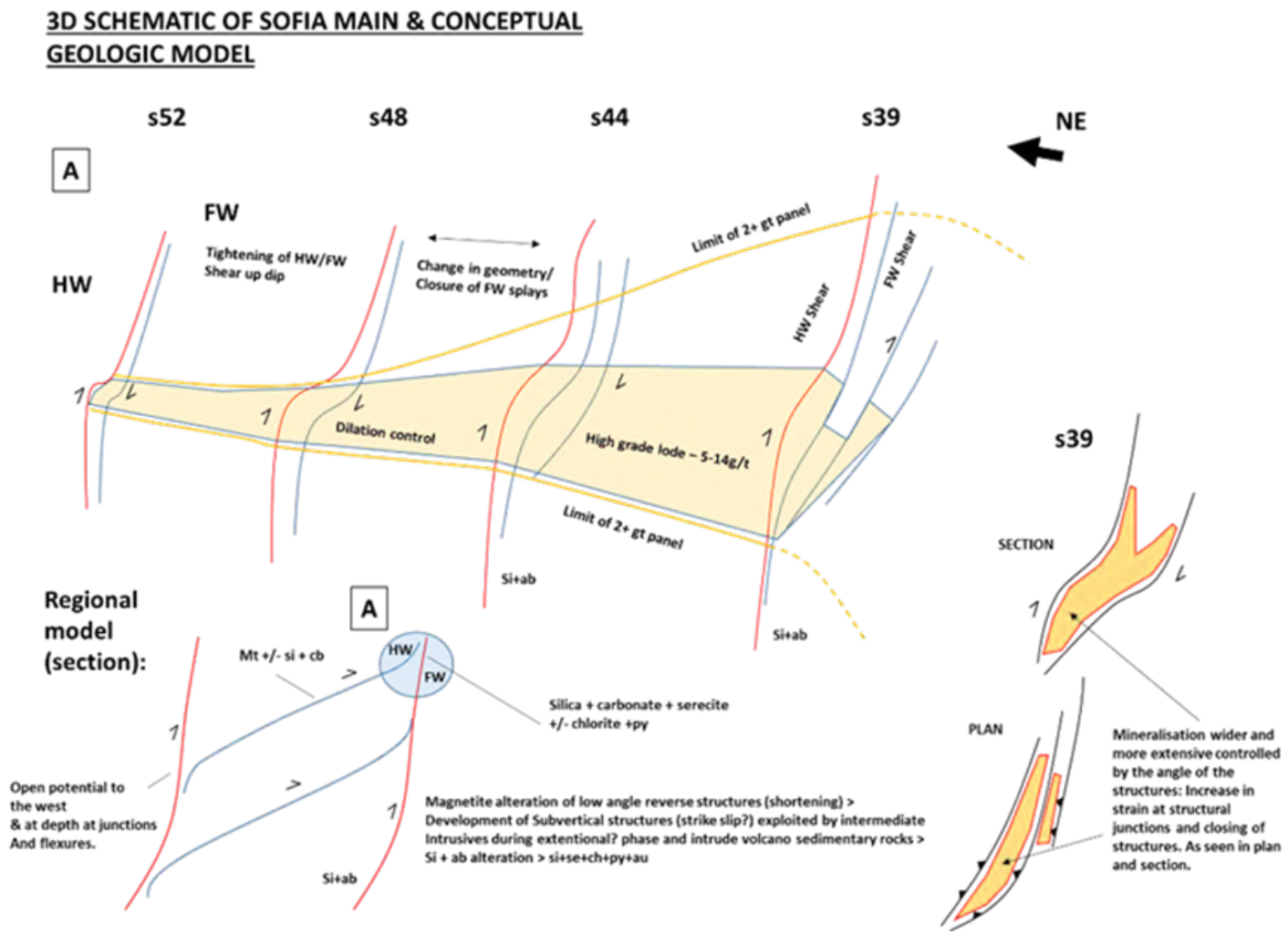


Figure 7-16 3D Schematic of Sofia Main: To the SW of Sofia Main

The mineralisation is thought to be structurally controlled where an apparent more tabular geometry comes to the surface (over 2 g/t Au mineralisation), that is controlled by a series of splays that are modelled to terminate on the FW structure, where the geometry at depth is the same along strike. Mineralisation is observed to increase at the junctions of these structures. Moving further to the NE, the FW splays close with the mineralisation and shears become tighter and more acute. Here a high tenor of gold is thought to be controlled by the dilatational/extensional HW geometry.

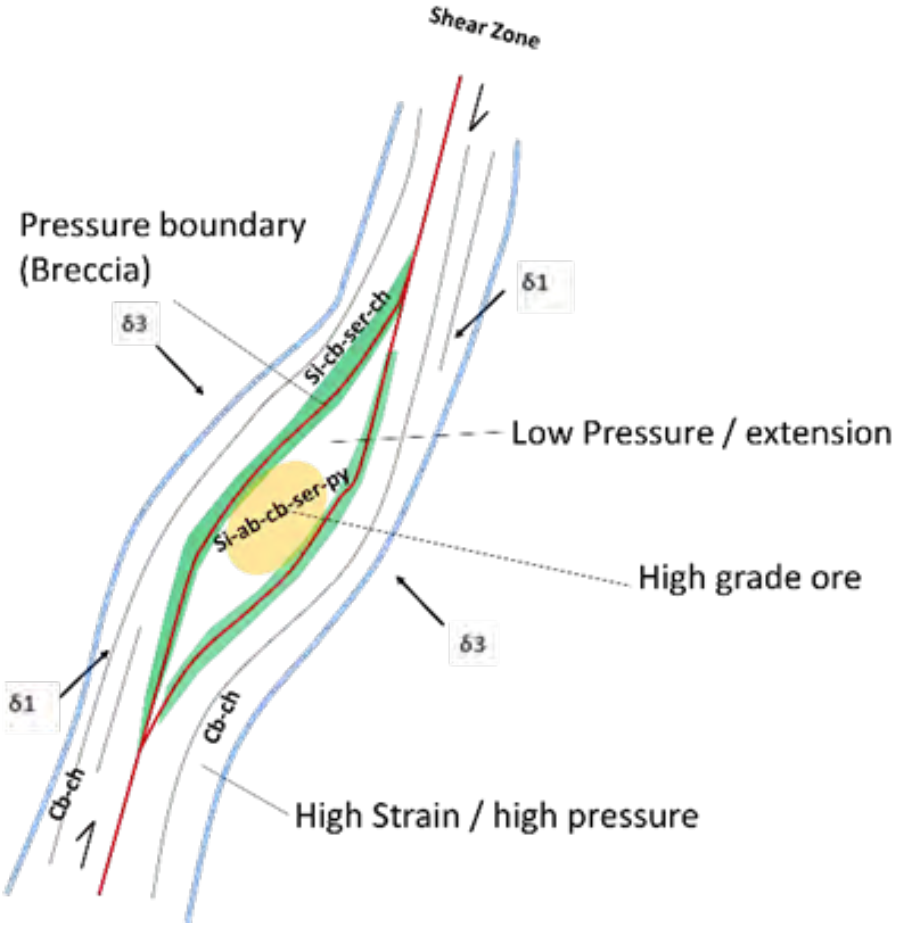


Figure 7-17 Simplified 3D Schematic of the Dilatational Geometry Modelled to Control the High-Grade Lode of the MZ at Sofia Main

Main Zone (MZ)

Mineralisation in the MZ is predominantly hosted in a quartz-diorite intrusive that has been strongly altered by albite-silica and overprinted by late silica-carbonate-sericite-pyrite mineralisation. The silica-carbonate bleaching is distinct and can pervasively alter the dark magnetite rich 'Fe–stones' directly implicated at the HW of the high-grade mineralisation. High grade is generally developed within stockworks and brecciated zones in the quartz diorite host, with the ductile strained zones characterised by high sericite content with lower Au content. The albite-silica phase of alteration is thought to act as a ridged body between the bounding structures under high strain with slip along the shear facilitated by a high fluid pressure model causing brecciation. Albite-silica sericite alteration developed in the HW (Sofia structure 2/SF2) and FW zones (Sofia Structure 1/SF1) can display progressively high strain with sulphides up to 5%, however, they do not display the high gold grades associated with the MZ and are generally below 1 g/t Au.

The steeply west dipping (65° to 80°) immediate FW geology is commonly composed of a NE (035° to 045°) strike continuous mafic sequence (gabbro with subordinate basalts and tuffs) that is sub-parallel to the main mineralized lode. Discontinuous zones of high-grade mineralisation are commonly hosted in silica albite chlorite veins where the strain transfers and continues into the FW of the system.

The mineralogy is dominated by pyrite with trace Cu-sulphides. Gold is predominately free milling, associated with pyrite and silicate gangue. Gold grain compositions consist on average of 93% Au with 7% Ag.

Footwall Zone (FWZ)

In the south of the Sofia Main, a lode of high grade (above 3 g/t Au) mineralisation is developed in the FW gabbro, immediately east of the bounding FW shear of the MZ. Over 300 m in strike, the mineralisation is modelled to be related to lower angle splays off the steeper FW shear. From the south, the splays open out to the east with a higher tenor of gold where the structures close at the angle of the FW shear down dip. Along strike to the north, these secondary structures begin to merge with the FW shear, again increasing the tenor of gold. This suggests that there is strong structural control on the mineralisation in the south, with strain and grade increasing at structural intersection.

Two styles of mineralisation are evident:

- Silica-albite-chlorite-pyrite alteration commonly forms at the sheared western margin of the FW gabbro. The silica-albite alteration is less intense than the MZ, whereas chlorite is more abundant. The alteration typically displays a banded appearance and is not completely textually destructive.
- Silica-carbonate ± albite veining. It is characterised by typically weakly foliated rocks with high-grade intercepts associated with localized higher strain zones. The wallrocks appear to display chlorite-magnetite alteration typically developed within fine-grained contact of the gabbro. This mineralisation style tends to occur at the intersection between the FW shear and the secondary lower angle splay.

The structural model prescribes the fault linkage and shear geometry in the HW as the likely primary structural control of the mineralisation. This suggests that the mineralisation can develop where these high strain jogs occur with relative low fluid pressures (extension) facilitating fluid influx and slip along the shear associated with brittle deformation. In addition, the interaction of the intra-shear faults in the HW are likely implicated. Fluid interactions with rock within the FW gabbro and the 'Fe-stone' may localise mineralisation at the contact. The lithological variations also may have a rheological control.

7.7.6. Sofia North Deposit

Sofia North has a strike length of 2.3 km and consists of one main NNE trending mineralized structure and a discontinuous FW lode. Mineralisation is mainly hosted in moderately altered quartz diorites, andesitic ash tuffs, and gabbros (western mafics). The northern zone is sub-divided in two mineralisation styles based on structure (strain/alteration) and vein hosted styles (intensity). The main mineralised structure that controls the strain and alteration is developed at the major eastern contact of the Western Mafics that turns in strike from 040º to 010º and has been delineated over more than 2 km by 200 m spaced trenching and RC drilling. The structure is strike continuous from Sofia Main. The structure is steep to west dipping and planar with the western mafic stratigraphy.

At Sofia North, the diorite is comparatively narrower compared to Sofia Main with elevated grades above the redox boundary suggesting a supergene influence. A second, high-grade (>5 g/t Au) but narrow mineralised zone is hosted in silica carbonate veins commonly developed within the HW stratigraphy lithological contacts (gabbros/lavas), suggesting that the strain has transferred from the FW (Sofia Main) to the HW.

The mineralisation is defined by a strong occurrence of fine disseminated pyrite (1%) accompanied by strong quartz, albite, and carbonate alteration. Mineralisation occurs in quartz-albite altered intermediate intrusives as at Sofia Main, with quartz ± magnetite ± chlorite. Vein hosted mineralisation in the mafic host is spatially associated with chlorite-rich shears with silica carbonate veining.

7.8. Delya Deposit

7.8.1. Regolith

Delya is located along the MTZ, approximately 10 km north of Massawa. Detailed soil sampling along with detailed mapping and litho-sampling across successive exploration campaigns defined a linear 6 km, 100 m wide, 030° trending greater than 100 ppb Au anomaly confirmed by subsequent trenching and drilling. The dominant regolith comprises Cuirasse Plateaus (ferricrete, Fp2)) and Erosional Outcrops (Es) orientated broadly NE sub-parallel to topography and the trend of the MTZ.

7.8.2. Lithology

The lithological sequence is composed of massive and foliated andesite to the west, and an eastern sedimentary package consisting of shales, carbonaceous shales, lithic, and feldspar-wacke. These units are intruded by a wide gabbro unit. These lithologies have undergone strong ductile deformation, with structures trending NE (030° to 040°) and steeply dipping to the east. The main Delya mineralised structure is located at the contact between the gabbro and the sedimentary package and is bounded by two carbonaceous shale units.

The dominant lithologies at Delya are shown in Figure 7-18.

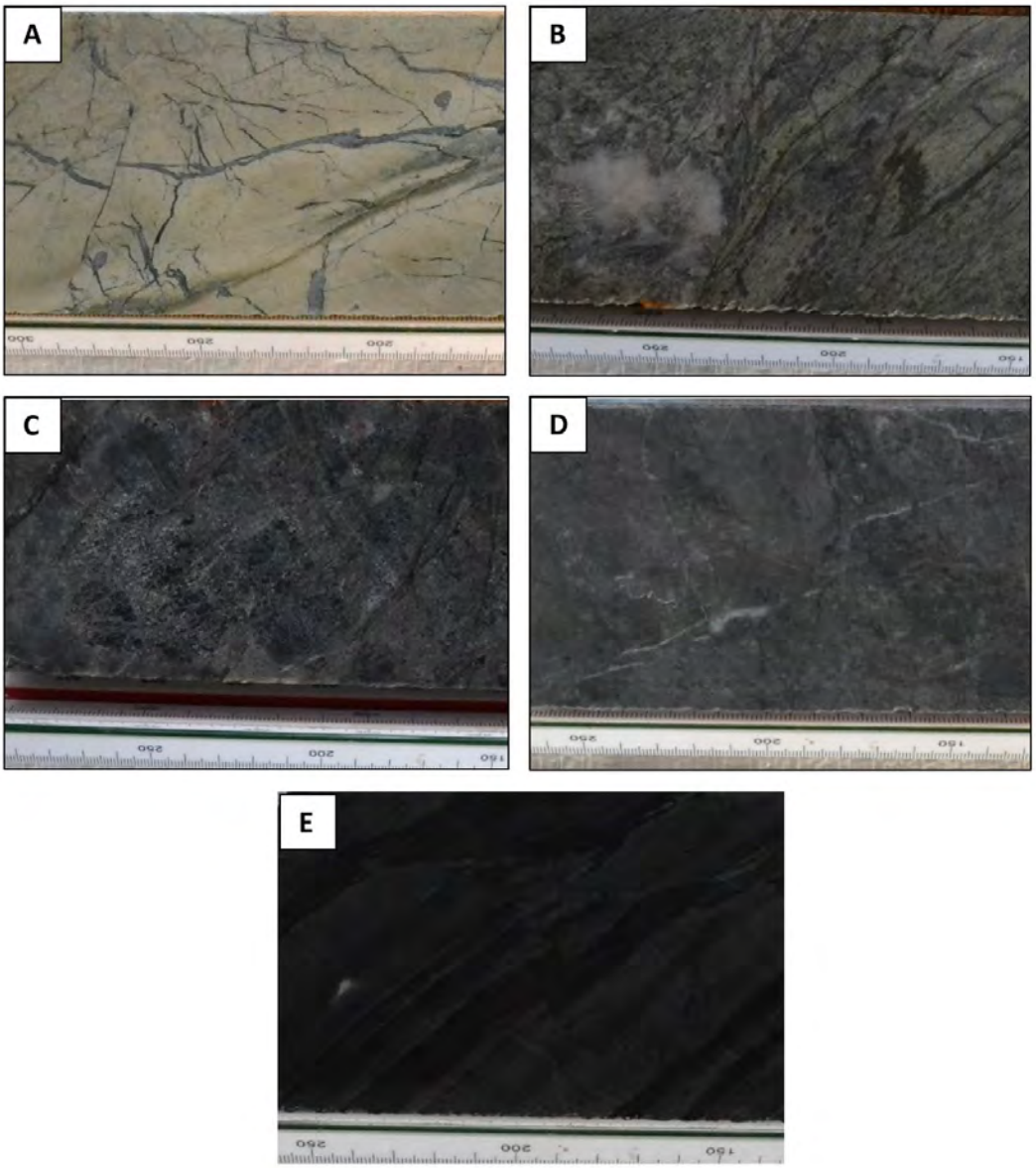


Figure 7-18 Core Photographs of the Dominant Lithologies at Delya

A- Sericite altered fine grained andesite tuff; B- Sericite and fuchsite altered mafic dyke; C- Brecciated carbonaceous shale; D- Brecciated chert; E- Carbonaceous shale.

7.8.3. Structure

Three parallel zones of mineralisation have been defined at Delya, over a one-kilometre strike length. The main zone of mineralisation is hosted at the lower margin of the gabbro within highly sheared, silicified and sericitised schist. The mineralized zone varies in thickness from 3 m to 10 m (average of 5 m), contains higher grades (up to 5 g/t Au), and dips to the east at 85°. The other branches are located to the west and have an average dip of 84° to the west. Mineralisation has been drill tested to a vertical depth of 150 m below the surface.

7.8.4. Mineralisation

The mineralisation and alteration assemblage consists of sericite-silica-carbonate and chlorite alteration, associated with strong disseminated fine arsenopyrite and pyrite (Figure 7-19). Arsenopyrite is dominant over pyrite (similar ratios to the Massawa NZ). Gold is largely refractory in nature and locked up in the crystal lattice of arsenopyrite.

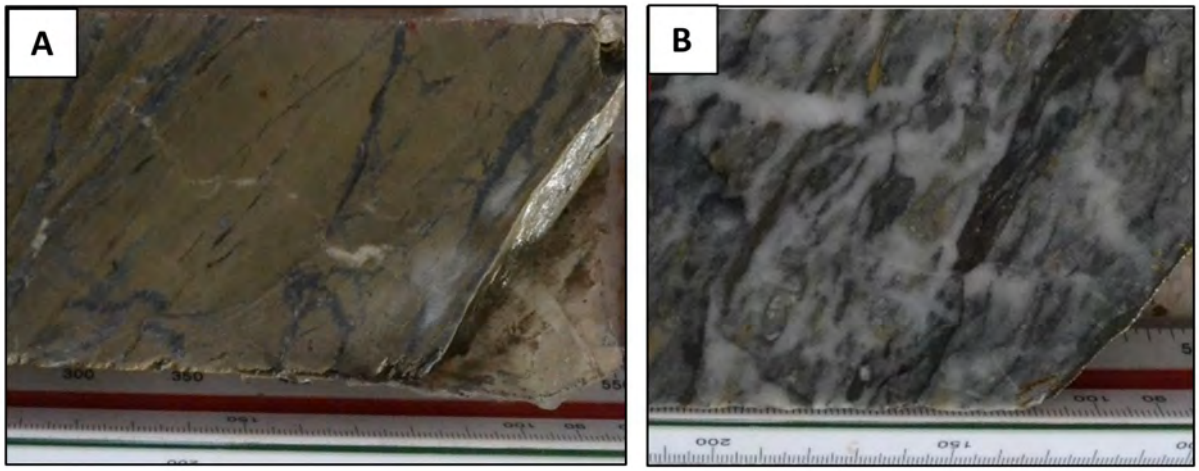


Figure 7-19 Core Photographs of Mineralisation Styles at Delya

A- Sericite schist host unit; B- Silica alteration and brecciation overprinting early sericite alteration. Note high sulphide content.

7.9. Satellite Deposits

Two satellite deposits occur in a 15 km radius around Massawa (Tina and Bambaraya) (Figure 7-20).

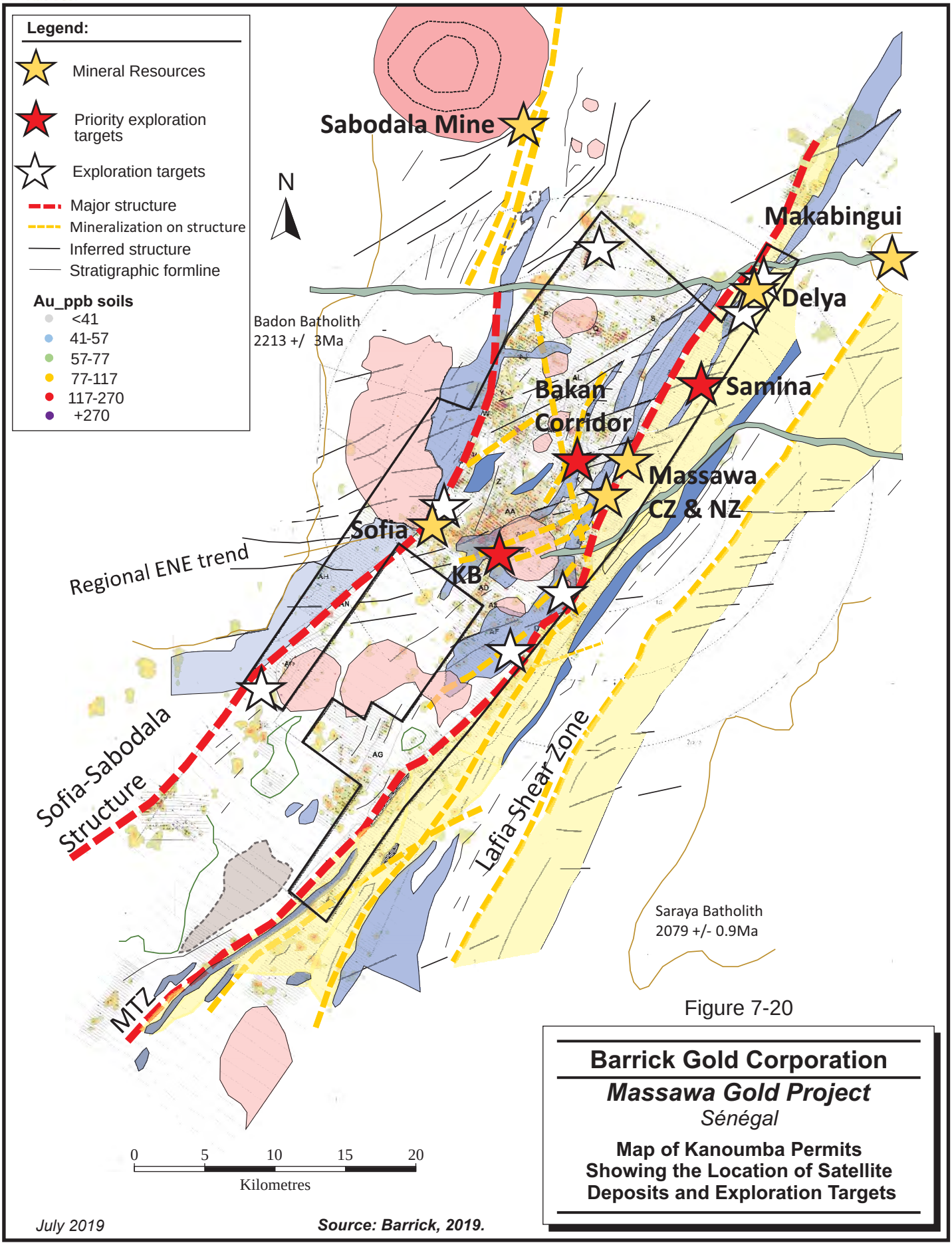


Figure 7-20

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Map of Kanoumba Permits Showing the Location of Satellite Deposits and Exploration Targets

July 2019

Source: Barrick, 2019.

7.9.1. Tina Deposit

The Tina deposit is located along the Bakan Corridor that groups together a number of anomalous gold in soil and rock anomalies (Tiwana, Tizia, Tina, Bakan, and Khosa). Mineralisation along this corridor is controlled by the over 10 km long Kossanto Shear. This belt parallel structure is reactivated at Tina by a north-south belt discordant dextral fault that runs from Teranga Gold Corporation's Goulouma targets to the north down to Massawa in the south. The Tina deposit is defined by a 1,500 m long and up to 20 ppb Au NE trending soil anomaly.

The geology of the Tina region comprises of a NE sequence of felsic and intermediate volcanic rocks (andesite, dacites, and rhyodacites), tuff, cherts, and felsic intrusive suites ranging from diorite to monzonites. Field structural studies show a dominant NE foliation, discrete NS foliation, and three mains directions of quartz vein: NE, NS, and ESE.

Disseminated pyrite-Au mineralisation has so far been intersected in drill core over a 700 m strike length, with an average width of 15 m and remains open in all directions and at depth. Two mineralized branches have been identified in strongly sheared and altered (silica, sericite, and K-feldspar) felsic intrusive and gossanous tuffs:

- A main western branch, which is steeply dipping, occurs on several sub-parallel sets (although two are more continuous), and is hosted by a felsic intrusion.
- An eastern lower grade mineralised branch, which also forms several narrow sets, dips 75° to the west, and is related to shear tuffs and gossanous rocks.

Recent interpretation indicated a potential upside towards the south (still open in all directions) and within the tuffs, outside the felsic intrusive. The structure continuity coincides in the field with gossanous rocks and a strong soil anomaly over at least one kilometre along strike that remains largely untested.

7.9.2. Bambaraya Deposit

Bambaraya is located in the NW corner of the Kounemba Permit along the Sabodala Shear Corridor, approximately 18 km to the north of Sofia. The NE trending (approximately 032°) shear zone marks the brecciated contact between pillowed basalts and massive and foliated andesites. These volcanic rocks have been intruded by gabbro, dolerite, and felsic plutonic rocks near the deposit.

The deposit is distinguished by a 50 m to 150 m wide, 2.2 km long gold in soil anomaly (50 ppb Au). To date, bedrock mineralisation has been intersected over 900 m. Bambaraya is characterised by Loulo-style mineralisation, with gold linked to quartz-tourmaline veins. These veins run oblique (010º) to the main shear geometry. An updated geological and structural model distinguishes two mineralised bands: a thicker western branch with more consistent grades, which dip steeply to the west (approximately 70°); and an eastern branch which moderately dips to the east (approximately 60°). The eastern mineralisation is in a potential dilatational zone where NE structures were dextrally reactivated by two north-south structures. Within this corridor,

the mineralisation is east dipping, whereas it dips towards the west outside the reactivated corridor.

Additional detailed geological mapping along the Bambaraya corridor has proved the continuity of the quartz-tourmaline veins towards the south. Subsequently, this work highlighted potential upsides on both the eastern and western branches. A strike length of 300 m remains untested along the western branch, whereas the updated geological model suggests a possible southern extension of the eastern branch over at least 450 m.

The mineralisation assemblages are dominated by pyrite, with gold mineralisation located within the quartz-tourmaline veins (visible gold) and/or in the alterations selvages around these veins. Alteration consists of silica-tourmaline-iron carbonate-sericite.

7.10. Advanced Targets

7.10.1. KB Target

The KB target is located 5 km from Massawa CZ along ENE trending mineralisation located between the NE Sofia-Sabodala and MTZ structures proximal to the Tinkoto granite. Dominant lithologies include gabbro, diorite and mafic volcanics with sedimentary component increasing toward the north.

Mineralisation at KB comprises of multiple ENE/EW mineralised trends coincident with +1,175 ppb soil anomalism proximal to intrusives. The mineralisation assemblage is highlighted by the presence of quartz-carbonate veining with fine-disseminated pyrite and strong carbonate alteration. Variation in the style of alteration and mineralisation is observed including stockwork, brecciated and semi-massive veining interpreted to highlight proximity to fluid source.

Additional drilling is currently in progress to define the exploration upside of the multiple targets at KB and to test for additional targets.

8. Deposit Types

Mineralogical, structural, and geochemical evidence collected at Massawa suggests that the mineralised body represents an epizonal-style orogenic gold deposit (c.f. Groves et al., 1998). Fluid inclusion and stable isotope analyses (Treloar et al., 2014) indicate mineralizing pressure and temperature conditions of 220°C to 315°C at 1 kbar to 1.65 kbar, at crustal depths of 3.5 km to 6.5 km, with gold mineralisation directly linked to the intrusion of the felsic porphyries. Massawa shares some features with shallow level orogenic Au-Sb deposits in the Murchison Greenstone Belt of South Africa (Ashley et al., 2000; Jaguin et al., 2012) and Au-Sb mineralization associated with the Himalayan orogen in Tibet (Zhai et al., 2014).

In contrast, orogenic gold mineralisation at Sofia appears to have developed at deeper crustal levels, however, to date no detailed research has been carried out on the mineralised body.

9. Exploration

9.1. Massawa Central and North Zone

Massawa was first identified in early 2004 after completion of the Kounemba Permit regional soil survey. The ground was selected based on a mineralised structure that was interpreted from Landsat imagery to extend south from the Sabodala gold deposit and Niamia Permit in the north, where thick sequences of deformed volcaniclastics including andesitic lithic tuff were found. The regional soil sampling programme at 1,000 m by 100 m spacing took place between late 2003 and early 2004. A total of eleven targets were identified, among which seven were ranked as a priority for detailed work.

Due to the low tenor of the Massawa anomaly, it was originally selected as a secondary target. A detailed soil grid at Massawa was completed in mid-2005 which identified a 3.5 km long, 100 m to 400 m wide gold in soil anomaly at greater than 50 ppb Au. Subsequent soil sampling in 2008 extended the anomaly to the south and north by a further 3.4 km for a total strike of 6.2 km.

The first trench was positioned over the anomaly in November 2006. MWTR001 was located on the SW part of the soil anomaly and returned an encouraging result of 10.9 m at 2.03 g/t Au which was followed up by exploratory RAB drilling. Positive results from the RAB holes resulted in a series of DD phases.

The southern portion of the Massawa target is within the original Kanoumering Permit, which was taken over from AngloGold Ashanti in 2002. This area had been subject to a considerable amount of exploration work involving reconnaissance and detailed soil sampling and mapping, airborne survey and drilling over selected targets, but no drilling over the Massawa anomaly. Table 9-1 summarises the work completed by Randgold to date by exploration campaign.

Table 9-1 Summary of Exploration at Massawa CZ and NZ

Year	Work Programme
2004	• Regional soil programme returned anomalous results.
2005	• Detailed soil sampling 200 m by 50 m returned a 3.5 km long, 0.1 km to 0.4 km wide anomaly at >50 ppb Au.
2006	• Field validation of soil anomalies and rock chip sampling. The first trench MWTR001 returned 10.9 m at 2.03 g/t Au.
2007	• Detailed geological mapping and litho-sampling over entire Massawa target. • RAB Phase 1: 400 m spaced lines, 95 holes for 3,291 m confirmed bedrock mineralisation over 2.8 km. • Second phase of trenching (three trenches completed). • DD Phase 1: 400 m spaced lines, seven holes (MWDDH001 to MWDDH 007) for 1,645 m. Results included: MWDDH002 – 12.16 m at 2.93 g/t Au from 135.94 m; MWDDH003 – 9.3 m at 3.79 g/t Au from 78 m, 5.96 m at 5.08 g/t Au from 94.04 m; MWDDH006 – 16.13 m at 4.15 g/t Au from 132.45 m; MWDDH007- 10.75 m at 9.08 g/t Au from 159 m.
2008	• Extended detailed soil grid to south and north, mapping and litho-sampling: o Southern extension: 379 soil samples. o Northern extension: 293 soil samples extended the soil anomaly from 3.5 km to 6.2 km long. • Infill RAB: line spacing closed to 200 m, 65 holes for 2,399 m (MWRAB096 to MWRAB160). • Southern extension: 146 holes for 5,175 m (MWRAB160 to MWRAB306). • Northern extension: 67 holes for 2,583 m (MWRAB307 to MWRAB373).

Year	Work Programme
	• DD Phase 2: 36 holes (MWDDH008 to 043) for 6,395 m, completed over 7.7 km comprising deep and shallow drill holes.
2009	• DD Phase 3: 51 holes (MWDDH044 to MWDDH094) for 9,844 m. 100 m by 50 m DDH spacing and exploration drilling to the north, update of Mineral Resource estimates using MWDDH001 to MWDDH081. • DD: 193 holes for 48,624 m at 50 m by 50 m spacing completed into two phases: o Phase 1 testing the geological continuity. o Phase 2 investigating at deeper mineralisation from first phase and comprise both exploration and first geotechnical and density drill holes. • RC drilling Phase 1: 84 holes for 6,272 m (MWRC001 to MWRC090) testing oxide material at shallower depth.
2010	• Scoping Study: Deep DD Phase 1: Eight holes (MWDHH452 to MWDDH455 and MWDDH459 to MWDDH462) for 6,099 m at 300 m by 700 m spacing, testing underground potential. • Scoping Study: Step-out drilling includes four holes for 1,605 m testing the mineralisation continuity along strike in north of and at Massawa South. • Scoping Study: Infill deep DD Phase 2: Six holes totalling 3,615 m to confirm Phase 1 drill programme. • Scoping Study: Twin DD drilling: Six DD twin holes for 1,369 m, to test the mineralisation variability in the CZ. • Scoping Study: RC drilling Phase 2: 104 holes for 7,169 m subdivided in two phases. o 78 holes (5,096 m) to confirm mineralisation continuity at surface. o 26 holes (2,073 m) twinning DH with poor recovery in oxide material.
2013	• Pre-feasibility: RC grade control (GC) drilling: 640 holes for 34,542 m at 5 m by 5 m spacing, testing continuity of high-grade quartz-stibnite-Au veins in the CZ. • Pre-feasibility: PQ (85 mm) DD twin drilling: Five representative RCGC holes in the CZ were twinned by five PQ holes (MWDDH510 to MWDDH514) to investigate sample size bias. • Pre-feasibility RC: RC GC drilling - 217 holes totalling 8,370 m in NZ2 (10 m by 10 m spacing).
2014	• A CZ trenching programme aimed to better understand the orientation of the high-grade quartz-stibnite-Au veins and improve geological model across the mineralisation. 16 trenches totalling 1,131 m. • CZ Twin Holes: Six RC holes were twinned with six DD holes to investigate sample size bias and assay techniques (fire assay vs LeachWELL vs screen fire).
2015	• Pre-feasibility RC drilling in CZ: Drill orientation drilling (10 m by 15 m spacing) over 60 m strike of CZ and down to a vertical depth of 100 m, to determine optimum drill spacing for drill out of the CZ. • NZ2 Sulphide RC Drilling: 14 holes for 1,495 m drilled over two fences at 10 m by 15 m spacing to test down-dip continuity of high-grade mineralisation. • CZ Remodel: Remodel of CZ separating phase 1 and phase 2 domains.
2016	• Pre-feasibility RC GC Drilling in the CZ: Four GC grids (Blocks A to D) were completed along the strike of the CZ. Grid spacing of 10 m by 15 m, testing mineralisation down to a vertical depth of 100 m, and included 128 holes for 11,583 m. Aims of the programme: o To test the 2015 geological model. o Provide material for metallurgical and geochemical testwork. • Trench Programme: 10 trenches (817.5 m) excavated across the strike length of the deposit to trace the surface expression of the mineralisation and to test the revised structural model. • NZ Oxide RC Drilling: 61 holes for 2,619 m across the NZ to test oxide mineralisation in zones not tested by trenches (>4 m thick laterite cover).
2017	• CZ Pre-feasibility: Trench programme at CZ: Five trenches (290.75 m) excavated at CZ across the strike length of the deposit to trace the surface expression of the mineralisation and to test the revised structural model. • CZ Pre-feasibility: RC drilling at CZ: 1 GC grid (Block G); 86 holes completed for 8,648 m and spacing 15 m by 10 m. • NZ Pre-feasibility: RC drilling at NZ1: Two GC grids (Block F and H) were completed along the strike of NZ1. Grid spacing of 15 m by 10 m and included 138 holes for 14,706 m. • CZ Feasibility: RC drilling at CZ: Four GC grid for pilot plant (PP1, PP2, PP3, PP4) completed along the strike of CZ within $1,000 pit shell. Grid spacing of 15 m x 10 m, testing mineralisation down to a vertical depth of 100 m. The aims were: • To test the geological and mineralisation model. • Provide material for metallurgical and geochemical testwork. • This programme included 293 holes for 40,303 m.

Year	Work Programme
	• Trench Programme at NZ1: one trench excavated along the strike at Block H, NZ1 to better understand the control and orientations of the high grade intersected during the RC drilling at Block H. • NZ Pre-feasibility: RC GC drilling at NZ2: 64 holes completed totalling 7,224 m at 10 m by 5 m spacing for two blocks (Block I and Block J), testing continuity of high grade and mineralisation variability in NZ2. • NZ Pre-feasibility: Phase 2 metallurgical test DD holes at NZ2: six holes completed for 942 m. • CZ Feasibility: DDH Phase 1 Camp 20: Metallurgical test + Twins Holes: six DDH completed for 942 m over block G. • NZ Pre-feasibility: Camp 27-29: Two DD holes completed totalling 111 m for metallurgical test work. • CZ Feasibility: RC Drilling at CZ: Two GC grids (Block K, Block L) completed along the strike of CZ within $1000 pit shell. Grid spacing of 15 m by 10 m, to a vertical depth of 100 m, and included 59 holes for 7,735 m.
2018	• CZ Feasibility: RC GC drilling: 749 holes totalling 75,538 m completed along the strike of CZ within $1,000 pit shell with different grid spacing from south to the north: 30 m x 10 m; 15 m by 10 m; 20 m by 10 m; 30 m by 20 m. • The aims were: • To test and confirm the geology and ore model. • Provide material for metallurgical and geochemical test work. • CZ Feasibility: DD Phase 2 Camp 20: Metallurgical test: four DD holes completed for 679 m over Block A, Block C, and PP3. • Trench Programme at CZ: Three trenches excavated at CZ across the strike length of the deposit to trace the surface expression of the mineralisation and to confirm the revised model. • CZ Feasibility: RC GC drilling: CZFW Phase 1 drilling to test the potential in the eastern part (FW mineralised zones); 25 holes spacing 45 m by 10 m for a total depth of 1,789 m. • CZ Feasibility: RC GC drilling: CZFW Phase 2 drilling to confirm the potential in the eastern part (FW mineralised zones); 50 holes spacing 15 m by 10 m over 240 m strike for a total depth of 2,986 m. • Scoping Study: RC Deep Drilling Programme over CZ: To test potential (over 8 g/t Au) at 20 m RL, beyond the $1,000 pit shell. seven holes completed for 2,017.5 m • Sterilisation: Waste Dump: Air core programme: 25 holes for 757 m completed to sterilise the eastern part of the $1,000 pit shell.

9.2. Sofia

Other than the regional soil geochemistry from AngloGold Ashanti, all of the work at Sofia has been completed by Randgold over several years of exploration, as summarised in Table 9-2. Exploration between 2004 and 2015 is described for both Sofia Main and Sofia North.

Table 9-2 Summary of Exploration at Sofia Main and North

Year	Work Programme
2004	• A detailed soil survey programme was undertaken by Randgold resulting in anomalous zones defined by background of 30 ppb Au. • A total of 25 pits and eight trenches were completed over this target with encouraging results.
2005	• Results of the five trenches and eight DDH performed over the target at a 400 m space lines outlined a low-grade heterogeneous mineralisation varying between 44 m at 2.0 g/t Au from 51 m in SFDDH002 and 5 m at 0.4 g/t Au from 64.42 m in SFDDH007. • Two phases of DD were completed during this field campaign of five holes each (SFDDH001 to SFDDH005 totalling 1,007.4 m and SFDDH006 to SFDDH008 totalling 895.5 m).
2006	• During the Phase 2 DD programme, a total of three DDH under trenches were completed: SFDDH006 to SFT014, SFDDH007 to SFT011, and SFDDH008 to SFT013.
2010	• An intensive RC exploration drilling programme with 4,900 m drilled at 100 m by 200 m drill spacing. Drilling included infill lines and drilling underneath previous RAB holes or trenches.

Year	Work Programme
Sofia Main	
2015	• Sofia Main RC drill programme: Five holes totalling 884 m, targeting the high-grade mineralisation in sulphide material. The high-grade zone was extended to a 350 m strike length at Sofia Main. • Sofia Main Trenching Programme: Aimed to express the mineralisation at surface and to understand structural control of the high-grade gold. A total of four trenches for 561.7 m were excavated. • Sofia Main RC drilling: Eight holes drilled to extend the high-grade panel at 80 RL up to 650 m strike.
2016	• Trenching Programme: Aimed to better express the mineralisation at surface and to understand structural control of the high-grade gold. A total of 30 trenches, at 50 m by 50 m spacing, for 2,529.25 m were excavated. • Seven DDH to test the high-grade mineralisation at depth (0 RL). No success and model review for additional drilling. • Exploration DD: 18 holes (SFDDH016 to SFDDH033) totalling 4,500 m to investigate revised geological model of flat high-grade mineralisation at 80 RL. • Infill RC drilling: 53 holes for 8,062 m at 40 m by 40 m spacing for Indicated Mineral Resource estimation.
2017	• Exploration DD: Eight holes (SFDDH034 to SFDDH041) totalling 2,191 m to test high-grade mineralisation at depth and extension. • Step-out and Infill RC Drilling: 25 holes totalling 3,508 m divided into step-out drilling at 40 m by 40 m spacing for testing the extension north and south of the Indicated Mineral Resource and 20 m by 20 m infill drilling (three holes for 1,369 m). • Metallurgical Programme: 19 DDH (SFMET005 to SFMET023) totalling 1,952 m. Provide material for metallurgical testwork. • Re-drilling RC holes: Seven holes for 1,016 m after survey programme. • Resource and Infill DD: Seven DDH (SFDDH042 to SFDDH047 and SFDDH051) totalling 1,538 m. • Provide material for metallurgical testwork: One DD hole for 65 m. • Provide material for metallurgical testwork: One DD hole for 40 m.
2018	• Provide material for metallurgical test work: One DD hole for 45 m. • Resource Drilling: One DD hole for 130 m.
Sofia North	
2015	• Trenching Programme: Aimed to express the mineralisation at surface of the north-south branch of the Sofia structure. A total of two trenches for 385.08 m were excavated.
2016	• Trenching Programme: Aimed to test the continuity of the mineralisation at surface. A total of three trenches for 153.8 m were excavated.
2017	• Infill Trenching Programme: Aimed to express the continuity of the mineralisation at surface and to understand structural control. A total of eight trenches for 667.5 m were excavated. • RC drilling: 95 holes totalling 1,148 m comprising: o A grid spacing of 30 m by 30 m infill drilling designed over 600 m on the +2 g/t Au to 5 g/t Au area, testing mineralisation down to a vertical depth of 100 m, and included 50 holes totalling 6,508 m for potential conversion to Indicated Mineral Resources. o Step-out drilling at the northern and southern parts of 30 m by 30 m grid to test continuity of the mineralisation with 45 holes for 5,360 m. • DD Programme: Six holes totalling 1,154 m comprising: o Two twin holes (SFDDH048 and SFDDH049) for 310 m to characterize the mineralisation and provide material for metallurgical testwork; o One exploration hole for 230 m to test the mineralisation at depth; o Three resource holes for 614 m. • Infill RC Drilling: two holes for 210 m for testing mineralisation under SFTR021. • Metallurgical Programme: eight holes totalling 515.5 m to provide material for metallurgical testwork.
2018	• Fourteen RC holes for 1,048 m completed targeting potential additional indicated resources. Over 2 g/t Au mineralisation confirmed within the 600 m Indicated Resource area. Mineralisation confirmed in the north for 3 koz potential. Lower grade than expected intersected in the south.

9.3. Delya

Delya is located south of the Mandinka target and was generated from Kounemba Permit regional soil survey in early 2004. Delya target was characterised by +100 ppb Au soil anomalism over

4 km by 400 m grid underlain by strongly silicified quartzite with a fair amount of pyrite occurrences. First pass of litho-sampling have been carried out over these ridges and a detailed soil grid was implemented. Exploration carried out at Delya is summarised in Table 9-3.

Table 9-3 Summary of Exploration at Delya

Year	Work Programme
2004	• Kounemba Regional Soil Programme: Discovery of the Delya soil anomaly, anomalous points >100 ppb Au on regional lines (100 m by 100 m) over 4 km
2005	• Detailed soil sampling 200 m by 50 m. Defined a +1.8 km long, 0.1 km wide anomaly at >100 ppb Au trending N030°.
2006	• Detailed mapping and litho-sampling. • Pitting (26) and first trenching (2) completed. • Follow-up trenching (eight trenches) delineated two parallel bedrock mineralised zones over at least 1,000 m. • Additional detail soil grid to south with detailed mapping and litho-sampling extended the soil anomaly from 3 km to 6 km long. • DD Phase1: Five holes (DLDDH001 to DLDDH 005) completed at 100 m to 200 m spacing for 966.84 m over 1 km strike length. Results included: DLDH001 - 9.83 m at 1.80 g/t Au, DLDH002 - 12.44 m at 5.07 g/t Au, including 7.00 m at 8.19 g/t Au; DLDH003 - 3.00 m at 1.80 g/t Au, and DLDH004 - 3.8 m at 4.80 g/t Au. • First trench (DLTR011) testing strong soil anomalous values to the south.
2007	• RAB Phase 1: Two heel to toe RAB lines, 35 holes (DLRAB001 to DLRAB035) for 959 m allowed extending to the north and south the above-mentioned 1 km zone of mineralisation within the + 6 km soil anomaly. • DD Phase 2: DLDDH006 completed underneath the DLRAB030 (21 m at 4.87 g/t Au) to the southern extension. • RAB Phase 2: Four heel to toe RAB lines at 400 m to 600 m spacing, 35 holes (DLRAB36 to DLRAB070) for 1,401 m completed to test the southern soil anomaly.
2010	• Phase 1 Exploration RC Drilling Programme: 32 holes (DLRC001 to DLRC032) completed at 100 m drill spacing for 2,671 m.
2017	• Phase 2 Infill RC Drilling Programme: 27 holes (DLRC033 to DLRC059) drilled at 50 m by 50 m spacing for 2,822 m over a 1 km strike length of the main Delya system to understand the potential and increase the resolution of the geologic model. • Twin Holes: One RC hole DLRC013 was twinned with one DDH (DLDDH007) with the aim to ascertain the geology and the controls of the high-grade mineralisation. • Metallurgical Drilling: One DD hole (DLDDH008) drilled for bacterial oxidation batch amenability tests (BIOX BAT). • Phase 1 Scout Step-Out RC Drilling Programme: Aimed at testing southern strike extension of the Delya Main mineralised system. o 17 RC holes (DLRC059 to DLRC090) for 2,011 m completed at Delya South for strike extension of the Delya Main system. • Shallow Oxide DD Programme: Ten shallow DDH for 418.5 m at 100 m spacing completed in the main zone for oxide density measurements. • RC Conversion Drilling Programme: 57 holes (DLRC091 to DLRC148) drilled at 25 m by 20 m spacing for 4,714 m over a 1 km strike length for Indicated Mineral Resource estimation. • Delya Main Trenching Programme: aimed to confirm both the grade and geometry of the geology at surface of the MOZ and HWOZ mineralisation. A total of 22 trenches for 917.10 m were excavated at 50 m spacing.
2018	• Phased Step-out Trenching Programme: Aimed at testing strike extension of the Delya Main mineralised system toward the south and north: o Delya South: 13 trenches for 928.10 m at 100 m spacing brought the surface resolution of the Delya shear extension to 100 m spacing over an 800 m strike length. o Delya North: 12 trenches for 897.8 m at 100 m to 200 m spacing over 1.4 km long soil anomaly +50 ppb Au. Trenching confirmed the northern continuity of the Delya HW lode and defined the locality of the sub-parallel NE striking main Delya shear located further east.

9.4. Tina

Tina is located in the Bakan Corridor located NW of Massawa CZ. It has an Inferred Mineral Resource of 50 koz grading 1.1 g/t Au. Exploration carried out at Tina is summarised in Table 9-4.

Table 9-4 Summary of Exploration at Tina

Year	Work Programme
2007	• Detailed geological mapping. • Detailed soil sampling with 1,219 samples collected. • Extensive litho-sampling and follow-up trenches (TNTR001, TNTR002, TNTR003, and TNTR004) for 336.7 m revealed mineralisation associated with deformed (semi-ductile to brittle) and altered felsic intrusive: TNTR002: 32 m at 1.15 g/t Au, and TNTR003: 32 m at 1.22 g/t Au. • Additional three trenches (TNTR005, TNTR006, and TNTR007) completed for 235 m to identify the shape of the mineralised felsic intrusive defined from litho-sampling and previous trenching.
2008	• Detailed soil sampling extension of the Tina soil anomaly to the south: 265 samples collected.
2011	• Exploration RC Drilling: Eighteen RC holes (TNRC001 to TNRC018) for 1,470 m completed over a 700 m strike length at 100 m to 200 m spacing to test underneath a shear hosted gold mineralisation associated with a felsic intrusive.
2018	• Thirty-four grooves for 534.2 m testing over 275 m NE strike of outcropping mineralisation and potential eastern branch. Results confirm NNE orientation. • Litho-sampling and mapping confirm geological setting. • Eight RC holes for 1,204 m completed testing over a 275 m NE potential. Results include: 52 m at 2.23 g/t Au from 68 m (TNRC023A) and 43 m at 2.02 g/t Au from 84 m (TNRC020). Confirms NNE orientation. • Two trenches for 300 m completed to gain structural measurements and observations to the geological setting to mineralisation. NNE and NNW orientations to mineralisation observed – controls relating to vein intensity, disseminated sulphides, proximity, and location of granodiorite intrusives.

9.5. Bambaraya

The Bambaraya deposit is located in the NW corner of the Kounemba Permit proximal to the north-south Sabodala Shear Corridor approximately 18 km to the north of Sofia deposit. Exploration carried out at Delya is summarised in Table 9-5.

Table 9-5 Summary of Exploration at Bambaraya

Year	Work Programme
2004	• Detailed soil sampling on a 200 m by 50 m grid (837 samples) defined three anomalous areas (> 100 ppb Au) zone east, zone south, and zone west (QT zone). • Three historical trenches (BBTR001, BBTR002, and BBTR003) for 64 m previously dug in the early 1970s across the vein-ridge, were re-opened, deepened, and re-sampled. • DD: Five holes (BBDDH001 to BBDDH005) for 1,001 m drilled across the +1,800 m long +100 ppb Au soil anomaly and anomalous trenches.
2005	• A fresh detailed mapping and target assessment: Selective sampling carried out over old trenches aimed to differentiate gold distribution from ferruginous and fresh QT. • Three new trenches: BBTR004 to BBTR006 for 282.2 m completed throughout the target with the aim to assist in geological mapping and assess QT and grade continuity. • Detailed soil: extended to the south of the target with 129 samples collected. • Two new trenches: BBTR007 and BBTR008 completed for 199.5 m each: BBTR007 100 m north of Trench BBT004 (6 m at 1.76 g/t Au, 4 m at 5.48 g/t Au, and 12 m at 4.06 g/t Au) and BBTR008 100 m south of Trench BBT006.

Year	Work Programme
2006	• Trenching: BBTR009 for a 150 m length. • Pitting: 14 pits for 28.5 m. • Infill Trenching: Four trenches (BBTR010, BBTR011, BBTR012, BBTR013) for 542 m aimed at accurately defining the geometry of the different mineralisation sets and investigating for lateral extension of the mineralisation. • Validation of anomalous soils undertaken in the NE part of the grid. • Intensive litho-sampling from the few outcrops confirms the latest values in a N140° massive quartz vein (250 m by 5 m) with anomalous values ranging from 0.69 g/t Au up to 105 g/t Au. Further to the north, a stream bed saprolite of felsic intrusive with N130° quartz veins assayed 38.6 g/t Au.
2007	• Two trenches (BBNTR001 and BBNTR002) for 99 m to the NE of the target completed over a N130° oriented and mineralised felsic intrusive and over N350° quartz vein bearing visible gold. • Four DDH (BBDDH006 to BBDDH009) totalling 761.93 m were completed, aiming to investigate the defined three to four sub-parallel zones of mineralisation at surface, over a strike length of 800 m.
2010	• Seven RC drill holes (BBRC003 to BBRC005, BBRC007 to BBRC010) for 588 m were drilled under or between existing DD holes in an attempt to confirm the main mineralised zone.

9.6. 2019 Planned Exploration

The main exploration objective at Massawa is to increase Ore Reserves to greater than 3 Moz of gold. For this purpose, the potential exploration discovery and definition of an additional 600 koz of Measured and Indicated Mineral Resources are required.

For 2019, the priority exploration target areas are KB, Samina (Delya strike extensions), Tina, and Tiwana South (Bakan Corridor). Each of these target areas are associated with anomalous soils and litho-samples and, apart from Samina, are located on transfer and/or second-order ENE, NE, and NNW structures. RC and Air Core drilling in combination with trenching and additional fieldwork is planned for 2019 to further define potential at these priority target areas.

10. Drilling

10.1. Methods and Procedures

All soil sampling, mapping, trenching, and geological supervision of drilling has been conducted by Randgold geologists. Table 10-1 details the DD, RC, and trenches undertaken on the property up until November 2018.

Table 10-1 Massawa Project Drilling and Trenching Summary

Zone	DD Drilling		RC Drilling		Trenches	
	No	Metres	No	Metres	No	Metres
Massawa CZ	198	43,537	2,205	199,308	22	2,133
Massawa NZ1	85	18,093	139	14,842	8	751
Massawa NZ2	169	37,936	383	22,655	5	208
Sofia Main	44	11,049	126	17,308	49	4,626
Sofia North	11	2,354	141	15,838	19	1,711
Delya Main	21	2,205	111	9,589	28	1,414
Bambaraya Deposit	9	1,766	7	588	11	1,008
Tina Deposit	-	-	18	1,470	5	447

10.2. Survey Grids

The Massawa Mine uses the UTM Zone 28N datum WGS84 grid for drill hole coordinates. At Delya, UTM Zone 29N datum WGS84 is converted into the same grid used at Massawa (UTM Zone 28N datum WGS84).

10.3. Drill Planning and Site Preparation

Drill holes are principally planned to intersect the mineralisation perpendicular to the main body of mineralisation. A few early drill holes were orientated to follow the main azimuth of the mineralisation trend to test the continuity of the mineralisation. Most of the holes are drilled on a 50º to 60º dip angle to cut the steeply dipping mineralised lenses at a high angle.

Initially, the planned collar location is marked by a surveyor using a differential global positioning system (DGPS). An appropriate size drill pad is cleared around the collar marker to ensure sufficient room for the drill rig, auxiliary vehicles, and sample collection. If the drill hole is to be completed using DD, then a sump is dug in one corner of the pad to collect the drilling returns. The sump is fenced off with a temporary barrier and security tape to help prevent accidents. Upon completion of the drilling, the sumps are backfilled and all efforts are made to remediate the drill site as far as possible whilst retaining the collar concrete pad and polyvinyl chloride (PVC) pipe casing.

10.4. Downhole Surveying

Since 2016, all drill holes have been surveyed with Reflex EZ-GYRO multi shot system with dip and azimuth measurements taken every 30 m down hole during drilling and then re-surveyed every 10 m down hole upon completion of the drill hole. All surveying is completed by the drilling

contractor. If there is more than 3.0º change in azimuth since the previous sample, a multi shot EZ-GYRO survey is undertaken every six or nine metres which overlaps with at least two 'good' single survey measurements.

The Reflex EZ-GYRO units on site are returned to the manufacturer for calibration on an annual basis. The tools are also checked once per week on the known survey tool test stand that has been erected within the camp (Figure 10-1).

Figure 10-1 Survey Tool Test Stand at Massawa Camp

Once the surveys have been taken, they are stored on a handheld Panasonic controller which connects to the Reflex Hub Management software that uploads the surveys directly to the Datashed database, thereby removing any potential human errors during transcription.

Prior to 2016, drill holes were measured with Reflex EZ-TRAC or Reflex conventional GYRO and manually transcribed to the database.

10.5. Collar Surveys

Once a drill hole or surface trench is completed, and all machinery has moved away from site, the drill hole is surveyed using a DGPS by Randgold surveyors. The drill collar is plugged and a concrete collar surround is placed which is labelled with the Hole ID and date of survey. The sumps are then backfilled and the drill site is remediated. Trenches are surveyed with a DGPS starting at the collar (initial cut) and then additional point measurements are taken every 2 m.

10.6. Diamond Drilling

All DD has been conducted by Boart Longyear, Canada. DD is utilised at Massawa for exploration and resource development. PQ rods (85.0 mm diameter) and HQ rods (63.3 mm diameter) are used in the weathered saprolite (oxides) with NQ (47.6 mm diameter) being utilised after the

contact with fresh consolidated rock. The quality of DD is considered to be of industry accepted standard with average core recoveries of 84% in the saprolite, 94% in the transition, and >96% in the fresh material.

10.6.1. Diamond Drilling Procedure

A Massawa field geologist must be on site prior to drilling commencing and ensure that the drill rig is lined up as per the drill plan, as well as supervising drilling, core orientation, and down hole surveying. Once each drilling run is complete, the drill core is removed from the drill rod and placed in an angle iron rack to mark up an orientation line with red chinagraph pencil or crayon from the data received from a Reflex ACT II Core Orientation Tool (ACT). The apex of the structure is marked on the core in a blue chinagraph pencil or crayon by the core technician. If the orientation and apex lines are overlapping, then the apex line is offset by 5 mm.

DD core is transferred to the core trays and a plastic down hole depth marker is placed at the beginning and end of each core run with the depth marked on it. All areas of core loss are identified, and the core is marked up for core recovery. Each drill core box is marked with the Hole ID, top and bottom depth of the core, and the box number. The core is then transferred to the core yard facility by Randgold staff for logging and sampling.

10.6.2. Diamond Core Logging

From the commencement of the feasibility programme in 2017, all drill holes have been logged digitally into Maxwell LogChief software running on rugged logging tablets. Prior to this, logging was completed on paper logs which were transcribed into the database at a later time. The drill core is logged for alteration, structure, mineralisation, and geology in detail by a trained geologist before being photographed. All structural data including alpha and beta angles of any structures are also recorded at this stage.

10.7. Reverse Circulation Drilling

RC drilling is used at Massawa for GC spaced drilling, as well as exploration/resource development. All RC drilling has been conducted by Boart Longyear, Foraco, IDC, and FTE. RC drill holes to date have been drilled at a 5.5 in./140 mm diameter which produces approximately 25 kg of material per one-metre sample interval. The RC drill rods are 6 m in length.

10.7.1. Reverse Circulation Drilling Procedures

A Massawa geologist is present during RC drilling. Prior to commencement of any drilling, cross sections for all planned holes are printed indicating the expected geology and mineralisation to be intercepted. The geologist lays out the site to ensure that, where possible, the drilling and sampling operations will not interfere with each other and that the sampling is not taking place down wind of the drilling.

A 6 m PVC casing is used to collar the hole to help prevent drill hole collapse and sample contamination. Once the drill hole has been collared, the geologist ensures that the drill hole is

cleaned to remove any material that may have ingressed during collaring before drilling is resumed.

If the drill hole intersects the water table, an auxiliary booster is utilised to ensure that the samples are dry. After each rod change, air is blown down the hole prior to recommencing drilling to dry it out. If the samples are consistently wet over more than 3 m of drilling within the fresh rock and the driller cannot increase the air pressure any further to dry out the samples, then the drill hole is terminated and the samples are not analysed.

10.7.2. Reverse Circulation Sampling and Splitting Procedure

In soft saprolite or transition material, the cyclone splitter is removed and the entire one metre (approximate weight of 25 kg) sample is collected in the dropbox and inspected by the geologist on site to ensure that it is dry. The dropbox is cleaned at the end of each drill string. The entire one-metre sample within the dropbox is transferred to the splitter.

If the sample is moist or damp, the bulk sample is weighed and recorded as a ‘wet sample weight’. The moist or damp samples are transported to the sampling area and dried. Once dry the sample is re-weighed and the dry weight recorded in addition to the previously recorded wet weight.

Prior to Q1 2018, the dry soft saprolite or transitional sample was poured evenly through the conventional 3:1 riffle splitter twice to obtain a minimum of 2 kg sample (up to a maximum acceptable sample weight of 3 kg). If the sample was underweight (below 2 kg), additional material is added by passing the total rejects back into the splitter until the desired sample weight is collected. The riffle splitter was cleaned with compressed air hose after every single use. In hard transitional or fresh rock drilling, the on-board cyclone and rotary splitter was used to collect a 2 kg sample and a reject sample in separate bags. The cyclone splitter was cleaned with a high-pressure air hose after every two drill rods or whenever material is observed to be sticking to the internal walls. If the RC samples were not entirely dry, the cyclone and rotary splitter were disengaged and the dropbox method used instead.

Since Q1 2018, the samples are taken directly from the dropbox and a 12.5 kg split sample is produced from the initial 25 kg RC sample through the primary Gilson splitter. This 12.5 kg samples are then passed through the Gilson splitter again (except for CZ where a secondary Gilson splitter is used due to the higher nuggety nature of the mineralisation) to produce a 3 kg to 4 kg sample which can be submitted for analysis. If a duplicate field sample is required, this process is then repeated with the reject material.

After splitting the sample, both the primary sample and rejects are weighed and recorded. A small amount of material is taken from the sample rejects and placed in a slotted plastic rock chip sample tray, which are pre-labelled with the drill hole ID and sequence numbered drill hole intervals in each sample slot. RC sample recovery is measured by weighing the total weight of sample collected over one-metre sample and comparing this to the theoretical expected weight for the lithological unit and weathering type.

Once the primary samples have been taken, the remaining reject sample is riffle split to create a 3 kg sample for long term sample storage for any potential future analysis and the remainder is disposed of.

The RC samples are believed to be representative of mineralisation present at Massawa and Sofia. The quality of RC drilling is considered to be of industry accepted standard, with average sample recoveries of 92% in the saprolite, 94% in the transition, and 96% in the fresh material.

10.7.3. Reverse Circulation Logging

All RC samples are logged on one-metre intervals as per the sample length received from the RC rig. Since commencement of the feasibility programme in 2017, all drill holes are logged digitally into LogChief software running on rugged logging tablets. Prior to this, the logging was completed on paper logs which were transcribed into the database at a later time. For each drill hole lithology, visible mineralisation, vein intensity, alteration, oxidisation, and depth of water table are logged as a minimum.

10.8. Trenching

Most the trenches were excavated perpendicular to strike of the known mineralisation, with two exceptions in the CZ which were cut along the mineralised zone to test the continuity of the proximal domain mineralisation along strike. The trenches are excavated through all pedolith material until at least 50 cm of insitu saprolite is exposed at the base of the trench. All trenches were geologically mapped in detail by a trained geologist on both walls. As part of the mapping, the orientation, distribution, and mineralogy of any visible veins was recorded, to better understand the formation of the late gold stibnite phase mineralisation. Any veins which contained visible sulphide phases were selectively sampled.

The standard sampling procedure was such that a shallow cut, approximately 30 cm wide channel (equivalent sample mass to an RC sample) is excavated at the bottom of the northern face of the trench and sampled on geological and mineralisation contacts. The dimensions of the channel are kept constant through the entire sample length (Figure 10-2).

The sample is split on site using a riffle splitter and a 3 kg representative sample placed in cloth sample bags with a sample ticket and sent to the laboratory. Additional RC mass equivalent channel samples were taken from both the north and south walls of the trenches to quantify the closely spaced grade variability. The results show that there is as much as a 40% variance in grade within the 1.0 m to 1.5 m along strike range. This high level of close space variability in the CZ is directly attributed to the occurrence of coarse gold as part of the proximal domain mineralisation.

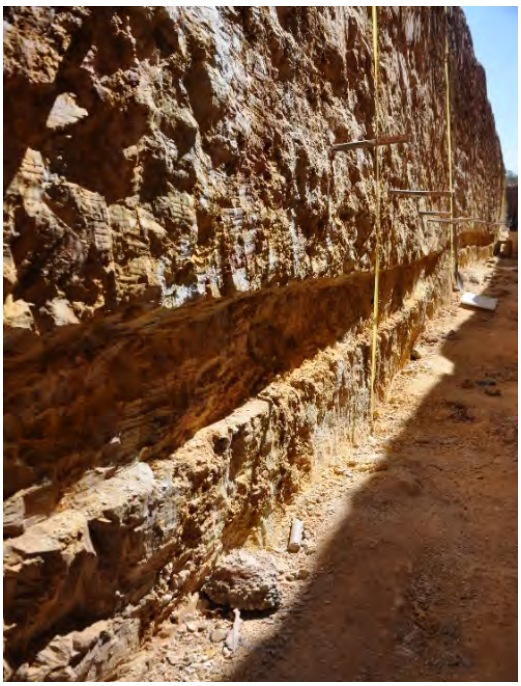


Figure 10-2 Photograph of RC Channel Sample Taken from Trench in 2014

10.9. Other Sampling Methods

RAB and Air Core drilling has previously been used at Massawa for exploration and sterilisation purposes, however, neither is used for resource estimation purposes.

10.10. Sample Volume Variance Bias

Initial investigations into potential sample volume variance were commenced during 2013 when a review of several twin holes showed a low to moderate level of repeatability within the Massawa CZ. This had not been previously identified on the Project. However, as further field duplicate data was obtained, the CZ assay data correlation decreased. During 2013, as specific sample volume and assay method variability studies were completed on each individual geological zone, it became apparent that the relative correlations in the CZ were much lower than within the other Mineral Resources. The CZ trenching and GC orientation programmes in 2016-2017 highlighted and confirmed the strike length continuity of the high-grade (proximal) nuggety gold systems and subsequent, orientation grids have consistently demonstrated the relative gold distribution between the broad low-grade refractory mineralisation zones and the narrow nugget high-grade (proximal) mineralisation zones.

10.11. Drill Twinning Studies

10.11.1. Massawa Central Zone

Gold deportment studies have identified that the gold mineralisation within the Massawa CZ is spatially associated with low-grade disseminated sulphides and is largely present as native gold inclusions within pyrite and arsenopyrite, as well as lattice bound gold in disseminated arsenopyrite. Conversely, the native gold within high-grade zones (proximal) mineralisation

domains occurs mainly as free gold in quartz and carbonate gangue, and occasionally as grains attached to or marginally embedded in Sb sulphides. Free gold grain sizes range from less than 20 µm and 1.2 mm in length and show a bimodal size distribution with peaks between 10 µm to 30 µm and 50 µm to 70 µm, thus giving the high-grade zones (proximal phase) its nuggety nature.

A series of RC, PQ, HQ, and NQ diamond twin holes were drilled in 2013 to conduct bias tests between equally represented areas of DD and RC drilling within the CZ Mineral Resources (Table 10-2). The primary aim of the drill twin analysis was to investigate any potential smearing of grade due to coarse gold and to determine if there was a volume variance effect related to the sample size (RC vs DDH).

Table 10-2 Massawa Central Zone Twinned Holes

Section	Hole	Core size	From (m)	To (m)	Interval (m)	Au (g/t)	RC Twin Hole	From (m)	To (m)	Interval (m)	Au (g/t)
S67	MWDDH178	HQ	69.00	79.00	10.00	1.16	MWRCGC952	74.00	81.00	7.00	0.43
S67	MWDDH178	HQ/NQ	105.00	127.20	22.20	3.25	MWRCGC952	115.00	132.00	17.00	1.79
S67	MWDDH179	HQ	25.00	29.20	4.20	3.34	MWRC197	27.00	31.00	4.00	4.55
S67	MWDDH179	HQ	40.50	48.50	8.00	2.33	MWRC197	42.00	51.00	9.00	2.78
S67	MWDDH179	HQ	52.70	60.80	8.10	0.68	MWRC197	54.00	62.00	8.00	1.32
S62	MWDDH187	HQ	65.10	73.00	7.90	0.28	MWRCGC924	66.00	73.00	7.00	8.66
S62	MWDDH187	NQ	127.70	169.90	42.20	2.26	MWRCGC924	126.00	156.00	30.00	0.60
S40	MWDDH198	HQ	19.00	36.80	17.80	5.07	MWRCGC1286	20.00	36.00	16.00	1.44
S40	MWDDH198	HQ	43.70	55.20	11.50	2.71	MWRCGC1286	44.00	57.00	13.00	4.12
S40	MWDDH198	HQ/NQ	72.00	87.00	15.00	0.40	MWRCGC1286	75.00	88.00	13.00	0.78
S60	MWDDH375	HQ	2.50	7.70	5.20	1.28	MWRCGC1471	4.00	9.00	5.00	4.30
S60	MWDDH375	HQ	54.00	76.80	22.80	1.11	MWRCGC1471	54.00	78.00	24.00	15.49
S60	MWDDH375	HQ/NQ	102.27	107.00	4.73	7.62	MWRCGC1471	99.00	105.00	6.00	1.40
S19	MWDDH548	HQ	60.50	79.80	19.30	1.92	MWRCGC498	60.00	82.00	22.00	7.83
S24	MWDDH014	HQ	8.70	18.70	10.00	0.90	MWRCGC1141	10.00	20.00	10.00	0.57
S24	MWDDH014	HQ	30.70	41.00	10.30	5.45	MWRCGC1141	32.00	41.00	9.00	10.00
S24	MWDDH014	NQ	86.50	91.25	4.75	0.52	MWRCGC1141	87.00	93.00	6.00	1.19
S24	MWDDH014	NQ	114.60	130.50	15.90	1.04	MWRCGC1141	115.00	132.00	17.00	2.09
S56	MWDDH477	HQ	36.00	58.30	22.30	1.80	MWRCGC1418	42.00	60.00	18.00	2.40
S56	MWDDH477	HQ/NQ	62.35	73.50	11.15	2.16	MWRCGC1418	67.00	81.00	14.00	1.93
S56	MWDDH477	NQ	87.60	100.70	13.10	0.68	MWRCGC1418	84.00	99.00	15.00	0.80
S56	MWDDH477	NQ	121.60	136.50	14.90	1.27	MWRCGC1418	123.00	137.00	14.00	0.85
S63	MWDDH077	HQ	10.50	14.40	3.90	2.08	MWRCGC1496	11.00	17.00	6.00	3.77
S63	MWDDH077	HQ/NQ	73.70	92.60	18.90	0.48	MWRCGC1496	75.00	102.00	27.00	2.09
S63	MWDDH077	NQ	118.40	135.90	17.50	0.60	MWRCGC1496	122.00	138.00	16.00	0.51
S72	MWDDH550	HQ	85.15	103.80	18.65	1.43	MWRCGC369	81.00	105.00	24.00	2.21
S75	MWDDH076	HQ	52.50	72.50	20.00	1.19	MWRCGC1649	53.00	75.00	22.00	2.27
S75	MWDDH076	HQ/NQ	79.40	110.90	31.50	2.04	MWRCGC1649	83.00	100.00	17.00	2.64
S73	MWDDH173	HQ	72.70	79.70	7.00	1.16	MWRCGC394	72.00	78.00	6.00	1.08
S73	MWDDH174	HQ	28.50	34.20	5.70	2.96	MWRCGC393	30.00	36.00	6.00	0.76
S73	MWDDH174	HQ	48.70	67.20	18.50	3.04	MWRCGC393	51.00	70.00	19.00	2.15
S15	MWDDH547	HQ	57.00	99.75	42.75	1.63	MWRCGC665	52.00	100.00	48.00	12.73
S18	MWDDH549	HQ	88.70	102.90	14.20	3.62	MWRCGC529	89.00	108.00	19.00	2.36
S29	MWDDH518	HQ	16.00	28.00	12.00	1.92	MWRCGC231	18.00	32.00	14.00	1.21
S29	MWDDH518	HQ	68.00	74.00	6.00	0.93	MWRCGC231	71.00	80.00	9.00	0.25
S31	MWDDH015	HQ	29.70	44.70	15.00	2.15	MWRCGC287	28.00	55.00	27.00	94.30
S31	MWDDH015	NQ	86.00	91.80	5.80	0.31	MWRCGC287	86.00	94.00	8.00	0.44
S31	MWDDH515	HQ	23.00	59.00	36.00	4.97	MWRCGC226	23.00	56.00	33.00	1.34
S31	MWDDH515	HQ	73.00	80.00	7.00	1.12	MWRCGC226	68.00	77.00	9.00	1.59
S31	MWDDH516	HQ	18.00	25.00	7.00	1.16	MWRCGC227	15.00	27.00	12.00	4.73
S31	MWDDH516	HQ	79.00	85.00	6.00	0.57	MWRCGC227	76.00	80.00	4.00	0.50
S34	MWDDH202	HQ/NQ	1.40	63.00	61.60	0.71	MWRCGC752	3.00	56.00	53.00	1.39
S34	MWDDH202	NQ	84.20	92.00	7.80	0.65	MWRCGC752	83.00	88.00	5.00	27.28
S51	MWDDH211	HQ/NQ	36.50	118.20	81.70	1.83	MWRCGC337	50.00	121.00	71.00	7.17
S54	MWDDH189	HQ	12.50	43.30	29.80	3.65	MWRCGC1833	11.00	48.00	37.00	2.36
S71	MWDDH018	HQ	7.70	21.70	12.30	1.55	MWRCGC1620	16.00	30.00	14.00	2.54

Section	Hole	Core size	From (m)	To (m)	Interval (m)	Au (g/t)
S71	MWDDH018	HQ	41.70	51.60	9.90	0.58
S75	MWDDH047	NQ	113.00	133.00	20.00	1.90
S82	MWDDH019	HQ	19.70	23.70	4.00	1.05
S82	MWDDH019	HQ	40.40	53.45	13.05	6.40
S82	MWDDH019	HQ	63.90	71.70	7.80	1.05
S85	MWDDH167	NQ	109.10	115.10	6.00	1.23
S87	MWDDH075	HQ	52.80	55.20	2.40	3.23
S87	MWDDH075	HQ	60.60	63.20	2.60	3.99

RC Twin Hole	From (m)	To (m)	Interval (m)	Au (g/t)
MWRCGC1620	39.00	53.00	14.00	1.71
MWRCGC1654	96.00	122.00	26.00	1.61
MWRCGC1674	13.00	18.00	5.00	1.27
MWRCGC1674	22.00	38.00	16.00	2.27
MWRCGC1674	57.00	63.00	6.00	1.32
MWRCGC422	99.00	118.00	19.00	5.72
MWRC053	52.00	56.00	4.00	3.26
MWRC053	61.00	64.00	3.00	1.79

The results indicated that, when compared to RC drilling, smaller diameter core such as NQ and HQ core do not provide a fully representative sample that is repeatable, due to the high nugget factor within the high-grade (proximal) zones, regardless of the analysis method selected (fire assay or bulk LeachWELL plus tail fire assay).

A direct comparison of the DD data with 50 g fire assay dataset and RC drill data with bulk LeachWELL plus fire assay tails demonstrates a significant negative skew in the grade distribution of the former, for both the high-grade zones (proximal) and the low-grade zones (distal) domains.

The Massawa CZ 2016 Mineral Resource estimation dataset comprises a combination of DD data with 50 g fire assay determinations and RC drill data with bulk LeachWELL plus tail fire assay determination, even with the known volume variance bias.

There is no significant bias between the mineralised RC wet samples and the rest of the mineralised RC data indicating that the wet samples are not inducing grade smearing in the RC holes. Considering only the RC data, the percentage of the mineralised wet data is 4.1%. A review of all the data (including RC and DDH samples), indicates that the overall percentage of the mineralised wet data is 3.7%.

At the current stage of the Project, the negative bias of the lower volume samples is considered to be an acceptable level of conservatism within the resource estimate and subsequent economic analysis.

The 2017 and 2018 drilling programme within Massawa CZ Mineral Resource has effectively replaced all historic DD data utilising 50 g fire assay dataset with new RC drill data with bulk LeachWELL plus tail fire assay, such that future Mineral Resource updates will exclude all historic DD data with 50 g fire assay determination. Pilot Plant campaigns have been conducted on subsequent RC drill areas with the aim to reconcile the estimate against the Pilot Plant campaigns.

10.11.2. Massawa Northern Zone

NZ deportment testwork has demonstrated that the gold mineralisation is largely refractory to direct cyanide leaching developed within the crystal lattice of arsenopyrite and, to a lesser extent, arsenian pyrite. There is no direct evidence in any work completed to date that could suggest that the NZ mineralisation contains a significant coarse free gold component. This is supported

by the repeatability of sample results within both twin drill holes and field duplicates to date using conventional NQ and HQ DD core sample volumes plus 50 g fire assay analysis.

Drill sample size bias tests between equally represented areas of DD and RC drilling in NZ Mineral Resources have not indicated the presence of any significant bias associated with sample volume differences (Table 10-3), due to significantly lower quantities of coarse visible gold associated with the mineralisation. Additional gold deportment studies will be completed on NZ as part of the feasibility work programme.

Table 10-3 Massawa Northern Zone Drill Twinning

DDH Hole ID	DDH Diameter	DDH Interval		DDH Interval Length (m)	RC Twin ID	RC Interval		RC Interval Length (m)	Zone	DDH Au Fire Assay (g/t)	RC Au Fire Assay (g/t)
		From	To			From	To				
MWDDH149	HQ	64.50	68.40	3.90	MWRC186	67	71	4	NZ1 MOZ	1.16	0.66
MWDDH228	HQ	57.50	80.00	22.50	MWRC182	61	81	20	NZ2 MOZ	3.31	3.38
MWDDH236	HQ	89.00	98.60	9.60	MWRC178	88	96	8	NZ2 MOZ	3.24	2.54
MWDDH239	HQ	87.65	101.70	14.05	MWRC180	90	104	14	NZ2 MOZ	5.58	7.23
MWDDH334	HQ	61.45	70.00	8.55	MWRC177	62	69	7	NZ2 MOZ	17.30	10.04
MWDDH335	HQ	62.00	76.90	14.90	MWRC176	63	87	24	NZ2 MOZ	10.69	7.51
MWDDH336	HQ	49.20	76.15	26.95	MWRC175	52	77	25	NZ2 MOZ	2.19	2.91

10.11.3. Sofia Main and Sofia North

Gold deportment testwork at Sofia has shown that the gold mineralisation is entirely free milling, associated with pyrite sulphide assemblages. Modelled nugget of 0.21 at Sofia Main and 0.22 at Sofia North are good representations of the average variance (0.21) between twin DDH and RC holes (Table 10-4).

Table 10-4 Sofia Main and Sofia North Drill Twinning

DDH Hole ID	DDH Interval		DDH Interval Length (m)	RC Twin ID	RC Interval		RC Interval Length (m)	Zone	DDH Au Fire Assay (g/t)	RC Au Fire Assay (g/t)
	From	To			From	To				
SFDDH013	115.6	131.6	16.00	SFRC063	115	132	17	Sofia Main	8.8	6.94
SFDDH013	133.5	142.15	8.65	SFRC063	132	140	8	Sofia Main	0.94	0.81
SFMET025	50.00	60.00	10.00	SFRC222	51	63	12	Sofia North	1.8	1.3
SFMET025	65.00	74.70	9.70	SFRC222	68	78	10	Sofia North	3.1	2.3
SFDDH049	68.00	76.60	8.60	SFRC197	70	80	10	Sofia North	2.12	2.1
SFMET027	71.00	96.45	25.45	SFRC231	56	80	24	Sofia North	2.78	2.6
SFDDH048	84.80	99.16	14.36	SFRC201	86	103	17	Sofia North	3.92	5.1
SFDDH048	108.9	128.14	19.24	SFRC201	110	129	19	Sofia North	3.33	2.8

10.11.4. Delya Main

Delya Main is located to the northeast of Massawa CZ along the MTZ. Twin hole analysis comparing RC to DD holes highlight an average variance of 20% compared to a modelled nugget of 0.16 (Table 10-5). Samples above cut-off grade show a slight positive bias toward DD.

Table 10-5 Delya Main Drill Twinning

DDH Hole ID	DDH Interval		DDH Interval Length (m)	RC Twin ID	RC Interval		RC Interval Length (m)	Zone	DDH Au Fire Assay (g/t)	RC Au Fire Assay (g/t)
	From	To			From	To				
DLDDH007	50	52	2	DLRC013	49	51	2	Delya Main	1.17	1.31
DLDDH007	63	74	17	DLRC013	63	75	17	Delya Main	4.82	6.2

10.11.5. Twinning Summary

In the QP's opinion the drilling and sampling methodologies are of a suitable quality and the samples are representative of the mineralisation.

It is fully understood that in using the smaller volume, the sample data is creating a negative bias within the Mineral Resource estimate of the Massawa CZ. Both the DD data with 50 g fire assay determinations and RC drill data with bulk sample LeachWELL plus tail assay determination are spatially distributed across the entire of the Massawa CZ domain (Figure 10-3) and as such there is no known spatial bias that is skewing the sample size distribution for each of the sample types.

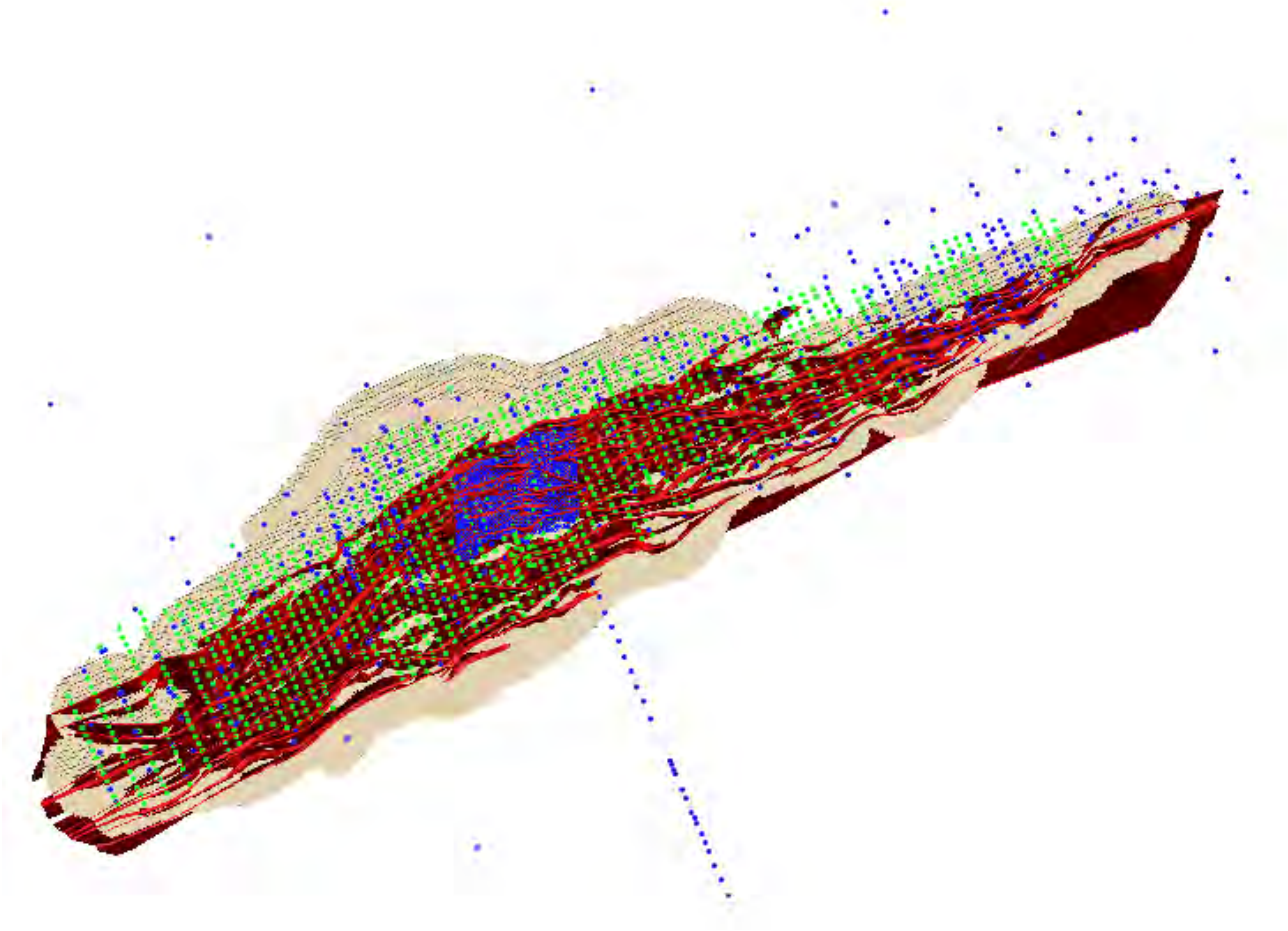


Figure 10-3 Spatial Distribution of 50 g Fire Assay Sample (Blue) and Bulk LeachWELL Plus Tail Fire Assay (Green) with Mineralisation Wireframes (Red) and Pit (Brown)

10.12. Drill Spacing Optimisation

10.12.1. Massawa Grade Control

Due to the high variability of gold grades associated with the multiple vein sets along the strike of Massawa, a close space GC drill programme was undertaken. This drill programme was designed to optimise the GC drill spacing by analysing the potential change in tonnage and grade associated with a change in the density of close space drilling.

A 150 m strike length portion of both CZ and NZ2 was selected for the GC drill programme. The CZ GC zone was drilled to a 5 m by 5 m (X by Y) spacing, whereas the NZ2 GC portion was drilled to a 5 m by 10 m (X by Y) spacing.

10.12.2. Central Zone Grade Control

The GC drilled portion of the CZ was selected where abundant phase 2 (proximal) domain quartz-stibnite veins is present adjacent to a porphyritic intrusion. The NE orientation of the CZ meant that the GC drill holes were drilled along a 320° strike and at -50° dip in order to effectively test 40 vertical meters of mineralisation. This drill grid design also accounted for the general SE dip of the phase 2 (proximal) domain quartz-stibnite veins. A total of 640 drill holes (34,542 m) were drilled along 29 lines. In total, the GC drilling intersected five of the primary phase 2 (proximal) mineralised domains and two sets of discontinuous brittle zones.

Subsequent change of support GC drill spacing tests were completed by progressively removing lines of drilling from the estimation dataset and then creating new mineralisation wireframes for the remaining data and creating a new GC model for the reduced spacing.

All change of support models were generated using the same parameters as definitions as that of the 2016 Massawa CZ Resource model. The results of the GC drilling indicate minimal change in ounces with a 15 m Y by 5 m X spacing for GC and a 5% to 6% reduction in total contained metal at a 15 m Y by 10 m X spacing. All wider drill spacing showed a significant under estimation of Au ounces (Figure 10-4). Subsequently a 15 m Y by 10 m X spacing was selected as an appropriate advanced GC (AdvGC) spacing to delineate the long thin anastomosing nature of the mineralised lodes and complete a reasonable grade estimate.

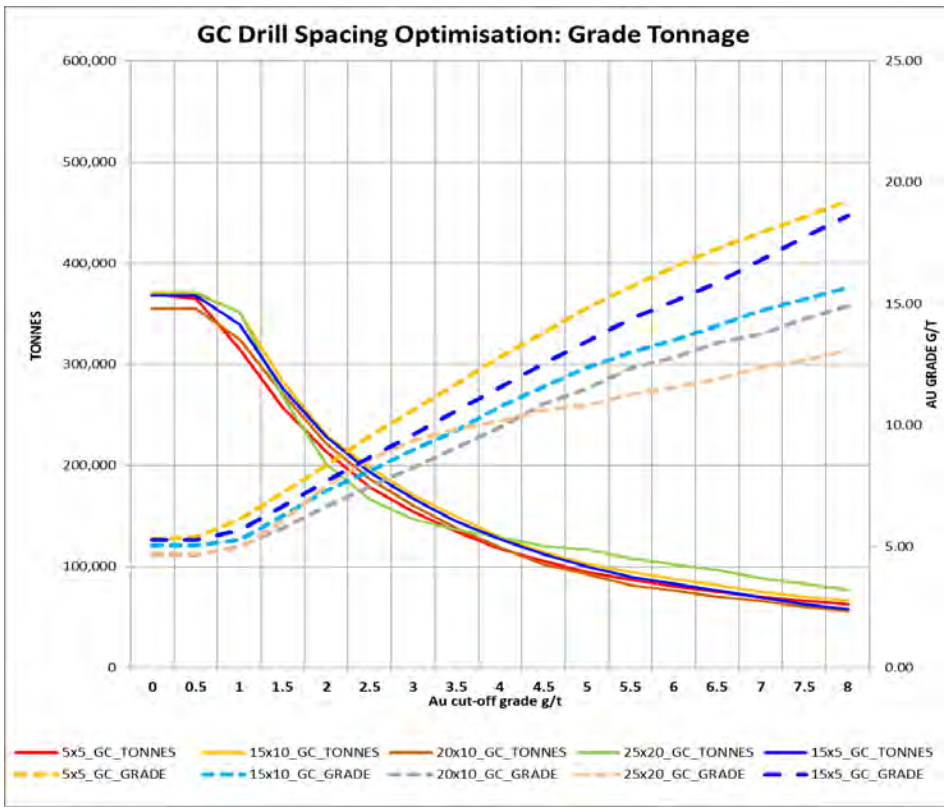


Figure 10-4 Grade Tonnage Curves for CZ1 GC Spacing Optimisation Models

10.12.3. Central Zone Mineral Resource

The majority of the CZ is drilled on 40 m to 50 m sections with a mixture of RC and DD (see Figure 7-5). There are six discrete blocks of RC 15 m by 10 m drilling spread along the strike of the CZ which have been drilled to test the impact of the AdvGC spaced drill definition against the original model, all of which has consistently resulted in a decrease in tonnes and increase in grade with minimal impact on ounces. There is also one large block of 5 m by 5 m RC, which was drilled as part of the drill spacing optimisation.

10.12.4. Northern Zone 2 Grade Control

The GC drilled portion of NZ2 was selected for drill spacing optimisation in order to intersect the known high-grade gold zones. The NE orientation of NZ2 also meant that the GC drill holes were drilled along a 300° strike and at -50° dip to effectively test 40 vertical metres of mineralisation. A total of 217 drill holes (8,370 m) were completed along 16 lines. The drilling intersected both the high-grade domain and the main mineralisation zone, as well as defining some extensions of small discontinuous HW brittle zones of mineralisation.

Subsequent change of support, GC drill spacing tests were completed by progressively removing lines of drilling from the estimation dataset and then creating new mineralisation wireframes for the remaining data and creating a new GC model for the reduced spacing.

All change of support models were generated using the same process parameters and definitions as the 2016 Massawa NZ Resource model. The results of the GC change of support drilling

indicate that 20 m Y by 10 m X spacing resulted in a 2% over estimation of total contained metal relative to the 10 m Y by 5 m X spacing. Subsequently, 20 m Y by 10 m X spacing was selected as an appropriate AdvGC spacing to delineate the long thin anastomosing nature of the mineralised lodes and complete a reasonable grade estimate. All wider drill spacing showed a significant over estimation of Au ounces (Figure 10-5).

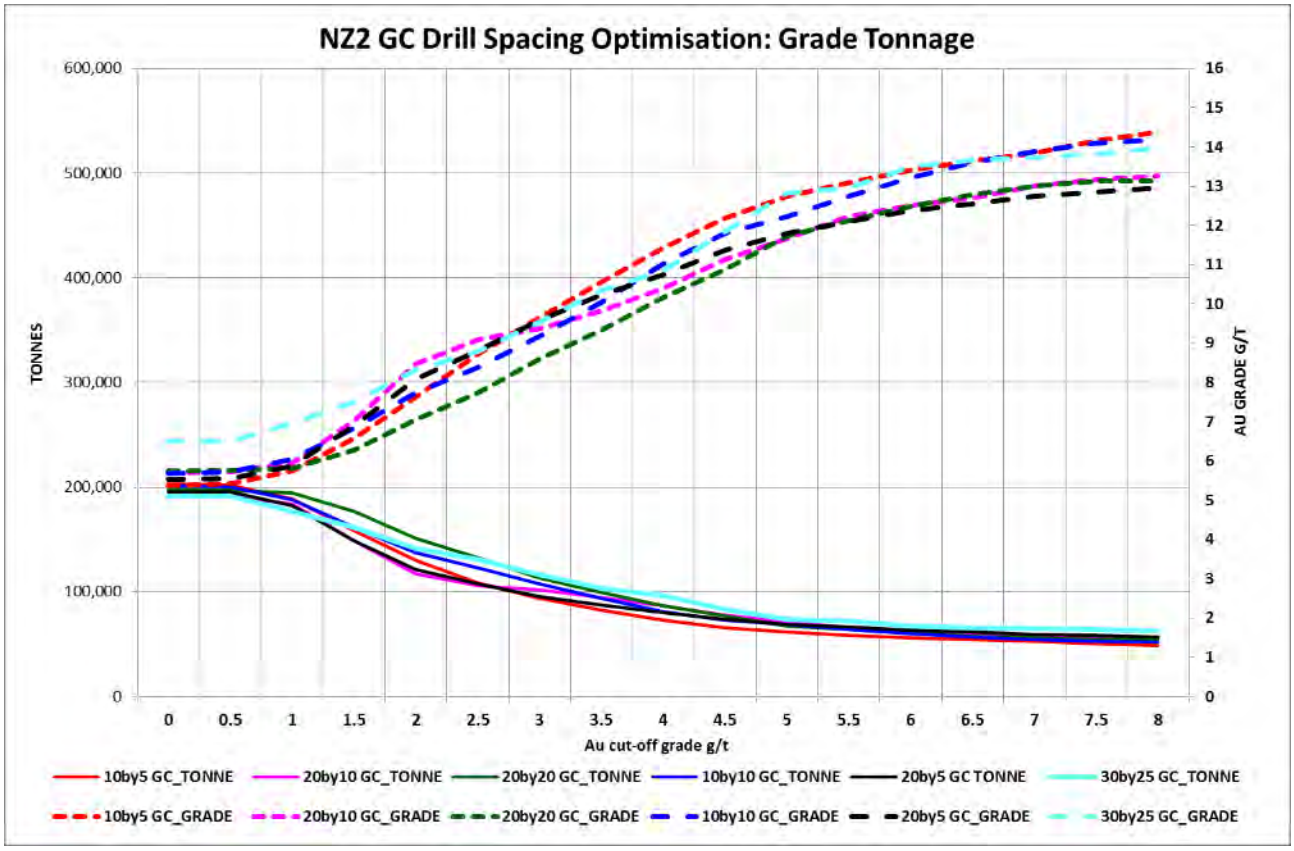


Figure 10-5 Grade Tonnage Curves for NZ2 GC Spacing Optimisation Models

10.12.5. Northern Zone Mineral Resource

Most the CZ is drilled on 40 m to 50 m sections with a mixture of RC and DD. There are two discrete blocks of RC drilling at 15 m by 10 m, one in NZ1 and one in NZ2 which have been drilled to test the impact of the AdvGC spaced drill definition against the original model. There is also one large block of 5 m by 5 m RC which was drilled as part of the drill spacing optimisation.

The QP is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

11. Sample Preparation, Analysis and Security

This chapter summarises the analytical techniques used at Massawa, the quality assurance and quality control (QA/QC) performance of the assay data, and the various re-assay protocols that were implemented for the feasibility programme. In addition, results from the various round robin programmes for the different analytical methods are also discussed.

11.1. Sample Preparation

Minimal sample preparation is carried out on site. Core samples are cut in half with a diamond saw and broken into smaller rock pieces for packaging. Approximately 3 kg of material per sample is sent to the laboratory for assay. Laboratory sample preparation includes weighing and drying, and crushing to 75% passing 2 mm. A 250 g or 1,000 g split (for fire assay and bulk respectively) is then pulverised to 85% passing 75 µm.

11.2. Sample Analysis

11.2.1. Massawa Northern Zone and Satellite Zones

For the Massawa NZ and satellite deposits, all samples were assayed by conventional 50 g Fire Assay (FA) with an Atomic Absorption Spectrometry (AAS) finish.

Massawa NZ samples are analysed by SGS Loulo laboratory at the Loulo mine site in Mali, where samples are analysed using lead collection 50 g fire assay with atomic absorption finish with a gravimetric finish for any samples reporting above 100 g/t Au.

Massawa CZ samples are analysed by SGS Ouagadougou in Burkina Faso. SGS Ouagadougou is used for all bulk 1 kg sample LeachWELL analysis (LWL69M) and 50 g fire assay of the remaining tail concentrate (FAA505T).

All samples for the Sofia Mineral Resource are analysed by SGS Loulo laboratory at the neighbouring Loulo mine site in Mali or SGS Bamako laboratory also in Mali. SGS Bamako is used for sample overflow, and for analysis that cannot be completed at SGS Loulo. Both laboratories are certified and operated independently by SGS. All Sofia Mineral Resource samples are analysed using lead collection 50 g fire assay with atomic absorption finish with a gravimetric finish for any samples reporting above 100 g/t Au.

All laboratories are certified and operated independently by SGS.

In addition, 500 g Metallic Screen (+/- 75 µm) fire assays (MSFAS30K) were also carried out at SGS Ouagadougou on the NZ samples. No significant assay bias was observed, likely due to lower amounts of coarse gold. No such testwork has been completed on Sofia and Delya samples to date, however, recent studies have shown a different sulphide assemblage with gold mineralisation directly associated with pyrite and no occurrence of free gold. In the QP's opinion, FA remains the most appropriate assay technique for the Sofia, Delya, and Massawa NZ.

11.2.2. Massawa Central Zone

For the CZ, where nuggety gold is present, a series of heterogeneity tests were completed on RC samples to determine the most appropriate laboratory analytical methodology to provide the most representative quantification of gold content. As part of the testwork, duplicate bulk one-kilogram sample LeachWELL analyses, in addition to standard 50 g FA, were completed at SGS Ouagadougou to test what sample size would provide more representative grades for samples with significant coarse gold content. LeachWELL is a reagent grade catalyst formulated for fast cyanide leach gold assaying, developed by Mineral Process Control (MPC). In addition to assay of the LeachWELL solutions, the residues (FA tails) are also assayed by 50 g FA to calculate a total gold grade (LeachWELL+FAtails).

The testwork samples were separated based upon the logged redox boundary in order to determine if there were any significant differences in the sample size requirements for assay of oxidised mineralisation and reduced fresh (sulphide) mineralisation. Results from this testwork indicated that for oxide samples the data shows a consistent bias for all grade ranges, with LeachWELL data showing a 24% increase in average grades and 28% increase in median grades relative to the FA data (Figure 11-1). For sulphide samples, the bias between the two assay techniques is less significant with LeachWELL data showing 8% increase in median grades relative to FA. The variance is more obvious above 5 g/t Au.

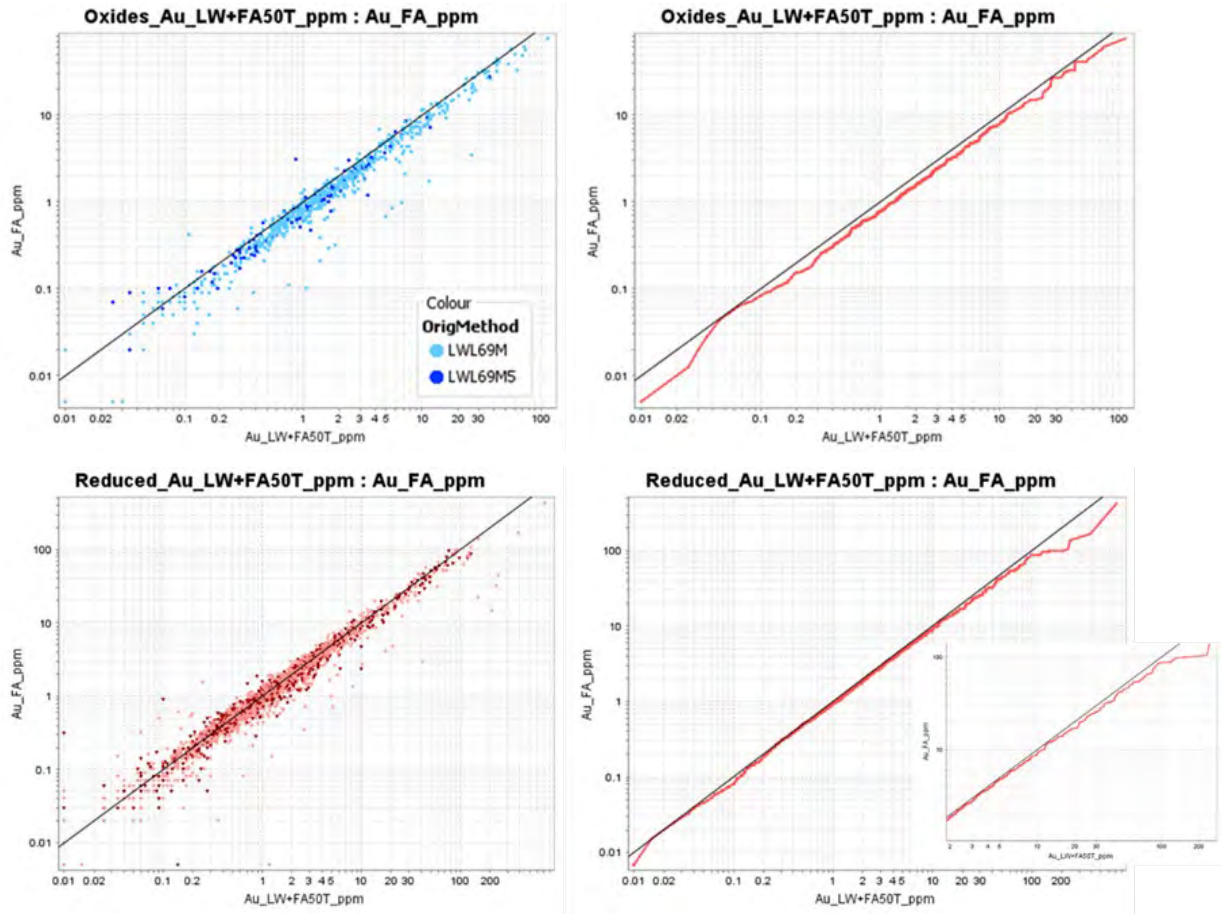


Figure 11-1 Scatter and QQ Plots Showing the Variance Between Fire and Assay and LeachWELL Analysis for Representative Oxide and Sulphide Samples

In addition to this, field duplicate analysis of multiple splits from the same RC sample showed a correlation in excess of 85% for LeachWELL, which is considered an acceptable level of repeatability. In contrast, both 50 g FA showed low levels of repeatability (<80% correlation of field duplicates). As a result, the LeachWELL method was chosen as the preferred assay method for the CZ FS to limit the sample volume variance of the nuggety ore. Although significant, the assay bias is minimal compared to the bias caused by the sample volume variance of RC vs DDH samples as described in Section 10.15.

Equivalent heterogeneity tests utilising metallic screen fire assays (MSFAS30K) returned lower grades than the LeachWELL analysis. Based upon this and the inefficiency of the technique for large sample volumes, this method was not chosen for the CZ FS.

11.2.3. LeachWELL Protocols

Over the course of the feasibility programme, the LeachWELL parameters at SGS Ouagadougou were modified and procedures revised after optimisation cyanide leaching testwork and guidelines set out by MPC. The three LeachWELL methods used during the FS are summarised in Table 11-1.

Table 11-1 Summary of the Various of LeachWELL Methods Used by SGS During the Massawa Feasibility

Parameters	LeachWELL Methodology		
	LWL69M	**LWL69M5**	**LWL69M5RG**
Sample Weight/Screening	1 kg pulverised to 85% <75 µm	1 kg pulverised to 85% <75 µm	1 kg pulverised to 85% <75 µm
Residence Time	24 hrs	24 hrs	12 hrs
No. of LeachWELL Tablets	1 x 10 g tab	5 x 10 g tab	5 x10 g tab
Litres of Water	2 litres	2 litres	2 litres
% Solids	33%	33%	33%
pH Controls	Lime added	No lime added, pH recorded every sample	No lime added, pH recorded every sample
Volume of Aliquot of Solution for AAS	40 ml	40 ml	40 ml
Standing Time Between	Excess of 6 to 8 hours	Excess of 6 to 8 hours	Within 6 to 8 hours
Tails Re-preparation Procedures	Slurry is washed and filtered 2 to 3 times, dried at 105°C, re-pulverised and solids analysed by 50 g FAAS	Slurry is washed and filtered 2 to 3 times, dried at 105°C, re-pulverised and solids analysed by duplicate 50 g FAAS	Slurry is washed and filtered thoroughly 3 times, dried at 105°C, re-pulverised and solids analysed by duplicate 50 g FAAS

The initial LeachWELL method used at SGS (LWL69M) followed the pilot LeachWELL testwork at BIGS Global Ouagadougou laboratory in Q4 2014. A small programme of 33 sulphide samples from twin RC and DDH drilling from the CZ were analysed on a 12 hr and 24 hr leach. Approximately 50% of the samples leached for 12 hrs show notably lower dissolutions and higher tail grades (mean grade of 15.1 g/t Au, median grade of 3.04 g/t Au) than samples leached for 24 hrs (mean grade of 6.6 g/t Au, median grade of 2.74 g/t Au). This bias is observed for all grade ranges from <1 g/t Au to >100 g/t Au. Results are presented below in Table 11-2 and Figure 11-2. The higher tail grades in the 12 hr LeachWELL samples relative to the 24 hr LeachWELL splits, and the variability in the total grades, was explained by incomplete dissolution of coarse gold grains over a 12 hr leach period, so the decision was made to use 24 hr leaching times for LeachWELL analysis going forward. The incomplete dissolution of the coarse gold grains over

12 hrs was confirmed by gold deportment studies, which showed frequent moth-eaten partial dissolution textures.

Table 11-2 Summary Statistics for Q4'2014 LeachWELL Orientation Work at BIGS Global Ouagadougou (BLC105)

	LW Solution_g/t		FAT_g/t		LW+FAT_g/t		Dissolution_%	
	BIGS_12 hr	BIGS_24 hr	BIGS_12 hr	BIGS_24 hr	BIGS_12 hr	BIGS_24 hr	BIGS_12 hr	BIGS_24 hr
Count	33	33	33	33	33	33	33	33
Minimum	0.63	0.36	0.31	0.16	3.36	0.94	6.43	10.62
Maximum	140.58	198.66	224.97	73.04	256.63	244.87	96.94	98.18
Mean	21.38	28.90	15.11	6.62	36.49	35.52	68.73	72.32
Median	8.79	10.01	3.04	2.74	12.56	11.66	82.27	83.74
Standard Deviation	28.49	45.35	43.72	14.36	52.91	52.01	29.81	28.23
Variance	811.6	2057.0	1911.8	206.2	2799.0	2704.8	888.6	797.2
Coefficient of Variation	1.33	1.57	2.89	2.17	1.45	1.46	0.43	0.39

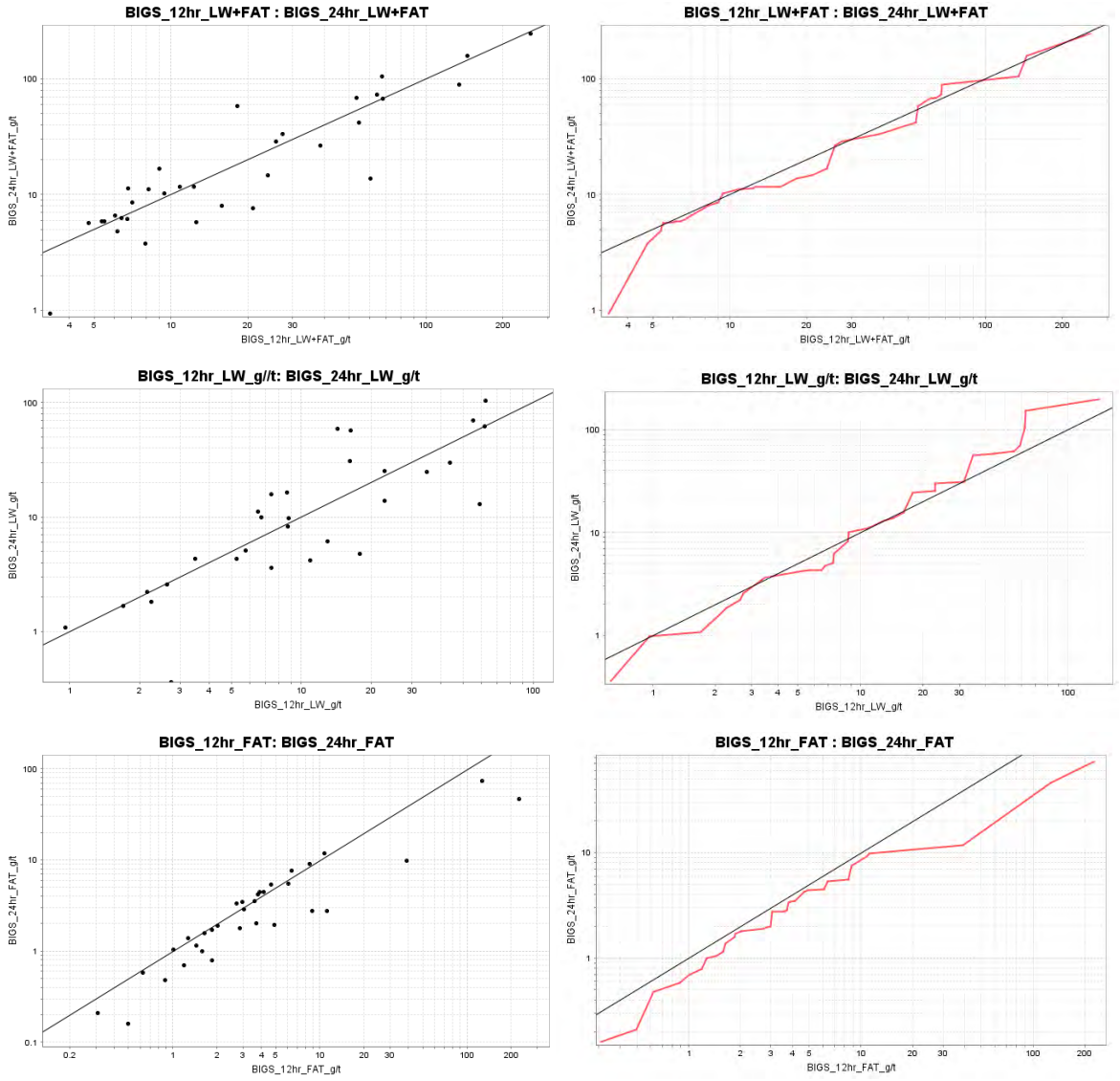
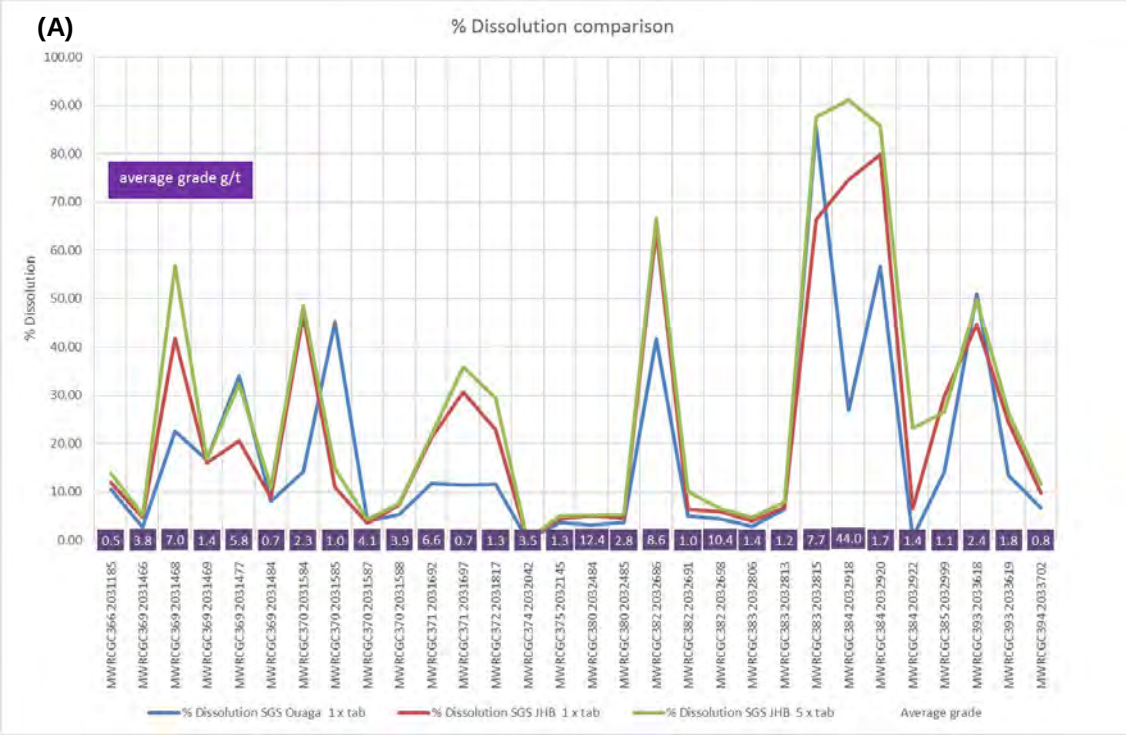


Figure 11-2 Scatter Plots (Left) and QQ Plots (Right) Showing the Q4'2014 LeachWELL Orientation Results for the Central Zone

A to B show Comparison of the Total Grades (solution plus the tails). The variability in total grades is the result of the nugget effect still being present in the tails.
C to D display the LeachWELL solution grades; E to F show the Fire Assay tail grades. Black line on all plots denotes the 1:1 line.

During the FS, the LeachWELL protocols were modified to meet the guidelines set out by MPC, and to standardise the procedure ahead of round robin testwork. The new SGS LeachWELL method (LWL69M5) included the following changes: (1) increasing the cyanide $(CN)^-$ concentration (five LeachWELL tablets instead of one tablet; so one tablet per 200 g of sample and 400 mL of water); (2) no addition of lime to prevent pH from rising above 11.5, as high levels of Sb can retard cyanidation in highly alkaline, neutral and acidic solutions (optimum pH 10-11.5); (3) more stringent controls on recording pH for every sample; and (4) duplicate 50 g FA analysis of the tails to validate for any sample bias in residues. In addition, external laboratory audits were carried out by senior Randgold geologists which led to alterations to several steps during sample preparation and analysis to reduce the risk of cross contamination and improve efficiency.

The change to five LeachWELL tablets (one tablet per 200 mL of water; five tablets for 1,000 g of sample at 50% solids), instead of one tablet, was aimed to maximise Au dissolution over a 24 hr residence time to better quantify the amenability of the ore to direct CN leaching (Figure 11-3A). The change in cyanide concentration has no effect on the total assay, only Au dissolution (Figure 11-3B).



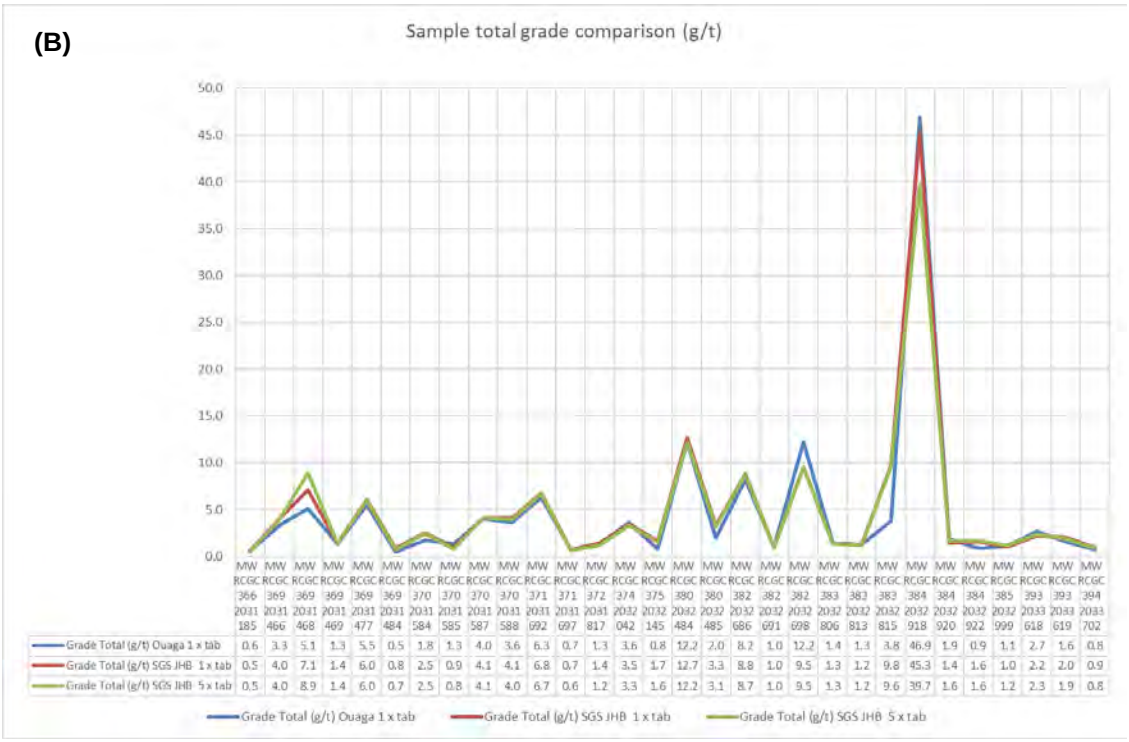


Figure 11-3 (A) One vs Five LeachWELL Tablet Dissolution of Selected Central Zone Mineralised Samples; (B) One vs Five LeachWELL Total Gold Grades

In March 2018, a final revision of the LeachWELL procedure was made (LWL69M5RG). Two phases of testwork at SGS Randfontein and SGS Ouagadougou were designed to test whether the residence time could be decreased from 24 hr to 12 hr, and thus increase turnaround at the laboratory. MPC suggests that the majority of gold should be leached within four hours using the LeachWELL method; although early pilot testwork at BIGS Global Burkina (BIGS Global) highlighted incomplete leaching at 12 hr. However, it is noted that this work used lower CN- concentrations (one LeachWELL tablet per 2 kg of sample) than suggested by MPC (one tablet per 200 g of sample).

The first phase of testwork was undertaken at SGS Randfontein and included kinetic LeachWELL analysis of six composites with intermediate solutions taken at 4 hr, 8 hr, 12 hr, and 18 hr with a total residence time of 24 hr, with the same parameters as outlined in the previous section. The composites included saprolite, saprock, and sulphide mineralisation with variable grades from 0.8 g/t Au to 30 g/t Au. The kinetic tests show that on average 95% of the gold dissolution occurred in the first four hours (Figure 11-4; Table 11-3), as anticipated by MPC. The kinetic curves also do not show a progressive reduction in dissolution with time, suggesting that the material is not preg-robbing.

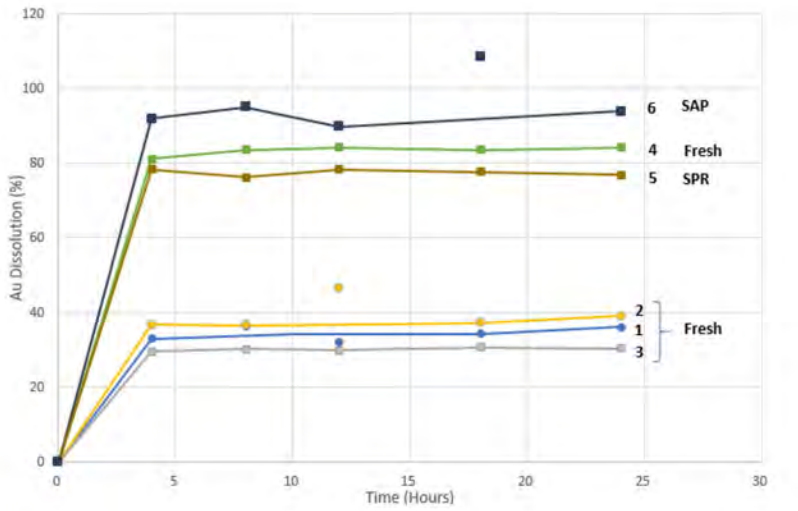


Figure 11-4 Gold Dissolution Kinetics for Composites 1 to 6, Showing Little Difference in Au Dissolution from 4 hours to 24 hours.

Table 11-3 Results from the Kinetic LeachWELL Tests at SGS Randfontein

Sample	Sample Interval	Residual NaCN				Final pH	Final DO	Residue Au (g/t)	Solution Au (mg/L)	Calculated Head Au (g/t)	Assayed Head Au (g/t)	Au Dissolution based on calc head Based on Solutions (%)	Au Dissolution based on calc head Based on solids (%)
		Titre (ml)	Aliquot (ml)	NaCN (g/L)	NaCN End (g)								
Comp 1	0	-				8.05				0.77	0.78	0.0	-
	4	50.80	5	73.68	139.90	10.69	9.58		0.127			33.1	-
	8	45.20	5	65.56	121.11	10.69	7.66		0.140			36.5	-
	12	43.20	5	62.66	111.37	10.72	8.25		0.124			32.3	-
	18	43.20	5	62.66	107.38	10.74	8.03		0.132			34.4	-
	24	40.80	5	59.18	101.41	10.75	8.34	0.49	0.139			36.2	36.2
Comp 2	0	-				8.24				1.95	2.23	0.0	-
	4	52.40	5	76.00	134.09	10.72	8.45		0.358			36.9	-
	8	49.20	5	71.36	122.29	10.71	8.59		0.356			36.7	-
	12	48.80	5	70.78	116.73	10.74	8.76		0.452			46.5	-
	18	48.40	5	70.20	111.27	10.73	8.71		0.363			37.4	-
	24	44.80	5	64.98	102.99	10.82	7.98	1.19	0.379			39.0	39.0
Comp 3	0	-				8.14				4.49	5.08	0.0	-
	4	49.20	5	71.36	137.59	10.66	7.22		0.661			29.5	-
	8	44.00	5	63.82	119.85	10.63	7.80		0.675			30.2	-
	12	44.00	5	63.82	114.97	10.58	7.03		0.670			29.9	-
	18	42.00	5	60.92	105.91	10.60	7.50		0.688			30.8	-
	24	41.60	5	60.34	104.90	10.64	9.02	3.12	0.682			30.5	30.5
Comp 4	0	-				8.26				31.85	30.85	0.0	-
	4	53.60	5	77.74	131.76	10.68	7.46		12.900			81.0	-
	8	49.60	5	71.94	118.31	10.67	7.68		13.300			83.5	-
	12	49.20	5	71.36	112.48	10.65	7.81		13.400			84.2	-
	18	48.40	5	70.20	106.25	10.67	7.60		13.300			83.5	-
	24	45.60	5	66.14	100.11	10.72	7.97	5.04	13.400			84.2	84.2
Comp 5	0	-				8.27				2.81	2.47	0.0	
	4	46.00	5	66.72	134.75	10.51	9.34		1.100			78.3	
	8	41.60	5	60.34	118.78	10.35	8.46		1.070			76.2	
	12	41.20	5	59.76	112.90	10.47	8.34		1.100			78.3	
	18	40.80	5	59.18	108.11	10.44	8.00		1.090			77.6	
	24	40.40	5	58.60	107.05	10.52	8.80	0.65	1.080			76.9	76.9
Comp 6	0	-				8.43	6.07			1.71	1.93	0.0	
	4	72.80	5	105.59	164.89	10.79	8.32		0.774			91.8	
	8	52.80	5	76.58	115.41	10.65	8.04		0.801			95.0	
	12	43.60	5	63.24	91.62	10.75	8.43		0.757			89.8	
	18	41.60	5	60.34	84.31	10.83	8.20		0.915			108.6	
	24	40.80	5	59.18	82.69	10.81	8.67	0.11	0.791			93.9	93.9

The second phase of testwork at SGS Ouagadougou included duplicate analysis LeachWELL (12 hr vs 24 hr) analysis of 75 selected mineralisation samples from the CZ Pilot Plant blocks.

Results show no bias between the 24 hr and 12 hr datasets, with median weighted total grades within 5% variance. Results are highlighted in the scatter and QQ plots in Figure 11-5. Based on these results, the residence time in LWL69M5RG update was decreased from 24 hr to 12 hr. Other changes to the protocol included reducing standing time to below six to eight hours after agitation has ceased and aliquot of solution taken for AAS, and filtration of the tails. Also, a more thorough filtration and washing procedure was implemented on the preparation of the tail samples. These changes were made to limit the chance of over reporting in the tails analysis.

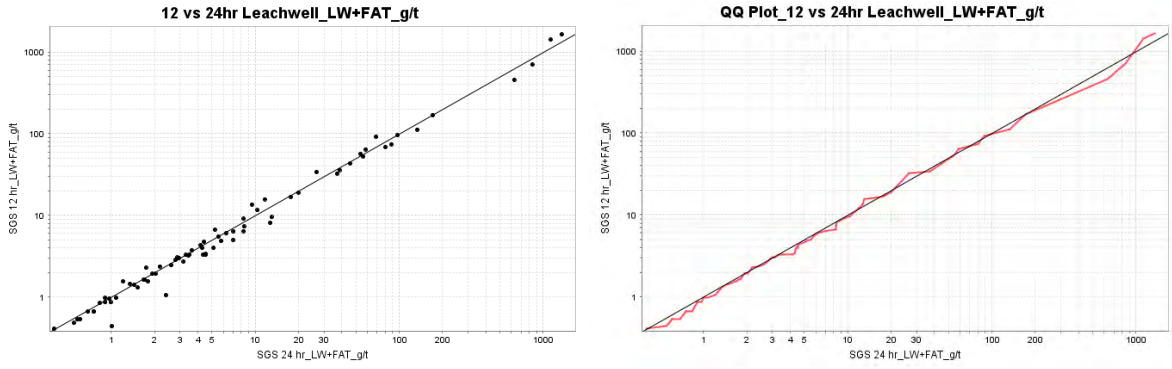


Figure 11-5 Scatter Plot (Left) and QQ plot (Right) Showing the Results of the Q1'2018 Repeat 12 hr LeachWELL Analysis Compared Against the Original 24 hr LeachWELL Data

Note: Black line refers to the 1:1 line.

The revised March 2018 LeachWELL protocol (LWL69M5RG) is summarised further in the flow chart illustrated in Figure 11-6.

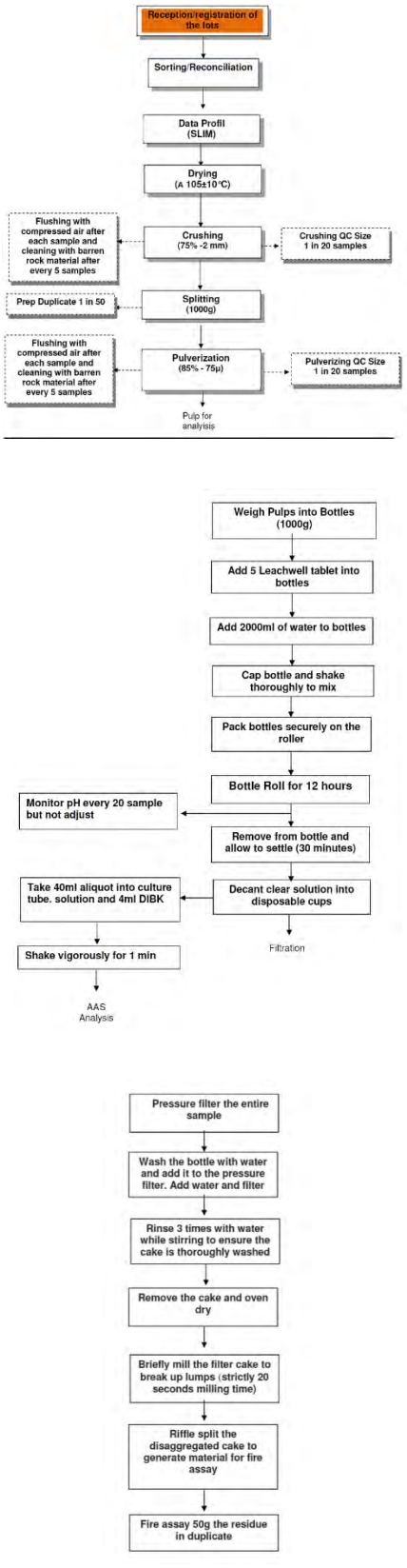


Figure 11-6 LeachWELL LWL69M5RG Flowchart

11.3. Quality Assurance and Quality Control

In order to ensure that the assay results are reliable, the Project has a robust QA/QC system in place to minimise errors at each stage, as well as procedures to be followed when errors are identified. This section covers the Delya, Sofia, and Massawa QA/QC analysis, from 20th December 2016 to 31st October 2018 (QA/QC review period). The majority of the samples were analysed by SGS at various locations including the Bamako, Ouagadougou, and Tarkwa commercial laboratories, in addition to the Loulo Mine laboratory. Analysis included conventional 50 g FA/AAS (FAA550) and one-kilogram accelerated bulk leach (LeachWELL; LWL69M5, LWL69M5RG). A minority of samples were assayed at BIGS Global laboratory in Ouagadougou.

11.3.1. Field Duplicates

Field duplicates were used to measure the precision of the whole sampling chain from primary sample collection, sub-sampling, through to laboratory sample preparation and analysis. Duplicates were inserted at a rate of one in every 20 samples. Mid-way through the FS, this was changed to one in every 20 samples within the mineralisation. A total of 1,346 field duplicates were analysed from the Massawa NZ and the satellite zones by FAA550, and 1,094 field duplicates were analysed from the Massawa CZ using LeachWELL (Table 11-4). Field duplicates show good correlation for all samples assayed by all the laboratories, with >93% correlation for FAA550 analysis method and >91% correlation for LWL69M5+LWL69M5RG analysis method. Such correlations can indicate that there is little or no bias in the results. Overall the results returned are acceptable.

Table 11-4 Summary of the Field Duplicate Samples Analysed by All Four Laboratories

Deposit	Laboratory	Sample Count	Analysis Method	Correlation
Delya	SGS Loulo	119	FAA550	98.67%
Delya	SGS Ouagadougou	16	FAA550	99.30%
Sofia	SGS Bamako	91	FAA550	99.44%
Sofia	SGS Loulo	101	FAA550	96.95%
Sofia	SGS Ouagadougou	43	FAA550	97.78%
Massawa	SGS Bamako	269	FAA550	97.61%
Massawa	SGS Loulo	32	FAA550	99.49%
Massawa	SGS Tarkwa	129	FAA550	93.21%
Massawa	SGS Tarkwa	126	LWL69M5/LWL69M5RG	96.28%
Massawa	SGS Ouagadougou	546	FAA550	97.71%
Massawa	SGS Ouagadougou	919	LWL69M5/LWL69M5RG	97.26%
Massawa	BIGS Ouagadougou	49	BLC155	91.50%

Figure 11-7 and Figure 11-8 provide an example of the graphical representations of original vs field duplicate samples analysed by all laboratories for the period. A breakdown of all field duplicates by individual laboratory and analysis type are presented in Appendix 30.1.

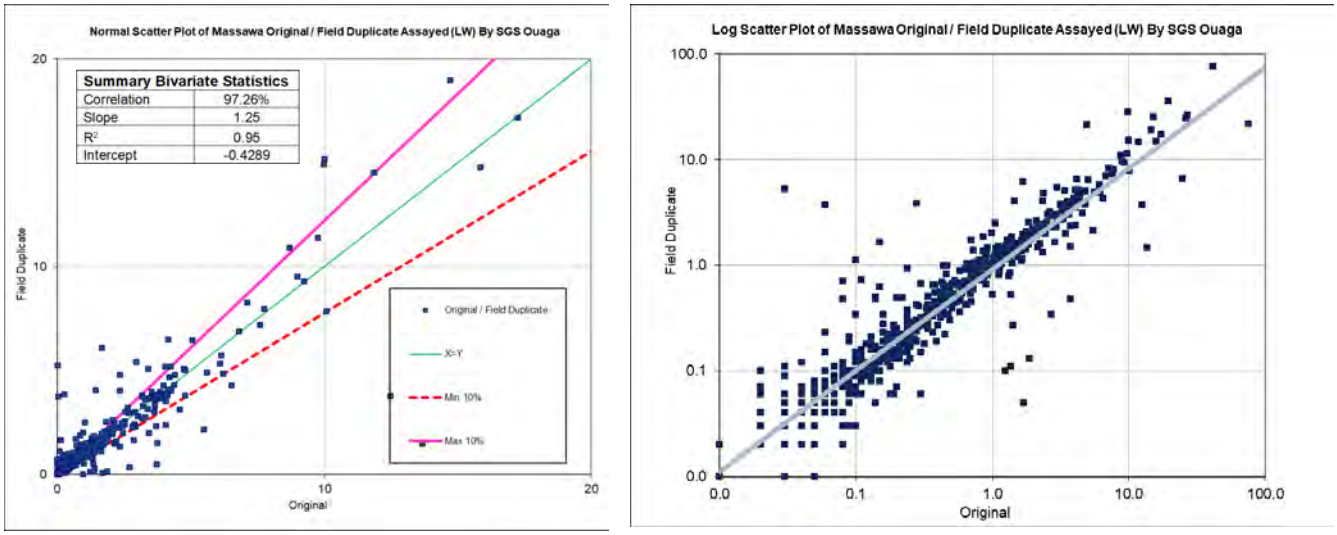


Figure 11-7 Normal and Log Scatter Plots of Massawa Field Duplicates Assayed (LeachWELL) by SGS Ouagadougou

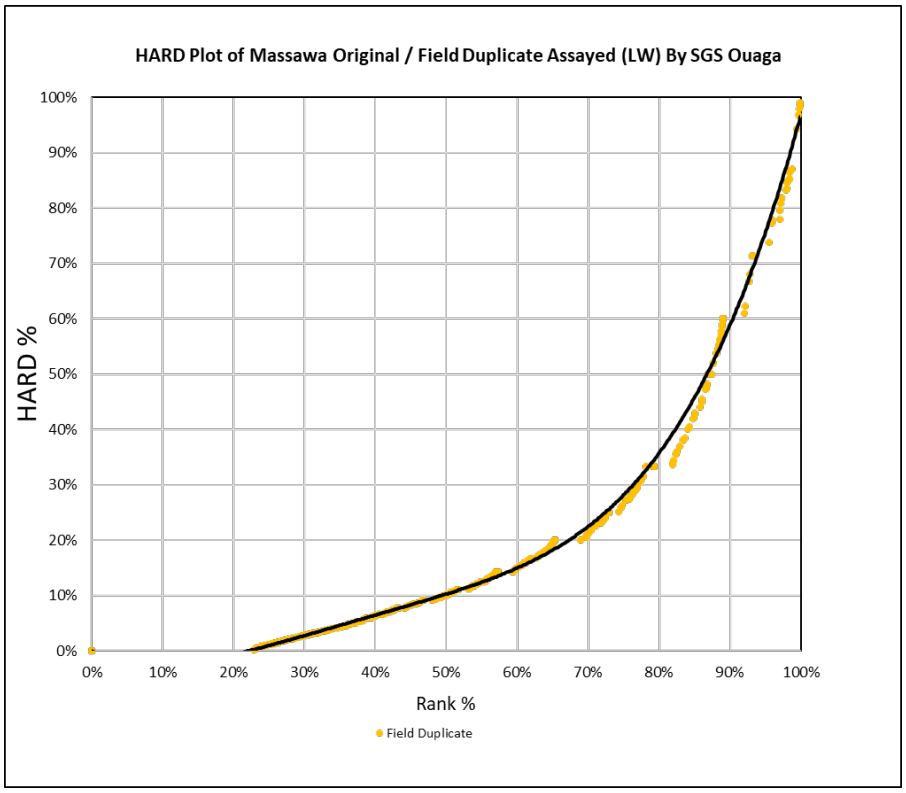


Figure 11-8 HARD Plot of Massawa Field Duplicates Assayed (LeachWELL) by SGS Ouagadougou

11.3.2. Duplicate Analysis of LeachWELL Residues

LeachWELL methods LWL69M5 and LWL69MRG include duplicate FA/AAS analysis of the residues for every sample. This was implemented to establish if any sample bias is still present in the 50 g tail assays. The tail duplicates for both SGS Ouagadougou and SGS Tarkwa show excellent repeatability for both laboratories with a correlation of 0.98, and thus no sample volume variance is present in the residue analysis (Figure 11-9 and Figure 11-10).

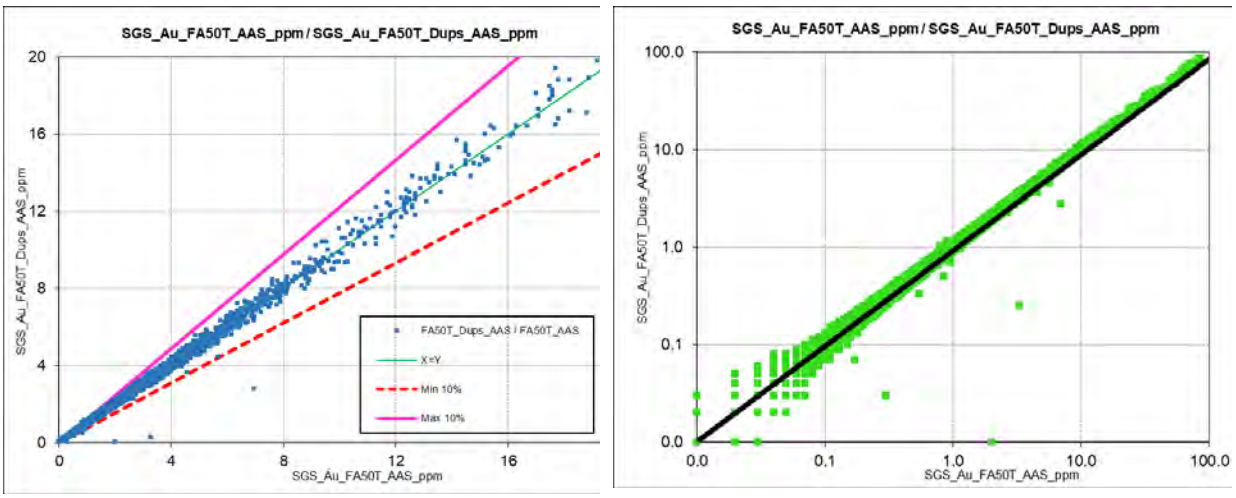


Figure 11-9 Normal and Log Scatter Plots of Massawa LeachWELL Residues Assayed by SGS Ouagadougou

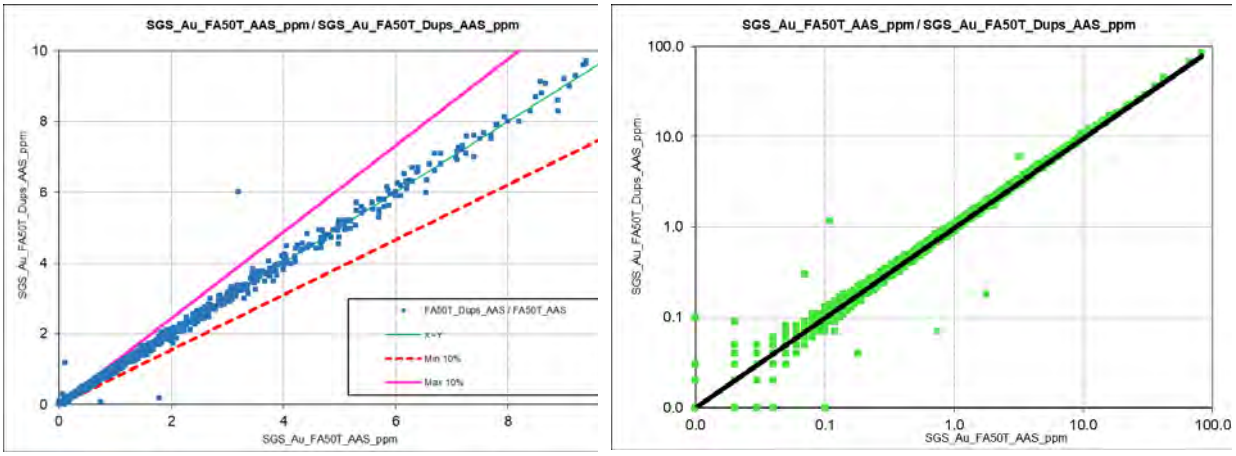


Figure 11-10 Normal and Log Scatter Plots of Massawa LeachWELL Residues Assayed by SGS Tarkwa

11.3.3. Blanks

Blank samples are free media (Au-free for these analyses) assayed to help ensure no false-positives are obtained from the laboratories and to check for contamination. Commercial blank OREAS27b was used for the FS. Blanks were inserted into the sample stream at a rate of 1:20 samples for LWL69M5+LWL69M5RG (1 kg) and 1:20 samples for FAA550 (approximately 200 g). These samples undergo the same sample preparation as the drill samples and are used to detect contamination due to poor cleaning of sample preparation equipment throughout the various sub-sampling processes.

A total of 4,115 samples were analysed using FAA550 and 3,067 were analysed using LWL69M5+LWL69M5RG. All blanks samples were evaluated against 2 to 3 times the standard deviation (SD) as the acceptable limit. Blank samples performed well within the period and sample contamination is within the acceptable limits at all the laboratories. The overall performance shows that approximately 99% of the blanks assayed are within acceptable limits (Table 11-5).

An example of the graphical representation of the performance Massawa LeachWELL blank sample results analysed by SGS Ouagadougou during the period is presented in Figure 11-11. Individual plots for each laboratory and analysis type are presented in Appendix 30.2.

Table 11-5 Summary of Blank Samples

Deposit	SGS Laboratory	Analysis Type	Max Assay	No. Samples	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Delya	Loulo	FAA550	0.24	797	3 (0.38%)	1 (0.13%)	793 (99.50%)	794 (99.62%)	3 (0.38%)
	Ouagadougou	FAA551	0.005	36	0	0	36 (100%)	36 (100%)	0
Sofia	Bamako	FAA552	0.01	269	0	0	269 (100%)	269 (100%)	0
	Loulo	FAA553	0.02	844	0	2 (0.24%)	842 (99.76%)	844 (100%)	0
	Ouagadougou	FAA554	0.005	141	0	0	141 (100%)	141 (100%)	0
NZ / CZ	Bamako	FAA555	0.02	635	0	13 (2.1%)	622 (98.0%)	635 (100%)	0
	Loulo	FAA556	0.01	229	0	0	229 (100%)	229 (100%)	0
	Tarkwa	FAA557	0.02	220	0	11 (5.0%)	209 (95.0%)	220 (100%)	0
		LWL69M5/ LWL69M5RG	0.02	345	0	1 (0.3%)	344 (99.7%)	345 (100%)	0
	Ouagadougou	FAA556	2.09	943	1 (0.1%)	11 (1.2%)	932 (98.8%)	942 (99.9%)	1 (0.1%)
		LWL69M5 /LWL69M5RG	2.07	2471	3 (0.1%)	14 (0.6%)	2457 (99.4%)	2468 (99.9%)	3 (0.1%)
			0.0005	251	0	0	251 (100%)	251 (100%)	0

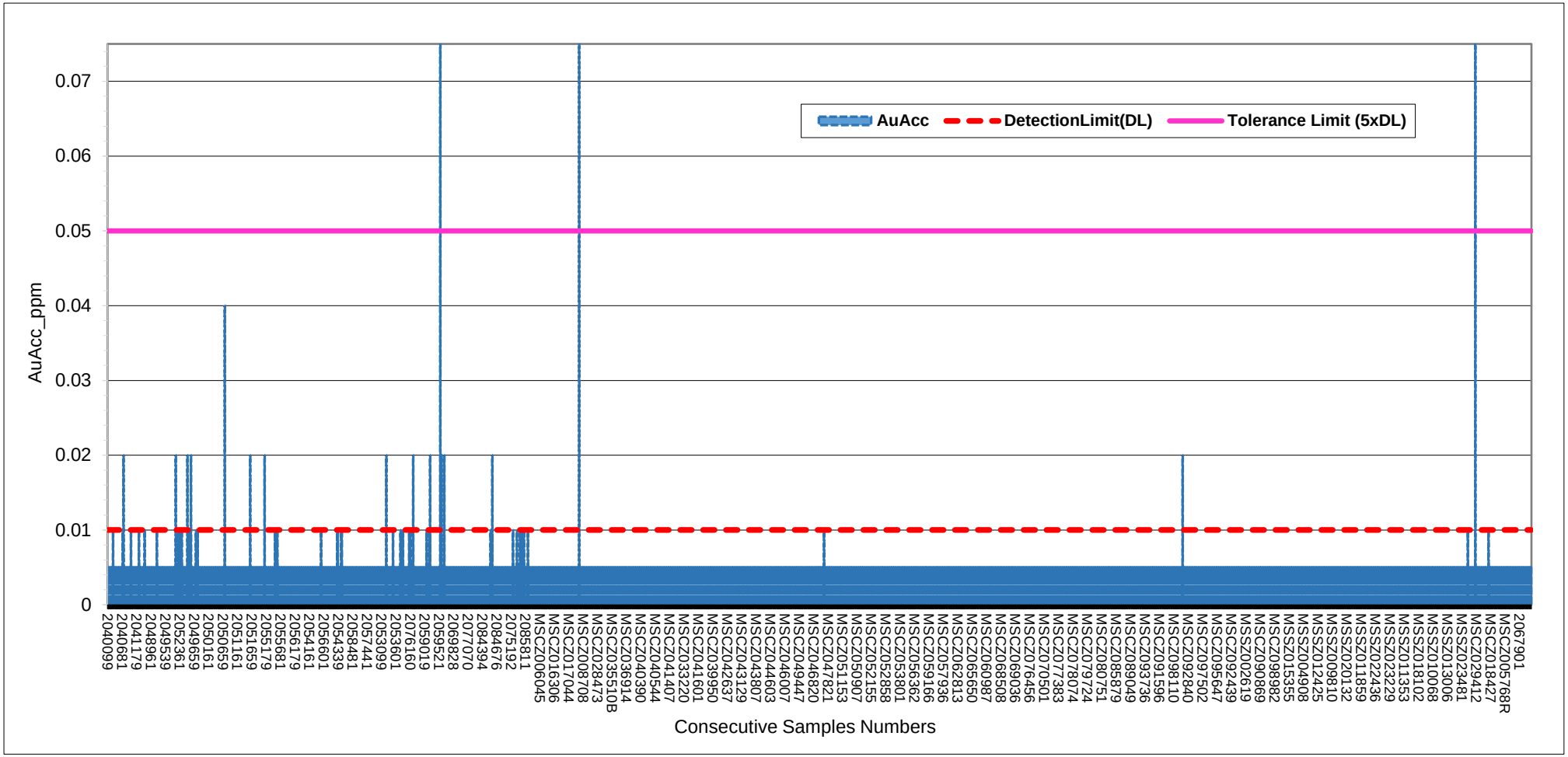


Figure 11-11 Massawa Blank Samples Assayed (LeachWELL) by SGS Ouagadougou

11.3.4. Certified Reference Materials

Certified Reference Materials (CRM) were inserted into each batch at a frequency of one in 20 mineralised samples for LWL69M5/LWL69M5RG and one in 20 samples for FAA550, in order to validate results reported by the laboratory and also monitor the control and calibration of the instruments used by the laboratory.

CRMs used within the QA/QC review period (20th December 2016 to 31st October 2018) were sourced from OREAS Australia, with 60 g sachets used for FAA550 and one-kilogram material used for LeachWELL. CRMs performed well, however, there is room for improvement. A total of 3,843 CRMs consisting of 25 types of CRMs were analysed using FAA550 and 2,280 CRMs consisting of 11 types were analysed using LWL69M5/LWL69M5RG during the period (Table 11-6).

Table 11-6 CRM Summary

Deposit	Laboratory	Sample Count	Analysis Method
Delya	SGS Loulo	782	FAA550
Delya	SGS Ouagadougou	33	FAA550
Sofia	SGS Bamako	174	FAA550
Sofia	SGS Loulo	763	FAA550
Sofia	SGS Ouagadougou	136	FAA550
Massawa	SGS Bamako	601	FAA550
Massawa	SGS Loulo	225	FAA550
Massawa	SGS Ouagadougou	924	FAA550
Massawa	SGS Ouagadougou	1711	LWL69M5/LWL69M5RG
Massawa	SGS Tarkwa	205	FAA550
Massawa	SGS Tarkwa	319	LWL69M5/LWL69M5RG
Massawa	BIGS Ouagadougou	250	BLC155

An example of the tram line plot and results of the CRMs used for LeachWELL analysis of Massawa samples at SGS Ouagadougou during the period is provided in Figure 11-12. Tram line plots and results for each individual laboratory and analysis type is provided in Appendix 30.3.

SUMMARY OF MASSAWA CRMs ASSAYED (LW) BY SGS OUAGA DURING THE PERIOD								
CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.49	5.946	5.034	5.11	5.69	152
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.84	0.925	243
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.06	1.168	0.952	1.02	1.155	281
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.15	2.309	1.991	2.03	2.26	146
OREAS 228	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	8.73	9.567	7.893	8.645	9.16	63
OREAS 229	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	12.11	12.728	11.492	12.075	12.305	4
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.645	0.715	195
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.352	1.088	1.185	1.31	167
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.55	2.778	2.322	2.485	2.665	173
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.08	4.341	3.819	3.965	4.245	172
OREAS 257	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	14.18	14.972	13.388	13.88	14.55	115
								1711

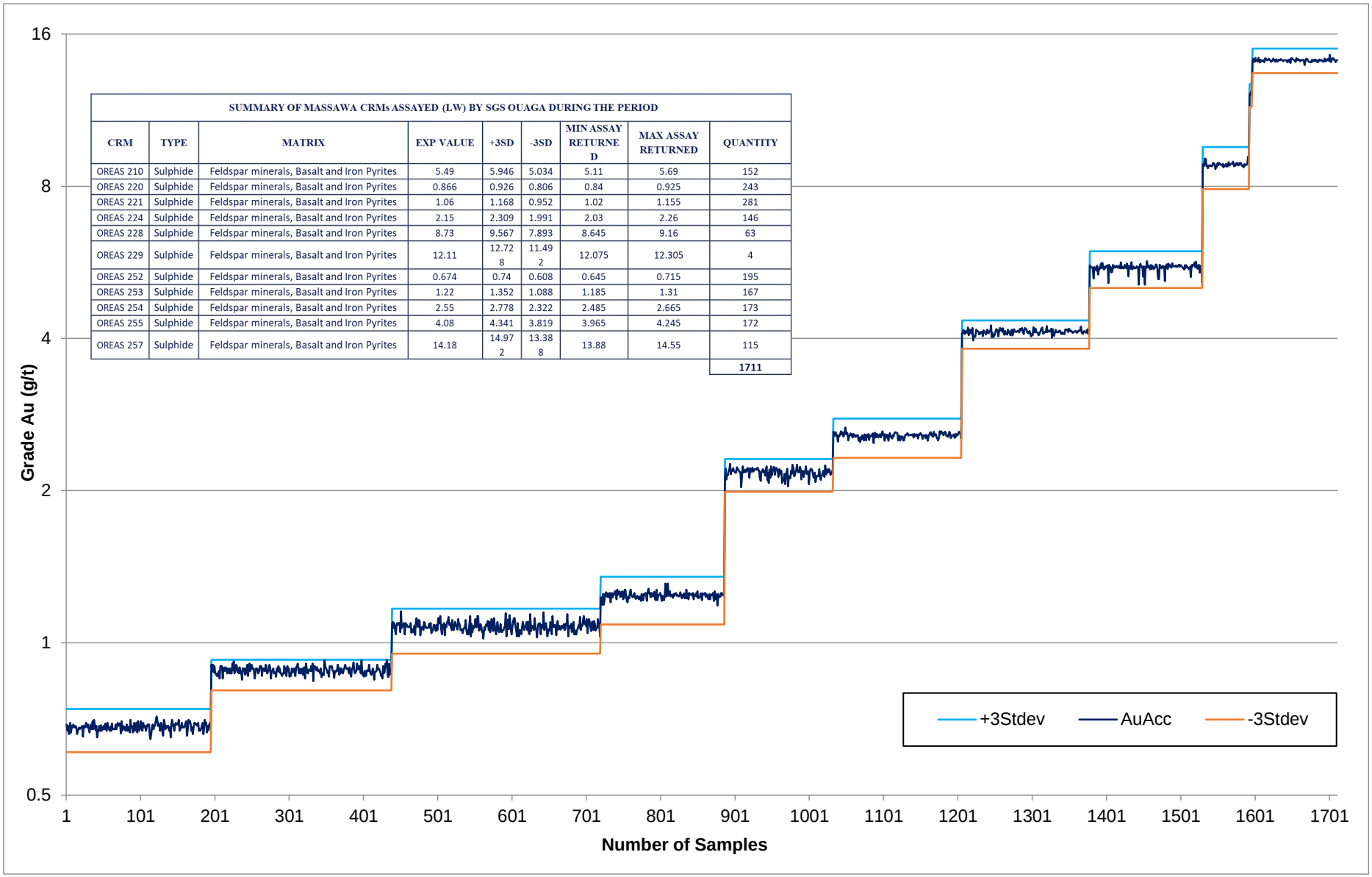


Figure 11-12 Tram Line of Massawa CRMs Assayed (LeachWELL) by SGS Ouagadougou Laboratory

11.4. Re-Assay Protocols

Out of 1,109 batches of results received from 20th December 2016 to 31st October 2018, 57 batches failed QA/QC and were re-assayed. Both original and re-assayed batches have been stored in the database with the re-assay data being prioritized. Although the QA/QC failures are evaluated on a case-by-case basis, the guidelines below were used for the FS (Table 11-7 and Table 11-8).

Table 11-7 Failed Fire Assay QA/QC Procedure

FSE Quality Control				Response
Rule	**Response**	**QC Measure**	**<Failure definition>**	
1	Blank	Contamination	A blank greater than 0.09 ppm [90 ppb] for Au is a failure. Blank .0.07 ppm [70 ppb] <0.09 ppm [90 ppb] warning but not a failure. A CRM > 3RSD (relative standard deviations) from the expected value is a failure.	**1x Failure per batch = Tray re-assay** (74 samples + 10 lab internal CRMs) **>1x Failure per batch = Whole batch re-assay** (200 samples + lab internal CRMs)
2	Standard	Accuracy	A CRM > 2RSD but <3SD from the expected value is a warning but not a failure	**1x Failure per batch = bracketed re-assay** (~10 samples either side of failure + lab internal CRMs) **2x consecutive failures per batch** = Tray re-assay (74 samples + 10 lab internal CRMs) **>2x consecutive failure / 2x non-consecutive failures per batch = Whole batch re-assay** (200 samples + lab internal CRMs) – indicates contamination
3	Standard	Accuracy	2x consecutive standards >2RSD but <3RSD on the same side (i.e. both +ve or –ve) of the expected value is a failure (indicate bias).	**1x Failure per batch = Tray re-assay** (74 samples + 10 lab internal CRMs)
4	Field Duplicates	Precision (field+lab)	% Difference >20% to be investigated and discussed with the laboratory % Difference >35% to be is a failure	**1x Failure per batch – bracketed re-assay** (~10 samples either side of failure + lab internal CRMs) **2x consecutive failures per batch** = Tray re-assay (74 samples + 10 lab internal CRMs) **>2x consecutive failure / 2x non-consecutive failures per batch = Whole batch re-assay** (200 samples + lab internal CRMs) – indicates contamination
5	Laboratory Duplicates	Precision (prep+analysis)	% Difference >20% to be investigated and discussed with the laboratory. % Difference >35% to be is a failure.	**1x Failure per batch = +/1 10 sample re-assay** (20 samples either side of failure + lab internal CRMs) **2x consecutive failures per batch** = Tray re-assay (74 samples + 10 lab internal CRMs) **>2x consecutive failure / 2x non-consecutive failures per batch = Whole batch re-assay** (200 samples + lab internal CRMs) – indicates contamination
6	Standard/ Blanks	Accuracy	Highly anomalous CRM value probably caused by sample mix-up should be investigated.	**CRM swap caused on-site = No re-assay** – CRM re-assigned **1x CRM swap caused at lab = Tray re-assay** (74 samples + 10 lab internal (CRMs) **2x CRM swap caused at lab = Whole batch re-assay** (200 samples + internal CRMS)

Table 11-8 Failed LeachWELL QA/QC Procedure

FSE Quality Control				Response
Rule	**QC_Type**	**QC_Measure**	**<Failure definition>**	
1	Blank	Contamination	A blank greater than 0.09 ppm [90 ppb] for Au is a failure. Blank 0.07 ppm [70 ppb] <0.09 ppm [90 ppb] warning but not a failure.	**1x Failure per batch = bracketed re-assay** (typically 20-25 samples either side of failure + lab internal CRMs) **>1x Failure per batch = Whole batch re-assay** (200 samples + lab internal CRMs)
2	Standard	Accuracy	A CRM > 3RSD (relative standard deviations) from the expected value is a failure. A CRM > 2RSD but <3SD from the expected value is a warning but not a failure.	**1x Failure per batch = bracketed re-assay** (typically 20-25 samples either side of failure + lab internal CRMs) **2x consecutive failures per batch** = **ext. bracketed re-assay** (ass samples above and below failed CRMs up to and including closet passed CRMs + blanks) **>2x consecutive failure / 2x non-consecutive failures per batch = Whole batch re-assay** (200 samples + lab internal CRMs) – indicates contamination
3	Standard	Accuracy	2x consecutive standards >2RSD but <3RSD on the same side (i.e. both +ve or –ve) of the expected value is a failure (indicate bias).	**1x Failure per batch = ext. bracketed re-assay** all samples above and below failed CRMs up to and including closet passed CRMs + blanks)
4	Field Duplicates	Precision (field+lab)	% Difference >20% to be investigated and discussed with the laboratory. % Difference >35% to be is a failure.	**1x Failure per batch – bracketed re-assay** (typically 20-25 samples either side of failure + lab internal CRMs) **2x consecutive failures per batch = ext. bracketed re-assay** (all samples above and below failed CRMs up to and including closest passed CRMs + blanks) **>2x consecutive failure / 2x non-consecutive failures per batch = Whole batch re-assay** (200 samples + lab internal CRMs) – indicates contamination
5	Laboratory Duplicates	Precision (prep+analysis)	% Difference >20% to be investigated and discussed with the laboratory. % Difference >35% to be is a failure.	**1x Failure per batch = bracketed re-assay** (typically 20-25 samples either side of failure + lab internal CRMs) **2x consecutive failures per batch = ext. bracketed re-assay** (ass samples above and below failed CRMs up to and including closest passed CRMs + blanks) **>2x consecutive failure / 2x non-consecutive failures per batch = Whole batch re-assay** (200 samples + lab internal CRMs) – indicates contamination
6	Standard/ Blanks	Accuracy	Highly anomalous CRM value probably caused by sample mix-up should be investigated.	CRM swap **caused on-site = No re-assay** – CRM re-assigned **1x CRM swap caused at lab = ext. bracketed re-assay** (all samples above and below failed CRMs up to and including closet passed CRMs + blanks) **2x CRM swap caused at lab = Whole batch re-assay** (200 samples + internal CRMS)

Definitions used in Table 11-7 and Table 11-8:

- **Bracketed re-assay:** samples above and below the failed QA/QC samples must be re-analysed up to and including halfway to the closest passed QA/QC samples. The number of samples for re-assay depends on the insertion ratio.

- Example (1): Fire assay (QA/QC insertion ratio 1/20) 10 samples above and below the failed QA/QC samples should be re-assayed.
- Example (2): LeachWELL (>120 m hole – in ore: 1/25 QA/QC insertion ratio) 13 samples above and below the failed QA/QC samples should be re-assayed.

- **Extended (ext.) Bracketed re-assay:** all samples above and below the failed QA/QC samples must be re-analysed up to and including the closest passed QA/QC samples. The number of samples for re-assay depends on the insertion ratio.

  - Example (1): LeachWELL (<120 m hole – in ore: 1/20 QA/QC insertion ratio) 20 samples above and below + passed QA/QC samples (CRM-blank-duplicate) totalling approximately 46 samples should be re-assayed.
  - Example (2): LeachWELL (>120 m hole – in waste: 1/50 QA/QC insertion ratio) 50 samples above and below + passed QA/QC samples (CRM-blank) totalling approximately 104 samples should be re-assayed.

- **Tray re-assay:** all samples run in a single FA furnace tray (typically approximately 74 samples) should be re-assayed. The number of samples to be confirmed with laboratory before re-assay requested. This should include all QA/QC samples in the batch.
- **Whole Batch re-assay:** all samples within the batch (100 to 200 samples) including QA/QC samples (CRMs-blanks-duplicates) must be analysed.

11.5. Round Robin Studies

Various campaigns of round robin testwork hаmy been carried out on the Massawa deposit during the pre-feasibility and feasibility programmes. These studies largely concentrated on the CZ and tested the reproducibility and reliability of the LeachWELL analysis and the SGS Ouagadougou laboratory.

11.5.1. Pre-feasibility Programmes

In Q1 2015, an initial phase of LeachWELL round robin testwork was completed. A total of 50 representative oxide ore samples from the 5 m by 5 m GC grid of the CZ were selected for analysis at SGS Ouagadougou and BIGS Global. The LeachWELL parameters for both laboratories at the time of analysis are summarised in Table 11-9, the most notable difference being the sample weight.

Table 11-9 BIGS (BLC105) and SGS (LWL69M) LeachWELL Procedures

Parameters	BIGS Global Ouagadougou (BLC105)	SGS Ouagadougou (LWL69M)
Sample Weight/Screening	1.5-2 kg pulverised to 90% <75 µm	1 kg pulverised to 85% <75µm
Residence Time	24 hrs	24 hrs
No. of LeachWELL Tablets	1 x10 g tab	1 x10 g tab
Litres of Water	2 litres	2 litres
% Solids	33%	33%
Lime Added?	Yes	Yes
Volume of Aliquot of Solution for AAS	100 ml	40 ml
Standing Time	Excess of 6 to 8 hours	Excess of 6 to 8 hours
Tails Re-preparation Procedures	Slurry is washed and filtered 2 times, dried at 115°C, re-pulverised and solids analysed by FAAS	Slurry is washed and filtered at least 3 times, dried at 105°C, re-pulverised and solids analysed by FAAS

Results from the round robin work are presented in Figure 11-13 and Table 11-10. The 2 kg BIGS Global samples showed lower dissolutions and higher tail grades compared to the 1 kg SGS samples. In total, 47 of the 50 (94%) samples analysed by BIGS Global displayed higher tail grades than those analysed by SGS, with mean tails grades 87.5% higher (3.44 g/t Au). The total grades reflect the bias in the tails grades, with consistently higher total grades reported from the BIGS Global samples.

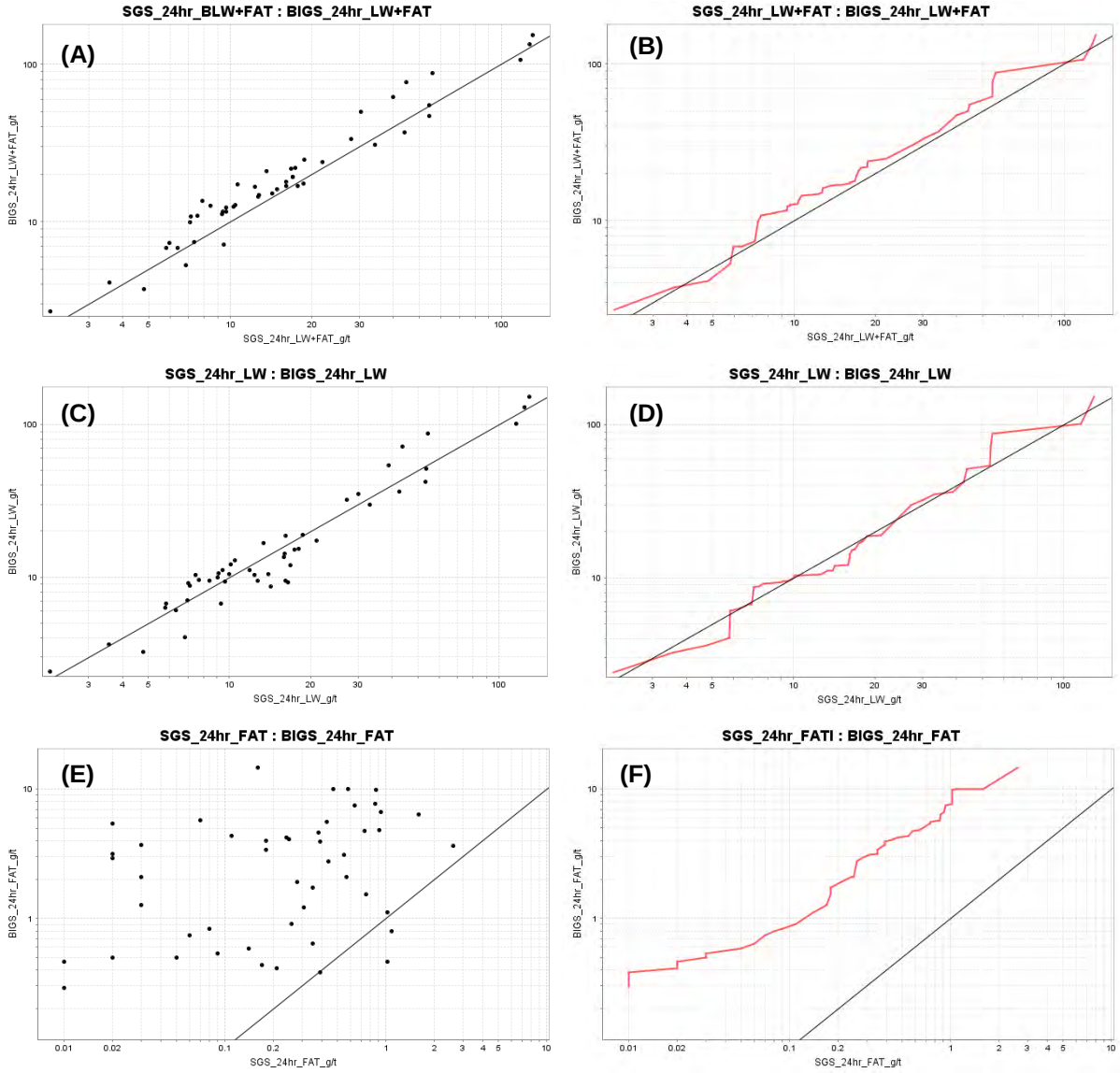


Figure 11-13 Scatter Plots (Left) and QQ Plots (Right) Showing the Initial Q1 2015 Round Robin Results for the Central Zone

A to B show comparison of the total grades (solution plus the tails). C to D display the LeachWELL solution grades; E to F show the FA tail grades. Black line on all plots denotes the 1:1 line.

Table 11-10 Summary Statistics for Q1 2015 LeachWELL Initial Round Robin Studies

Parameters	LW solution_g/t		FAT_g/t		LW+FAT_g/t		Dissolution_%	
	BIGS_24 hr	SGS_24 hr	BIGS_24 hr	SGS_24 hr	BIGS_24 hr	SGS_24 hr	BIGS_24 hr	SGS_24 hr
Count	50	50	50	50	50	50	50	50
Minimum	2.40	2.16	0.29	0.01	2.70	2.17	48.03	94.71
Maximum	151.13	130.00	14.76	2.63	153.88	130.44	99.48	99.84
Mean	24.56	23.62	3.44	0.43	28.00	24.05	82.97	98.08
Median	11.13	13.40	2.91	0.28	16.73	13.64	87.98	98.03
Standard Deviation	32.12	29.12	3.18	0.49	32.65	29.49	13.49	1.34
Variance	1031.8	848.2	10.1	0.2	1066.1	869.4	181.9	1.8
Coefficient of Variation	1.31	1.23	0.92	1.13	1.17	1.23	0.16	0.01

The higher residue grades in the BIGS Global samples were initially explained as lower degrees of dissolution in the larger 2 kg samples relative to the 1 kg samples analysed at SGS Ouagadougou (twice the material to leach for the same CN^- concentration). The cyanide dissolution issue at BIGS Global were later confirmed by detailed gold deportment studies on selected high-grade (over 10 g/t Au) tail samples. Petrographic analysis of residue gold grains shows pronounced moth-eaten textures indicative of partial leaching. However, the amount of variance in the tail grades and the bias seen in the total grades suggested an additional factor was involved, which was investigated in more detail during the FS round robin programmes. The BIGS Global laboratory was resistant to change its LeachWELL procedures and so the decision was made to use SGS Ouagadougou as the primary assay laboratory (SGS Tarkwa as overflow laboratory) for the CZ pre-feasibility and feasibility programmes.

11.5.2. Feasibility Programmes

During Q4 2017 to Q3'2018, a phased inter-laboratory round robin testwork programme was carried out on the Massawa CZ. A total of 600 representative mineralised samples from across the strike length of the CZ were selected and analysed at ALS Ouagadougou and Bureau Veritas (BV) Abidjan to test for the reproducibility of the feasibility SGS Ouagadougou LeachWELL data, and to test for any sample bias. The representative sample suite included saprolite, oxidised+reduced saprock, and sulphide mineralisation in an approximate 20/15/65 ratio.

Comparisons of the total grades (LeachWELL solution plus the tail FA) show variability on a sample-by-sample basis (a third of the samples show greater than 25% variance); however, mean total grades for all campaigns are within 9% variance. Some of the more nuggety parts of the CZ system show higher variance with mean grades up to 25% higher in the original SGS analysis (Table 11-11). This bias is more evident in samples below 10 g/t Au (Figure 11-14A-B). To help explain this bias, a statistical comparison was carried out on both the individual solution and FA residue assays. Slightly higher LeachWELL solutions grades are recorded at the round robin laboratories (Figure 11-14C-D), but the main difference is seen in the FA residue data (Figure 11-14E-F). A consistent bias is seen for all grade ranges, with significantly higher (average 30% variance) residue grades reported at SGS (weighted average of 3.15 ± 0.5 g/t Au) compared to ALS/BV (weighted average of 2.20 ± 0.43 g/t Au). This bias is seen in all campaigns except the northernmost part of the CZ (Block K/L) (Table 11-11). As a result, lower dissolutions are reported in the SGS data relative to the repeat testwork.

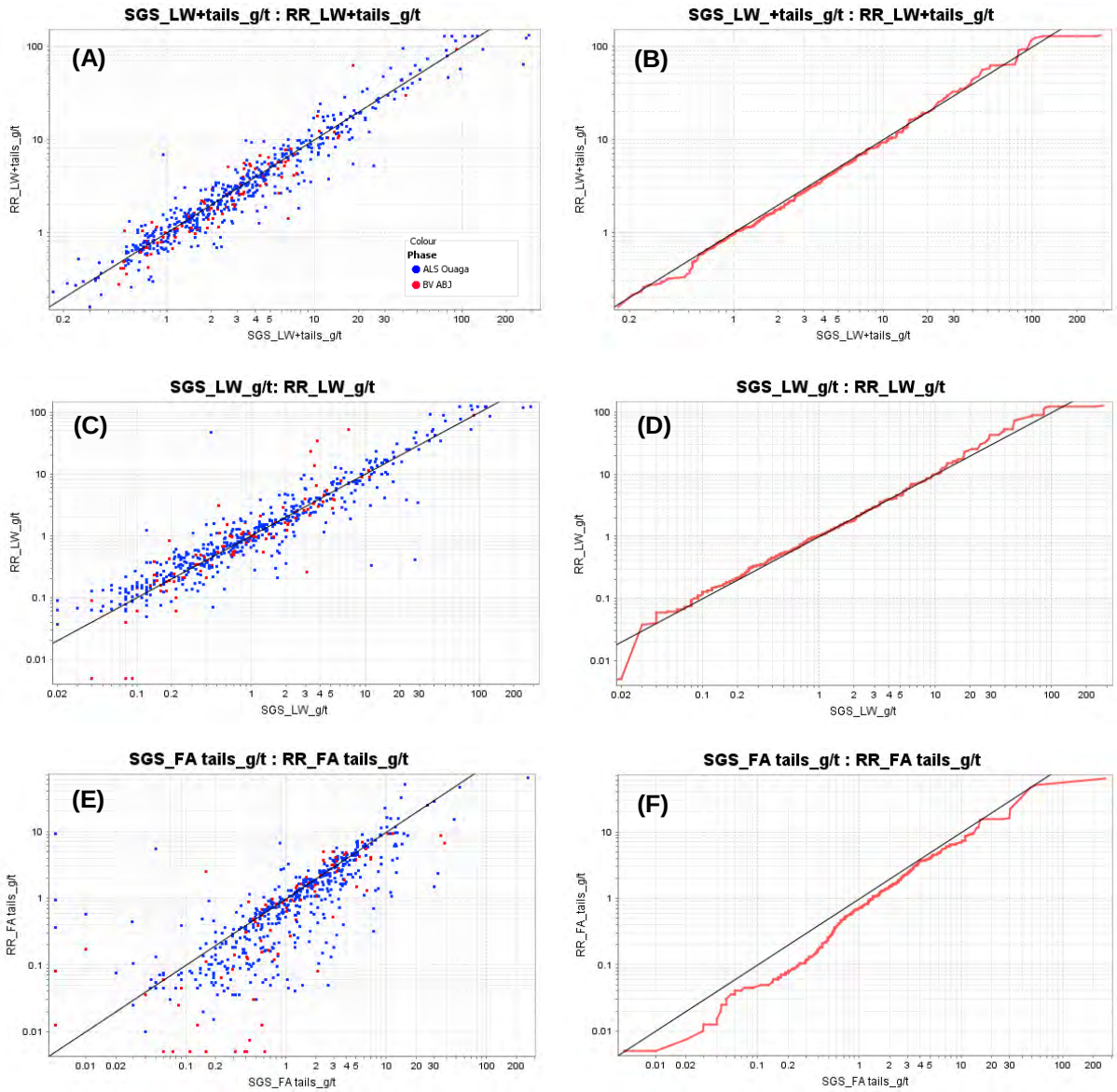


Figure 11-14 Scatter Plots (Left) and QQ plots (Right) Showing the Feasibility LeachWELL Round Robin Results for the Central Zone

A to B show comparison of the total grades (solution plus the tails). C to D display the LeachWELL solution grades. E to F show the FA tail grades. Original SGS data is displayed along the x-axis, and the repeat testwork along the y-axis.
Block G results are not shown in plots C to F as the original LeachWELL parameters are slightly different to the repeat test work (1 vs 5 LeachWELL tablets) for this campaign. Black line on all plots denotes the 1:1 line.

Table 11-11 Summary Statistics (Mean, Median) for the Six Round Robin Feasibility Campaigns

Average	Straight FA_g/t		Variance		LW_g/t		Variance		FA tails_g/t		Variance		LW+FA tails_g/t		Variance
	SGS	RR	%		SGS	RR	%		SGS	RR	%		SGS	RR	%
Block G	7.93	8.72	10.1										8.30	9.12	9.9
PP2	8.21	8.53	3.9		7.89	7.96	0.9		2.99	2.15	-28.1		10.88	10.11	-7.1
PP3	7.98	7.08	-11.4		6.19	5.54	-10.5		2.53	1.86	-26.6		8.72	7.40	-15.2
PP4					2.53	3.58	41.5		2.69	1.78	-33.8		5.22	5.76	10.3
Block K/L					2.36	2.21	-6.4		2.83	3.01	6.4		5.13	5.22	1.8
Block 1/2R					8.71	7.98	-8.4		5.78	2.80	-51.6		14.50	10.78	-25.7
Total					**5.63**	**5.82**	**3.4**		**3.15**	**2.20**	**-30.2**		**8.79**	**8.02**	**-8.8**

Median	Straight FA_g/t		Variance		LW_g/t		Variance		FA tails_g/t		Variance		LW+FA tails_g/t		Variance
	SGS	RR	%		SGS	RR	%		SGS	RR	%		SGS	RR	%
Block G	2.46	2.75	11.8										2.82	2.68	-5.0
PP2	2.99	2.91	-2.7		0.88	1.01	15.7		1.05	0.69	-34.5		2.92	2.97	1.7
PP3	2.46	2.27	-7.5		1.14	1.03	-9.3		1.27	0.95	-24.7		2.86	2.18	-23.5
PP4					0.68	0.80	17.6		1.06	0.86	-18.9		2.54	2.09	-17.7
Block K/L					0.44	0.52	18.2		1.35	1.33	-1.1		2.43	2.52	3.7
Block 1/2R					0.86	0.82	-4.7		1.60	0.84	-47.5		2.96	2.68	-9.5
Total					**0.82**	**0.91**	**11.0**		**1.26**	**0.92**	**-27.0**		**2.75**	**2.54**	**-7.6**

LeachWELL solution grades and tail assays for Block G are not compared in this table as the leaching parameters for this campaign were slightly different between SGS and ALS (original Block G assays collected before the change in LeachWELL procedure outlined in the opening paragraph of the feasibility round robin section).
PP4 round robin analysis was carried out at BV Abidjan, all other round robin campaigns were completed at ALS Ouagadougou.
RR – round robin.

Some of the variability shown in Figure 11-57 and Table 11-11 can be explained by the sample bias; in particular, PP3 is located in the highest grade (and most nuggety) part of the CZ. However, an apparent over reporting of the tail grades was flagged to SGS in early 2018. Discussions with SGS technical staff, other laboratories, and external laboratory audits highlighted that excessive standing time (>8 hours) between the end of rolling and filtration of the residues, and poor washing and filtration of the tails were possible causes. In March 2018, the LeachWELL procedure was updated (LWL69M5RG) and lower batch volumes initiated so SGS Ouagadougou could filter the residues thoroughly within six to eight hours after agitation has ceased and the aliquot of solution taken for AAS analysis.

11.5.3. Fire Assay Umpire Analysis

An umpire assaying programme was previously completed on a significant drill programme in 2010. This umpire analysis tested the SGS Loulo and SGS Kayes (Mali) laboratories against the OMAC Laboratory, Ireland. These results indicated that the OMAC umpire results returned higher grades than those of SGS Loulo and SGS Kayes, although they were impacted by one significant outlier. Once the outlier was removed, the results were significantly closer overall. A slight negative bias was outlined by Randgold against the SGS Loulo and SGS Kayes laboratory, however, both were considered as being acceptable, with the SGS Loulo laboratory noted for its higher precision. The more recent round-robin LeachWELL analysis has provided confidence in the general SGS Ouagadougou laboratory procedures and analysis in light of any umpire analysis.

11.6. Sample Security

Labelled samples are placed into large bags (10 at a time) and these are sealed. The samples are placed in a crate which are transported to the warehouse and trucked to the relevant laboratory by Randgold personnel.

All laboratory sample backlogs are actively monitored on a weekly basis and the number of samples dispatched is adjusted accordingly to ensure that the laboratory backlog does not become more than one month's sample processing capability.

In the instance that the laboratory backlog does exceed that of one week's sample processing capability, all samples are retained onsite in Massawa in a secured locked container until they can be dispatched.

Results from all laboratories are emailed to a project email group and are later imported into the database by the Database Administrator. A paper certificate is mailed at a later date.

Until the end of 2016 all project data had been stored in MS Access databases. During the first quarter of 2017 all project data has been migrated and secured in industry standard Maxwell Datashed SQL database for optimal validation through constraints, library tables, triggers, and stored procedures. All site software application databases will be set up to link back to the main database for information retrieval via and Open Database Connectivity (ODBC) link.

11.7. Audit

A full external audit of the sampling methods and procedures was undertaken by Roscoe Postle Associates Inc. (RPA) in 2017. RPA did not identify any material issues. Two high priority issues were identified, the collection of bulk densities which did not follow Randgold standard operating procedures (SOP) from other sites, and laboratory certification and selection with regard to LeachWELL analysis. Bulk density sampling has been updated to meet the same standard as at other Barrick Africa and Middle East sites. LeachWELL certification does not currently exist, however, Randgold undertook a programme of laboratory round robin assaying using its current assaying procedures and 1 kg custom OREAS CRM standards which indicated there to be no issue with repeatability of these samples. All other minor 'housekeeping' issues outlined in the RPA audit have been reviewed and acted upon where required.

In the QP's opinion, the sample preparation, analysis, and security procedures at the Project are adequate for use in the estimation of Mineral Resources. The QA/QC programmes as designed and implemented by Randgold are adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.

12. Data Verification

All QPs authoring this technical report have been directly involved with the development of the Massawa Project since 2012. Consequently, all QPs have been directly responsible for reviewing all forms of Project data used for the estimation of Mineral Resources and Reserves.

The Massawa Mineral Resource databases are considered to be appropriate to be used for the estimation of Mineral Resources and Ore Reserves. The inclusion of sample data that is negatively biased within the Massawa Mineral Resource is considered to provide an acceptable level of conservatism within the estimation of the Mineral Resources and Ore Reserves, and economic evaluation.

The Sofia Mineral Resource databases are considered to be appropriate to be used for the estimation of Mineral Resources and Ore Reserves.

In April 2017, a dedicated Massawa based database administrator was appointed who is overseen by the Loulo regional database administrator.

Up until Q1 2017, logging took place on paper logging sheets with the geologist later transcribing the data into a custom Microsoft (MS) Excel template that contained fixed drop-down options to reduce human error. Once completed the Loulo database manager was responsible for verifying the data and uploading it to the MS Access database. For this Mineral Resource estimate, a selection of assay values in the database were checked against the original assay certificates and no errors were observed.

Since the commencement of the feasibility study in Q1 2017, all logging has been undertaken directly into Maxwell LogChief running on Dell Tough Tablets to help reduce transcription errors. LogChief also includes many cross-validation tools to ensure that the data is correct. The data is automatically uploaded from LogChief to a buffer table pending Senior Geologist's approval prior to uploading to the central database.

During Q1 2017, all Project data was migrated and secured in industry standard Maxwell Datashed SQL database for optimal validation through constraints, library tables, triggers, and stored procedures. All site software application databases will be set up to link back to the main database for information retrieval via Open Database Connectivity (ODBC).

During the migration to SQL database, initially all assay data was migrated from the access database. Subsequently, all assay data has been re-imported directly from assay certificates from the laboratory and ranked such that they will have a higher priority than the MS Access imported data. Since the upgrade to Datashed, all assay values are directly imported from the raw laboratory certificates.

In the QP's opinion, database verification procedures for the Project are adequate for the purposes of Mineral Resource estimation.

13. Mineral Processing and Metallurgical Testing

13.1. Summary of Metallurgical Testing Programmes

The Massawa concession currently holds three major gold deposits. These are the Massawa (CZ and NZ), Sofia (Sofia Main and North), and Delya deposits. There are several other satellite deposits within the permit area.

Free-milling ore occurs in Sofia and most of CZ, and there is refractory mineralisation in the northern part of CZ as well as NZ and Delya. The refractory ores have been proven to be highly recoverable through BIOX as an oxidative step.

The CZ and the refractory ore contain deleterious elements such as arsenic (As) and antimony (Sb). Where the CZ is not processed through BIOX, testwork has proven that the As and Sb can be removed successfully through a precipitation plant designed by Multotec. During the refractory processing phase, the neutralisation section of the BIOX plant will remove the precipitate and these elements. As an additional mitigation, the Multotec precipitation plant will process any liquor that is released from the tailings storage facility (TSF) to the environment.

There have been a number of testwork programmes completed on Massawa ores. Some testwork was performed to help validate the results of historic studies. A summary of the testwork to date can be found in Table 13-1, Table 13-2, and Table 13-3.

Table 13-1 Scoping Phase Campaign Summaries

Scoping Campaigns				
Campaign	**Testwork Description**	**Laboratory**	**Report ID or Number**	**High Level Outcomes**
Campaign 0 - 2008	Central and Northern Zone bottle roll tests.	Loulo metallurgical laboratory	Loulo lab Excel sheets	CZ Leach Au dissolution: Oxide 94 -97%, Transition 91.9 - 95.5%, Fresh high sulphide 18.7 - 25.9%
Campaign 1 - 2008	Central Zone high sulphide fresh gravity recovery (GRG), Extended-GRG modified diagnostic leach and mineralogical testwork.	SGS Randburg, Peacocke & Simpson (MPS)	Peacocke & Simpson - RANDGOLD/01/08 NOVEMBER 2008. SGS - Randgold Min 0608 -121&166, METY 08/Z37	High coarse gravity recoverable gold present, including large gold grains. EGRG at 80% - 75 µm (unpanned) approximately 47- 52% Au. Diagnostic leach CIL + Oxidative step for low grade and high grade can achieve 61% and 71% Au dissolution. This confirms high gravity component and refractory nature of gravity tails (gtails).
Campaign 2 - 2008	Extended-GRG (CZ). CZ and NZ composite float optimisation.	SGS Randburg, Peacocke & Simpson (MPS)	Peacocke & Simpson - Randgold 05-09 Massawa Sulphide EGRG report. SGS - Flotation report 08-321	EGRG (CZ high sulphide) 80% - 80 µm (unpanned) 31% Au. CZ + NZ composite float recovery 96.2%; 20.5% mass pull (MP); Tails grade 0.14 g/t.
Campaign 3 - 2009	Central and Northern Zone BBWi (150 & 106 µm) BRWi (1,180 µm) Ai.	SGS Randburg	SGS - PROP 09/006 rev 1 excel results	BBWi 106 µm CZ 13.6 (Phase 1) to 23.6 (phase 2) kWh/t. NZ 106 µm CZ 13 (phase 1) to 18.6 (phase 2) kWh/t. BBRWi 1,180 µm (phase 1) CZ = 12.5 and NZ = 12.3 kWh/t. Ai = 0.1 to 0.25.
Campaign 4 - 2009	Northern Zone carbonaceous leach and preg-robbing testwork.	SGS Randburg	SGS - CC92 (Prop 09/249 r1)	Low CIL recoveries. Less than 10% Au dissolution. Tests indicated preg robbing potential.
Campaign 5 - 2009/2010	Environmental Acid Base Accounting (ABA), Acid Rain Leach (ARLP) and column leach test on flotation tails from CZ and NZ (campaign 2).	SGS Environmental Services	SGS - ENV10 - 00019ABA and ARLP. SGS-ENV10-0068/94/110/137/155/173	CZ and NZ high acid generation potential. ARLP indicate potential over-limit tenors for arsenic, calcium, iron, magnesium, manganese, lead, sulphur, silica, and strontium. Flotation tails leach tests indicate high arsenic and Iron levels over 76 weeks.
Campaign 6 - 2009	Central and Northern Zone flotation and leach OX testwork.	SGS Randburg. Maelgwyn Mineral Services Africa (MMSA)	SGS - MET 09/CC65. SGS - MET 09-277. MMS - REPORT No. 09/12	Float = CZ 86.7 % Au, 14% MP, 0.53 g/t Tails. NZ 91.4 % Au, 16% MP, 0.57 g/t Tails. Float Conc CIL 6.1% and 5.8% for CZ and NZ respectively. Leach-OX (UFG) 28.6% and 29.9% for CZ and NZ respectively. Leach-OX not effective. Diagnostic leach on float conc +UFG + Oxidative CZ = 36.7% and NZ = 35.4%.

Table 13-2 Pre-Feasibility Campaign Summaries

Pre-feasibility Campaigns				
Campaign	**Testwork Description**	**Laboratory**	**Report ID or Number**	**High Level Outcomes**
Campaign 7 - 2010	Central and Northern Zone flotation, pressure oxidation (POX), roasting, High Shear Atomaer and Intense leach.	SGS Canada (POX), Roasting Hazen USA, High Shear Atomaer RSA, Intense leach Mintek	SGS - 12296-001. Hazenresearch Met report 11053. Mintek - Report no 5526	Diagnostic leach (ROM) CIL + Oxidative step - CZ = 15.4%, NZ = 16.21%. Graphite pre-float: CZ = 12% NZ = 4% Au. Sulphide float: CZ = 84% NZ = 87.7% Au. CIL on conc: CZ = 12.9% NZ = 8.4%. Results on sulphide conc: POX: CZ = 98.5% NZ = 92.6%. Roast CZ = 79.9% NZ = 80.6%. HSR Aachen (10µm): CZ = 23% NZ = 14.2%. Float tails: CZ = 39.5% NZ = 14.2%. POX out performed other technologies.
Campaign 8 - 2010 /2011	Central and Northern Zone. Bulk gravity, flotation, POX, roasting, High Shear Atomaer RSA and intense leach. BIOX amenability.	SGS Canada (POX), Roasting Hazen USA, High Shear Atomaer RSA, Intense leach Mintek. BIOX Gold fields	SGS -Massawa draft report 10-230 campaign 8. SGS - Canada - Report 12296-002. Hazenresearch Report 11053. MMS REP 10-34 Massawa. Goldfields BIOX - Report 11_02	Gravity 50% - 75 µm: CZ=20% ROM Float = 93.6% Gtails float = 89.2%. POX: CZ (ROM float) = 99%, Gtails Float = 98.8%. Leach-OX (Aachen 10 µm) = 39%. BIOX = 94% ROM, 94.1% Gtails. Roast: 72% ROM, 67% Gtails. Gravity 50% - 75 µm: NZ=14% ROM Float = 93.8% Gravity Tails float = 95%. POX: NZ (ROM float) = 90.2%, Gtails Float = 85.4%. Leach-OX (Aachen 10 µm) = 39%. Roast: 77.7% ROM, 78.8% Gtails.
Campaign 9 -2010/2011	Central Zone bulk gravity, float optimisation and bulk POX and roast testwork. Thickening and rheology testwork.	Knelson Africa, MMSA, SGS Canada Africa Patterson and Cooke	MMS - MMSA PROPOSAL 10/30 Excel sheets. Knelson Africa EGRG 17-10 report. SGS Canada Proposal #100510 report and Excel results. Patterson and Cooke SEN-MAS-8216 R01 rev 1	Gravity 80% - 106µm CZ1= 31.5% and CZ2=35.2%. Float Rec 80% - 75µm: CZ1= 94.7% CZ2=96%. POX ROM conc: CZ1= 99% CZ2=97%, POX Gtails: CZ1=98% CZ2=98%. Roast ROM conc: CZ1= 89% CZ2=85.2%, POX Gtails: CZ1=93% CZ2=82.9%.
Campaign 10 - 2010	Central Zone gold deportment high sulphide.	AMTEL	AMTEL REPORTS 10-22 and 10-38	Central Zone high sulphide direct cyanidation is not an option for the Massawa gold ore. Since most of the sub-microscopic gold is in solid solution in sulphides, a pre-leach oxidation step is necessary. Given the good liberation of the sulphides and gold ROM rock at reasonable grinds (P80 = 120-130 µm) the recommended treatment option is flotation followed by BIOX or POX/CIL or UFG/Albion/CIL depending on the gold inventory.
Campaign 11 - 2010	Leach testwork on satellite pits. Bambaraya, Delya, Sofia and Kasara.	SGS Randburg	Excel results for Proposals 10-355 r2 and 10-264	CIL samples. Bambaraya 86%. Delya Oxide = 87%, Delya Fresh = 16.3%. Kawsara Fresh = 85%. Sofia Fresh 89.5%
Campaign 12 - 2011	Comminution on Central Zone, Northern Zone, Delya and Sofia	MINTEK RSA	Mintek - Massawa report 2011 - Project nr MPE - 57	BBWi (kWh/t) 106 µm: CZ=16.34 to 22. NZ =19.6 to 21.4. Delya=19.9, Sofia=18.88. BBWi (kWh/t) 150 µm: CZ=18 to 22.2. NZ =20.5 to 21.8. Delya=19, Sofia=19.8. Ai 0.07 to 0.34.

Pre-feasibility Campaigns				
Campaign	**Testwork Description**	**Laboratory**	**Report ID or Number**	**High Level Outcomes**
Campaign 13 - 2011	Mineralogy and chemical composition - Central Zone, Northern Zone and Soft Sulphides LE.	SGS Booysens	SGS report MIN 1010/181	Results used in geological models.
Campaign 14 - 2010	Gambia River water analysis.	SGS Environmental Services	SGS report ENV 2010-205	Water analysis from Gambia: Fe = 8.1 mg/L. Limited water analysis indicates some elements are above IFC/WHO standards.
Campaign 15 - 2011	Limestone analyses from Senegal and Mali.	SGS Booysens	SGS project nr. 10/262 r2	Local sourced limestone can be considered for use in the process plant.
Campaign 17 - 2014	Flotation optimisation on Northern Zone.	Hazen Research USA	Campaign 17 - Report - Massawa Final Flotation Report September 2014rev1Sept 24, 2014	Float results indicate >95% Au recovery, >93% As recovery, >4.9% CO_3 and >97% St. Adequate Fe/As ratios for arsenic precipitation. S_2^-/CO_3 ratios suitable to minimise gold chloride complex formation and precipitation of elemental Au onto organic carbon.
Campaign 18 - 2015 - 2018	Sofia Gravity and leach testwork. Sofia comminution.	Morila Metallurgical Lab and SGS Randburg	Morila Laboratory report in excel. SGS Randgold Sofia North Comminution Report 17_454	Sofia Fresh Gravity (1.7 mm) = 6.1%. Sofia Gtails CIL = 87.4%. Sofia Main = 89.1%. Sofia Oxide = 92.3%. Sofia Ox-Tr = 91.7%. Sofia Main BBWi (kWh/t) = 16.4 to 19.2, SMC Axb = 37 to 61, CWi= 7.2 to 18.9. Ai = 0.1 to 0.2.
Campaign 19 2015/2016	PFS Central Zone gravity, flotation and hydrometallurgical testwork. Composites and variability.	SGS Randburg	SGS Reports 15_622 (Block A), 16_092 (Block A and B), 16_505 (Block C) and 16_550 (Block D)	Modelled Gravity: Blocks A:49%, B=44%, C=42%, D=16%. Gravity + Gtails CIL Blocks: A=71%, B=72%, C=77%, D=70%. Gravity + Float Conc CIL (UFG 15 micron) + Float tails CIL blocks: A=81%, B =83%, C=80%, D=75%. Excess NaCN levels on all tests.

Table 13-3 Feasibility Campaign Summaries

Feasibility Campaigns				
Campaign	**Testwork Description**	**Laboratory**	**Report ID or Number**	**High Level Outcomes**
Campaign 20 - 2017/2018	Feasibility Central Zone Gravity, Flotation and Hydrometallurgical testwork. Comminution testwork. Gravity and flotation concentrate, gold deportment concentrate	SGS Randburg	SGS - 17/165 Revision 6. SGS 17-549	Gravity 212µm Blocks: G=25%, A=48%, B=40%, C=32%, D=8% ABCG= 34%. Gravity + Gtails CIL Blocks: G=50%, A=75%, B=70%, C=85.2%, D=22.9%, ABCG=65%. Flotation recovery blocks: G=90.6%, A=83.8%, B=89%, C=90.8%, D=92.1%. BBWi 106 µm (kWh/t) blocks: G=29.1, A=24.3, B=30.5, C=22.2, D=24.4. Hardness overstated due to sampling. Optimum leach conditions CIL 32 hrs, 600 ppm NaCN controlled.

Feasibility Campaigns				
Campaign	**Testwork Description**	**Laboratory**	**Report ID or Number**	**High Level Outcomes**
Campaign 21 -2017/2018	Sofia Main and North, comminution, gravity recovery. Hydrometallurgical testwork. Gold deportment studies.	SGS Randburg	SGS - 17/166 Revision 1 (Sofia Main). SGS 17/454 Revision 1 (Sofia North)	Gravity 212µm: Sofia Main Met 01 to 08 = 5.1%, Composite (Main) =6.2%. Sofia North SMET01 - 06=6.5%, Composite (North) =5.2%. Gravity + Gtails CIL: Sofia Main=88%, Sofia North=85%. Modified Diagnostic CIL + Oxidation step: Sofia Main=90.6%, Sofia North=87.6%. Further Aachen Oxidation testwork in Campaign 26. Optimal CIL: 32 hrs NaCN 600 ppm controlled. BBWi 106 µm (kWh/t): Sofia Main= 18, North= 16.5.
Campaign 22 - 2016 to 2018	Flotation optimisation on Northern Zone and Extension. Further flotation optimisation on NZ. Bulk flotation for BIOX campaign 1 and 2.	Hazen Research (USA)	Hazen Research report 12180 (Float optimisation), Campaign 1 and Campaign 2 report 12386	Optimal float regime: 150 g/t $CuSO_4$, 100 g/t Pax, 60 g/t Aero 404, W22C as required. Campaign 1 Conc generation =91.2%, Campaign 1 Variability = 87%, Campaign 2 Bulk float = 90.8% and Campaign 2 lead 300 kg = 88.2%. Average MP = 7.92%
Campaign 23 - 2016 to 2018	BIOX BAT Hazen Research samples project 12180. Pilot and BAT testwork on Campaign 1 and 2 from Hazen Research report 12386	SGS BIOX Pilot plant	Outotec (BAT) - BIOX® BAT Testwork Programme on Massawa Samples September 14, 2016. Outotec Massawa NZ BIOX Pilot Plant Run 1 (Campaign 23) and Massawa NZ BIOX Pilot Plant Run 2 (Campaign 23)	All refractory float concentrate from CZ, NZ and Delya is highly reactive with BIOX. Float recoveries from campaign 20 and 22 as reference. Float + BIOX Bat: CZ Block D=87.5%. BIOX Pilot Campaign 1: NZ=88.6%, Campaign 2 Pilot run: NZLG=88.3%, NZHG=88%, CZ/NZ blend=87.7%. Neutralisation highly efficient. As and Sb below IFC/WHO standards. Optimal reagent addition and dosing was determined.
Campaign 26 - 2017/2018	Aachen oxidation leach testwork on Central Zone, Sofia, and Sofia North. Pilot plant gravity and CIL testwork on Central Zone PP1 to PP4.	Maelgwyn Mineral Services Africa	CZ Block L Rep 18-051. CIL Pilot plant report 17-041. Sofia Main and North Excel results 2018	Aachen CIL CZ Block L Fresh=37%, Transition-Red=40.6%, Transition-Ox=72%, Oxide=87.6%. Aachen CIL (CIL SGS). Sofia Main = 93% (88.4%), Sofia North= 87.9% (85.3). Pilot Plant CZPP1 to CZPP4. Gravity: PP1= 50.2%, PP2=53.7%, PP3=57.9%, PP4=15.2%. Overall Gravity+Gtails CIL: PP1=83.3%, PP2=85.8%, PP3=82.6%, PP4=37.6%.
Campaign 27 - 2017/2018	Slurry behaviour and thickening testwork. On oxides, Sofia, Central Zone (pre- and post-leach) and Northern Zone BIOX samples (neutralised product and BIOX product)	Slurrytec Vietti	Reports SEN-MAS-8584	Oxide composite pre- and post-leach samples require 60-80 g/t flocculant and achieve solids flux rates of 0.25 $t/m^2.h$ to 0.4 $t/m^2.h$. Sofia pre- and post-CIL require 30 g/t to 60 g/t flocculant, and solids flux rates of 0.45 to 0.5 $t/m^2.h$. CZ WOL pre- and post-CIL require 40-50 g/t flocculant, and solids flux rates of 0.4 $t/m^2.h$ to 0.5 $t/m^2.h$. BIOX pre-CIL product 200 g/t flocculant, and a solids flux rate of 0.3 $t/m^2.h$.
Campaign 28 - 2017/2018	Bulk solid flow testwork on oxides from CZ, NZ, and Sofia Main.	Greentechnical	Report A786	For the oxide samples tested, for ROM, Apron feeders are best, for post-crushed, belt feeders are optimal.

Feasibility Campaigns				
Campaign	**Testwork Description**	**Laboratory**	**Report ID or Number**	**High Level Outcomes**
Campaign 29 - 2018	Cyanide destruction testwork on oxides, Sofia Main and North as well as Central Zone.	Cyplus	Report Pending. Results emailed pdf document email dated 27 November.	Cyanide destruction methods tested, INCO, Cold Caro's Acid and Combinox. Results pending on oxide, and Sofia Caro's and Inco very effective. For CZ SCN- is produced. Caro's acid destroys SCN-, Inco product left with 580 ppm SNC- post destruction.
Campaign 31 - 2018	Sofia Waste rock comminution testwork.	SGS Randburg		BBWi (kWh/t) 106 µm: 22.2. BRWi (kWh/t) 1180µm: 17.8. Ai 0.30. Waste is not considerable harder than Sofia Main.
Campaign 32 - 2018	Oxide composites Leach Kinetics. Composites from CZ, NZ, Sofia North, Sofia Main and Delya.	SGS Randburg	Excel results - 17/463 Rev 3	Optimal CIL determined to be 24hrs, 300ppm NaCN controlled. Gravity at 212 µm: CZ=48.5%, NZ=15.1%, Sofia Main=4.3%, Sofia North=8.4%, Delya=0.9%. ROM CIL: CZ=97.8%, NZ=90.4%, Sofia Main=91.4%%, Sofia North=88.9%%, Delya=96.5%.
Campaign 33 - 2018	Albion testwork.	Glencore Technologies	Report TA070-GT-DD5001	Flotation concentrate from Campaign 22 was UFG'ed to 10 µm and Albion applied. Only an 80.2% overall oxidation+CIL was achieved. Nor further work was completed as BIOX achieves > 97% on the same samples.
Campaign 34 - 2017	Orway Minerals 2 stream mill sizing and simulations.	Orway Mineral Consultants	Report 7860	Simulations and sizing based on running two parallel streams early in the BFS (single stream, of 150 tph to process CZ alone). Simulations indicated that 1 x Primary Jaw+1 x Secondary + 1 x 6.0 m ø 9 m effective grinding length (EGL). Mill can produce a 150 tph 80% - 75 µm product.
Campaign 36 - 2017/2018	Delya gravity and flotation optimisation. Concentrate through BIOX BAT test reported in campaign 23.	SGS Randburg	Gravity and Flotation report SGS 17/642	Delya gravity 212 µm = 2%. Float recoveries average = 96.2%, BIOX-CIL=95.5%. Overall recovery = 91.8%
Campaign 37	Multotec testwork on arsenic and antimony removal from simulated TSF liquor.	Multotec South Africa	Report Randgold Massawa Arsenic and Antimony Precipitation Benchtop Testwork WRQ053	Central Zone free-milling arsenic, antimony, and copper levels of 14.5 ppm, 52 ppm, and 13.1ppm, respectively, were leached into solution. Applying a 3-stage ferric chloride precipitation all elements can be precipitated and meet the stringent IFC/WHO standard for water release to the environment.

During the PFS and FS phases, several trade-off studies were completed. Process specific trade-off studies were generated and used as part of process development, and ultimately the design of the plant.

Table 13-4 Pre-feasibility and Feasibility Trade-off Summaries

Trade-off Title	Description	Outcome
PRESSURE OXIDATION VERSUS BACTERIAL OXIDATION - Memorandum dated 02 March 2017	High level assessment of POX vs BIOX using PFS assumptions.	POX and BIOX achieve the same levels of oxidation + CIL. The capital cost and associated operating cost indicate that BIOX is more cost effective to operate. Further risk of a POX unit in remote areas in Senegal is also highlighted as a risk.
POWER SUPPLY TRADE-OFF - Document Number SS 0655-0000-0W11-001 dated 21 April 2017	A high level trade-off exercise was completed to assess the possibility of using liquefied petroleum gas (LPG) instead of heavy fuel oil (HFO) for power generation for the Massawa Project.	The evaluation showed that the use of HFO or a combination of HFO and diesel results in greater economic benefits than the use of LGP power generators.
PROCESS ROUTE SELECTION TRADE-OFF STUDY - Document Number SS0672-0000-0W11-003 Rev 0B dated 25 June 2017	Determine optimal process routes for various ore types found at Massawa. Memorandum was based mostly on PFS results. Feasibility results have changed the optimal process route for Central Zone.	WOL at 75 µm is optimal for free milling ore. PFS assumptions assumed flotation of CZ and UFG on conc, as well as a leach of float tails (this is disproven in feasibility). NZ to include BIOX, leaching of float tails not optimal.
FLOWSHEET DEVELOPMENT AND VERIFICATION – EFFECT OF GRIND – MASSAWA CENTRAL ZONE - Memo dated 3 August 2017	To proceed with the CIL optimisation testwork, a preferred grind for the gravity tailings that will be leached is required. The decision to conduct such testwork will be based on the first set of data generated from Campaign 20.	All optimization testwork will thus be conducted at a grind of P80 of 75µm.
MASSAWA SOFIA FLOW SHEET DEVELOPMENT - EFFECT OF CYANIDE ADDITION - Memo dated 08 August 2017	Sofia ore was tested at 300, 400 and 600 ppm Controlled NaCN levels. An additional 3.5% dissolution is achieved at 600 vs 300 ppm. A financial impact was assessed based on higher dissolution and in turn higher NaCN consumption.	Leaching of Sofia ores at a controlled 600 ppm NaCN accepted as optimal.
ELUTION METHODS TRADE-OFF - Document Number SS 0672-0000-0W11-005 Rev 0C dated 15 August 2017	SENET completed a trade-off study for four elution options, Split Anglo American Research Laboratory (SpAARL), Straight Anglo American Research Laboratory (StAARL), the standard pressure ZADRA, and a short cycle (12h) Pressure ZADRA elution methods at the request of Randgold. The trade-off study was based on 12 ton carbon batch capacity for all four elution methods.	The two Pressure ZADRA methods have lower capital and operating costs when compared to the two AARL options. The short cycle Pressure ZADRA option presents the most optimal option for Massawa.
MASSAWA FLOWSHEET DEVELOPMENT – CONCENTRATE HANDLING (ALBION) - Memo dated 30 October 2017	Albion testwork was conducted on the NZ flotation concentrate produced at Hazen.	Even though the BIOX process showed higher operating and capital costs, gold recovery and contribution to operating margin were higher for the BIOX process compared to the Albion process.
CZ PROCESS ROUTE TRADE-OFF STUDY PFS RECOVERIES - SS 0672-0000-0W11-004 Rev 0C dated 12 February 2018	Three process routes for the CZ were identified from the PFS and current on-going testwork programme indicate that the proposed treatment route would require a trade-off study to identify the most suitable processing route before the BFS deliverables can commence.	The WOL of CZ is financially viable down to overall Au recoveries of 67%. This is measured against CZ being processed as a CZ and NZ blend, to lower operating cost. This necessitates the blending of 25% CZ and 75% NZ during flotation and utilises flotation tails recycling in the neutralisation stage.

MASSAWA CIL CIRCUIT CONFIGURATION - Memo dated 6 July 2018	Recently SENET was requested by Randgold to revaluate the proposed circuit and to develop a more simplistic flow sheet that would be able to treat both ore types in a single CIL circuit to reduce capital costs.	To conclude the 6 stage adsorption at 2500 m^3 each can bring down the dissolved gold losses to below the required 0.010 mg/L Au for the flotation concentrates (CZ and NZ). The only disadvantage is doubling up the carbon inventory. The elution batch size of 12 tons is suitable for all the scenarios.
MASSAWA SECONDARY CRUSHED ORE STORAGE TRADE-OFF - Doc nr SS 0672-0000-0W11-009 Rev 0A dated 19 July 2018	Determine if a 4,000 t live capacity stock pile is advantageous over a 500 t overflow bin with an emergency stock pile and reclaim facility. Trade-off based on cost advantages and Mineral Resources management of storage facilities by Randgold.	Financially the 500 t bin with over flow stockpile and emergency reclamation. Grade control is managed satisfactory.
MASSAWA OXYGEN PLANT RENTAL OR OWNER BOUGHT COMPARSION - SS 0672-0000-0W11-010 dated 17 August 2018	High level cost trade-off study to determine the life cycle cost of renting a 20 tpd oxygen plant compared to buying a unit and maintaining it as the owner.	The average LOM cost based on 24 Mt over 10 years with no escalation of discount is substantially less. If the plant is bought the operating cost is $0.14/t ore and owner operated $0.39/t ore. The operating cost for the owner bought plant is 60% less when compared to the rental option.

13.2. Summary of Results

The extensive metallurgical testwork campaigns demonstrate two distinct behavioural patterns. The first includes ore sources, in particular oxides as well as some fresh rock sulphides from Sofia Main, Sofia North and most of CZ, that exhibit free-milling characteristics suitable for gold extraction by a conventional CIL metallurgical process. These ores also contain gravity recoverable gold and, in CZ, GRG is as high as 50% of the gold.

The second ore type is highly refractory, with very low gravity recoverable gold, and is found in the north of the CZ pit, in the NZ pit, and Delya. Flotation and BIOX are combined to achieve overall recoveries in excess of 85% for this ore type. The flotation tailings grades achieved were below 0.4 g/t Au and exhibited very low CIL recoveries.

Based on the above, the Massawa plant design consists of two distinct processing circuits which will run sequentially:

- For the first seven years, the free-milling ore sources will run through a conventional gravity and CIL circuit, which will include an arsenic and antimony precipitation processing plant before the liquor is released from the TSF to the environment.
- From year 7 and onwards, the refractory ore sources are processed through a flotation circuit with a concentrate fine grinding step to P80 = 45 µm, followed by the BIOX step and CIL.

Sampling for testwork was completed by the geologists in conjunction with the modelled and simulated pits and mine plans. The samples were selected as to ensure both spatial and geological representativity.

Figure 13-1 shows the metallurgical sample location at Sofia North as an example.

Looking East

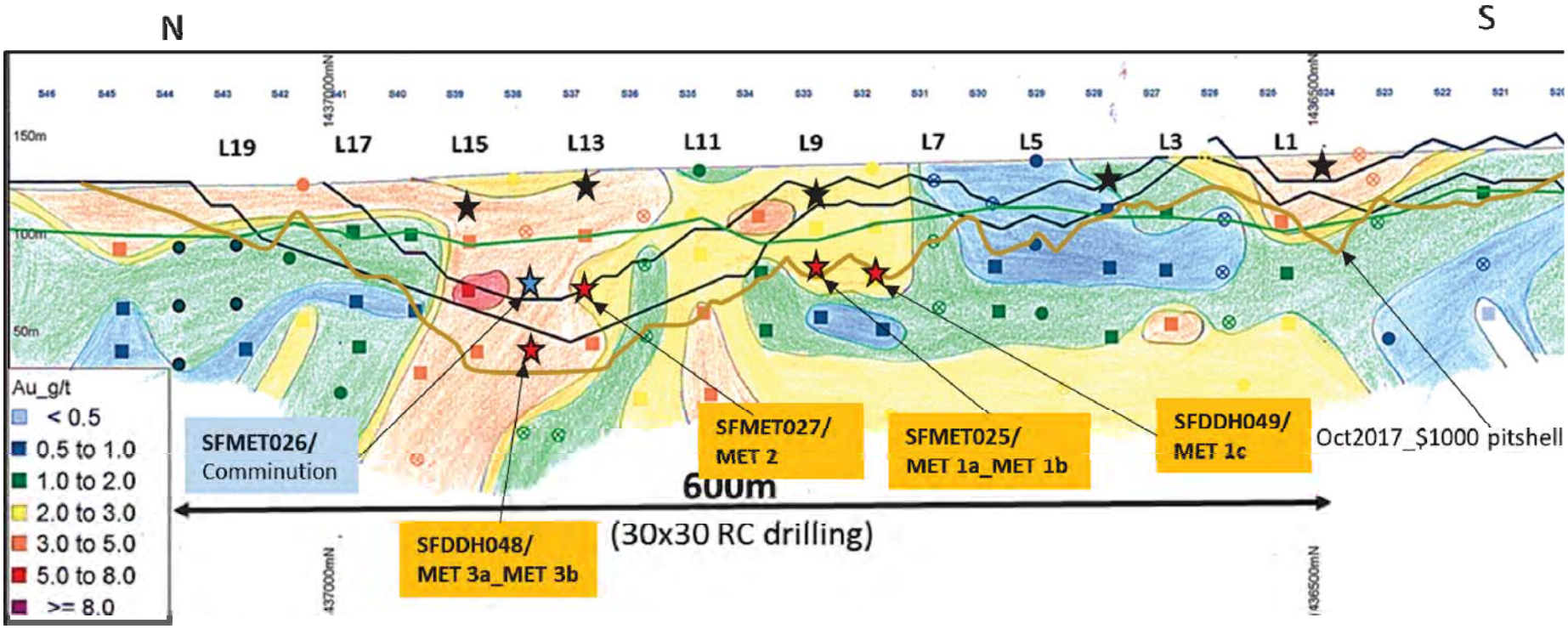


Figure 13-1

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Sofia North Metallurgical Sample Location

Legend:

- Drilled sulphide met
- Drilled oxide met/density holes
- Drilled Comminution

0 50 100 150 200 250

Metres

July 2019

Source: Randgold, 2018.

The samples selected were based on geological classification, grade, as well as representivity of the ore being tested. Scouting tests were then undertaken on a composite sample to determine optimum operating conditions in terms of grind, residence time and reagent additions. After this optimisation exercise, several samples were grouped to make up a series of variability sample tests, which were then tested at the afore-determined optimum conditions.

This approach ensured that the testwork completed was representative and that the optimal results could be incorporated with high confidence into the design criteria, and ultimately the plant design and sizing.

13.3. Recovery

The gold recovery achieved from the free-milling ores is highly dependent on the grade processed through the circuit. The relationship indicates that the lower the grade, the lower the overall recovery of the ore.

For example, 41 CIL results were available on the Sofia Main ROM leach pad with varied head grades. To determine the optimal recovery based on grade, the gold dissolution results were used to plot a scatter graph of head grade against tails grade distribution. A best fit linear curve was drawn to give predicted tails grade from head grade values.

The following tails–head grade relationship was established from the set of results, in the form of a typical linear equation.

$T = (M \times H) + C$

Where T = tails grade (g/t Au)
H = head grade (g/t Au)
M = slope
C = constant

Using the above relationship, predicted tails were calculated for each head grade and the corresponding gold dissolution was determined. The predicted gold dissolution for the given head assays allows predicting gold recovery based on its feed grade.

The head grade as determined by the mine plan is the average Sofia Main fresh head grade of 3.48 g/t Au. Applying the formula in Figure 13-2, the tails grade is calculated as T = 0.073 x (3.48) + 0.103 = 0.36 g/t Au. This value is applied to the head grade of 3.48 g/t Au, which results in a gold recovery of 89.7%.

This recovery of 89.7% was estimated for the ROM leach material but the same dataset also has a Gravity + CIL recovery, which was determined to be 88.7%. Therefore, the average gold recovery used for Sofia Main fresh material is 89%.

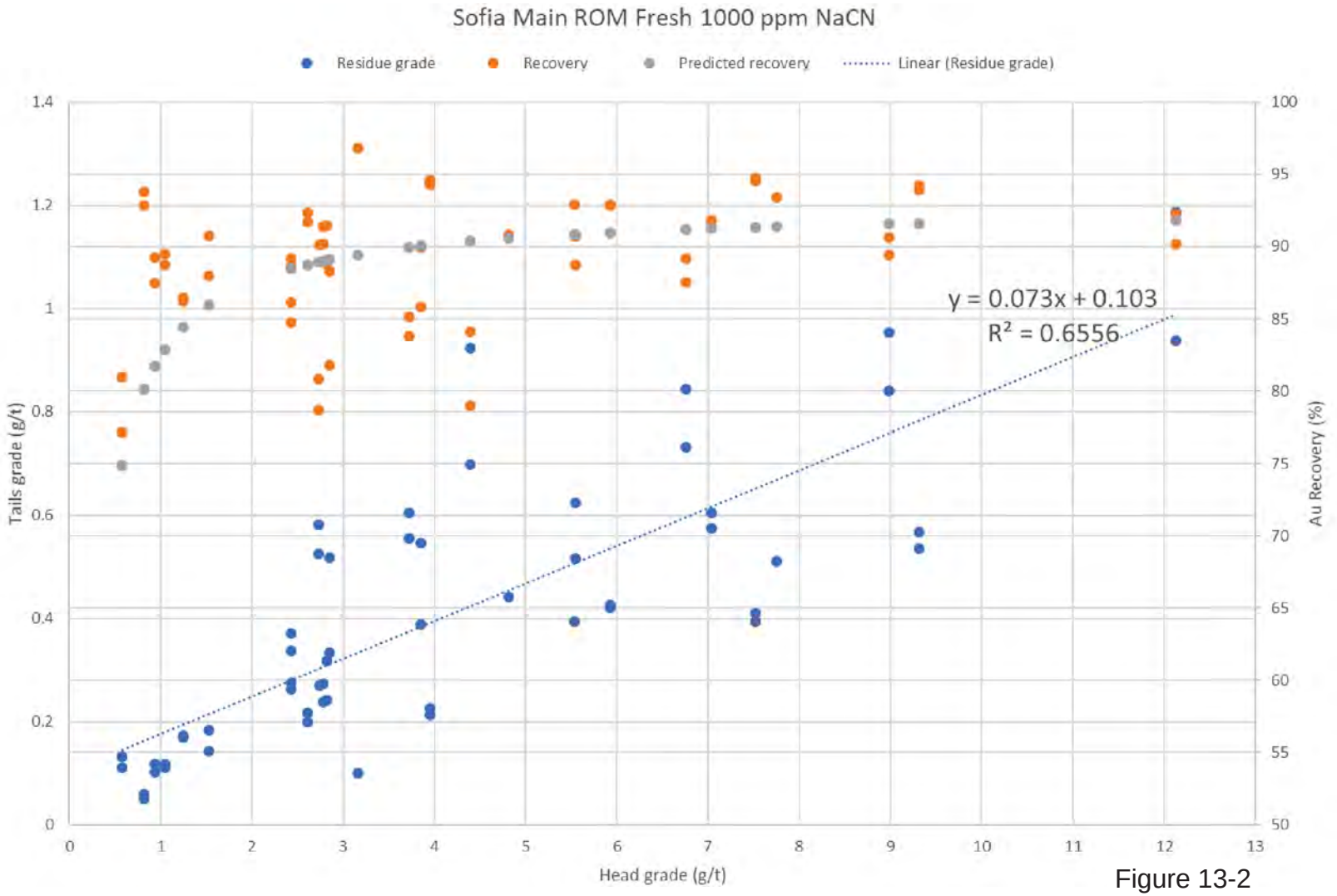


Figure 13-2

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Sofia Main Recovery Summary and Recovery Estimation Based on Head Grade

July 2019

Source: Randgold, 2018.

This method was applied to all datasets that had sufficient data points. If there were 15 or less data points, the straight average was used.

Table 13-5 summarises recoveries for each of the ore types and processing routes.

Table 13-5 Recoveries Summary per Ore Type and Weathering

WOL 2.7, 2.4 and BIOX 1.2 Mtpa		Central Zone			North Zone		Sofia Main	Sofia North	Delya		
		WOL	BIOX - 100% CZ	BIOX - 25% CZ / 75% NZ	WOL 100% NZ	BIOX 100% NZ	WOL	WOL	WOL 100% Delya	BIOX 100% Delya	BIOX 20% Delya 80% NZ
Recoveries	Oxide	94%	na	na	90%	na	92%	91%	94%	na	na
	Oxide Trans	94%	na	na	89%	na	92%	91%	90%	na	na
	Reduced Trans	Geomet	57%	57%	51%	57%	90%	89%	52%	57%	57%
	Fresh	Geomet	87%	88%	13%	88%	89%	85%	9%	92%	89%

13.4. Deleterious Elements

13.4.1. Cyanide

Massawa chooses to abide by the guidelines of the International Cyanide Management Code. Massawa will follow the requisite cyanide protocols. The TSF will be lined with an HDPE liner. Protocols call for limited threshold discharges to the TSF and cyanide discharge concentrations are controlled through use of an on-line cyanide analyser and controller. The current plant includes a cyanide destruction circuit before slurry is deposited on the TSF. Maximum water return from the TSF is employed, further mitigating release to the environment.

Further mitigation is included at the discharge end of the TSF, where a polishing cyanide destruction step can be included to ensure that legal requirements are met before release to the environment. During the refractory ore treatment, an additional ASTER cyanide destruction is added to destroy the thiocyanate to below the 7 ppm level. This might have to be installed earlier if further testwork indicates that thiocyanate might be building up in the circuit during CZ WOL processing.

13.4.2. Arsenic, Antimony and Copper

The CZ, NZ, and Delya ores contain arsenic, copper, and antimony. The NZ and Delya material is processed through a neutralisation step within the BIOX circuit of the main plant. Arsenic and other deleterious elements are to a large extent eradicated before any tailings are pumped to the TSF.

The higher risk during the LOM is posed by the CZ free-milling tailings for which testwork has shown arsenic, antimony, and copper levels of 14.5 ppm, 52 ppm, and 13.1 ppm, respectively. Multotec undertook confirmatory testwork to determine what type of process is required to remove and precipitate these deleterious elements. The testwork indicated that a three stage precipitation can remove these elements, with dosing 360 mg/L ferric chloride in the first stage,

100 mg/L in the second, and 50 mg/L in the third. The testwork has proven that all three deleterious elements can be precipitated and meet the stringent WHO/IFC standards.

When CZ fresh free-milling is processed, the arsenic precipitation plant designed by Multotec will be built. Any water released from the TSF to the diversion dam will be processed though this precipitation plant.

The pit dewatering and other waste dump run-off also indicated the presence of arsenic and antimony. These are at much lower levels and have been found at 0.15 mg/L As and 0.12 mg/L Sb concentrations. To remove these elements from the run-off, a simple greensand filtration system will be used, which is a simple adsorption option. The greensand will be replaced every two years. This approach is highly effective at these lower concentrations.

14. Mineral Resource Estimates

The Massawa Mineral Resources consist of Massawa CZ, Massawa NZ, Sofia, Tina, Delya, and Bambaraya. The CZ, NZ, Sofia, and Delya Mineral Resources were re-estimated during 2018 using updated geological interpretations generated from both resource definition and AdvGC drilling. The 2018 AdvGC drill programmes in CZ, Sofia, and Delya were designed to determine the optimal drill spacing required to outline an accurate local estimation, which could be used during the mining process.

All Massawa Mineral Resources have been estimated using a combination of trench, RC, and DD date up until November 2018.

Outside of the three principal deposits (Massawa, Sofia, and Delya), Inferred Mineral Resources exist for two satellite projects (Bambaraya and Tina). No exploration has taken place on Bambaraya and Tina since the previous Mineral Resource estimate in 2013. The satellite deposits have been estimated using inverse distance squared and as such have been classified as Inferred Mineral Resources. These satellite Inferred Mineral Resources represent less than 28% of the total declared tonnes and less than 7% of the contained gold ounces of the Project. Accordingly, they do not form part of the declared Ore Reserve and have no impact on the current economic evaluation of the Project. Further exploration programmes are planned to be completed on these satellites as part of the feasibility programme, to evaluate their prospectivity with additional resource development.

Additionally, modifying factors for reserve definition have been updated and refined, including updated mining costs, selectivity, and recoveries which are incorporated as part of the cut-off grade criteria. All Mineral Resource pit optimisations and associated underground reporting areas have been updated accordingly.

As at 31st December 2018, the open pit (OP) Indicated Mineral Resource is estimated to be 23 Mt at an average grade of 4.00 g/t Au containing 2.97 Moz of gold and the OP Inferred Mineral Resource is estimated to be 3.7 Mt at an average grade of 2.2 g/t Au for 0.26 Moz of gold. An underground Inferred Mineral Resource, situated below the NZ1 and NZ2 open pit solid, is estimated to be 2.6 Mt at an average grade of 4.1 g/t Au containing 0.35 Moz gold (Table 14-1).

Table 14-1 Massawa Project Mineral Resource Statement as at 31st December 2018

Mineral Resource	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Attributable Contained Gold (Moz)*
OP Measured	-	-	-	-
OP Indicated	23	4.00	3.0	2.5
Total Measured + Indicated	**23**	**4.00**	**3.0**	**2.5**
OP Inferred	3.7	2.2	0.26	0.22
UG Inferred	2.6	4.1	0.35	0.29
Total Inferred	**6.3**	**3.0**	**0.61**	**0.51**

* Attributable gold (Moz) refers to the quantity attributable to Barrick based on Barrick's 83.25% interest in the Massawa Project.
Open pit Mineral Resources are reported as the insitu mineral resources falling within the $1,500/oz pit shell reported at an average cut-off grade of 0.8 g/t Au.
Underground Mineral Resources are those insitu mineral resources below the $1,500/oz pit shell of the North Zone 2 deposit reported at a 2.5 g/t Au cut-off grade.
Mineral Resources are reported inclusive of Ore Reserves

Mineral Resources for Massawa were generated by Simon Bottoms, MGeol, FGS CGeol, FAusIMM, an employee of the company and Qualified Person.

Barrick is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

The Mineral Resource estimate has been prepared according to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves standards and guidelines published and maintained by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (the JORC (2012) Code). Barrick has reconciled the Mineral Resources to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated 10th May 2014 as incorporated into NI 43-101, and there are no material differences.

14.1. Geological Modelling

14.1.1. Modelling Techniques

Cross sections were generated for all deposits by drawing vertical sections approximately perpendicular to the strike direction of the mineralised zones in Maptek Vulcan, Seequent Leapfrog, and Micromine. Geological models, including the mineralisation wireframes, were generated by the on-site Massawa geology team who were directly responsible for the geological logging, thus ensuring that the wireframes are geologically representative of the mineralised bodies.

The mineralisation wireframes for Sofia, Delya, and Massawa NZ were generated through drawing a string outlining the edges of mineralisation intercepts on vertical sections in Vulcan. The vertical sections are generally spaced 10 m apart, except in areas where the GC orientation drilling has been completed which are spaced at half of the drill spacing (e.g. 5 m sections for 10 m along strike drill spacing). This allows for two intermediate section strings between each line of drilling. The use of intermediate strings allows for the wireframes to be smoothed between sections, particularly when the mineralised body anastomoses both horizontally and vertically.

The vertical section strings are snapped to the exact intersection within all forms of data including surface trenches, RC, and DD holes thereby ensuring that the points that define the wireframe intersect the drill hole or trench. Thereafter, the strings on each section are connected to form a 3D triangulated solid. Where a mineralised zone terminates, the wireframe is projected along strike by 50% of the drill spacing (mean distance of 25 m).

The wireframes are again checked against the hardcopy hand-drawn sections, and then cut to the 5 m resolution LIDAR topographic surface, for Massawa and Sofia, the sections are cut to the 10 m resolution DGPS topographic surface for Delya.

For the CZ, cross sections were hand drawn for each drill line and flitches were then generated every 10 m vertical depth. These interpretations were then scanned and georeferenced into

Seequent Leapfrog in order to create the 3D wireframes. The wireframes were generated in Seequent Leapfrog through selecting the geological intervals on vertical and horizontal sections and then using the vein-modelling tool to model these intercepts in 3D space. Intervals were assigned to veins based on the mineralised continuity seen on both vertical sections and horizontal flitches cut 10 m throughout the mineralised body. The vein meshes are snapped to the exact interval selections, which represent the exact mineralised intersection, within all forms of data including surface trenches, RC, and DD holes. The use of vein modelling in 3D allowed the wireframes to be smoothed between sections and to provide a more accurate 3D model, particularly where the mineralised body anastomoses, merges, and splits both horizontally and vertically.

14.1.2. Massawa Northern Zone

Massawa NZ1 is composed of a single discontinuous and bifurcating shear zone, with small discontinuous zones of mineralisation in the hanging wall and footwall (Figure 14-1) which are modelled as separate wireframe solids.

In contrast, the Massawa NZ2 mineralised zone is primarily composed of a single linear continuous shear zone, as well as small discontinuous hanging wall and footwall mineralisation zones which are also wireframed as separate domains (Figure 14-2).

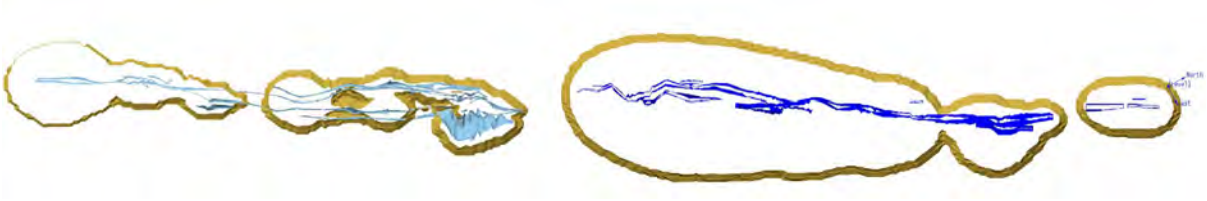


Figure 14-1 Massawa Northern Zone 1 (Light Blue) and Northern Zone 2

(Dark Blue) 2018 Updated Wireframes Within $1,500 Pit, in Plan View (Looking Down the Z Axis)

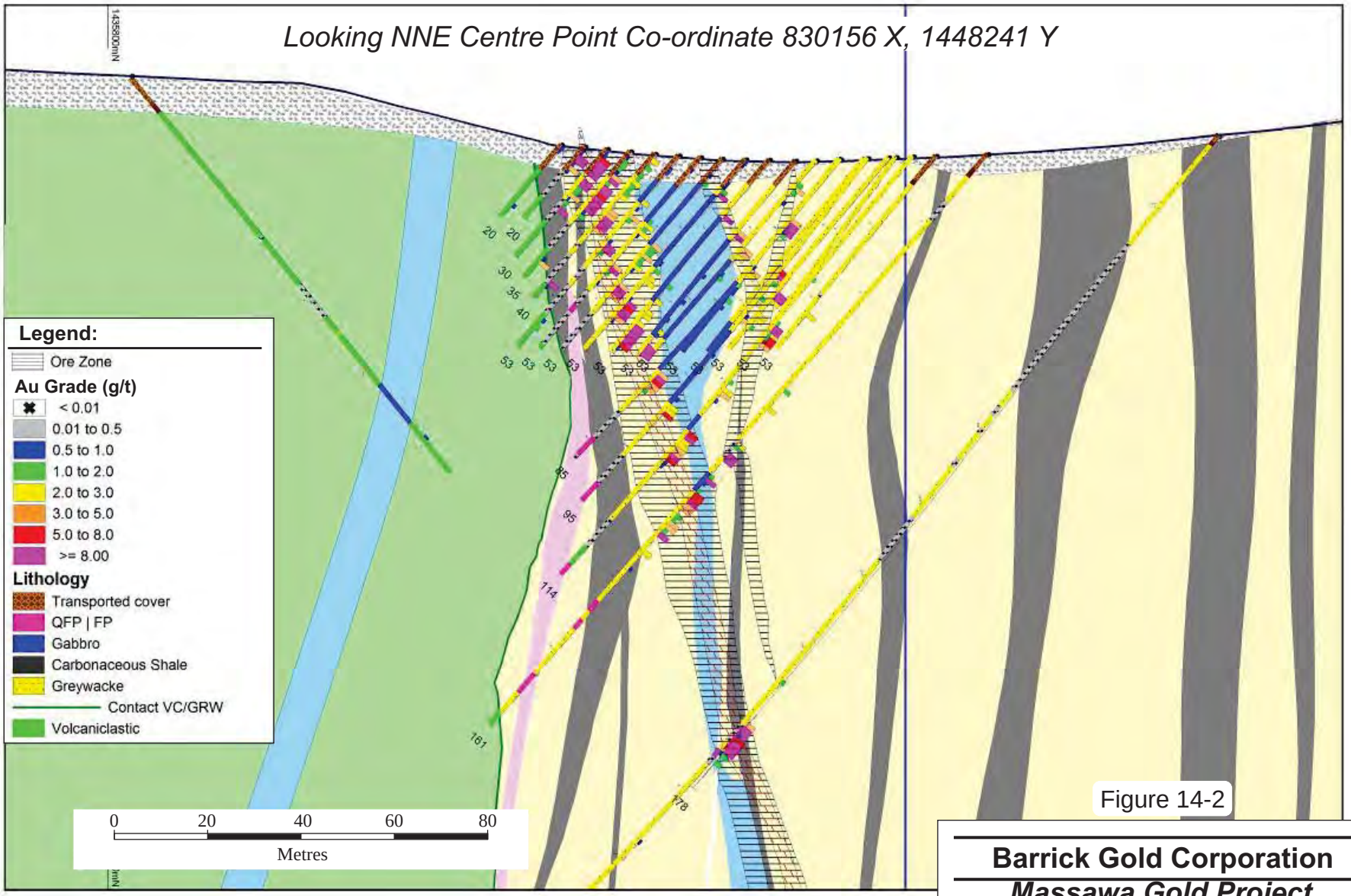


Figure 14-2

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Cross Section of Section 74 Through the North of North Massawa Zone 2 Showing the Mineralised Envelope and Lithologies

July 2019

Source: Barrick, 2019.

There is generally a sharp contact between the mineralised and waste material, with a strong correlation between the grade change and that of the alteration and structural deformation. This means that the mineralised wireframes are defined by geologically logged intersections.

Visual data correlations confirm that a suitable geological related cut-off grade was approximately 0.5 g/t Au in Massawa NZ resource wireframes. During interpretation, efforts were made to minimise the amount of sub-grade material included within each of the mineralised lode wireframes.

14.1.3. Massawa Central Zone

The CZ is defined by an envelope of altered rock containing 51, thin, NE trending, sub-vertical lodes. These mineralized lodes anastomose horizontally and vertically along strike, and are defined as the low-grade 0.5 g/t Au mineralisation envelopes (Figure 14-3).

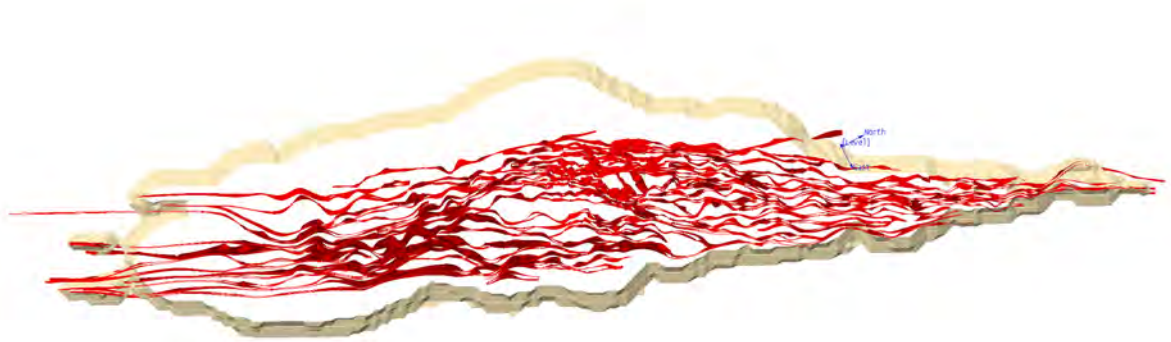


Figure 14-3 Surface Level Flitch at 175 m RL of Massawa Central Zone Mineralised Lode (Red) Low-Grade Wireframes Inside 2018 $1,500 Pit Shell

Individual lodes are 1 m to 25 m thick, with an average thickness of between 6 m and 10 m. Visual data correlations confirm that a suitable geological related cut-off grade was approximately 0.5 g/t Au for the low-grade Massawa CZ resource wireframes. During interpretation, efforts were made to minimise the amount of sub-grade material included within each of the mineralised lode wireframes.

Within 41 of the low-grade wireframes the mineralisation is sub-domained into a high-grade wireframe of brecciated, extensional shear veins with moderate to strong silica-carbonate alteration and sulphides. The relationship with the low-grade mineralisation and lithology is outlined in Figure 14-4 and the distinct grade distributions in Figure 14-5.

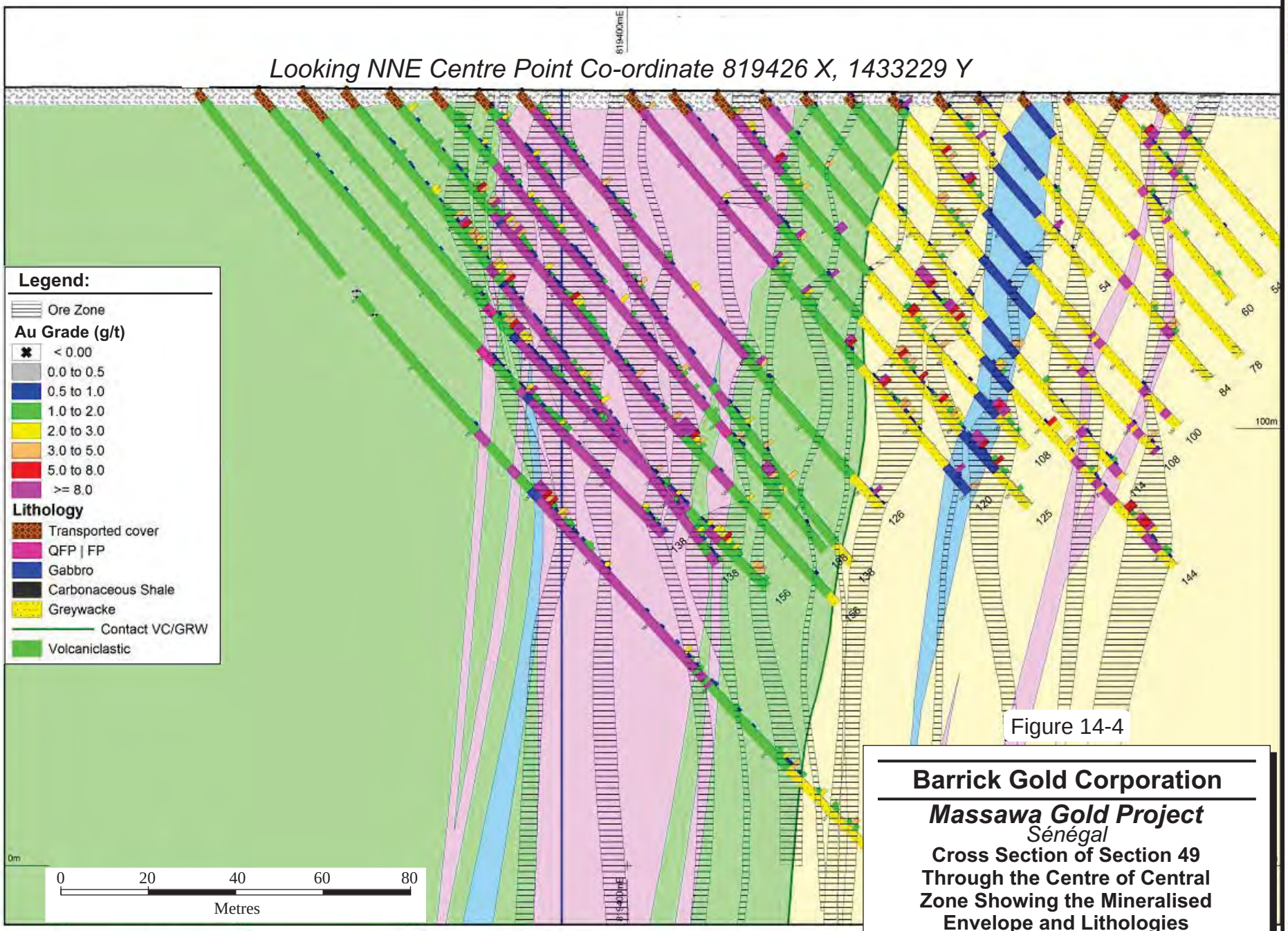


Figure 14-4

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Cross Section of Section 49 Through the Centre of Central Zone Showing the Mineralised Envelope and Lithologies

July 2019

Source: Barrick, 2019.

Figure 14-5 Massawa Central Zone Low-Grade (Right) versus High-Grade (Left) Box and Whisker Uncut Grade Distribution Comparison

The high-grade mineralisation contains 94% of the gold content, with coarse visible Au within the veins and Au inclusions within the arsenopyrite and arsenian pyrite disseminations. These high-grade domain wireframes have been modelled within 41 of the 51 low-grade envelopes along the entire strike length of the CZ as confirmed by the drilling and trenching.

The contact between the low-grade and high-grade mineralisation domains is very sharp with a strong correlation between the grade change and vein plus brecciation and sulphide abundance, which means that the high-grade intersections can be defined by both the geological logging and the final assay values.

14.1.4. Sofia

Sofia Main is defined by a linear altered shear zone, where the mineralisation is thought to be structurally controlled. The majority of the high-grade mineralisation (above 2 g/t Au) is situated within a plunging tabular dilation zone that is modelled to terminate on the FW structure. There are also several small FW splays of mineralisation. Consequently, the Sofia Main mineralisation wireframe depth projection is cropped by the modelled structural intersection of the alteration envelope and the FW gabbro (Figure 14-6).

Sectional interpretation correlations confirm that a suitable geological related cut-off grade was approximately 0.5 g/t Au for the Sofia Main resource wireframes (Figure 14-7). During interpretation, efforts were made to minimise the amount of sub-grade material included within each of the mineralised lode wireframes.

Looking NNE Centre Point Co-ordinate 808983 X, 1434691 Y

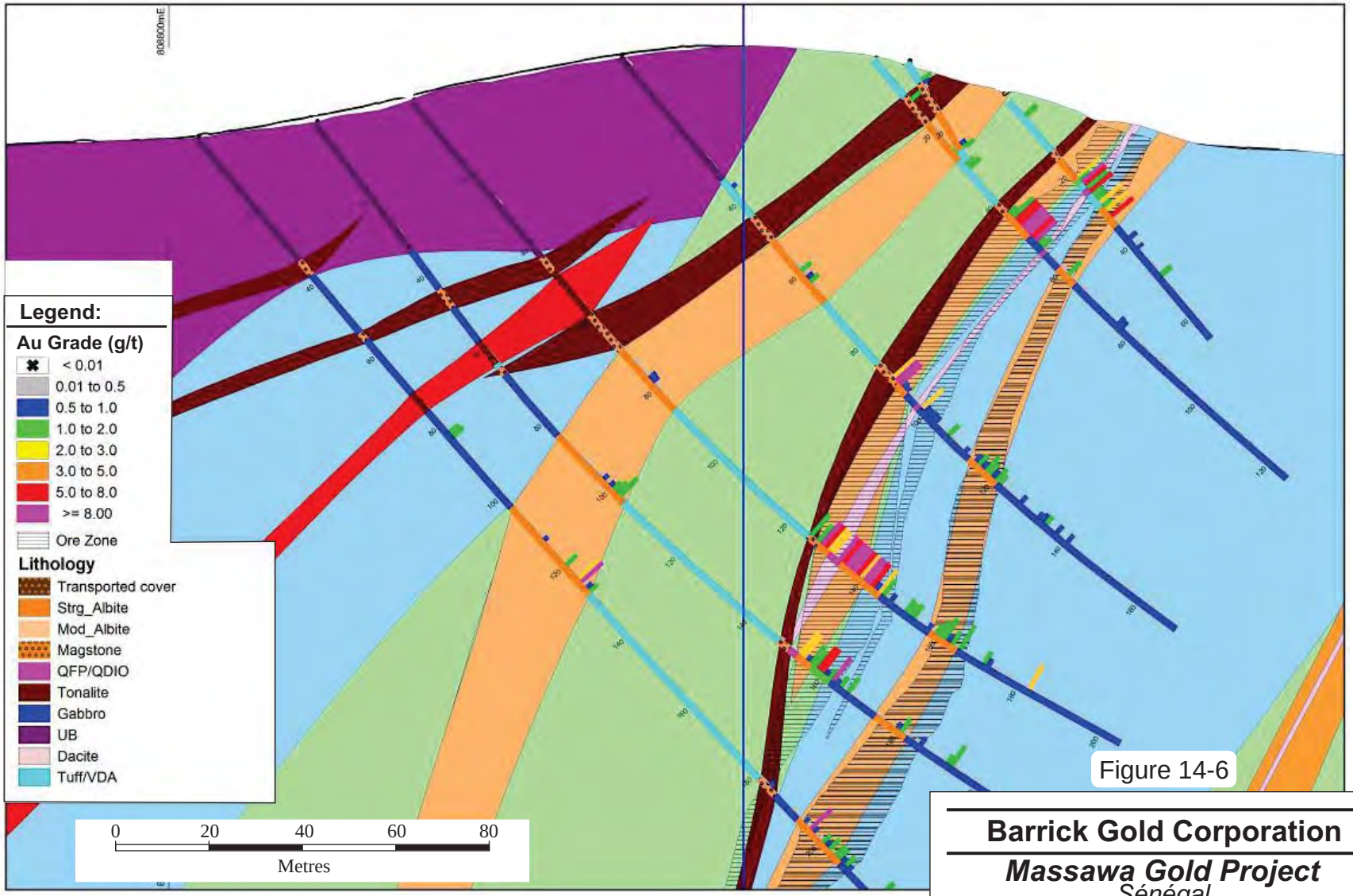


Figure 14-6

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Cross Section of Section 43 Through the Centre of Sofia Main Showing the Alteration Zone (Mineralised Envelope) and Lithologies

July 2019

Source: Barrick, 2019.

Looking NNE, Centre Point Co-ordinate is 809863 X, 1436593 Y

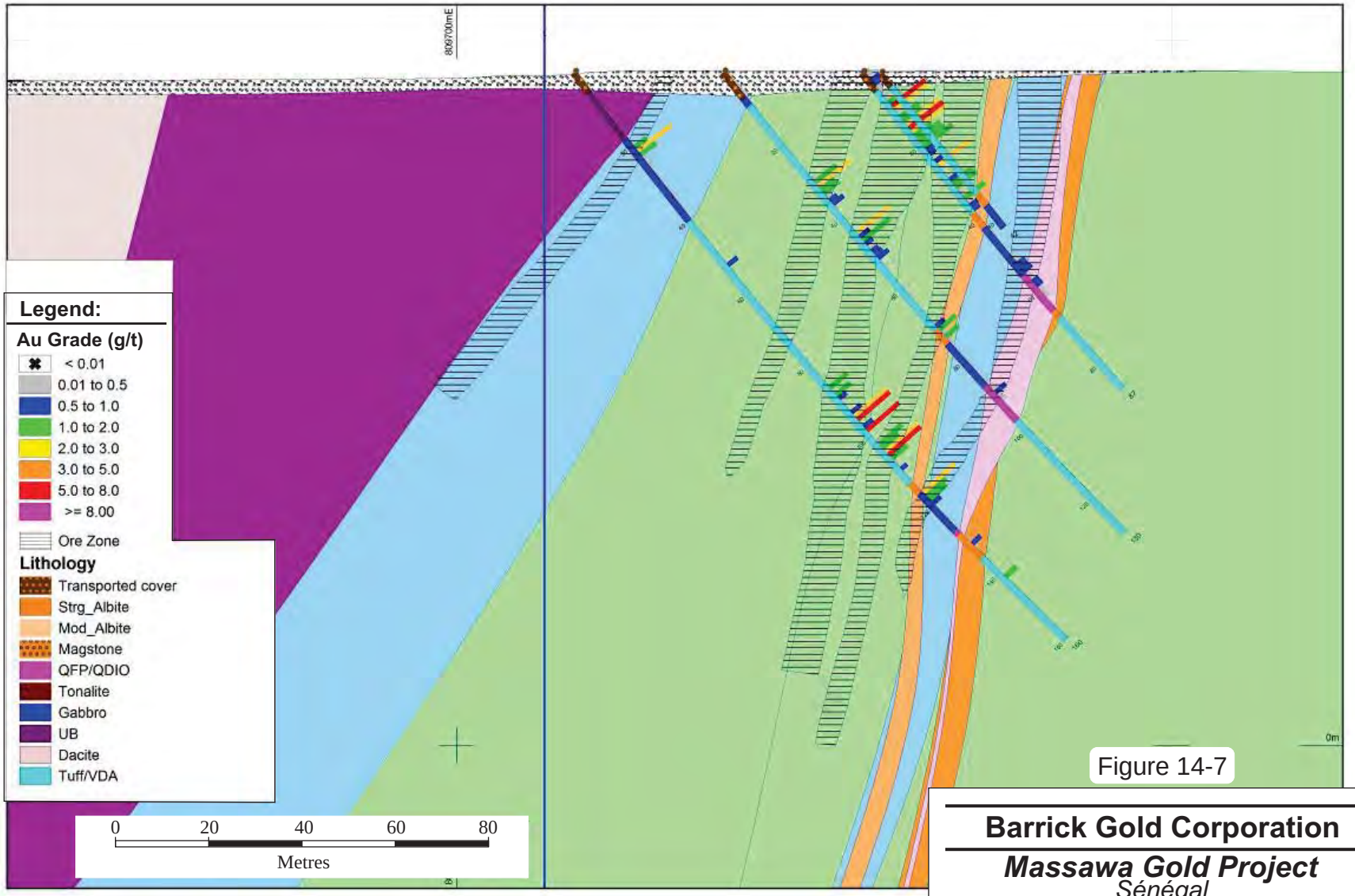


Figure 14-7

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Cross Section of Section 28
Through the Centre of Sofia North
Showing the Alteration Zone
(Mineralised Envelope) and Main Lithologies

July 2019

Source: Barrick, 2019.

Sofia North consists of one main NNE-trending mineralized structure and discontinuous FW and HW lodes. The main mineralised structure that controls the alteration is developed at the eastern contact of the Western Mafics where the strike rotates from 040º to 010º and has been delineated over more than 2 km by trenching and RC drilling. This structure continues along strike from Sofia Main.

The mineralised envelope contacts at Sofia are well defined by a strong correlation between grade and alteration intensity, meaning that the mineralisation intersections can be defined by both the geological logging as well as the assay values.

14.1.5. Delya

Delya is located along a major shear zone and can be defined by three steeply dipping, parallel zones of mineralisation, striking 030°, which are hosted in the lower margins of a series of gabbros within highly sheared and silicified sericitised schist. The mineralized zones are defined by an alteration assemblage of sericite-silica-carbonate and chlorite with fine disseminated pyrite and arsenopyrite. This altered unit contains higher grades (up to 5 g/t Au), and dips to the east at 85°. The other lodes are located to the west and have an average dip of 84° to the west (Figure 14-8).

Sectional interpretation correlations confirm that a suitable geological related cut-off grade was approximately 0.5 g/t Au for the Delya resource wireframes. During interpretation, efforts were made to minimise the amount of sub grade material included within each of the mineralised lode wireframes. The wireframes are again checked against the hardcopy hand-drawn sections, and then cut to the 10 m resolution topographic surface, generated from the DGPS points.

Looking NNE Central Point Co-ordinate 830157 X, 1448241 Y

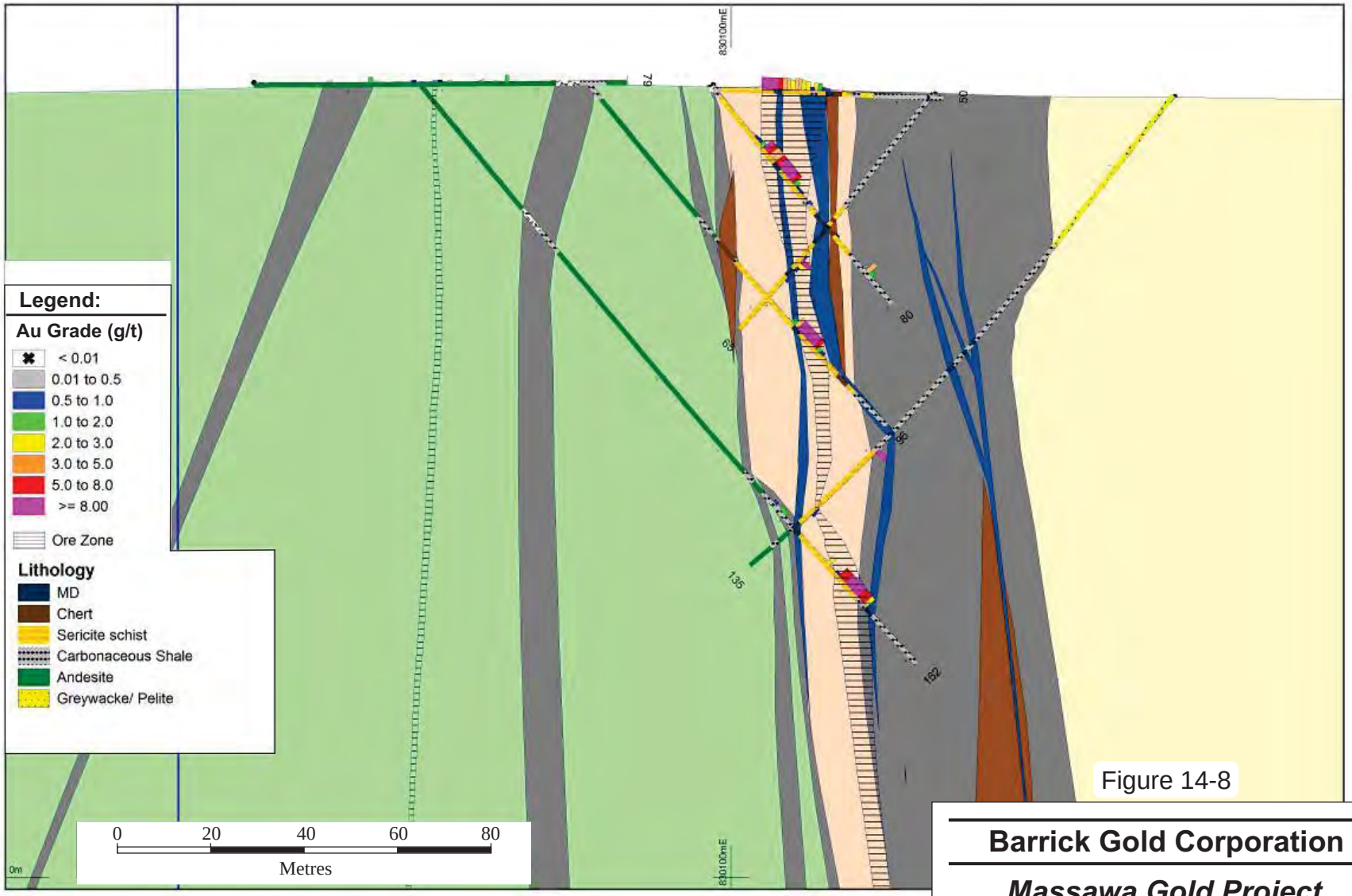


Figure 14-8

Barrick Gold Corporation

Massawa Gold Project
Sénégal

Cross Section of Section 24 Through the Centre of Delya Showing the Mineralised Envelope (Grey Hashed Solid) and Lithologies

July 2019

Source: Barrick, 2019.

14.1.6. Tina and Bambaraya

The Tina and Bambaraya satellite deposits have been interpreted using a combination of grade, alteration, and sulphide logging in vertical cross sections.

The distance between the sections is dependent upon the data spacing, typically 50 m to 100 m. The vertical section strings are snapped to the exact intersection within all forms of data including surface trenches, RC, and DD holes thereby ensuring that the points that define the wireframe intersect the drill hole or trench. Thereafter, the strings on each section are connected to form a 3D triangulated solid. Where a mineralised zone terminates, the wireframe is projected along strike by 50% of the drill spacing up to a maximum of 25 m. These wireframes have been built using Micromine software.

The wireframes are again checked against the hardcopy hand-drawn sections, and then cut to the 5 m resolution LIDAR topographic surface.

14.2. Data Analysis and Domaining

Since the Massawa and Sofia deposits have different drill spacing within different portions of the bodies, the wireframes are sub-domained by their relevant data spacing. This means that all geological and geostatistical domains are split into exploration and GC sub-domains. The application of this sub-domaining technique enables the application of different block sizes within each of the data density domains and the application of more localised estimation parameters within the GC domains relative to the exploration domains. This estimation methodology is applied throughout all other Barrick Mineral Resources in West Africa.

The GC sub-domains are defined by a wireframe basal surface situated at the base of the GC drilled areas. This basal surface is used to split any domain wireframes intersected by it, to create exploration and GC sub-domains.

All data spacing defined sub-domains use soft boundaries for the purposes of the Mineral Resource estimation, thus ensuring that their application within the resource model does not constrain the estimation from using sample support from an equivalent geological or statistical domain. Each data spacing sub-domain will only use its relevant search ellipse estimation constraints (size, number of samples, etc.), to take sample support from an equivalent geological or statistical domain.

14.2.1. Massawa North Zone

NZ1 is split into the following domains for the purposes of Mineral Resource estimation:

- 1103 = HWOZ
- 1104 = FWOZ Folded
- 1105 = FWOZ
- 1106 = FWOZ South
- 1107 = MOZ1

- 1108 = MOZ2

All drilling in NZ1 is either on an exploration or on a GC drill spacing. Thus, the NZ1 domains are sub-domained into exploration or GC domains based upon their sample data density.

The NZ1 main zone wireframes are discontinuous and bifurcating in nature meaning that they have a similar nature to that of the HW and FW domains and the wireframes all intersect one another in places. Subsequently, a soft boundary is applied between data in all three domains for the purposes of resource estimation. Further support is provided for this by the fact that there are no differences in the grade distributions in the main domains relative to the HW and FW lodes.

The resultant separate grade populations show a significant improvement in stationarity of domains relative to the original distribution for application during resource estimation (Figure 14-9).

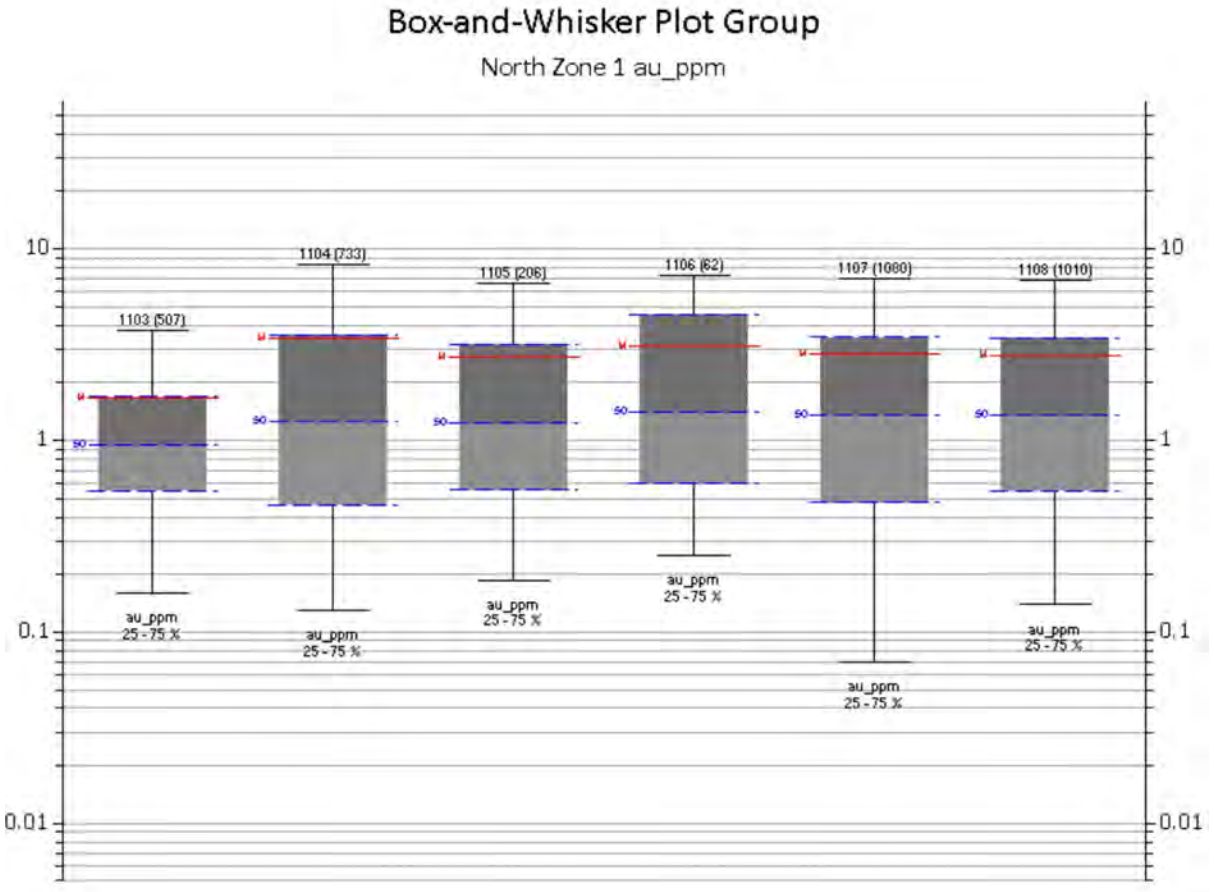


Figure 14-9 North Zone 1 Mineralisation Zone Domain Grade Distribution Box and Whisker Uncut Grade Distribution Comparison

NZ2 sample data is split into the following domains for the purposes of resource estimation:

- 2101 = MOZ Domain
- 2102 = FWOZ Domain
- 2103 = HWOZ Domain
- 2111 = MOZ HG Domain

- 10 = Exploration Variable Domain
- 11 = GC Variable Domain

The resultant separate grade populations show a significant improvement in stationarity of domain relative to the original distribution for application during resource estimation (Figure 14-10).

Compared to NZ1, the distribution of sample grades within the NZ2 wireframes is bi-modal. Consequently, the main mineralised zone wireframe is sub-domained into a high-grade domain which separates the strongly bi-modal distribution into two or more single stationary geostatistical population with consistent major directions of mineralisation. The definition of the high-grade domain model was based upon the following characteristics:

- 3% or more arsenopyrite sulphide mineralisation,
- 2% of less pyrite sulphide mineralisation,
- Arsenopyrite: pyrite ratio greater than 1.5,
- Grades mostly greater than 8 g/t Au.

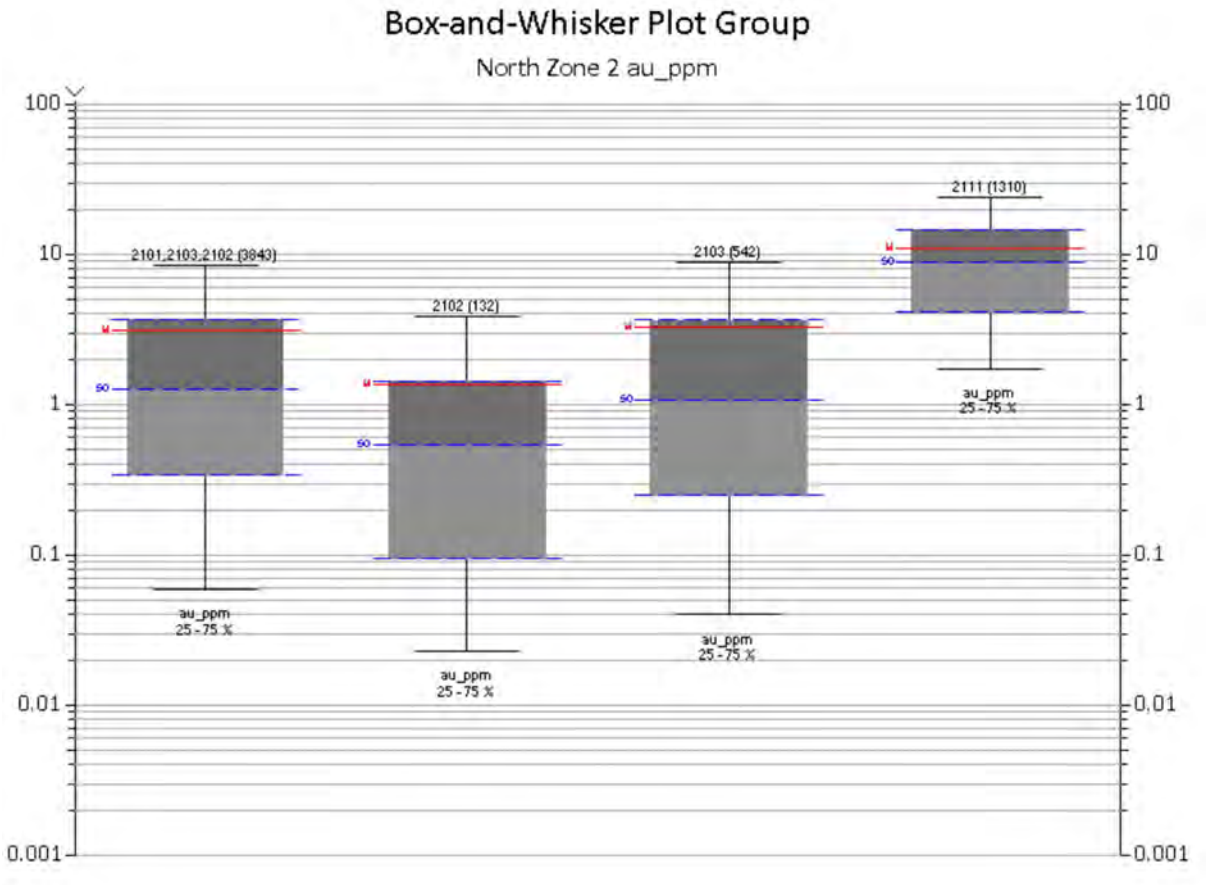


Figure 14-10 North Zone 2 Mineralisation Zone Domain Grade Distribution Box and Whisker Uncut Grade Distribution Comparison

The final high-grade domain model represents 17% of the total NZ2 wireframe volume (Figure 14-11).

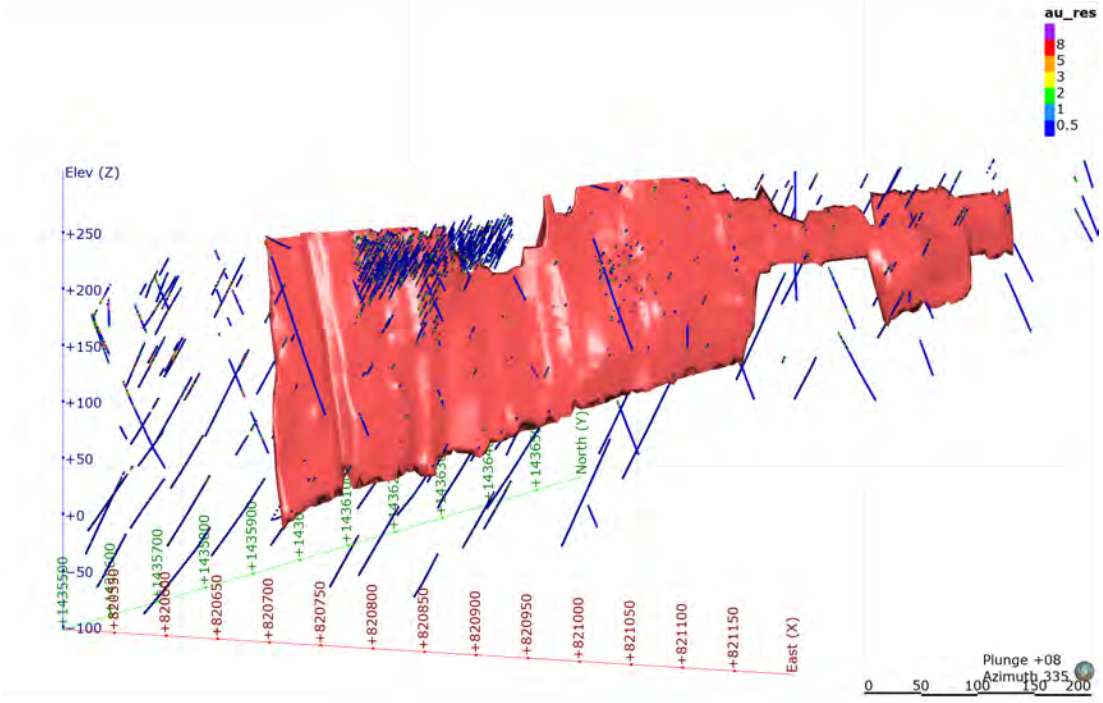


Figure 14-11 Massawa North Zone 2 High-Grade Domain Wireframe and Supporting Drill Samples

The resultant separate grade populations show a significant improvement in stationarity of domains relative to the original distribution for application during resource estimation (Figure 14-12).

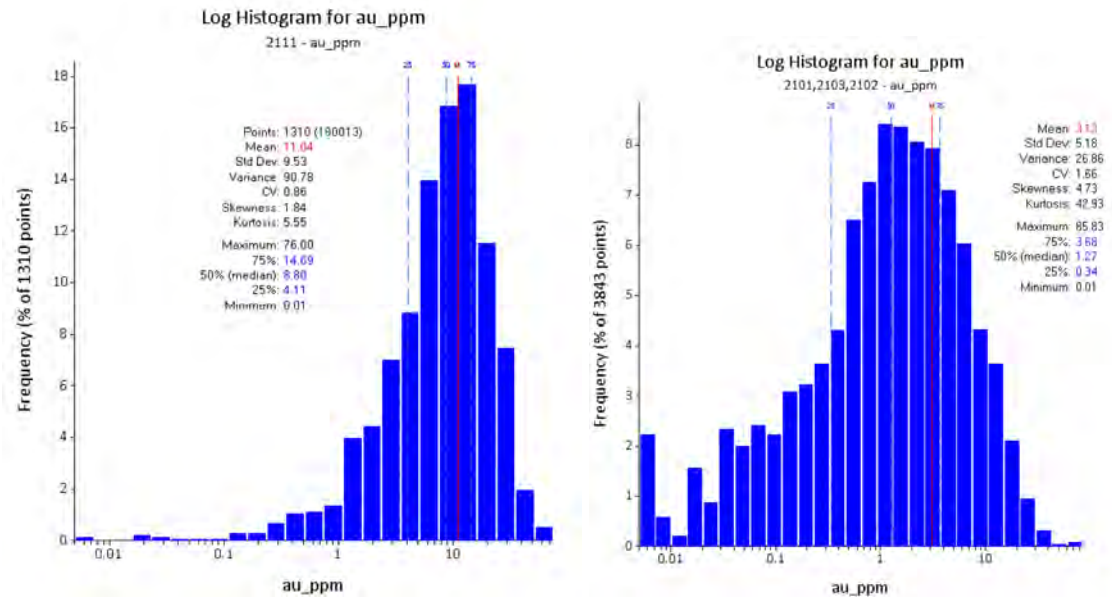


Figure 14-12 Massawa North Zone 2 Log Histograms for the Raw Sample Data Inside the High-Grade Domain (Left) and the Remainder of the Mineralised Wireframe - Low/Medium-Grade Domain (Right)

There are still some indications of mixed population within the low/medium-grade domain – likely to be caused by small isolated high-grade pods. However, with the current drill density this cannot be spatially constrained.

Contact profile analysis between the high-grade domain and the surrounding main mineralised zone shows a sharp change in grade across the boundary (Figure 14-13). However, geological observations show that the change in sulphide concentrations can often be gradational. Consequently, a hard boundary was applied between the outer mineralisation wireframe and the inner high-grade domain.

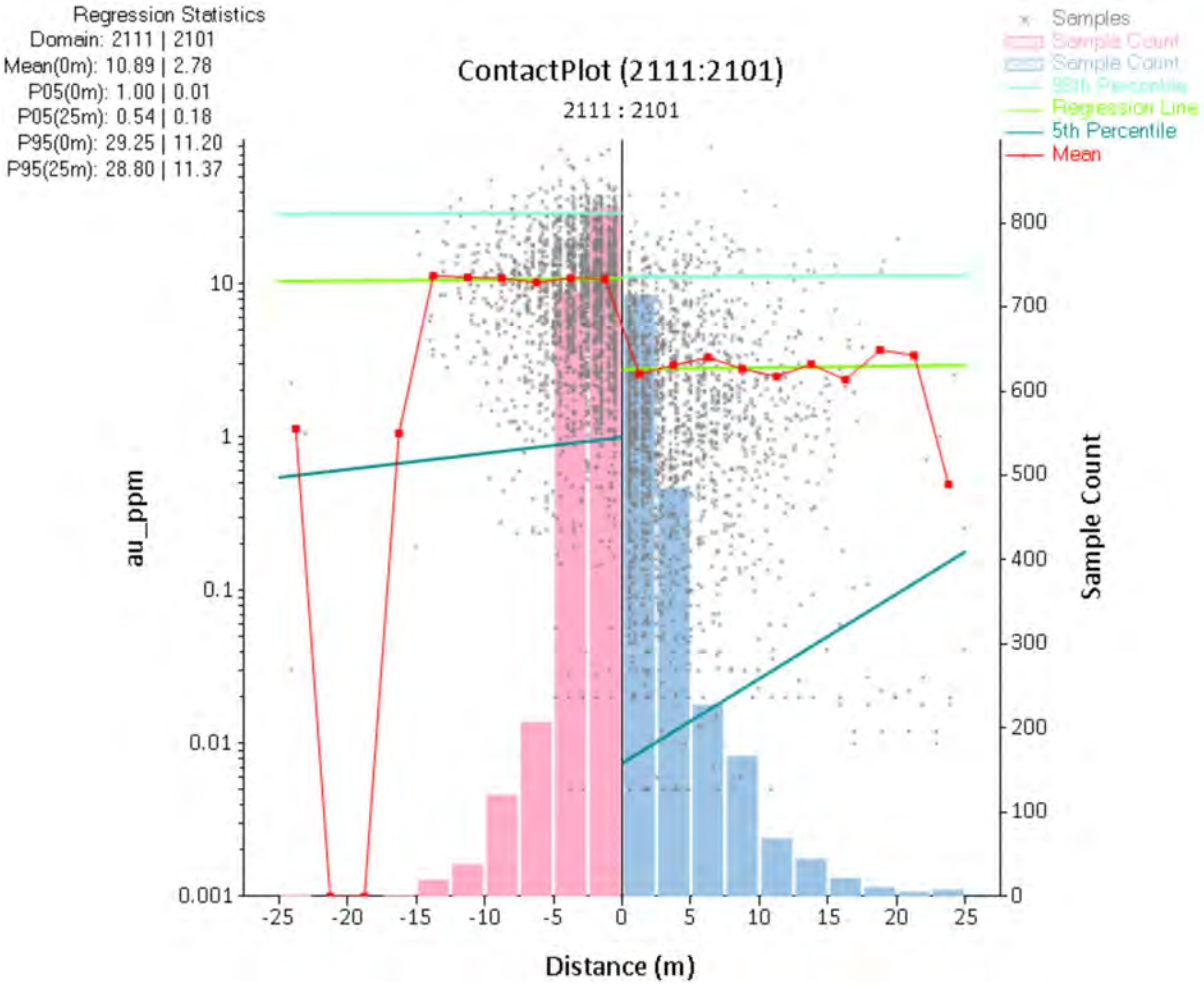


Figure 14-13 Contact Analysis Plot of North Zone 2 Domain 2101 (Main Zone Mineralisation) and Domain 2111 (Main Zone High-Grade Mineralisation)

14.2.2. Massawa Central Zone

Mineralisation in the individual lodes within the CZ is domained into: a low-grade, broader halo of weak to moderate shearing with silica-carbonate alteration and disseminated pyrite>>arsenopyrite and a high-grade zone of high strain including brecciation, extensional and shear veins with moderate to strong silica-carbonate alteration on sulphides. Arsenopyrite is the dominant sulphide associated with gold; arsenopyrite and pyrite are also observed as vein selvedges +/- visible gold. The high-grade mineralisation contains 80% of the grade in the CZ1 with coarse visible gold within the veins and gold inclusions within the arsenopyrite. As a direct consequence, the two domains have distinctly different grade distributions due to the sulphide assemblages and free gold occurrence within the vein system (Figure 14-14 and Figure 14-15).

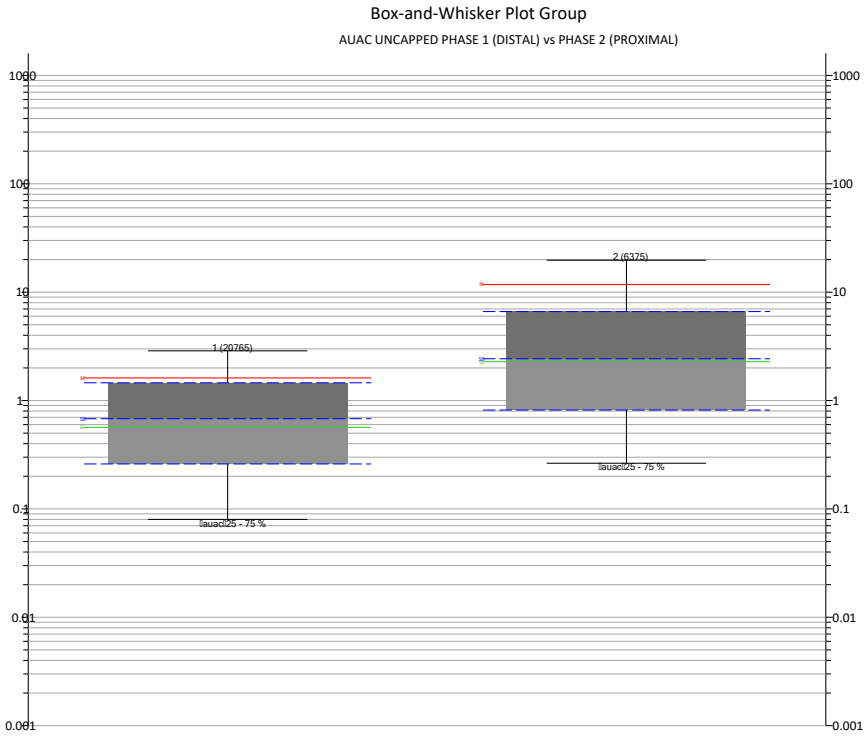


Figure 14-14 Massawa Central Zone Low-Grade (Left) vs High-Grade Proximal (Right) Box and Whisker Uncut Grade Distribution Comparison

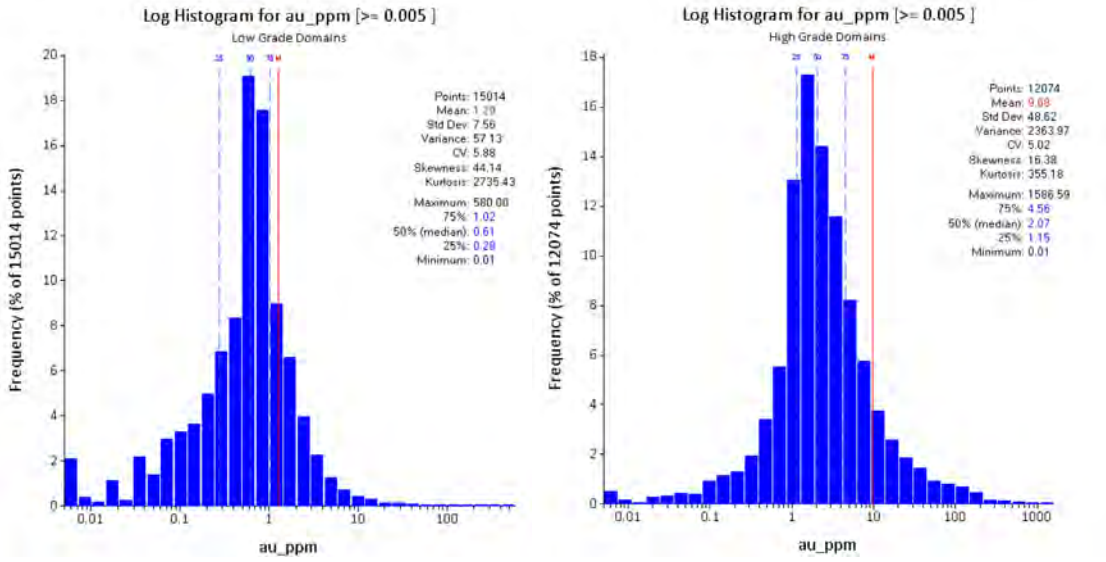


Figure 14-15 Massawa Central Zone Low-Grade (Left) vs High-Grade (Right) Log Histogram Uncut Grade Distribution Comparison

Contact profile analysis between the high-grade proximal domain and the surrounding low-grade domain zone (Figure 14-16) shows a sharp boundary and change in grade at the edges of the high-grade system. Accordingly, the two domains are wireframed separately and are hard bounded from one another for resource estimation purposes, which prevents the very high-grade samples, within the high-grade distribution, from influencing the estimation within the low-grade domains.

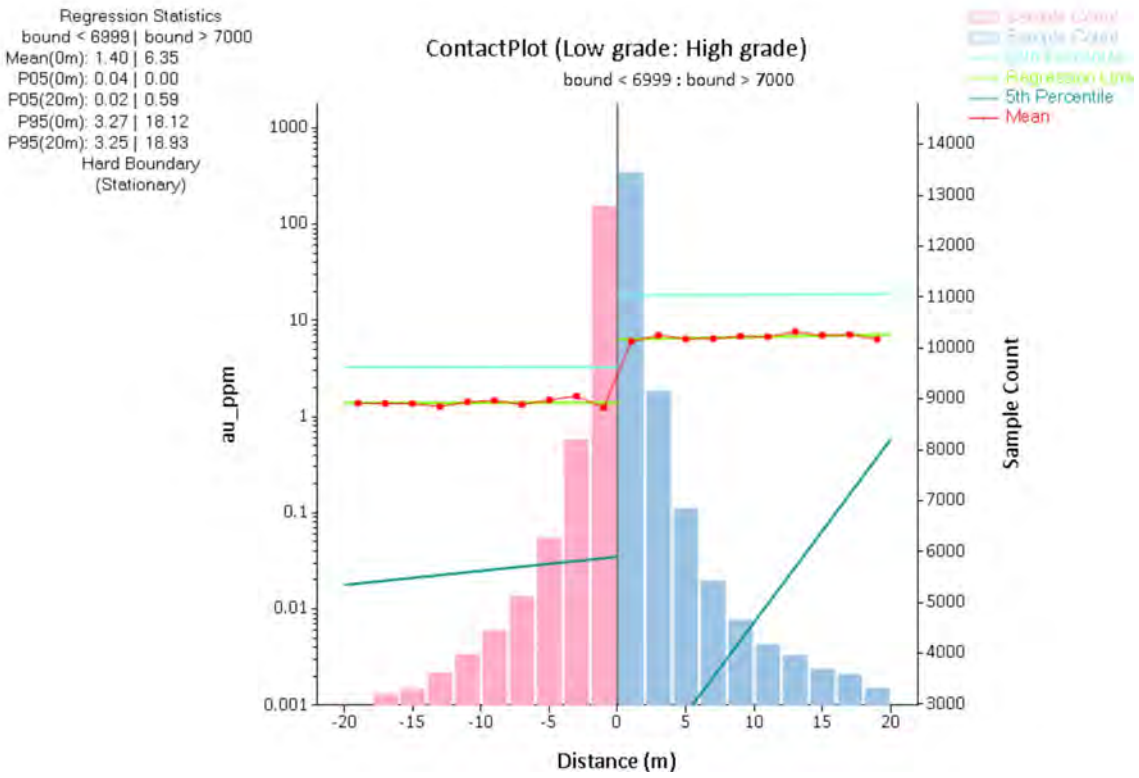


Figure 14-16 Contact Analysis Plot of Central Zone Low-Grade Mineralisation (All Domains) and Central Zone High-Grade Mineralisation (All Domains)

Each of the low-grade and high-grade lodes is effectively treated as its own separate domain which is hard bounded from other parallel lodes for the purpose of resource estimation. This is because the different lodes have distinctly different grade distributions (Figure 14-15).

The difference in the grade distributions of the different lodes both within low-grade and high-grade domains is controlled by the spatial association with the porphyritic intrusions as the brecciated contacts of the porphyry provide a rheological contrast, which focusses fluid flow and increases the abundance of mineralisation.

CZ sample data is split into domains, which are in turn spatially grouped geologically and by grade distributions for the purposes of top cutting and resource estimation whereby the estimate allows each group to see one group across, but no more, thus keeping the estimate tightly restricted across strike data continuity associated with geological domains. The domain groups are:

- 6101, 6102, 6103, 6104, 6105, 6110, 6120 and 6130 – Low-grade HW domains
- 6210,6301, 6304 and 6305 – Low-grade HW/Central Bridge domains
- 6410, 6411, 6420, 6434, 6436, 6441, 6443, 6444, 6450, 6470, 6471, 6472, 6473, 6474 and 6480 – Low-grade Central domains
- 6430, 6460, 6461, 6610, 6620, 6630, 6641 and 6641 – Low-grade Central/FW Bridge domains
- 6440, 6490, 6502, 6710, 6711, 6712, 6713, 6621, 6631, 6640, 6650, 6660, 6670, 6671, 6690 and 6691 – Low-grade FW domains
- 7101, 7102, 7104, 7110 and 7120 – High-grade HW domains

- 7210, 7304 and 7305 High-grade HW/Central Bridge domains
- 7410, 7420, 7434, 7441, 7443, 7444, 7450, 7470, 7471, 7472, 7473 and 7480 – High-grade Central domains
- 7430, 7460, 7501, 7610, 7630 – High-grade Central/FW domains
- 7440, 7490, 7502, 7621, 7631, 7640, 7650, 7660, 7670, 7671, 7690, 7691, 7710 and 7711 – High-grade FW domains

These domains are then further sub-domained into exploration or GC domains based upon their sample data density (Figure 14-17).

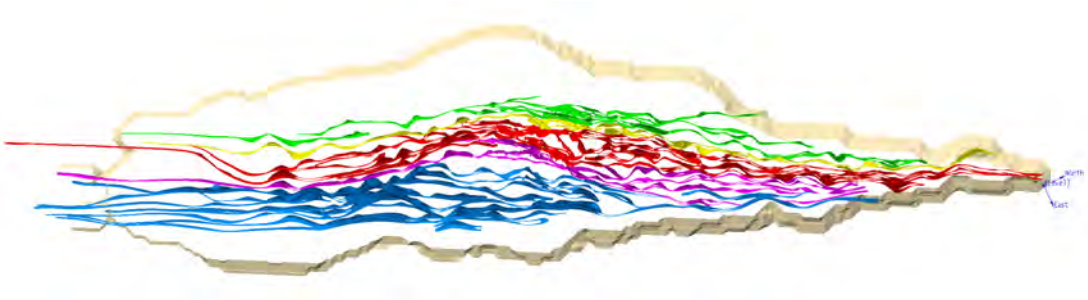


Figure 14-17 Massawa Central Zone Grouped Domains

Note: Green domains are low-grade hanging wall, yellow domains are the low-grade hanging wall/central bridge domains, red are central low-grade domains, purple are Central/ footwall bridge domains and blue are the footwall low-grade domains.

Testwork completed to date has indicated that historic half core diamond drill core data analysed with 50 g fire assay will underestimate the gold content relative to larger samples with bulk analysis methods. This data is currently used as part of the estimation dataset within the inferred portions of the CZ Mineral Resource only.

14.2.3. Sofia

Sofia is split into the following domains for Mineral Resource estimation purposes:

- 6000 = Sofia Main MOZ
- 6100 = Sofia Main FWOZ
- 6200 = Sofia Main HWOZ
- 6300 = Sofia Main MOZ North Extension
- 6400 = Sofia North FWOZ
- 6500 = Sofia North MOZ
- 6600 = Sofia North HWOZ
- 6700 = Sofia North HW Vein
- 6900 = Sofia North HWOZ1

The HW and FW mineralisation zones intersect the main mineralisation zone in Sofia Main and Sofia North. Subsequently, a soft boundary is applied between data in all three domains for the purposes of resource estimation. This is further supported by the fact that there are no differences in the grade distributions in the MOZ relative to the HWOZ and FWOZ.

The resultant separate grade populations show a significant improvement in stationarity of domains relative to the original distribution for application during resource estimation (Figure 14-18 and Figure 14-19).

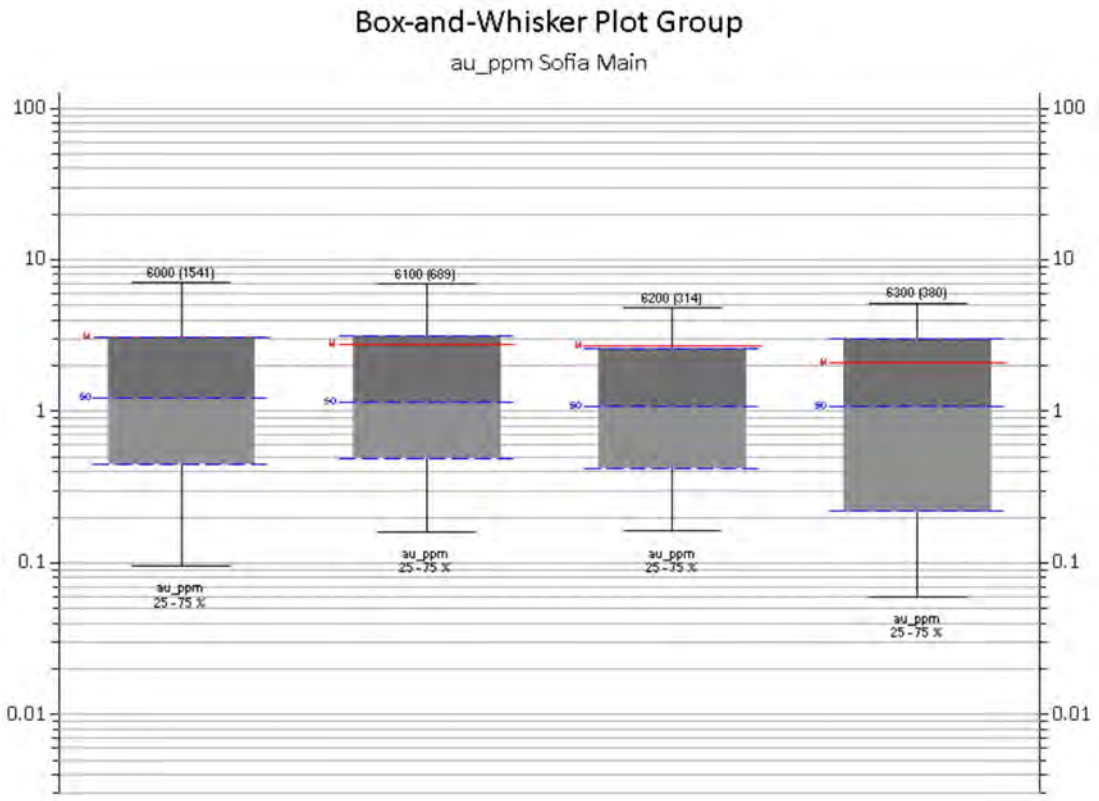


Figure 14-18 Sofia Main Mineralisation Zone Domain Grade Distribution Box and Whisker Uncut Grade Distribution Comparison

(6000 – MOZ, 6100 – FWOZ, 6200 – FW Vein, 6300 – MOZ North)

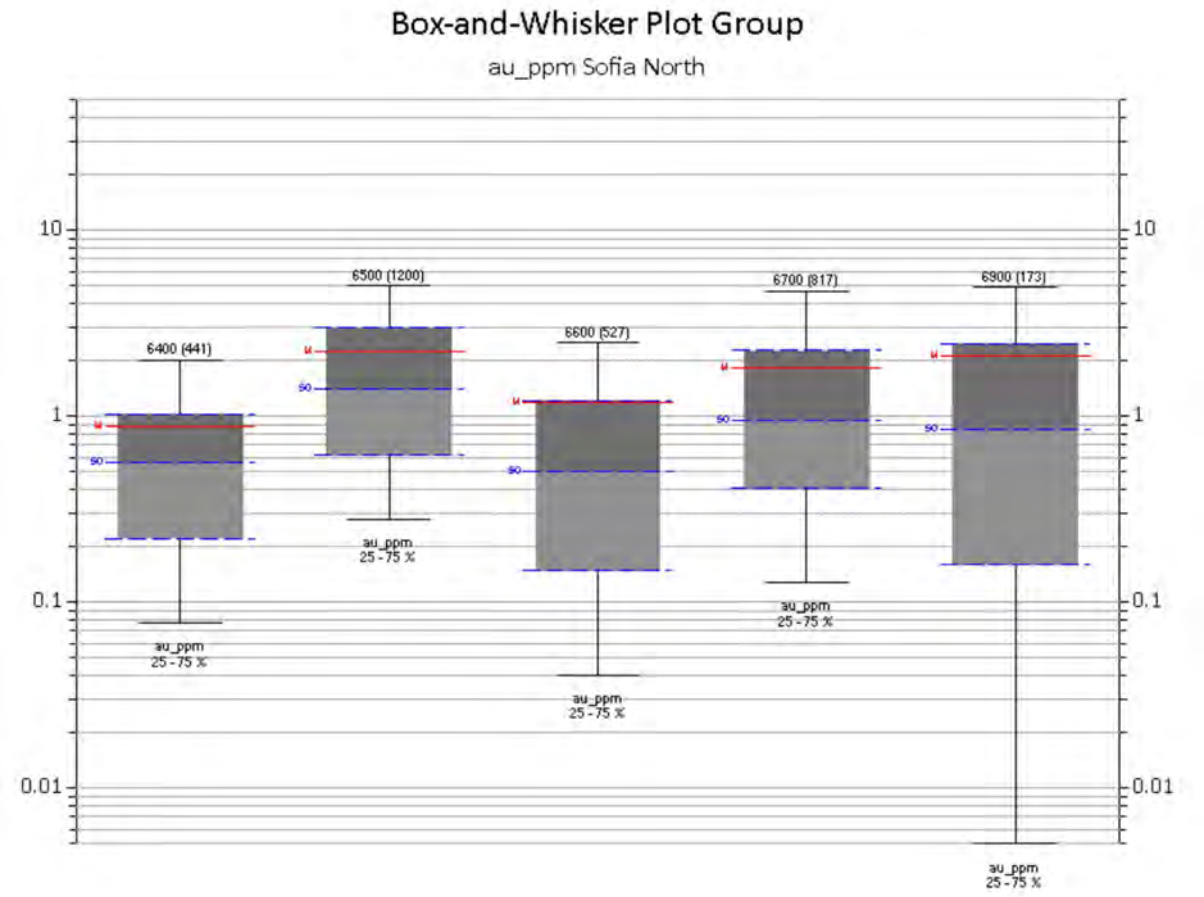


Figure 14-19 Sofia North Mineralisation Zone Domain Grade Distribution Box and Whisker Uncut Grade Distribution Comparison

(6400 – FWOZ, 6500 – MOZ, 6600 – HWOZ, 6700 – HW_Vein 6900 HWOZ1)

Contact profile analysis between the mineralised domains and the surrounding main mineralised zone shows minimal to no change in grade across the boundaries of the HW and FW mineralisation zones. These domains also intersect the main mineralisation zones in Sofia Main. Subsequently, a soft boundary is applied for data between domains 6000, 6100, and 6200 and a hard boundary for domains 6000 and 6300 for the purposes of Mineral Resource estimation (Figure 14-20). This is further supported by the fact that there are minimal differences in the grade distributions in the MOZ relative to the HWOZ and FWOZ.

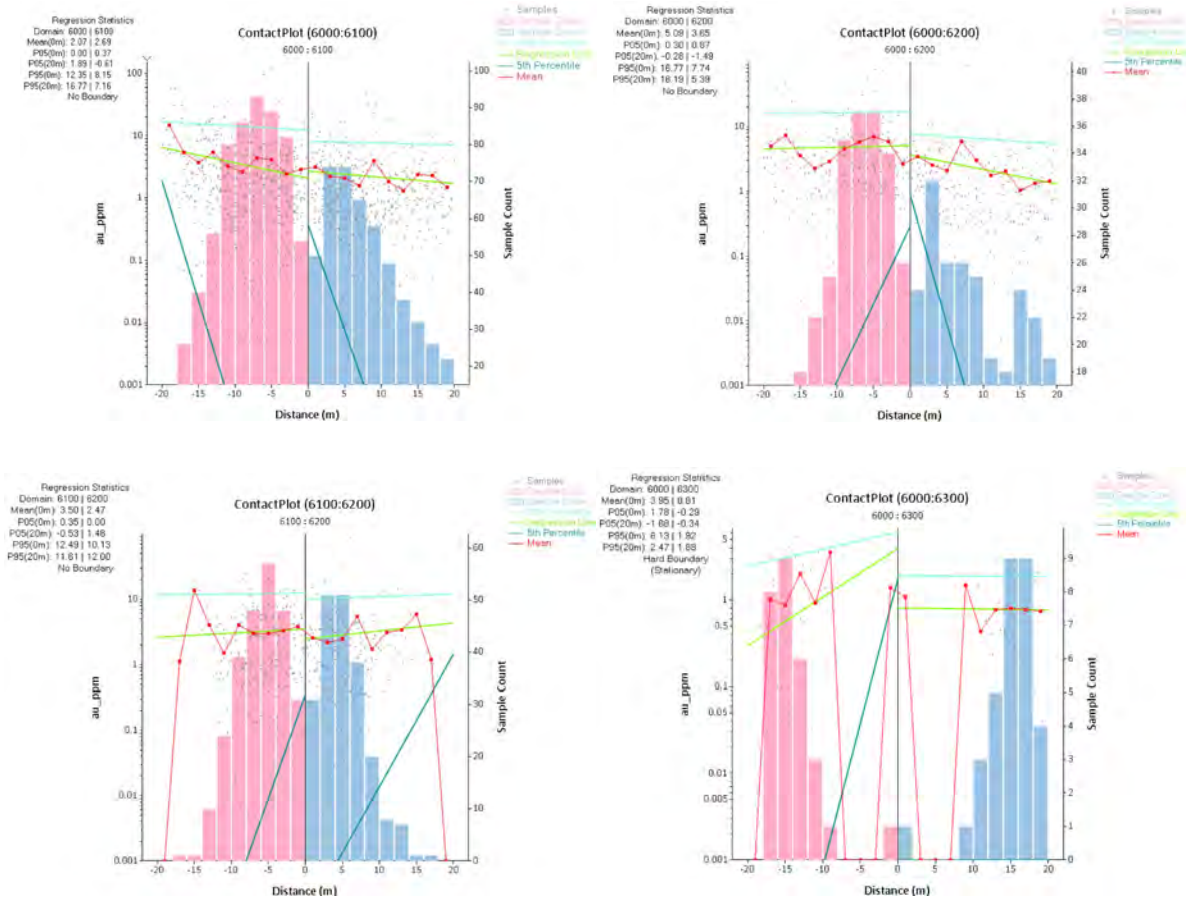


Figure 14-20 Contact Analysis Plots of Sofia Main Mineralised Domains (6000 – MOZ, 6100 – FWOZ, 6200 – FW Vein, 6300 – MOZ North)

There are still some indications of mixed population within the low/medium-grade domain – likely to be caused by small isolated high-grade pods. However, with the current drill density this cannot be spatially constrained.

Contact profile analysis (Figure 14-21) between the mineralised domains shows a gradational to no change in grade across the boundary. However, geological observations show that the change in mineralisation and alteration styles is distinct and, consequently, a hard boundary was applied between the mineralisation wireframes.

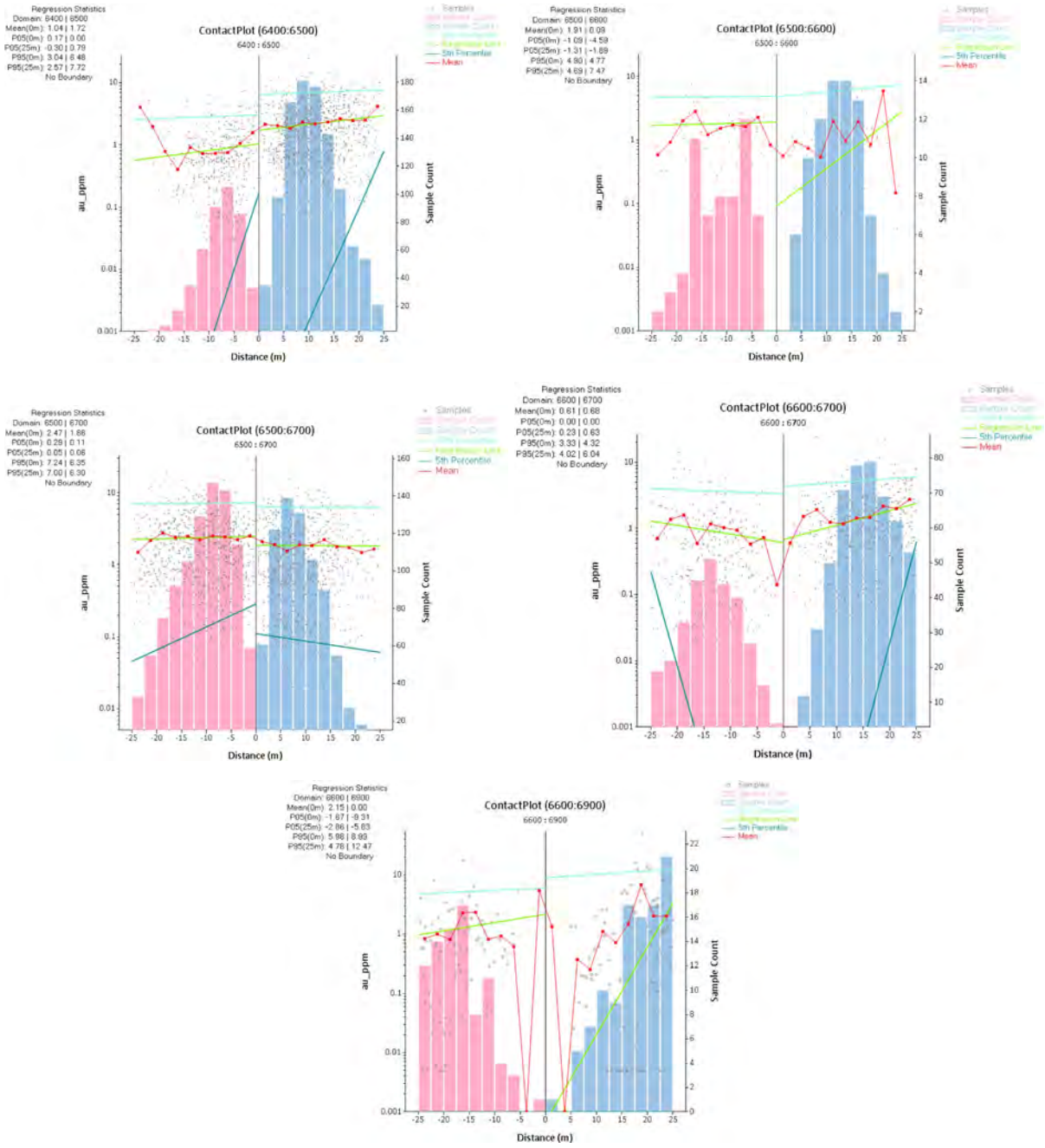


Figure 14-21 Contact Analysis Plot of Sofia North Mineralisation Domains (6400 – FWOZ, 6500 – MOZ, 6600 – HWOZ, 6700 – HW_Vein 6900 HWOZ1)

14.2.4. Delya

Delya is split into the following domains for resource estimation purposes:

- 6000 = Delya MOZ
- 6100 = Delya HWOZ
- 6200 = Delya 6200

The HW and FW mineralisation zones in Delya do not intersect and, subsequently, no soft boundary is applied between data.

The resultant separate grade populations show a significant improvement in stationarity of domains relative to the original distribution for application during resource estimation (Figure 14-22).

Figure 14-22 Delya Mineralisation Zone Domain Grade Distribution Box and Whisker Uncut Grade Distribution Comparison

(6000 – MOZ, 6100 – HWOZ, 6200 – HWOZ_2)

14.2.5. Tina and Bambaraya

All available data has been used to estimate each of these satellite deposits including DD, RC drilling, and trenches.

Table 14-2 summarises the important statistics of each of the deposits. Data density varies from 80 m to 100 m for the closest spaced drilled deposit to 200 m to 250 m for the widest space deposit.

Table 14-2 Estimation Database Statistics for Massawa Satellite Resources

Satellite	Number of Composites	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	CV (g/t Au)
Bambaraya	203	0.005	11.30	1.71	1.37
Tina	527	0.01	20.39	0.59	1.88

14.3. Compositing

Massawa, Sofia, and Delya Mineral Resources have been assessed independently to optimise the most appropriate compositing length, by checking the impact of the composite length on the mean, coefficient of variation, and skewness of the resultant composite distribution relative to that of the original sample distribution. Additional consideration is given to the relevant domain true widths and the average raw sample length.

All drill holes are effectively composited using estimation domain wireframes as boundaries to the compositing process, thereby ensuring that the resultant composites respect the modelled boundaries between any estimation domains, e.g. NZ main mineralisation wireframe and internal high-grade domain.

Massawa CZ trench and DD samples, as well as Massawa NZ, Sofia, and Delya Mineral Resource estimation sample datasets were composited to one metre composites within their respective estimation domain wireframes with a tolerance of 0.5 m. This results in a maximum composite size of 1.5 m and a minimum composite size of 0.5 m, however, over 95% of the composites are equal to 1.0 m. Any residual composites with a length below 0.5 m were excluded from the final estimation dataset.

Massawa CZ RC samples used in the Mineral Resource estimation sample datasets were taken on one metre intervals, therefore the samples were flagged within their respective estimation domain wireframes, giving one-metre composites.

A summary of the residuals removed from all datasets is outlined in Table 14-3. The total number of ignored data represents less than 0.02% of the total estimation dataset thereby ensuring that the effective exclusion of these residuals does not bias the grade distribution within the remaining dataset. This compositing methodology, and the flagging for Massawa RC samples, is considered to be the most appropriate approach, as it ensures that the resultant composite datasets comprise homogenous sample weights.

Table 14-3 Summary Table of Residuals Removed from All Massawa Datasets

Resource	Number of Residuals	Grade (g/t Au)
Massawa North Zone	111	4.40
Massawa Central Zone	22	2.08
Sofia	24	0.70
Delya	-	-

14.4. Top Cutting and Grade Restrictions

Top cutting is determined using a multifactor analysis, whereby an appropriate top cut is assessed utilising individual methods of analysis including raw composite percentile disintegration, analysis of the grade distribution, and log probability plots. Generally, the chosen top cutting value is expected to occur between the 95th and 99.9th percentiles. Subsequently, the final top cutting value is chosen taking into consideration the results of each of the methods of analysis.

Prior to applying the final top cutting value, all data points with results at or above the selected top cut value are spatially reviewed to determine if the sample points are spatially spread out illustrating typical outlier characteristics and cannot be domained as discrete high-grade zone with a separate distribution. If they are spatially associated with one another, then the application of a separate high-grade domain is tested and the top cutting assessments are repeated on each of the new domains.

14.4.1. North Zone

NZ sample grade distributions were assessed using conventional multifactor analysis methods. The grade distributions for NZ1 domains were grouped for the purposes of top cut optimisation. The grade distributions for NZ2 were also grouped for the purposes of top cut optimisation but split by domains with similar grade distributions.

The resultant grade distribution for NZ1 domain 1103 indicated that samples over 13 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 13 g/t Au was chosen as the appropriate top cutting value and a total of three samples were top cut between 13.5 g/t Au and 26.10 g/t Au. Top cutting reduced the average mean grade by 10% from 1.66 g/t Au to 1.62 g/t Au and resulted in a reduction of the coefficient of variation from 1.45 to 1.3.

The resultant grade distribution for NZ1 domain 1104 indicated that samples over 27 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 27 g/t Au was chosen as the appropriate top cutting value and a total of 11 samples were top cut between 26.5 g/t Au and 59.2 g/t Au. Top cutting reduced the average mean grade by 14% from 3.43 g/t Au to 3.22 g/t Au and resulted in a reduction of the coefficient of variation from 1.86 to 1.6.

The resultant grade distribution for NZ1 domain 1105, 1106, 1107, and 1108 indicated that samples over 21 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 21 g/t Au was chosen as the appropriate top cutting value and a total of 20 samples were top-cut between 21 g/t Au and 41.17 g/t Au. Top cutting reduced the average mean grade by 9% from 2.8 g/t Au to 2.73 g/t Au and resulted in a reduction of the coefficient of variation from 1.49 to 1.36.

Log histograms and log probability plots showing the overall grade distributions for the NZ domains are shown in Figure 14-23 to Figure 14-25.

An additional restricted search ellipse constraint has been placed on all samples over 12 g/t Au in domain 1104 and above 21 g/t Au in domains 1105, 1106, 1107, and 1108 where partial disintegration point exists within the top 3% to 4% of the total high-grade sample population. This search restriction represents 50% of the total variogram range for their respective domains.

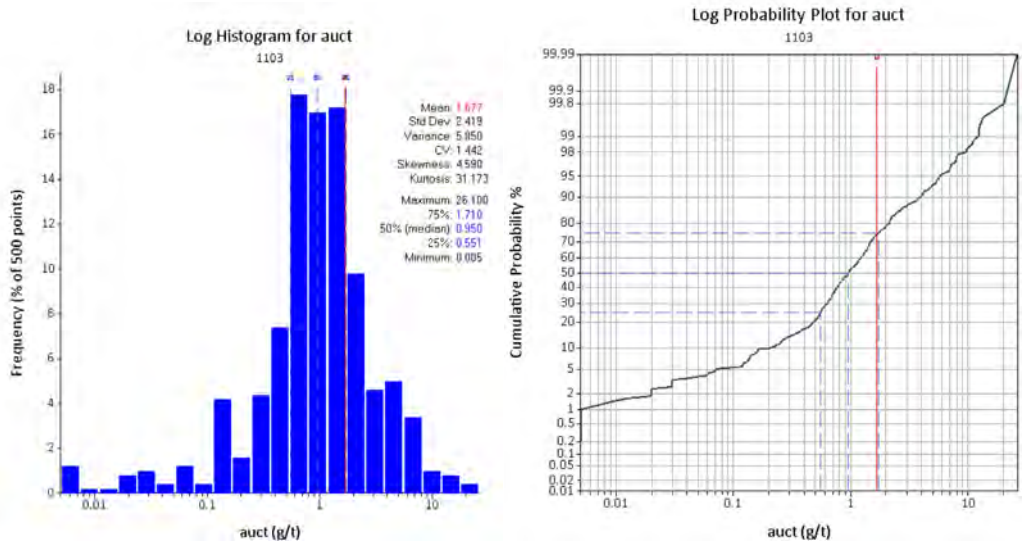


Figure 14-23 Massawa North Zone 1 Domain 1103 Log Histogram and Log Probability Plot of 1 m Composite with 0.5 m Merge Estimation Data Situated Within the NZ1 Mineralisation Wireframes

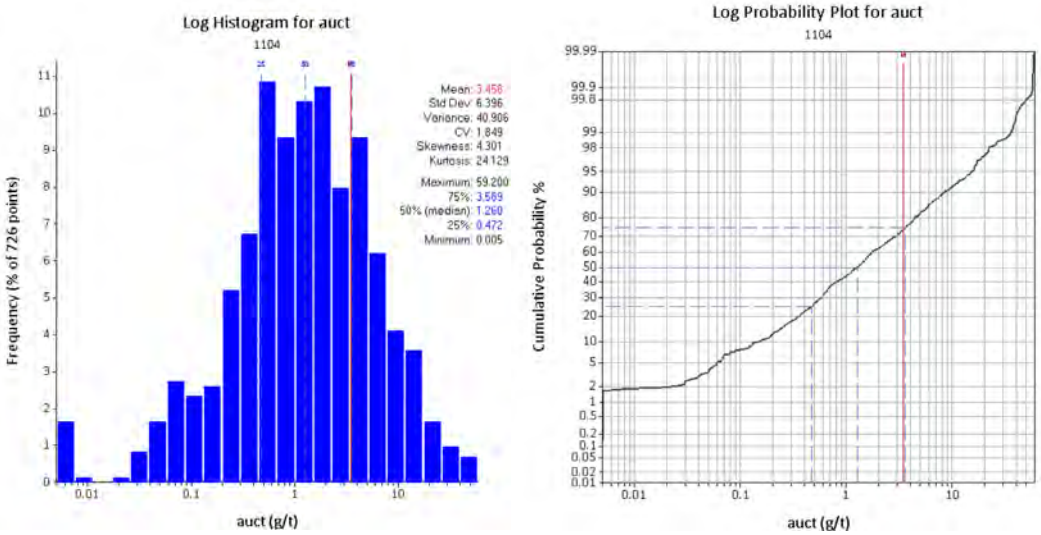


Figure 14-24 Massawa North Zone 1 Domain 1104 Log Histogram and Log Probability Plot of 1 m Composite With 0.5 m Merge Estimation Data Situated Within the NZ1 Mineralisation Wireframes

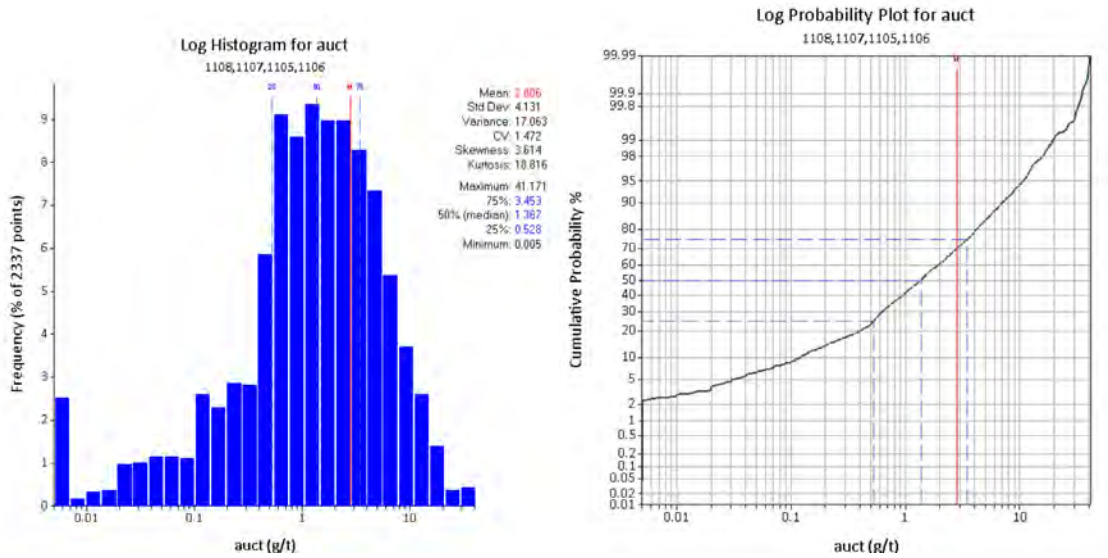


Figure 14-25 Massawa North Zone 1 Domains 1105, 1106, 1107 and 1108 Log Histogram and Log Probability Plot of 1 m Composite with 0.5 m Merge Estimation Data Situated Within the NZ1 Mineralisation Wireframes

The resultant grade distribution for NZ2 domain 2111 (high-grade) indicated that samples over 65 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 65 g/t Au was chosen as the appropriate top cutting value and a total of eight samples were top-cut between 65 g/t Au and 76 g/t Au. Top cutting reduced the average mean grade by 3% from 10.88 g/t Au to 10.86 g/t Au and resulted in a reduction of the coefficient of variation from 0.88 to 0.87.

The resultant grade distribution for NZ2 domains 2101 and 2103 indicated that samples over 39 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 39 g/t Au was chosen as the appropriate top cutting value and a total of six samples were top-cut between 39 g/t Au and 85 g/t Au. Top cutting reduced the average mean grade by 0.93% from 3.22 g/t Au to 3.19 g/t Au and resulted in a reduction of the coefficient of variation from 1.63 to 1.55.

The resultant grade distribution for the low-grade and poorly sampled NZ2 domain 2102 indicated that samples over 7 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 7 g/t Au was chosen as the appropriate top cutting value and a total of five samples were top-cut between 7 g/t Au and 17.58 g/t Au. Top cutting reduced the average mean grade by 9% from 1.41 g/t Au to 1.27 g/t Au and resulted in a reduction of the coefficient of variation from 1.7 to 1.3.

Log histograms and log probability plots showing the overall grade distributions for the NZ1 domains are shown in Figure 14-26 to Figure 14-28.

An additional restricted search ellipse constraint has been placed on all samples over 48 g/t Au in domain 2111 and above 21 g/t Au in domains 2101, 2102, 2103, 1105, 1106, 1107, and 1108 where partial disintegration point exists within the top 3% to 4% of the total high-grade sample population. This search restriction represents 50% of the total variogram range for their respective domains.

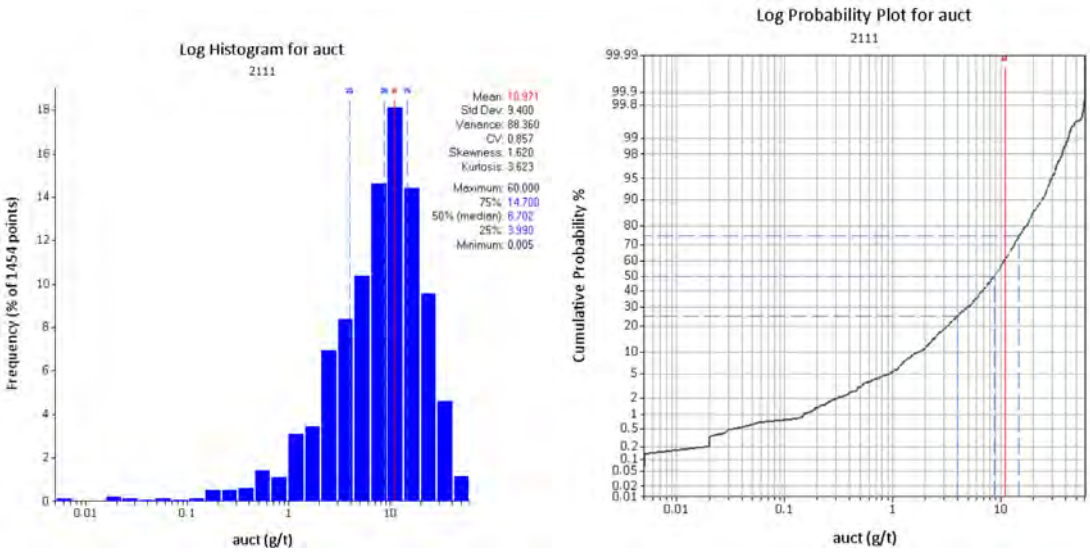


Figure 14-26 Massawa North Zone 2 Log Histogram and Log Probability Plot of 1 m Composite with 0.5 m Merge Estimation Data Situated Within NZ2 High-Grade Domain 2111

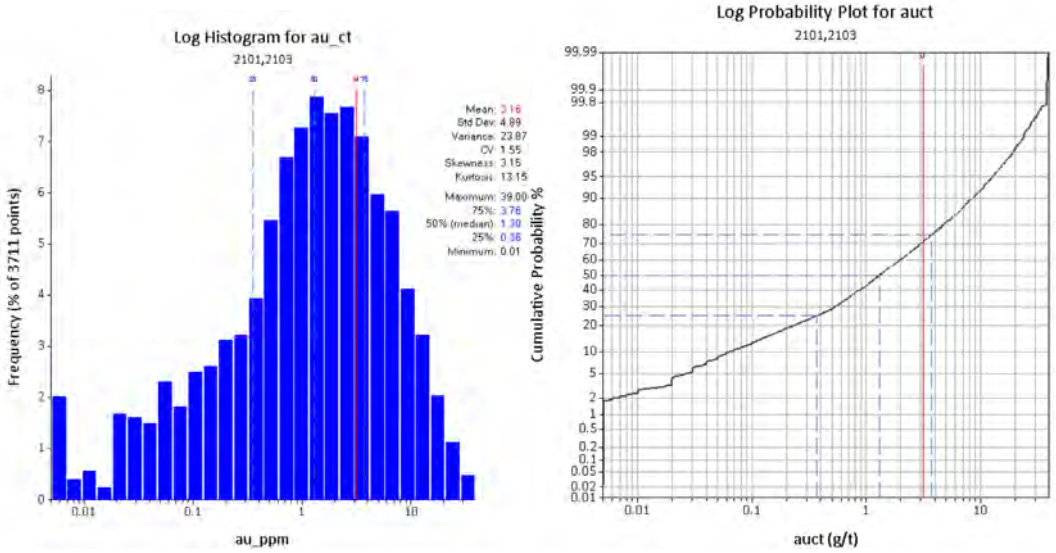


Figure 14-27 Massawa North Zone 2 Log Histogram and Log Probability Plot of 1 m Composite with 0.5 m Merge Estimation Data Situated Within Domains 2101 and 2103

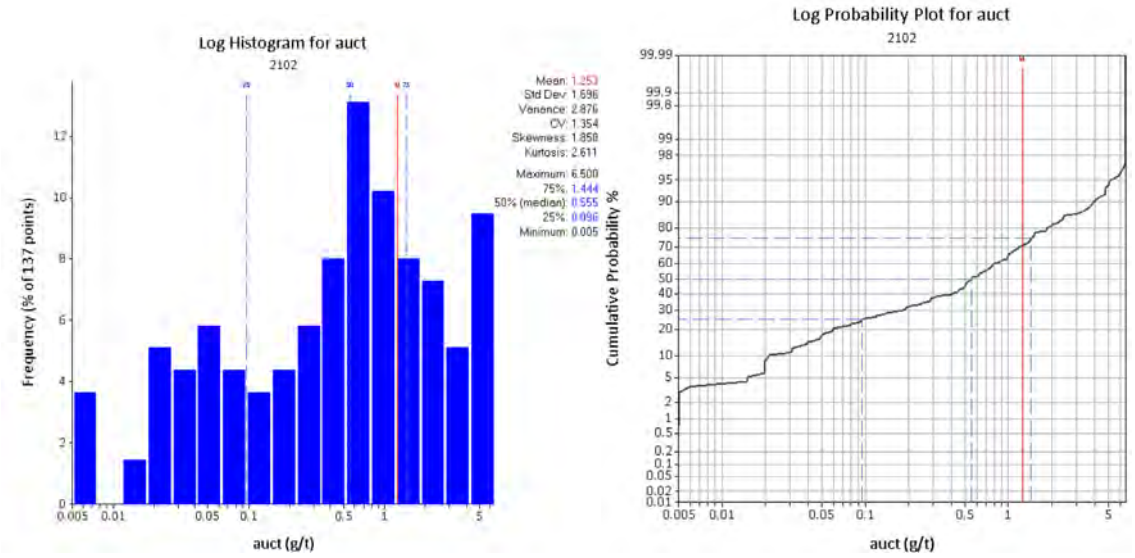


Figure 14-28 Massawa North Zone 2 Log Histogram and Log Probability Plot of 1 m Composite with 0.5 m Merge Estimation Data Situated Within Domain 2102

14.4.2. Central Zone

All CZ GC samples top cutting optimisations were restricted to RC drill data with bulk sample LeachWELL plus tail assay determination. As outlined in Section 11, due to the presence of nuggety gold in the Massawa CZ, bulk one-kilogram LeachWELL samples provide not only more representative quantification of gold content but also greater levels of repeatability (>85%) relative to 50 g FA data. In addition, top cutting was optimised with bulk sample LeachWELL plus FA data to ensure that the assessment of what is considered as an outlier from the main grade distribution was not biased or skewed by the known negative bias of sample grades within the smaller volume diamond drill data with 50 g FA. CZ top cutting outside the RC GC area, with no bulk sample LeachWELL data, included smaller volume DD data with the 50 g FA.

Due to the separate grade distributions of CZ mineralisation, the domains were grouped for top cutting based on similarities in grade distributions (Figure 14-29 and Figure 14-30). Domains in each top cutting group domain are outlined in Table 14-4.

Table 14-4 List of Mineralisation Domains in Top Cutting Groups

Top Cutting Group	Mineralisation Domains
1	All low-grade domains 6101 - 6712
2	7110, 7210, 7470, 7690, 7631, 7660, 7630, 7410, 7610, 7120, 7640, 7710, 7434, 7460, 7440, 7420, 7101, 7104, 7502, 7691, 7650, 7490, 7670, 7102, 7711, 7671, 7473,
3	7410, 7420, 7480, 7610, 7630, 7490, 7690, 7691, 7470, 7460, 7640, 7440, 7210, 7660, 7710, 7110, 7631, 7502, 7473, 7101, 7305, 7102, 7501, 7472, 7670, 7650, 7621, 7671, 7304, 7120, 7430, 7450, 7434
4	7470, 7430, 7450, 7434, 7410, 7472, 7471, 7443, 7441, 7502, 7473, 7444, 7420

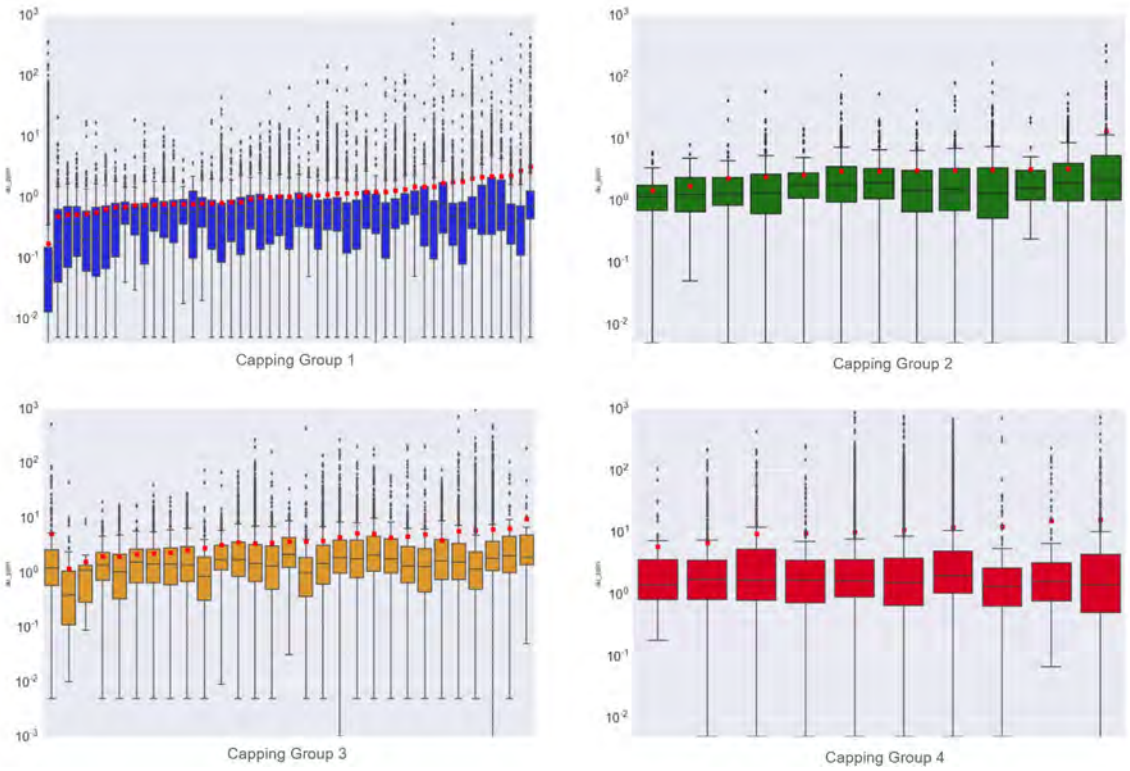


Figure 14-29 Grade Distributions of Grouped Central Zone Low-Grade and High-Grade Domains by Top Cutting Groups

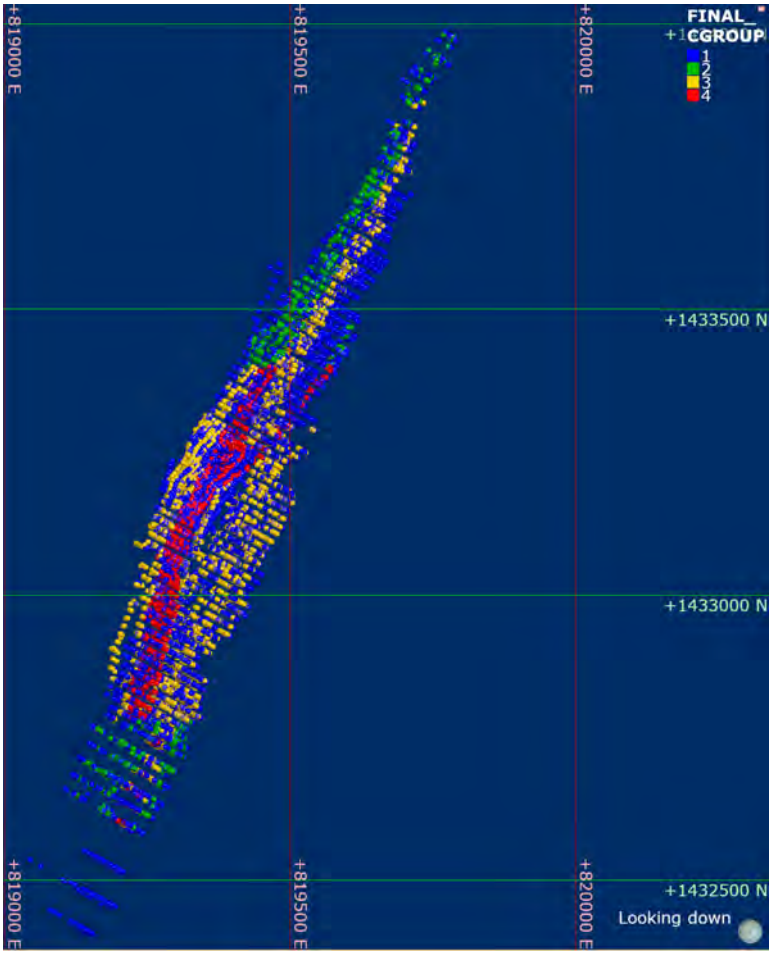


Figure 14-30 Spatial Distributions of Massawa Central Top Cutting Groups 1 to 4

Low Grade

Grade distributions in the CZ low-grade domains (top cutting group 1) were assessed using conventional multifactor analysis methods. Grade distributions were assessed both independently for each of the mineralised zones and also as grouped domains to compare the results of the top cut evaluations (Table 14-5).

Table 14-5 Massawa CZ Top Cutting Statistics by Top Cutting Group

Top Cutting Group	No of Samples	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	CV (g/t Au)	Top Cut Value (g/t Au)	Mean Top Cut (g/t Au)	CV Top Cut (g/t Au)	No of Top Cut Samples	Metal Reduction
1	29,159	0.00	716.6	1.40	6.80	55	1.17	3.13	75	-16%
2	6,722	0.01	712.2	3.30	4.11	150	3.09	2.33	8	-6%
3	14,645	0.00	957.0	4.60	3.97	150	4.30	2.78	37	-7%
4	8,601	0.01	1,636.5	11.23	5.10	465	10.12	3.86	25	-10%

An additional restricted search ellipse constraint has been placed on all low-grade samples above 28 g/t Au within the GC domains, where the search ellipse is restricted to 33% of the total variogram range (21 m), which is almost equal to the range of the third structure.

Within the wider spaced exploration domains, the search ellipse for estimation of all samples above 11 g/t Au is restricted to 66% of the total variogram range (42 m), which is almost equal to the range of the second structure. The search ellipse restriction is reduced in the exploration domains relative to the GC domains, due to the wider spacing of the drill sample data.

This combination of a restricted search and extreme outlier top cutting is considered to be the most appropriate method for estimation within a nuggety gold system, as it retains the areas of local very high-grade material but restricts their influence whilst also top cutting the actual outliers. The resultant impact on the contained metal ounces is the same as the impact from applying a straight top cut.

High Grade

The grade distributions were assessed both independently for each mineralised zone and then grouped to compare the results of the top cut evaluations. Due to the partial coverage of the RC data with bulk LeachWELL assay determination, some lodes were under sampled with respect to others and as such the grouped distribution was used to determine the relevant top cut.

Due to the nuggety nature of the high-grade (proximal) domains, the range of potential top cuts established using conventional multifactor analysis methods varied significantly (by >200%). Accordingly, it was necessary to run an additional ‘metal at risk’ simulation which has been utilised at other nugget vein gold mine operations. The metal at risk simulation was run on all 15 m by 10 m AdvGC RC drilling analysed using bulk LeachWELL analysis. The simulation was restricted to a tight estimation data analysis envelope area that is restricted to half the drill spacing away from the last GC grid intersection.

The simulation starts by using the lowest (harshest) ‘possible top cut’ that has been established from conventional multifactor analysis (histogram, log probability, disintegration etc.). Using this

'possible top cut', the 'top cut metal content' and simulated high-grade metal distribution are calculated. From the simulated high-grade metal distribution, the 'target metal content' is calculated using 20% percentile (Figure 14-31).

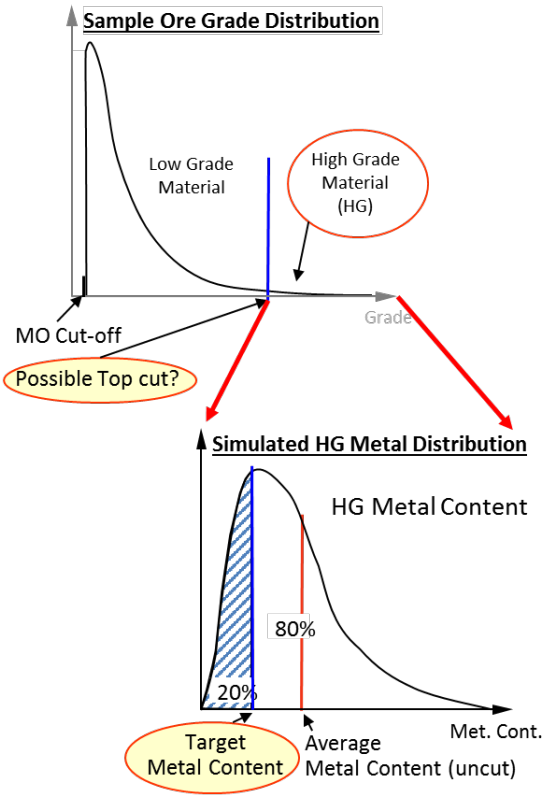


Figure 14-31 Metal at Risk Simulation Grade Distribution Algorithm Graphical Representation

The simulation tests if the 'top cut metal content' is greater than 'target metal content' and if it is, the risk of over estimation is considered to be too significant. The simulation process is repeated using different 'possible top cuts' until the 'cut metal content' is equal to 'target metal content'. The simulation 'target metal content' is calculated using 20% percentile of the simulated high-grade metal distribution – such that a minimum of 80% of the mine production will achieve or exceed the predicted grade of the model assuming no other modifying factors. Approximately 20% of the mine production will achieve or fall below the predicted grade of the model assuming no other modifying factors. The simulation runs 5,000 'realisations' for each given production rates. A number of production rates were tested, however, taking into consideration the current mine schedule, the 1.2 Mtpa simulation (average CZ feed over the LOM) where a maximum of 40% (480 kt) of the production tonnes would be from high-grade domains was selected. The results of these simulations are tabulated in Table 14-6 for each top cutting domain individually within the CZ except for domains 2 and 3 which were combined due to comparable grade distributions and variability plus geological setting. Selection of the final top cutting value took into consideration both results of the metal at risk simulation together with other methods (Table 14-6). Additional restricted estimation search ellipse radii were applied at approximately the 99% percentile to reduce the spatial influence of the upper most portion of the high-grade distribution tail.

Table 14-6 Massawa Central Zone Top Cutting Values by Method

Capping Group	MAR Cap (g/t)	Decile Analysis Cap Value (g/t)	Final Cap Value (g/t)	High Yield Search Restriction (g/t)
1	65	35	55	28
2 (HW & HW Bridge Domains Only)	38	17	150	28
2	38	17	150	-
3	173	150	150	-
4	488	460	465	230

14.4.1. Sofia

Sofia sample grade distributions were assessed using conventional multifactor analysis methods. The grade distributions for Sofia Main domains were grouped for the purposes of top cut optimisation. The grade distributions for Sofia North were split by geological domains with similar grade distributions for the purposes of top cut optimisation.

The resultant grade distribution for Sofia Main indicated that samples over 25 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 25 g/t Au was chosen as the appropriate top cutting value, and a total of 237 samples were top-cut between 27.79 g/t Au and 122.92 g/t Au. Top cutting reduced the average mean grade by 9% from 2.88 g/t Au to 2.63 g/t Au and resulted in a reduction of the coefficient of variation from 2.16 to 1.44. The resultant grade distribution and log probability curve are shown in Figure 14-32.

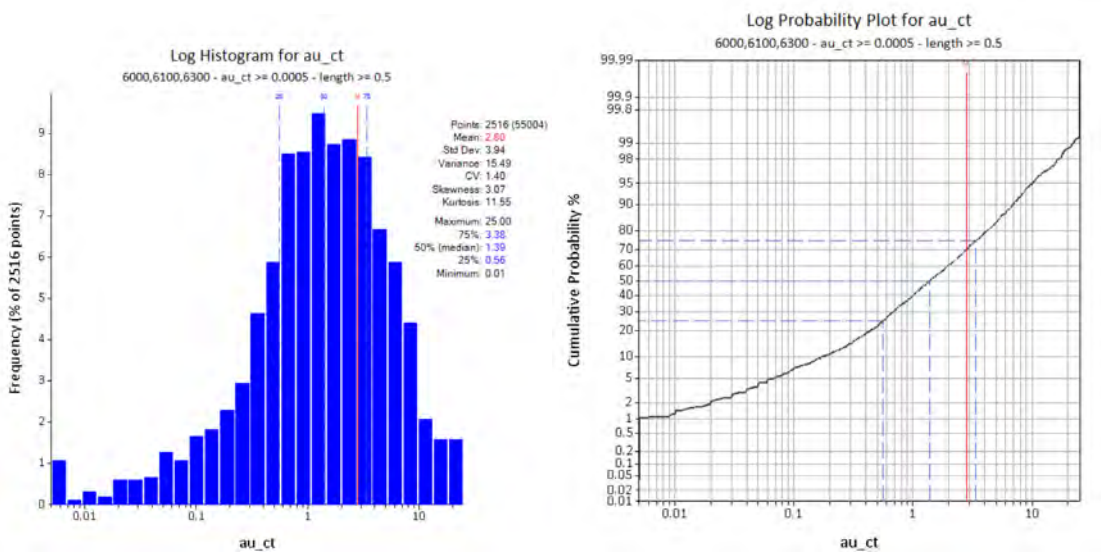


Figure 14-32 Massawa Sofia Main Log Histogram and Log Probability Plot of 1 m Composite with 0.5 m Merge Estimation Data Situated Within Mineralisation Wireframes

The resultant grade distribution for Sofia North was divided into domains with similar grade distributions. The grade distributions for domains 6500, 6700, and 6900 over 16 g/t Au and for domains 6400 and 6600 over 7 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, for domains 6500, 6700, and 6900, 16 g/t Au was chosen as the appropriate top cutting value and a total of nine samples were top-cut between 16.68 g/t Au and 90.47 g/t Au. Top cutting reduced the average mean grade by 2% from 1.84 g/t Au to 1.79 g/t Au and resulted in a reduction

of the coefficient of variation from 1.45 to 1.3. In total, the length weighted metal reduction impact of the top cut is also 2%. The resultant grade distribution and log probability curve are shown in Figure 14-33.

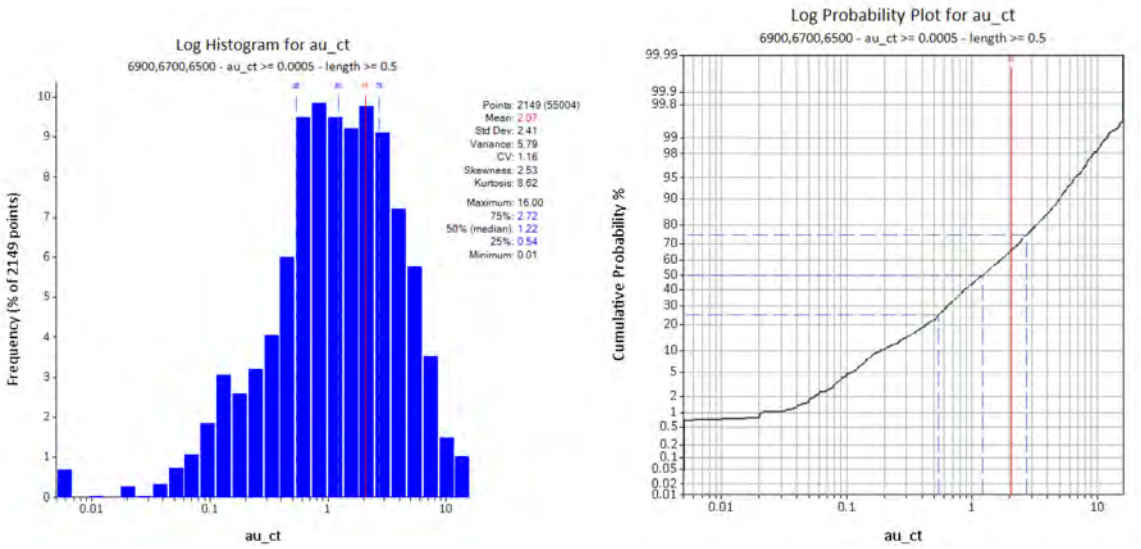


Figure 14-33 Massawa Sofia North Domains 6500, 6700 and 6900 Log Histogram and Log Probability Plot of 1 m Composite With 0.5 m Merge Estimation Data Situated Within Mineralisation Wireframes

For domains 6400 and 6600, 7 g/t Au was chosen as the appropriate top cut, and a total of 17 samples were removed between 7.51 g/t Au and 22.5 g/t Au. Top cutting reduced the average mean grade by 8.13% from 1.04 g/t Au to 0.97 g/t Au and resulted in a reduction of the coefficient of variation from 1.81 to 1.37. The resultant grade distribution and log probability curve are show in Figure 14-34.

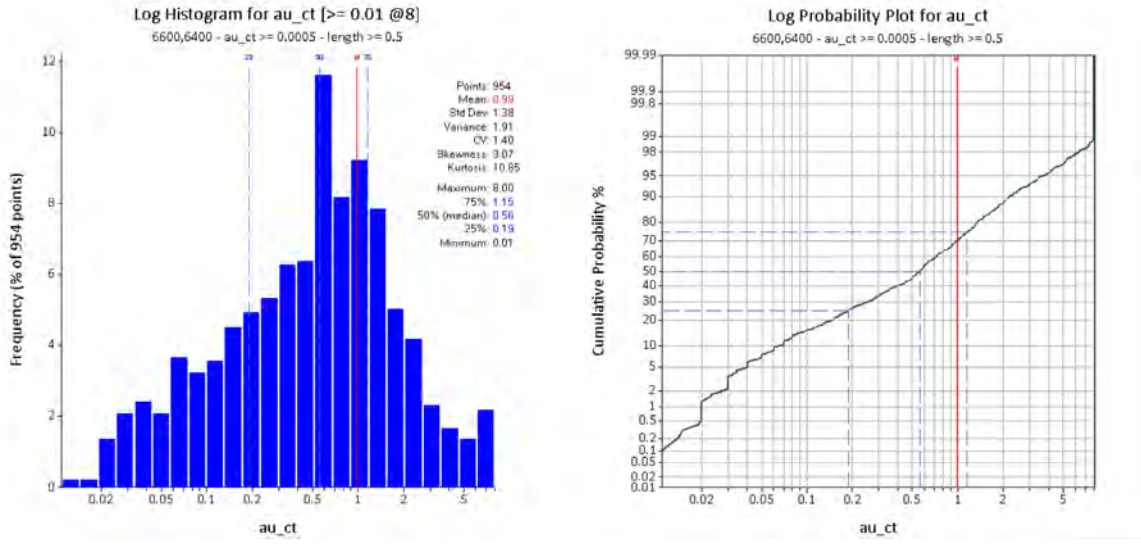


Figure 14-34 Massawa Sofia North Domains 6600 and 6400 Log Histogram and Log Probability Plot of 1 m Composite with 0.5 m Merge Estimation Data Situated Within Mineralisation Wireframes

An additional restricted search ellipse constraint has been placed on all samples over 17 g/t Au in Sofia Main and above 13 g/t Au in Sofia North domains 6500, 6700, and 6900, where a partial disintegration point exists within the top 3% to 4% of the total high-grade sample population. This

search restriction represents 50% of the total variogram range for Sofia Main or the ranges of the range of variogram structure for Sofia North.

14.4.2. Delya

Delya sample grade distributions were assessed using conventional multifactor analysis methods. The grade distributions for Delya domains 6200 and 6300 were grouped for the purposes of top cut optimisation.

The resultant grade distribution (Figure 14-35) for Delya domain 6000 (MOZ) indicated that samples over 22 g/t Au did not fit the main sample distribution and displayed typical outlier characteristics as they were spatially spread along the strike of the mineralised wireframe. As such, 22 g/t Au was chosen as the appropriate top cutting value and a total of 11 samples were top-cut between 22.3 g/t Au and 31.50 g/t Au. Top cutting reduced the average mean grade by 9% from 4.29 g/t Au to 4.25 g/t Au and resulted in a reduction of the coefficient of variation from 1.16 to 1.15. In total, the length weighted metal reduction impact of the top cut is also 2.3%. The grade distribution for combined Delya domains 6100 and 6200 (Figure 14-36) indicated that samples over 8.5 g/t Au did not fit the main sample distribution. As such 8.5 g/t Au was selected as the appropriate top cut value and a total of five samples were top-cut.

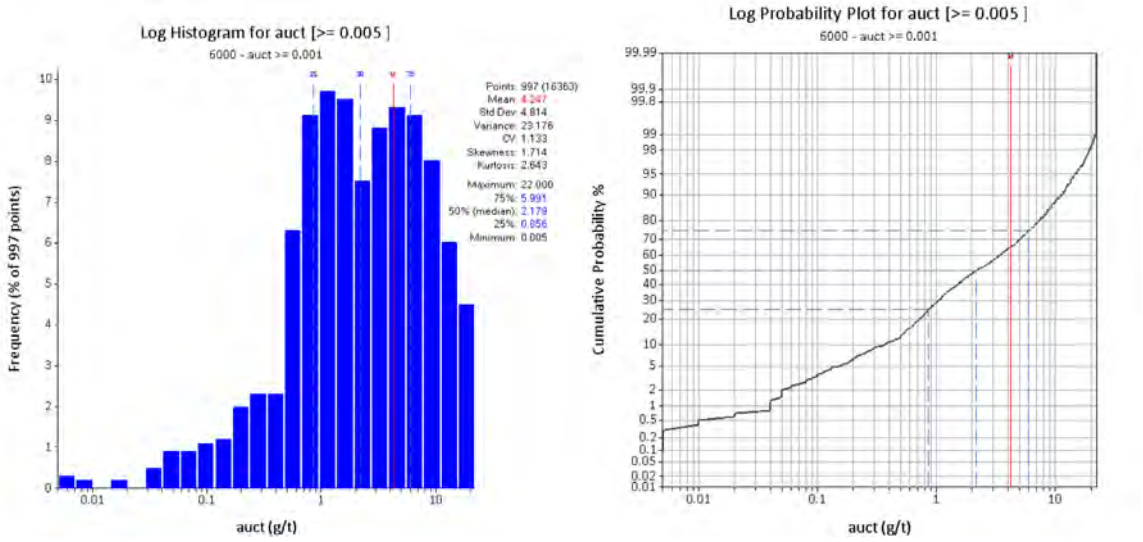


Figure 14-35 Delya Domain 6000 Log Histogram and Log Probability Plot of 1 m Composite With 0.5 m Merge Estimation Data Situated Within Mineralisation Wireframes

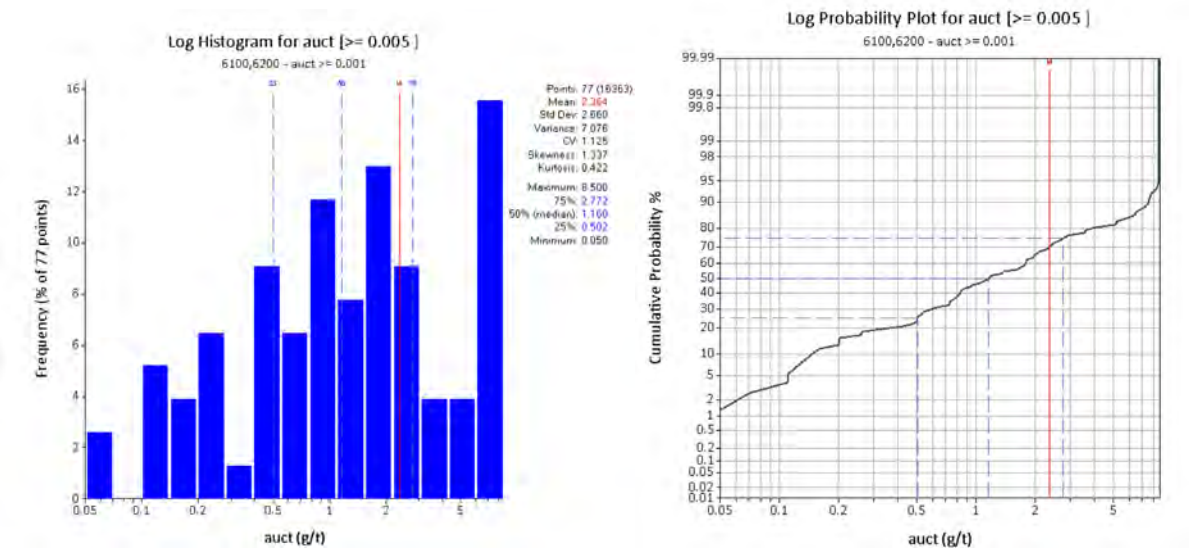


Figure 14-36 Delya Domain 6100 and 6200 Log Histogram and Log Probability Plot of 1 m Composite Width 0.5 m Merge Estimation Data Situated Within Mineralisation Wireframes

An additional restricted search ellipse constraint has been placed on all samples over 16 g/t Au in Delya domain 6000, where partial disintegration point exists within the top 3% to 4% of the total high-grade sample population. This search restriction represents 100% of the total variogram range for this domain.

14.5. Massawa, Sofia and Delya Estimation Dataset Declustering

The sample data in Sofia Main and Sofia North are not heavily clustered as the drilling is fairly evenly spaced and there are no areas of GC drilling. Subsequently, it was decided that no declustering correction was required.

The sample data in NZ, CZ, and Delya zones are heavily clustered in areas due to influence of GC drilling. Subsequently, a cell weighting declustering correction was applied to the domain to effectively account for the clustering of data within geostatistical analysis and variography. The cell weighting declustering technique applied used a linked Y and Z axis to effectively increase the accuracy of the declustering cell size selection.

Declustering correction is used within geostatistical analysis only and is not applied to estimation values because kriging will decluster the data. The final declustering cell size selections are detailed in Table 14-7, Table 14-8, and Table 14-9.

Table 14-7 Massawa North Zone Decluster Cell Size Results by Domain

Domain	Decluster Cell Size (X by Y by Z)	Raw Mean Grade (g/t Au)	Declustered Mean Grade (g/t Au)	% Change in Mean Grade
NZ1	6 m by 120 m by 6 m	2.82	2.46	-14
NZ2	8 m by 24 m by 80 m	4.83	4.77	1.26

Table 14-8 Massawa Central Zone Decluster Cell Size Results

Domain	Decluster Cell Size (X by Y by Z)	Raw Mean Grade (g/t Au)	Declustered Mean Grade (g/t Au)	% Change in Mean Grade
All	20 m by 12 m by 20 m	3.47	2.48	-28.5%

Table 14-9 Delya Decluster Cell Size Results

Domain	Decluster Cell Size (X by Y by Z)	Raw Mean Grade (g/t Au)	Declustered Mean Grade (g/t Au)	% Change in Mean Grade
Main(all)	54 m by 10 m by 10 m	4.11	3.79	-7.8%

14.6. Variography

Exploratory data analysis (EDA) was conducted by Randgold using Snowden Supervisor statistical software and all modelling and estimation was completed in Maptek Vulcan. Variography was completed using Snowden Supervisor v8; to analyse the spatial continuity and relation within each of the main domains to determine appropriate search strategy and estimation inputs. The variogram modelling process followed involved the following steps:

- A normal score transform was applied to all data prior to undertaking variography on the top cut, declustered composite dataset, where declustering was appropriate.
- Both the omni-directional and down hole variogram are initially modelled to characterise the nugget effect.
- Orientated variogram directions are modelled in three dimensions and verified with the known geological or structural controls to mineralisation to identify the plane of greatest continuity.
- Variogram fans within the plane of greatest continuity are modelled for each structure to identify the direction of maximum continuity within this plane.
- Experimental variograms are modelled in the direction of maximum continuity and the orthogonal directions.
- A back transform to all independent structure experimental variogram models was applied to obtain the appropriate nested variogram models for interpolation of raw composite data.

Prior to completing the final estimation, each semi-variogram model is cross validated to ensure that there is no or minimal bias in estimated grade relative to the actual sample grade. This was completed by plotting the resultant estimated grades from the first pass compared to the actual sample grade, to evaluate the resultant estimation for possible trends. In exploration data spaced sub-domains, there is level of smoothing in estimated grade compared to actual sample grade, but the overall estimated grade is checked to ensure that it is within 5% of the original sample grade and that there is no evidence of conditional bias.

The search strategy used was based on the modelled variogram ranges considering that in nugget shear hosted Au systems the full variogram range is often artificially extended for the last 3% to 5% of variability. Additionally, the estimation search strategy takes into consideration the data distribution for both the geological and data spacing sub-domains.

14.6.1. Massawa North Zone

Only one variogram model was constructed for NZ1, as there are no differences in the grade distributions in the MOZ relative to the HW and FW zones and, due to their bifurcating nature, the wireframes all intersect one another. Subsequently, a soft boundary was applied between data in all three domains for the purposes of resource estimation. The variogram model parameters as defined are included in Table 14-10.

Table 14-10 Massawa North Zone 1 Maptek Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.6	210	0	85	23	16	3
Spherical	0.18	210	0	85	32	30	10
Spherical	0.02	210	0	85	64	50	16
Nugget	**0.2**						
Total Sill	**1**						

Separate variogram models were constructed for NZ2 MOZ and the internal high-grade domain, as a function of the significant differences of the grade distributions and continuity within each of the two domains. Consequently, the high-grade domain variogram models exhibited shorter ranges of continuity relative to that of the MOZ domain (Table 14-11 and Table 14-12). This also supports the application of a smaller/restricted search ellipse within the high-grade domain.

The MOZ variogram model was applied to all HW and FW domains due to minimal difference in the grade distributions and continuity. Furthermore, the HW and FW wireframes do not contain enough data to form independent statistically valid domains due to their limited size and discontinuous nature. Accordingly, a soft boundary was applied between data in all three domains for the purposes of resource estimation.

Table 14-11 Massawa North Zone 2 High-Grade Maptek Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.46	217	5	75	20	5	2
Spherical	0.12	217	5	75	31	8	7
Spherical	0.18	217	5	75	101	55	9
Nugget	**0.24**						
Total Sill	**1**						

Table 14-12 Massawa North Zone Low-Grade Domain Maptek Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.46	210	0	-100	12	8	10
Spherical	0.12	210	0	-100	41	21	15
Spherical	0.18	210	0	-100	120	96	24
Nugget	**0.24**						
Total Sill	**1**						

14.6.2. Massawa Central Zone

Separate variogram models were constructed for low-grade and high-grade mineralisation domains, as a function of the significantly different grade distributions and continuity ranges within each of the two domains.

A singular variogram model was applied to all top cutting group 1 domains (Table 14-13) due the fact that some of the modelled structures do not contain enough RC drill data.

Similarly, singular variogram models were created for top cutting groups 2, 3, and 4; however, top cutting groups 2 and 3 were combined to use the same variogram due to similar grade

distributions (Table 14-14), where the individual variograms had almost identical nugget, modelled structure ranges, and directions. Top cutting group 4 (Table 14-15) was modelled separately as high-grade domains. The resultant variogram models were re-orientated where appropriate to fit the slight variations in strike and dip of each mineralised domain.

Table 14-13 Massawa Central Zone Top Cutting Group 1 Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.54	25	0	-90	15	15	8
Spherical	0.05	25	0	-90	350	350	40
Spherical	0	0	0	0	10	10	10
Nugget	**0.42**						
Total Sill	**1.01**						

Table 14-14 Massawa Central Zone Top Cutting Group 2 and 3 Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.52	25	0	-95	7	7	5
Spherical	0.03	25	0	-95	90	90	54
Spherical	0	0	0	0	10	10	10
Nugget	**0.44**						
Total Sill	**0.99**						

Table 14-15 Massawa Central Zone Top Cutting Group 4 Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.47	25	0	-95	12	12	9
Spherical	0.02	25	0	-95	80	80	18
Spherical	0	0	0	0	10	10	10
Nugget	**0.51**						
Total Sill	**1**						

14.6.3. Sofia

Separate variogram models were constructed for Sofia Main and Sofia North, as a function of the significant differences of the data spacing, resultant continuity directions, and ranges within each of the two Mineral Resources. The Sofia Main variogram model parameters modelled from domains 6000, 6100, and 6300 as defined are included in Table 14-16 and Sofia North variogram model parameters modelled from domain 6500 as defined are included in Table 14-17.

Table 14-16 Sofia Main Maptek Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.56	37.5	-4.32	59.91	52	16	9
Spherical	0.19	37.5	-4.32	59.91	93	48	27
Spherical	0.05	37.5	-4.32	59.91	140	75	34
Nugget	0.21						

Table 14-17 Sofia North Maptek Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.52	10	0	80	46	27	3
Spherical	0.17	10	0	80	78	34	15
Spherical	0.09	10	0	80	163	102	23
Nugget	0.21						
Total Sill	1						

14.6.4. Delya

A variogram model was constructed for Delya domain 6000, as a function of the significant differences of the data spacing, resultant continuity directions, and ranges within each of the two Mineral Resources. The Delya variogram model parameters modelled from domain 6000 as defined are included in Table 14-18.

Table 14-18 Delya Maptek Vulcan Variogram Model Parameters

Structure Model Type	Sill Differential	Bearing (º)	Plunge (º)	Dip (º)	Major Axis (º)	Semi Major Axis (º)	Minor Axis (º)
Spherical	0.34	35	-5	100	31	15	4
Spherical	0.31	35	-5	100	40	22	5
Spherical	0.16	35	-5	100	60	38	10
Nugget	**0.2**						
Total Sill	**1**						

14.7. Bulk Density

Bulk density values were measured by applying the Archimedean principles (density = weight (in air) ÷ (weight (in air) – weight (in water)). All results are grouped by weathering profile and lithology. The density measurement procedure differed slightly for saprolite, transition, and fresh material:

- Saprolitic density measurements were primarily obtained from trenches, although some drill hole core was used. For trench samples, cubes of approximately 25 cm by 25 cm by 25 cm were excavated and the insitu weight measured to estimate the moisture content. The sample is then dried out and wrapped in a waterproof membrane.
- Fresh and transition density measurements were primarily obtained from drill core. The procedure followed involved the selection of 10 cm to 15 cm pieces using the water immersion method.

Once the dry sample weight has been measured using a Mettler 3000 electronic balance, the saprolite samples are weighed in water using the lower hook of the balance.

All density data distributions were checked to remove both lower and upper outliers before the mean density value was calculated for each lithology split into weathering groups (Table 14-19, Table 14-20, and Table 14-21).

Table 14-19 Massawa Central and North Zone Model Densities Assigned per Lithology and Weathering Zone

Lithology	Saprolite Density	Transition Density	Fresh Rock Density
Porphyry	1.80	2.41	2.74
Gabbro	1.66	2.37	2.85
Volcaniclastics	1.77	2.33	2.75
Greywacke	1.78	2.25	2.77
Carbonaceous Shale	1.75	2.27	2.75
NZ2 Mineralisation	1.87	2.25	2.80
NZ2 HG Mineralisation	2.21	2.2	2.83
NZ1 Mineralisation	1.85	2.26	2.81
CZ Low grade	1.66	2.35	2.80
CZ High grade	1.66	2.25	2.72

Table 14-20 Sofia Model Densities Assigned per Lithology and Weathering Zone

Lithology	Saprolite Density	Transition Density	Fresh Rock Density
Transported Cover	1.89		
Quartz-Feldspar Porphyry	1.76	2.63	2.76
Quartz Diorite	1.88	2.62	2.8
Gabbro	1.85	2.57	2.84
Dolerite	1.81	2.55	2.83
Dacite	1.8	2.56	2.82
Tonalite	1.55	2.6	2.78
Tuff	1.79	2.53	2.79
Tuff SAB	1.7	2.57	2.78
Ultra Basic	1.86	2.6	2.82
Sofia Main Mineralisation	1.9	2.6	2.8
Sofia North Mineralisation	1.9	2.47	2.77

Table 14-21 Delya Densities Assigned per Lithology and Weathering Zone

Lithology	Saprolite Density	Transition Density	Fresh Rock Density
Dolerite			2.88
Mafic Dyke	1.85	2.38	2.81
Carbonaceous Shale	1.9	2.5	2.77
Chert	1.87	2.54	2.9
Andesite	1.9	2.77	2.86
Greywacke	1.86	2.56	2.71
Schist	1.81	2.54	2.83
Delya Mineralised Zone	1.77	2.34	2.82

Density values are hard coded into all block models based on the lithology and weathering. Where density data does not exist in inferred satellite resources (Inferred resources only), the density has been inferred from the nearest deposit on the same shear zone with the same weathering and lithology combination.

14.8. Block Models

All Mineral Resource block model limits are set such that the block model covers the entire Mineral Resource plus a 150 m to 300 m buffer radius, to ensure that any pit optimisation is not limited by the extent of the waste blocks within the model.

Consequently, the NZ block model does overlap with the model extents of the CZ. None of the blocks in the CZ portion of the NZ block models contain any grades and are only flagged by

connecting waste lithologies. The two block models have remained as separate files due to the different timing of updates and to limit the resultant file size of the block models.

All Massawa and Sofia Mineral Resource block models are sub-cell block models which enable accurate definition of geological and domained contacts.

A different block size was applied to the GC domains relative to the exploration domains. Subsequently, using a smaller GC sub-domain block size enables the resource model to be estimated through application of a more localised estimation in areas of GC drilling with a high sample density. The block sizes used within both GC and exploration areas are intended to reflect the average drill hole spacing in both scenarios. The exploration sub-domain block size is required to be an exact multiple of GC block size by Maptek Vulcan software, in order to use multiple block sizes within the resource model.

To ensure that all selected block sizes and sub-cell sizes provide an accurate representation of the mineralisation wireframes, the respective block volumes are validated against the original wireframe volumes to ensure that the variance is less than or equal to 1%. The 2.5% decrease in block model volume relative to the wireframe volume is a result of the narrow and anastomosing nature of the wireframes relative to the minimum appropriate block size (Table 14-22).

Table 14-22 Global Massawa Sofia and Delya Wireframe vs Block Model Volume Variance Reconciliation

Domain	Wireframe Volume (m^3)	Block Model Volume (m^3)	Variance
NZ1	9,719,028	9,721,759	-0.03%
NZ2	3,341,534	3,342,141	0.1%
CZ Low Grade	18,518,093	18,047,480	-2.5%
CZ High Grade	3,030,700	3,022,796	-0.3%
Sofia Main	3,255,997	3,255,951	0.01%
Sofia North	5,775,474	5,776,946	0.03%
Delya	847,553	851,443	-0.46%

Block sizes are optimised utilising Qualitative Kriging Neighbourhood Analysis (QKNA). This optimisation process is repeated for each sub-domain (exploration and GC splits), to ensure that the optimisation best reflects the relevant sample spacing. Additional factors including the true width of the respective mineralised wireframes and potential selectivity during mining are taken into consideration when selecting an appropriate block size considering the geology and domaining used.

Block sizes within the Massawa NZ block model are specified in Table 14-23. A block size limit has been applied for GC domains – which physically limits the parent cell size of the blocks within the GC areas.

Table 14-23 Massawa North Zone Block Model Extents

Block Extents	Easting (X)	Northing (Y)	Elevation (Z)
Minimum Coordinates (UTM 29N)	819,395	1,434,254	-650
End Point Offset (m)	900	3,080	1005
Parent Block Size (m)	1.5	8.0	5.0
Sub Cell Size (m)	0.5	1.6	1.0
GC Domain (2108, 2109) Block Size Limit (m)	0.5	1.6	1.0
Rotation	117°		

Due to the complex thin anastomosing shape of the high-grade wireframes, the CZ block model has been built entirely based upon the GC block size to achieve a close volume reconciliation between the block model and wireframes. Consequently, the exploration sub-domains use a parent cell size that is exactly double that of the GC block size, meaning that the estimation will assign the grades into groups of eight blocks when writing estimated grades to represent parent cell estimation (Table 14-24).

Table 14-24 Massawa Central Zone Block Model Extents

Block Extents	Easting (X)	Northing (Y)	Elevation (Z)
Minimum Coordinates (UTM 29N)	818,156.276	1,432,022.035	-175
End Point Offset (m)	976.5	2,650.5	400
Parent Block Size (m)	2.25	4.5	2.5
Sub Cell Size (m)	0.25	0.5	0.5
Rotation	117°		

Similarly, in the Sofia block model, the thin anastomosing wireframe shapes in Sofia Main dictate that the entire block model be built entirely based upon the GC block size to achieve a close volume reconciliation between the block model and wireframes. Consequently, the Sofia North sub-domains use a parent cell size that is exactly double that of the GC block size in strike length (Y axis) to account for the wider spacing along strike (Table 14-25). This means that the estimation will estimate a single grade into the groups of two blocks when writing estimated grades, representative of the parent cell size.

Table 14-25 Sofia Block Model Extents

Block Extents	Easting (X)	Northing (Y)	Elevation (Z)
Minimum Coordinates (UTM 29N)	80,7701	1,434,400	-200.0
End Point Offset (m)	1,650	4,800	540
Parent Block Size GC Domain (m)	3.0	15	12
Sub Cell Size (m)	0.5	1.5	1.2
Rotation	120°		

Block sizes within the Delya block model are specified in Table 14-26.

Table 14-26 Delya Block Model Extents

Block Extents	Easting (X)	Northing (Y)	Elevation (Z)
Minimum Coordinates (UTM 29N)	829,780	1,447,163	-35
End Point Offset (m)	1,556	801	245
Parent Block Size (m)	2	9	7
Sub Cell Size (m)	0.5	1.8	1.75
Rotation	35°		

14.9. Resource Estimation

14.9.1. Massawa, Sofia and Delya

All Massawa, Sofia, and Delya Mineral Resources are estimated using an Ordinary Kriging (OK) estimation methodology and apply parent cell estimation for the relevant GC or exploration parent cell size, thereby ensuring that sub-cells are effectively grouped to provide sufficient sample support per estimated block.

QKNA was utilised to determine the optimal block size, minimum number of samples, search radius, and block discretization for each pass, by evaluating the kriging efficiency (KE) and slope of regression (SR) across multiple block centroid locations to ensure that the selected parameters are robust and appropriate for different areas within the relevant domain.

All Massawa CZ search radii, number of samples used, and QKNA estimation parameter optimisations were restricted to RC drill data with bulk sample LeachWELL (plus tail assay) determination in order to ensure that the respective grade distributions and continuity were not skewed by the known negative bias of sample grades within the smaller volume DD. Block size optimisations were performed to take into account the mining SMUs.

The estimation search strategies for Sofia, Massawa NZ, and Delya were tested using QKNA based on the modelled variogram ranges taking into account that in nugget shear hosted gold systems the full variogram range is often artificially extended for the last 3% to 5% of variability. Additionally, the estimation search strategy takes into consideration the data distribution for both the geological and the data spacing sub-domains. Examples of the QKNA results for Sofia Main optimisation are shown in Figure 14-37 to Figure 14-40. The resultant search ellipsoid orientations are verified visually against the domain wireframe and, where appropriate, the search ellipse and associated variogram directions are locally re-orientated to reflect bifurcation and local changes in the strike and dip of the mineralisation wireframes.

In 2018, dynamic anisotropy (DA) was introduced to the Massawa CZ deposit to better reflect the anastomosing nature of the mineralisation and local changes in orientation across the deposit (Figure 14-41). Reference surfaces are generated automatically in Leapfrog geo-modelling software vein modelling tool for each domain using the centre point of the drill hole intercept. These reference surfaces are then flagged to the empty block models by providing a reference azimuth to provide the strike, dip, and plunge for interpolation.

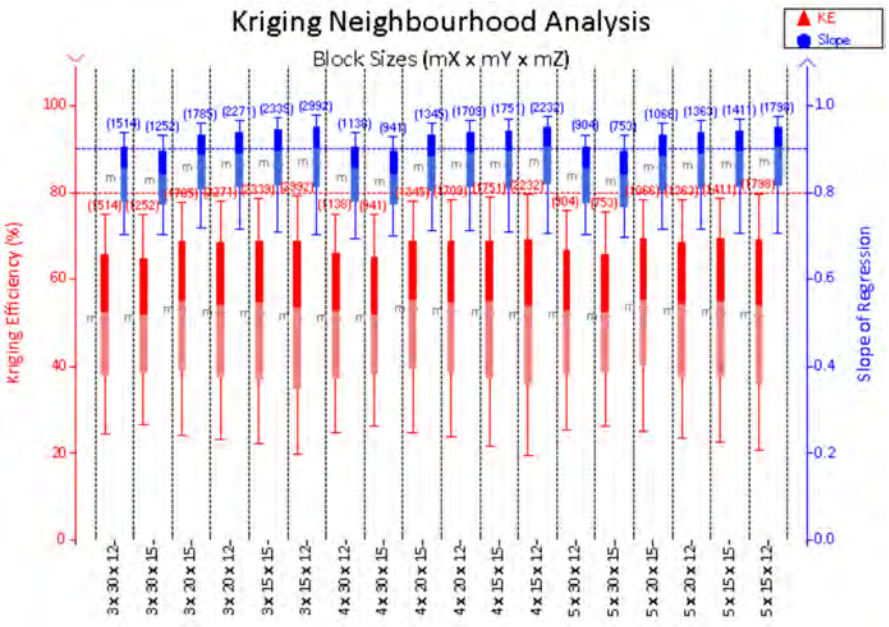


Figure 14-37 Sofia Main QKNA for Block Size

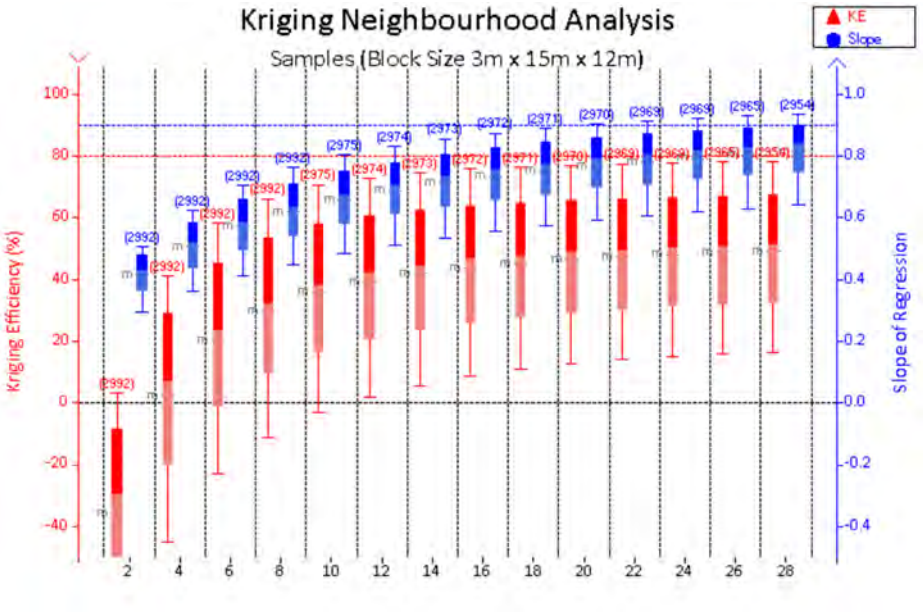


Figure 14-38 Sofia Main High-Grade (Proximal) QKNA for Sample Limits (Bottom)

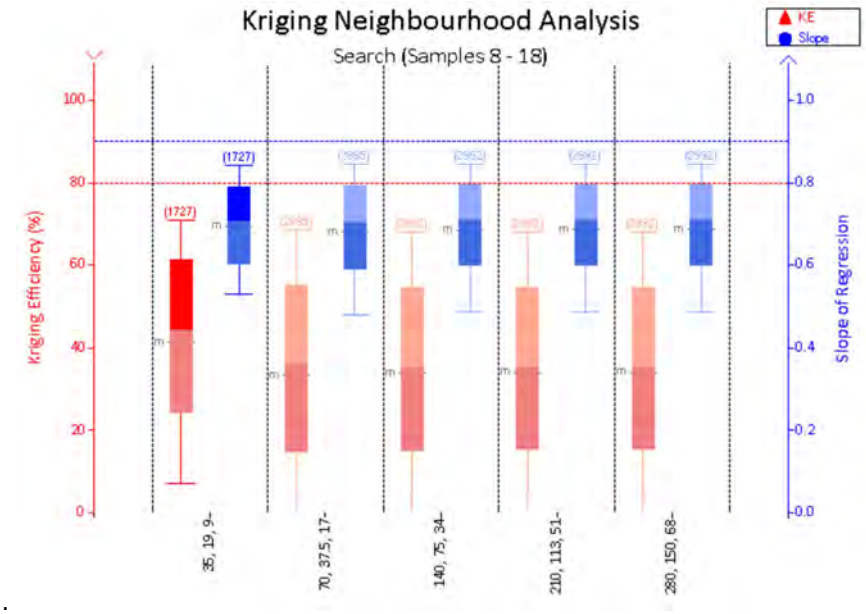


Figure 14-39 Sofia QKNA for Search Size

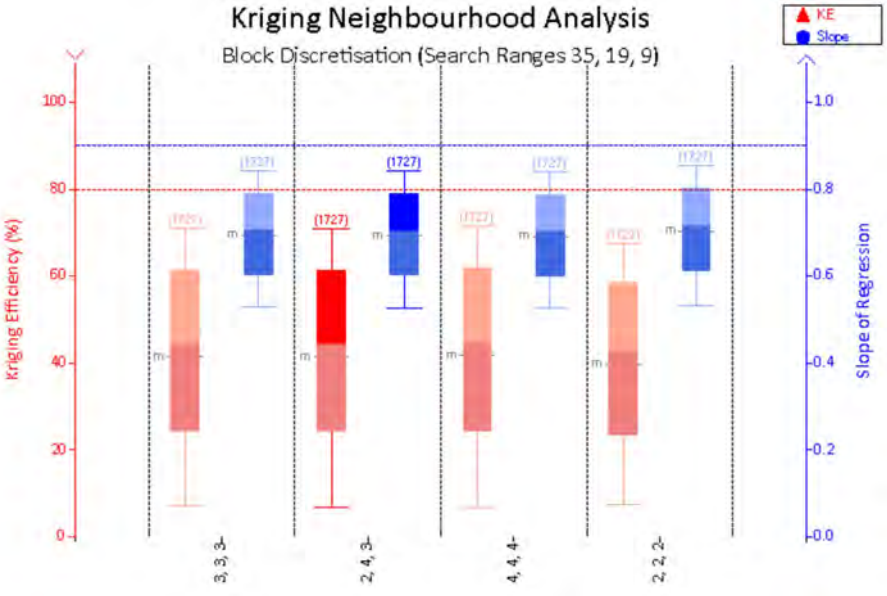


Figure 14-40 Sofia QKNA for Block Discretisation

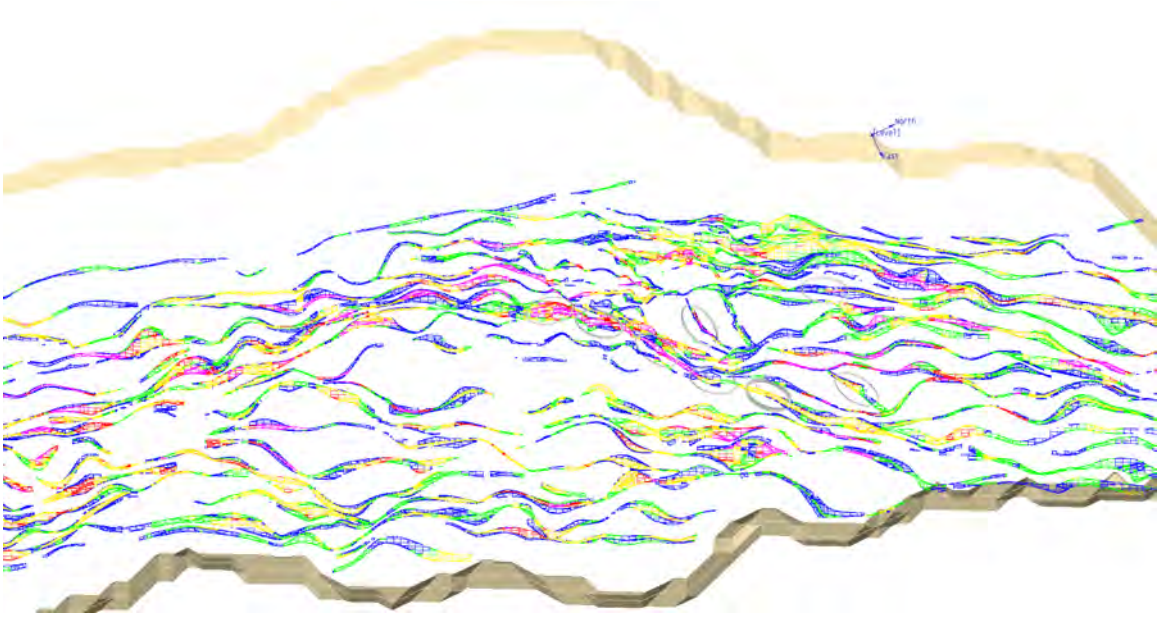


Figure 14-41 Localised Changes to Massawa Central Zone Search Ellipsoids (white) through Dynamic Anisotropy on the Estimated Mineralised Domains in the Block Model in the $1,500 Whittle Shell

All Mineral Resources use multi-pass estimation searches, in order to estimate all blocks, whereby each pass uses a varying degree of restrictions before any given block can be estimated. Accordingly, the Mineral Resource classification is adjusted per the search pass during which the blocks from an area were estimated. Estimation search pass parameters for Massawa NZ, CZ, Sofia, and Delya are shown in Table 14-27 to Table 14-32.

Table 14-27 Massawa North Zone Resource Estimation Parameters

Zones	Estimation Pass	Block Size			Search Orientation			Search Radius			No Samples per Run		Max Samples Per DH	Discretisation			Restricted Search (Global)			
		X	Y	Z	Z	X	Y	Major	Sec	Minor	Min	Max		X	Y	Z	Grade (g/t Au)	Major	Sec	Minor
NZ1 EXP	Pass 1 - (2nd variogram structure)	3	16	10	210	0	90	32	30	10	8	18	5	2	4	3	-			
	Pass 2 - (Full Variogram Range)	3	16	10	210	0	90	64	50	16	6	16	4	2	4	3	-			
	Pass 3 - (2 x Variogram Range)	3	16	10	210	0	90	128	100	32	4	14	-	2	4	3	21	64	50	16
	Pass 4 - (4 x Variogram Range)	3	16	10	210	0	90	256	200	64	3	10	-	2	4	3	21	64	50	16
NZ1 GC	Pass 1 - (1st Variogram structure)	1.5	8	5	210	0	90	23	16	3	6	16	4	2	4	3	-			
	Pass 2 - (2nd variogram structure)	1.5	8	5	210	0	90	32	30	10	4	14	3	2	4	3	-			
	Pass 3 - (Full Variogram Range)	1.5	8	5	210	0	90	64	50	16	4	12	-	2	4	3	-			
	Pass 4 - (2x Full Variogram Range)	1.5	8	5	210	0	90	96	75	24	3	6	-	2	4	3	21	64	50	16
NZ2 LG EXP	Pass 1 - (Half of range)	4.5	24	10	210	-7	-100	40	24	16	8	18	5	2	4	3	-			
	Pass 2 - (2nd structure)	4.5	24	10	210	-7	-100	70	40	20	6	16	4	2	4	3	30	40	24	16
	Pass 3 - (1 x Variogram Range)	4.5	24	10	210	-7	-100	120	80	30	4	14		2	4	3	30	40	24	16
	Pass 4 - (2 x Variogram Range)	4.5	24	10	210	-7	-100	240	160	60	3	12		2	4	3	30	40	24	16
NZ2 LG GC	Pass 1 - (1st Variogram structure)	1.5	8	5	210	-7	-100	20	12	12	6	16	4	2	4	3	-			
	Pass 2 - (half range)	1.5	8	5	210	-7	-100	40	24	16	4	14	3	2	4	3	-			
	Pass 3 - (2nd structure)	1.5	8	5	210	-7	-100	70	40	20	4	12		2	4	3	30	40	24	16
	Pass 4 - (1x Full Variogram Range)	1.5	8	5	210	-7	-100	120	80	30	3	10		2	4	3	30	40	24	16
NZ2 HG EXPL	Pass 1 - (Half range)	4.5	24	10	215	0	75	32	20	10	8	18	5	2	4	3	-			
	Pass 2 - (2nd variogram structure)	4.5	24	10	215	0	75	60	30	14	6	16	4	2	4	3	48	32	20	10
	Pass 3 - (Full Variogram Range)	4.5	24	10	215	0	75	120	60	28	4	14		2	4	3	48	32	20	10
	Pass 4 - (2x Full Variogram Range)	4.5	24	10	215	0	75	240	120	56	3	12		2	4	3	48	32	20	10
NZ2 HG GC	Pass 1 - (1st Variogram structure)	1.5	8	5	215	0	75	18	12	6	6	16	4	2	4	3	-			
	Pass 2 - (half range)	1.5	8	5	215	0	75	32	20	10	4	14	3	2	4	3	-			
	Pass 2 - (2nd variogram structure)	1.5	8	5	215	0	75	60	30	14	4	12		2	4	3	48	32	20	10
	Pass 4 - (1x Full Variogram Range)	1.5	8	5	215	0	75	120	60	28	3	10		2	4	3	48	32	20	10

Table 14-28 Massawa Central Zone Low-Grade Estimation Parameters - Low-Grade Domains

Low-Grade Zones	Estimation Pass	Estimation Run			No Samples Run		Samples Per DH	Discretisation			Restricted Search			
		Major	Sec	Minor	Min	Max	Max	X	Y	Z	Grade (g/t Au)	Major	Sec	Minor
Hanging Wall and Hanging Wall Bridge	Pass 1	15	15	8	6	18	4				-	-	-	-
	Pass 2	30	30	16	6	16	4				-	-	-	-
	Pass 3	90	90	24	4	14	3				28	30	30	16
	Pass 4	180	180	32	4	12					28	30	30	16
	Pass 5	270	270	40	4	12					28	30	30	16
Central	Pass 1	15	15	8	6	18	4	2	4	4	-	-	-	-
	Pass 2	30	30	16	6	16	4	2	4	4	-	-	-	-
	Pass 3	90	90	24	4	14	3	2	4	4	-	-	-	-
	Pass 4	180	180	32	4	14	-	2	4	4	28	30	30	16
	Pass 5	270	270	40	4	14	-	2	4	4	28	30	30	16
Footwall Bridge	Pass 1	15	15	8	6	18	4	2	4	4	-	-	-	-
	Pass 2	30	30	16	6	16	4	2	4	4	-	-	-	-
	Pass 3	90	90	24	4	14	3	2	4	4	28	30	30	16
	Pass 4	180	180	32	4	14	-	2	4	4	28	30	30	16
	Pass 5	270	270	40	4	14	-	2	4	4	28	30	30	16
Footwall	Pass 1	15	15	8	6	18	4	2	4	4	-	-	-	-
	Pass 2	30	30	16	6	16	4	2	4	4	-	-	-	-
	Pass 3	90	90	24	4	14	3	2	4	4	28	30	30	16
	Pass 4	180	180	32	4	14	-	2	4	4	28	30	30	16
	Pass 5	270	270	40	4	14	-	2	4	4	28	30	30	16

Table 14-29 Massawa Central Zone Low-Grade Estimation Parameters - High-Grade Domains

Low-Grade Zones	Estimation Pass	Estimation Run			No Samples Run		Samples Per DH	Discretisation			Restricted Search			
		Major	Sec	Minor	Min	Max	Max	X	Y	Z	Grade (g/t Au)	Major	Sec	Minor
CZ Low-Grade Hanging Wall and Hanging Wall Bridge	Pass 1	14	14	5	4	14	3				-	-	-	-
	Pass 2	28	28	10	4	12	3				-	-	-	-
	Pass 3	80	80	18	4	10	3				100	28	28	10
	Pass 4	160	160	20	3	10					100	28	28	10
	Pass 5	240	240	30	3	10					100	28	28	10
CZ High-Grade Central	Pass 1	14	8	8	4	14	4	2	4	4	-	-	-	-
	Pass 2	28	28	10	4	12	3	2	4	4	-	-	-	-
	Pass 3	90	90	20	4	10	3	2	4	4	230	28	28	10
	Pass 4	160	160	20	3	10	-	2	4	4	230	28	28	10
	Pass 5	240	240	30	3	10	-	2	4	4	230	28	28	10
CZ High-Grade Footwall Bridge	Pass 1	14	14	5	4	14	4	2	4	4	-	-	-	-
	Pass 2	28	28	10	4	12	3	2	4	4	-	-	-	-
	Pass 3	80	80	18	4	10	3	2	4	4	230	28	28	10
	Pass 4	160	160	20	3	10	-	2	4	4	230	28	28	10
	Pass 5	240	240	30	3	10	-	2	4	4	230	28	28	10
CZ High-Grade Footwall	Pass 1	14	14	5	4	14	4	2	4	4	-	-	-	-
	Pass 2	28	28	10	4	12	3	2	4	4	-	-	-	-
	Pass 3	80	80	18	4	10	3	2	4	4	230	30	30	16
	Pass 4	160	160	20	3	10	-	2	4	4	230	30	30	16
	Pass 5	240	240	30	3	10	-	2	4	4	230	30	30	16

Table 14-30 Sofia Estimation Parameters

Zones	Estimation Pass	Block Size			Estimation Run1			No Samples Run 1		Samples Per DH	Discretisation			Restricted Search (Domain 6000)			
		X	Y	Z	Major	Sec	Minor	Min	Max	Max	X	Y	Z	Au Grade (g/t Au)	Major	Sec	Minor
Sofia Main	Pass 1 - (25% range)	3	15	12	35	18.8	8.5	8	18	3	2	4	3	-			
	Pass 2 - (50% Variogram Range)	3	15	12	70	37.5	17	6	16	2	2	4	3	20	52	16	9
	Pass 3 - (Variogram Range)	3	15	12	140	75	34	4	14	2	2	4	3	20	52	16	9
	Pass 4 - (1.5 x Variogram Range)	3	15	12	210	113	51	4	12	-	2	4	3	20	52	16	9
	Pass 5 - (2 x Variogram Range)	3	15	12	280	150	68	3	10	-	2	4	3	20	52	16	9
Sofia North 6500, 6700, 6900	Pass 1 - (25% range)	3	15	12	40.8	25.5	5.75	8	18	3	2	4	3	-			
	Pass 2 - (50% Variogram Range)	3	15	12	81.5	51	11.5	6	16	2	2	4	3	13	48	26	6
	Pass 3 - (Variogram Range)	3	15	12	163	102	23	4	14	2	2	4	3	13	48	26	6
	Pass 4 - (1.5 x Variogram Range)	3	15	12	245	153	34.5	4	12	-	2	4	3	13	48	26	6
	Pass 5 - (2 x Variogram Range)	3	15	12	326	204	46	3	10	-	2	4	3	13	48	26	6
Sofia North 6400, 6600	Pass 1 - (25% range)	3	15	12	40.8	25.5	5.75	8	18	3	2	4	3				
	Pass 2 - (50% Variogram Range)	3	15	12	81.5	51	11.5	6	16	2	2	4	3				
	Pass 3 - (Variogram Range)	3	15	12	163	102	23	4	14	2	2	4	3				
	Pass 4 - (1.5 x Variogram Range)	3	15	12	245	153	34.5	4	12	-	2	4	3				
	Pass 5 - (2 x Variogram Range)	3	15	12	326	204	46	3	10	-	2	4	3				

Table 14-31 Delya Estimation Parameters

Zones	Estimation Pass	Block Size			Estimation Run1			No Samples Run 1		Samples Per DH	Discretisation			Restricted Search (Domain 6000)			
		X	Y	Z	Major	Sec	Minor	Min	Max	Max	X	Y	Z	Grade (g/t Au)	Major	Sec	Minor
DELYA 25X25	Pass 1 - (2nd structure)	2	9	7	40	22	5	6	16	4	2	4	3	-			
	Pass 2 - (100% Variogram Range)	2	9	7	60	38	10	4	14	-	2	4	3	-			
	Pass 3 - (2 x Variogram Range)	2	9	7	120	76	20	4	12	-	2	4	3	16	60	38	10
	Pass 4 - (3 x Variogram Range)	2	9	7	240	152	40	4	10	-	2	4	3	16	60	38	10

14.9.2. Tina and Bambaraya Satellite Deposits Block Size, Top Cutting and Estimation

Tina and Bambaraya have been estimated using inverse distance squared using one pass of the search ellipse. All searches have been orientated into the average strike of each of the deposits.

Tina and Bambaraya have log normal distributions. Table 14-32 below shows the top cutting, searches, block size, and orientations.

Table 14-32 Bambaraya and Tina Satellite Deposit Mineral Resource Estimation Parameters and Block Model Summary

Deposit	Search Radius (m)			Orientation (º)		Block Size (m)			Top Cut Value (g/t Au)	Number of Samples	
	Main	Second	Minor	Bearing	Dip	X	Y	Z		Min	Max
Bambaraya	120	15	30	150	20	8	50	25	11.3	4	20
Tina	120	15	30	168	30	10	90	12	5.9	4	20

14.10. Resource Classification

Massawa Gold Project Mineral Resources are classified based on geological continuity, data density, variogram range, and resultant estimation search pass, as well as estimation quality in form of SR and KE. The classification parameters are presented in Table 14-33.

Table 14-33 Massawa Gold Project Mineral Resource Classification

Statistic	Deposit	Measured	Indicated	Inferred
Minimum Samples	All	6	4	3
Minimum Consecutive Sections	All	6	4	Good Geological Continuity
Maximum Drilling Density	CZ	5 m by 5 m RC with LeachWELL	15 m by 10 m RC with LeachWELL	50 m by 50 m
	NZ	-	40 m by 40 m	80 m by 50 m
	Sofia and Delya	-	40 m by 40 m	80 m by 60 m
Estimation Search Pass	All	GC Pass 1 or GC Pass 2	GC Pass 2 or GC Pass 3 or EXP Pass 1 or EXP Pass 2	EXP Pass 2 or EXP Pass 3
Slope of Regression	CZ	-	-	-
	NZ	-	>0.6	<0.6
	Sofia	-	>0.6	<0.6
	Delya	-	>0.5	<0.5
Kriging Efficiency	CZ	-	-	-
	NZ	-	>0.5	<0.5
	Sofia	-	>0.6	<0.6
	Delya	-	>0.5	<0.5
Estimation Method	All	Kriging	Kriging	Kriging or IVD

Classification of the blocks was carried out by displaying the estimated blocks (Pass No, SR, and KE) on vertical cross sections and generating a wireframe basal surface for Measured, Indicated, and Inferred. All block models were then re-blocked to the relevant classification surfaces to ensure accurate representation within the sub-cell model.

Drill densities in CZ and NZ Mineral Resources have been optimised from specific drill spacing studies. For Indicated Mineral Resources, there are some allowances for areas where drilling density is lower but successive drilling campaigns have shown that there is grade and geological continuity.

Tina and Bambaraya are both classified as Inferred Mineral Resources as they have been estimated with inverse distance squared using the resource classification criteria shown in Table 14-33.

There are also areas that, although not part of declared Mineral Resources, show geological continuity within the current exploration regions. These areas are considered to be potential exploration targets which will be followed up and evaluated in future exploration programmes.

14.11. Resource Model Validation

Once the block model has been classified, the following procedure were undertaken to check the block models and estimated grades which can be used to indicate any major errors during the estimation process as well as testing the precision, accuracy, and any bias of the estimated grade:

i. A volume reconciliation between the block model estimation domains and related wireframes is undertaken.
ii. All models are visually checked against the composite data on both cross section and long section basis, to validate the estimation and ensure that estimated grades and metal show no clear bias or trends that would not be expected from the variography or estimation domaining.
iii. A check of the number of the blocks estimated using the negative kriging weight is completed. Any blocks estimated using negative kriging weight have been reset to the anisotropic nearest block grade of the closest sample.
iv. A comparison between the data minimum, maximum, mean, and the estimated grade at 0 g/t Au cut off for each of the domains (within the open pit or underground reporting areas) is created. This is completed to check for possible over or under estimation.
v. Swath plots are created for each geological domain to validate the estimated grade variability compared to the composite along X, Y, and Z axes. This is to check that the model estimate follows the trends seen in the data and that there is no general bias with over or under estimation. Areas with less data support are also highlighted for further drilling and geological work.

The swath plots for Massawa, Sofia, and Delya Mineral Resources show that the correlation between the estimation and the source data is acceptable and that conditional bias is minimised. Example swath plots for Massawa NZ2 Y axis (northing) Mineral Resources are shown in Figure 14-42 and Figure 14-43. Swath plots for Sofia Y axis (northing) are shown in Figure 14-44 and Figure 14-45 and Delya in Figure 14-46. CZ high-grade and low-grade swath plots are presented in Figure 14-47 and Figure 14-48, respectively.

The swath plot analysis does, however, highlight a significant increase in estimation smoothing within the exploration sub-domains relative to the GC sub-domains. This is considered acceptable and is a function of the wider data density and the fact that the estimations in the

exploration sub-domains use samples from a much wider search area relative to the more local GC sub-domains.

As part of the optimisation study a pilot plant scale test is planned for the CZ Mineral Resource, which will provide a source of reconciliation data to test against the GC blocks estimated within the Mineral Resource.

No external audit has been undertaken on the Massawa Mineral Resource models, however, the general procedures used to generate the Mineral Resources are common across Barrick Africa and Middle East operations which are externally audited on three-year cycles. The QP for the Mineral Resource (Mr Simon Bottoms) is the QP for all Barrick Africa and Middle East operations and has ensured the Massawa Mineral Resources.

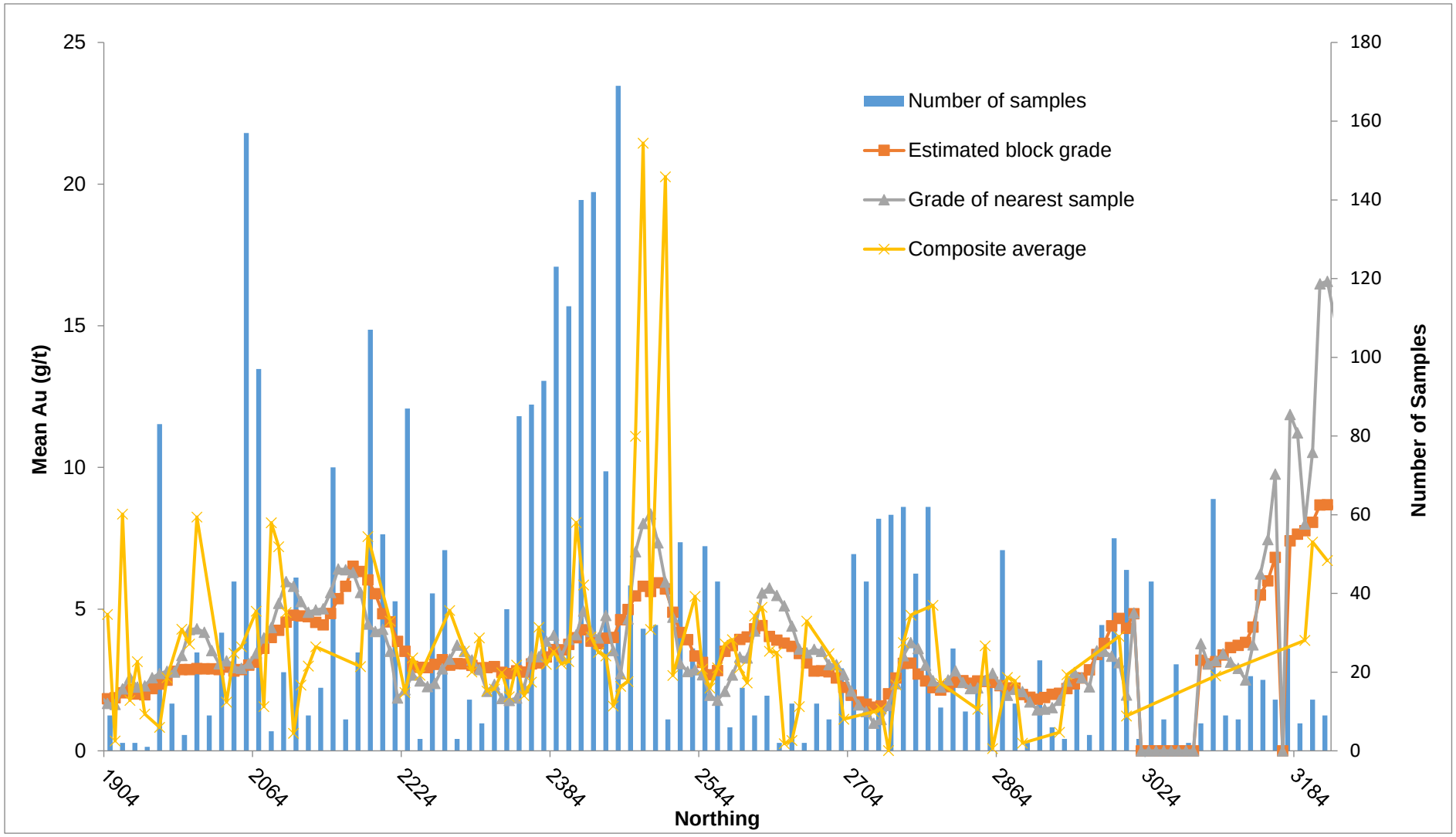


Figure 14-42 Massawa North Zone 2 Domain 2101 Swath Plot Y Axis (Northing)

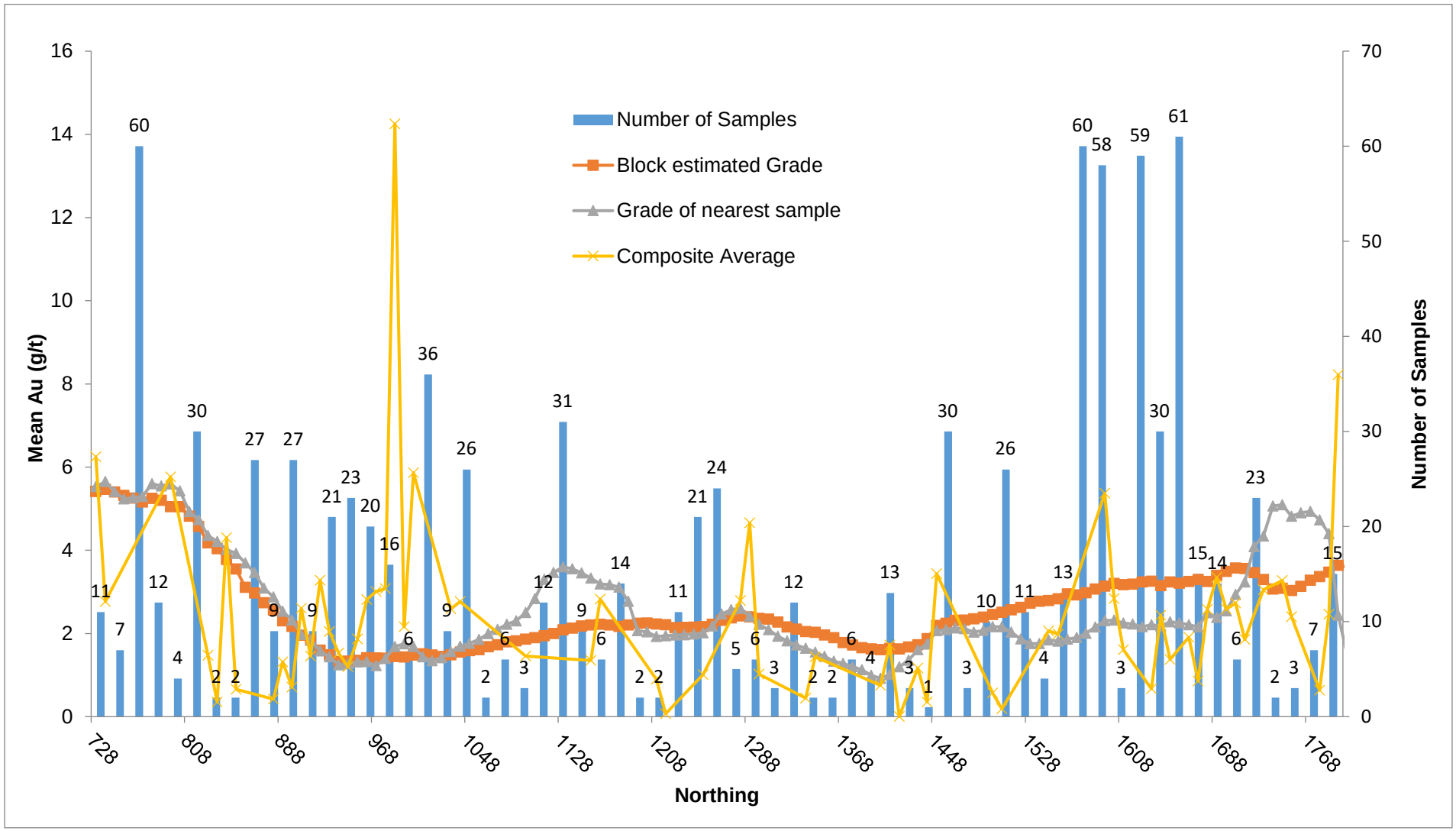


Figure 14-43 Massawa North Zone 1 Domain 1103 Swath Plot Y Axis (Northing)

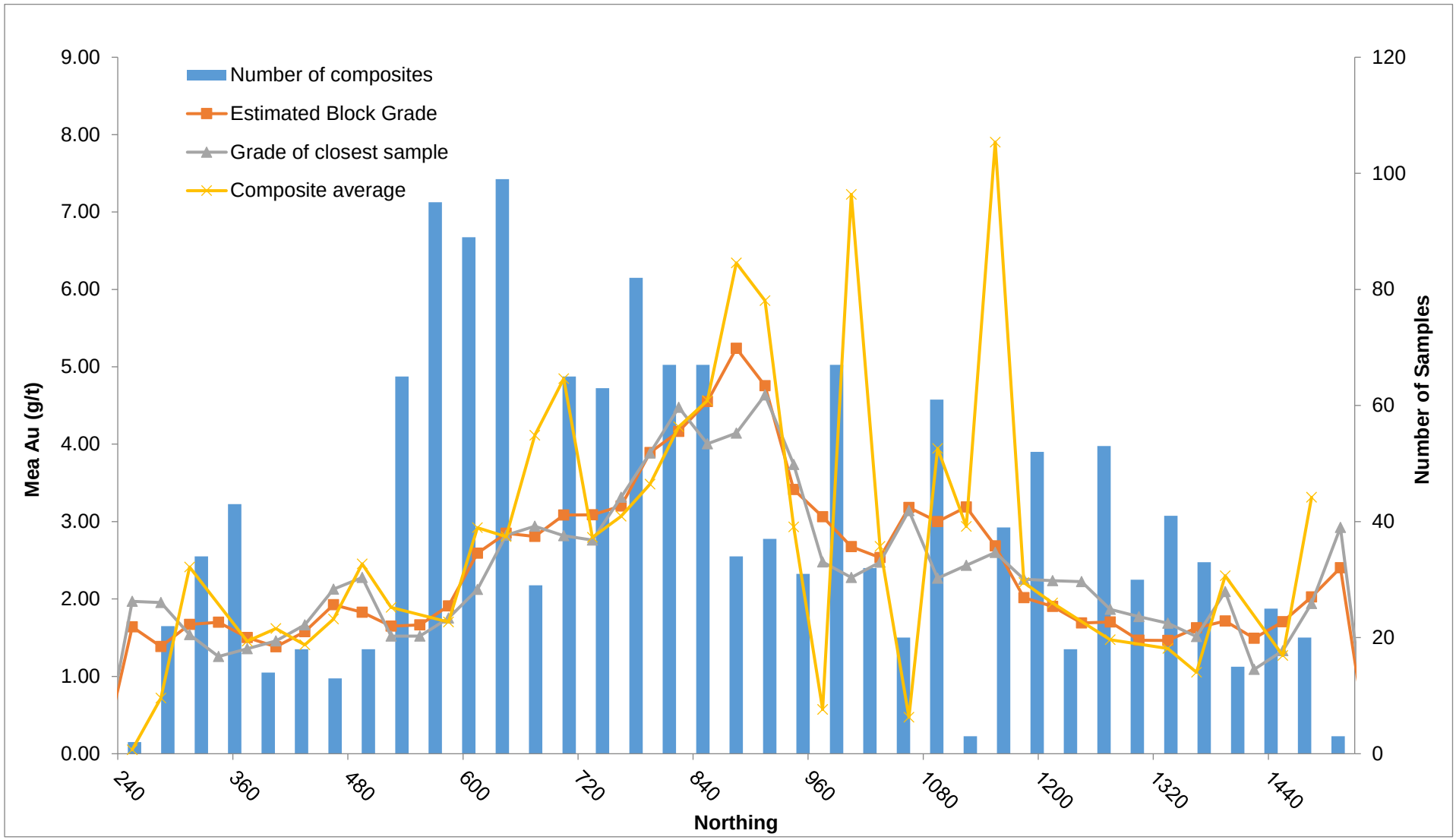


Figure 14-44 Sofia Main Domain 6000 Swath Plot Y Axis (Northing)

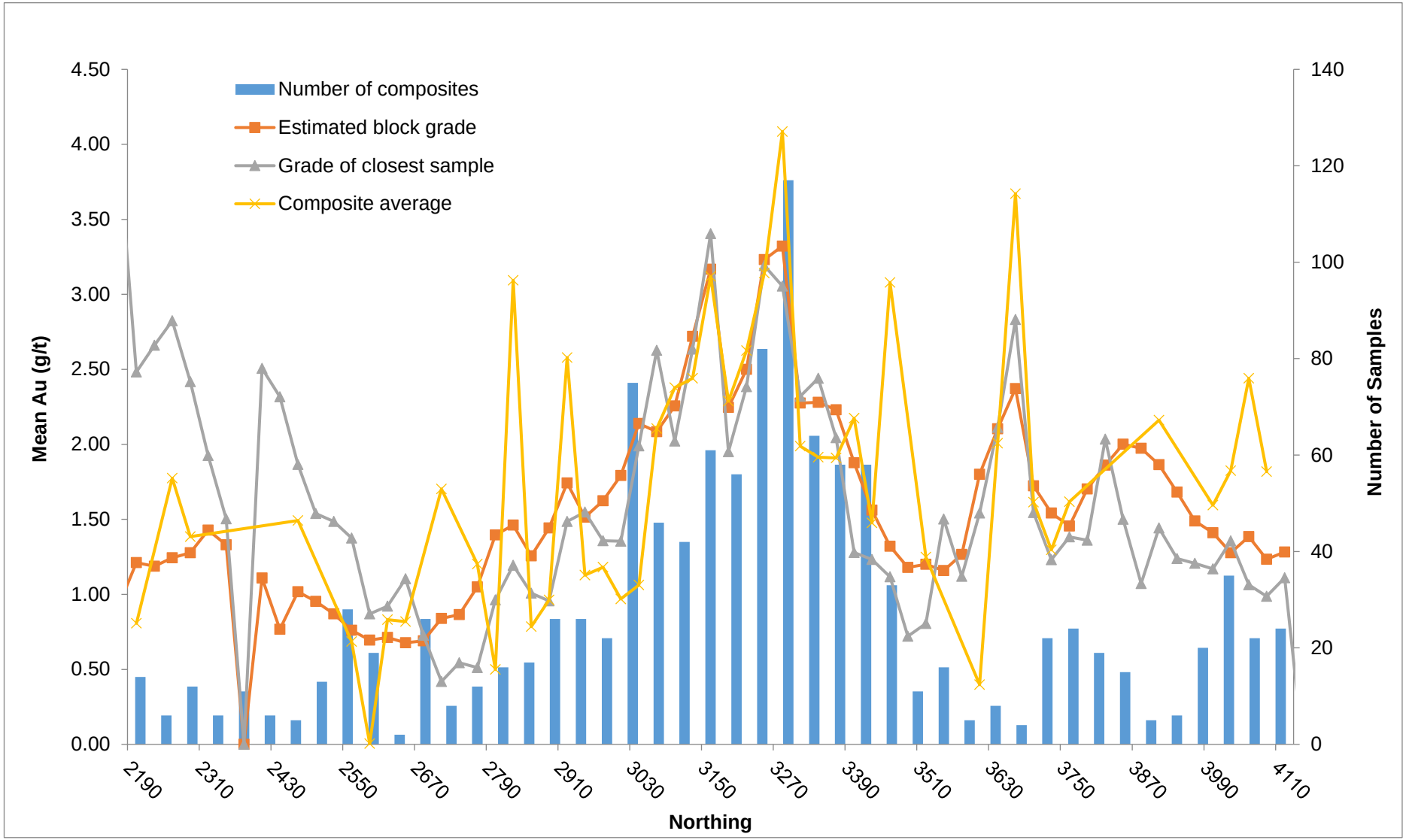


Figure 14-45 Sofia North Domain 6500 Swath Plot Y Axis (Northing)

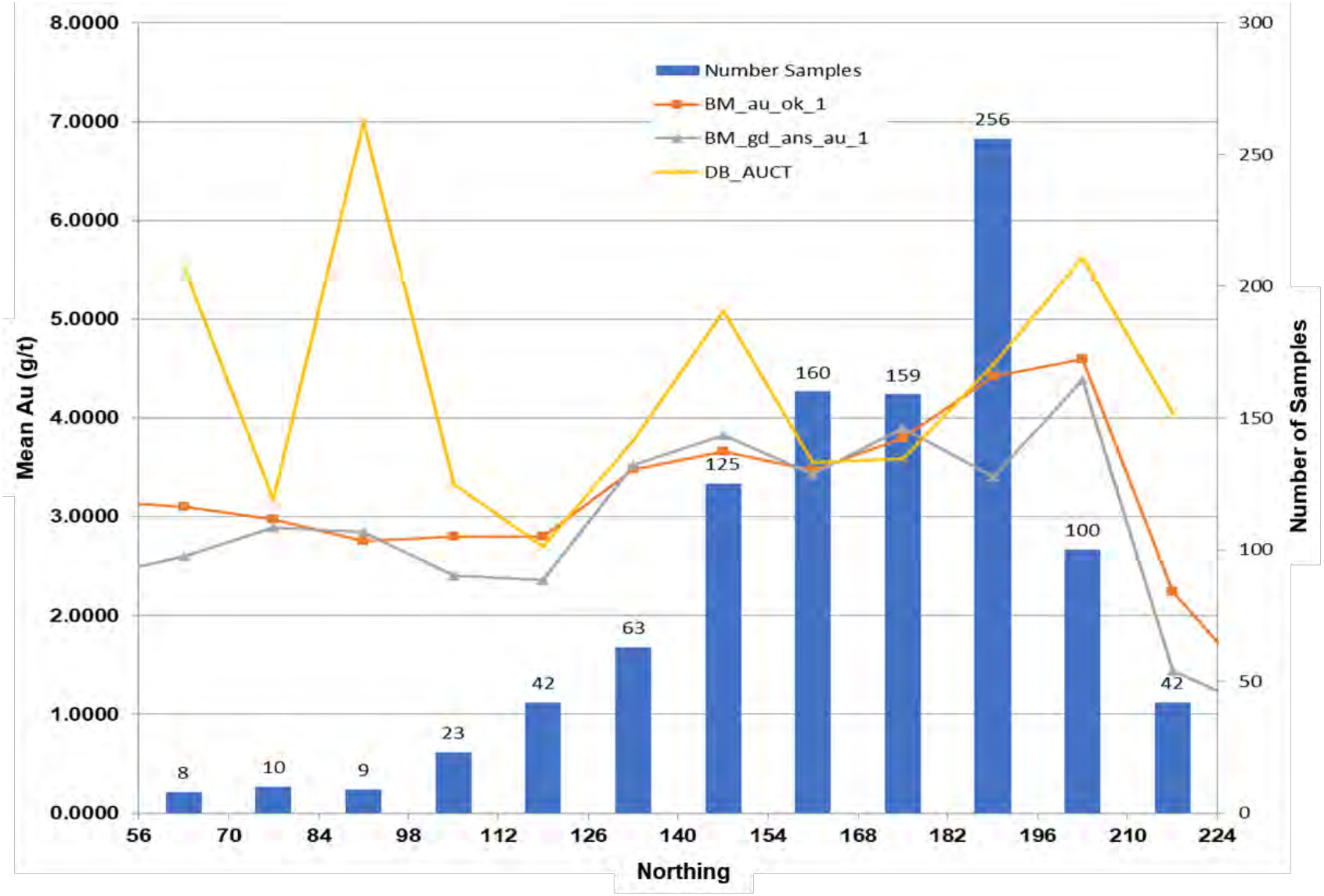


Figure 14-46 Delya Swath Plot Domain 6500 Z Axis (Down Dip)

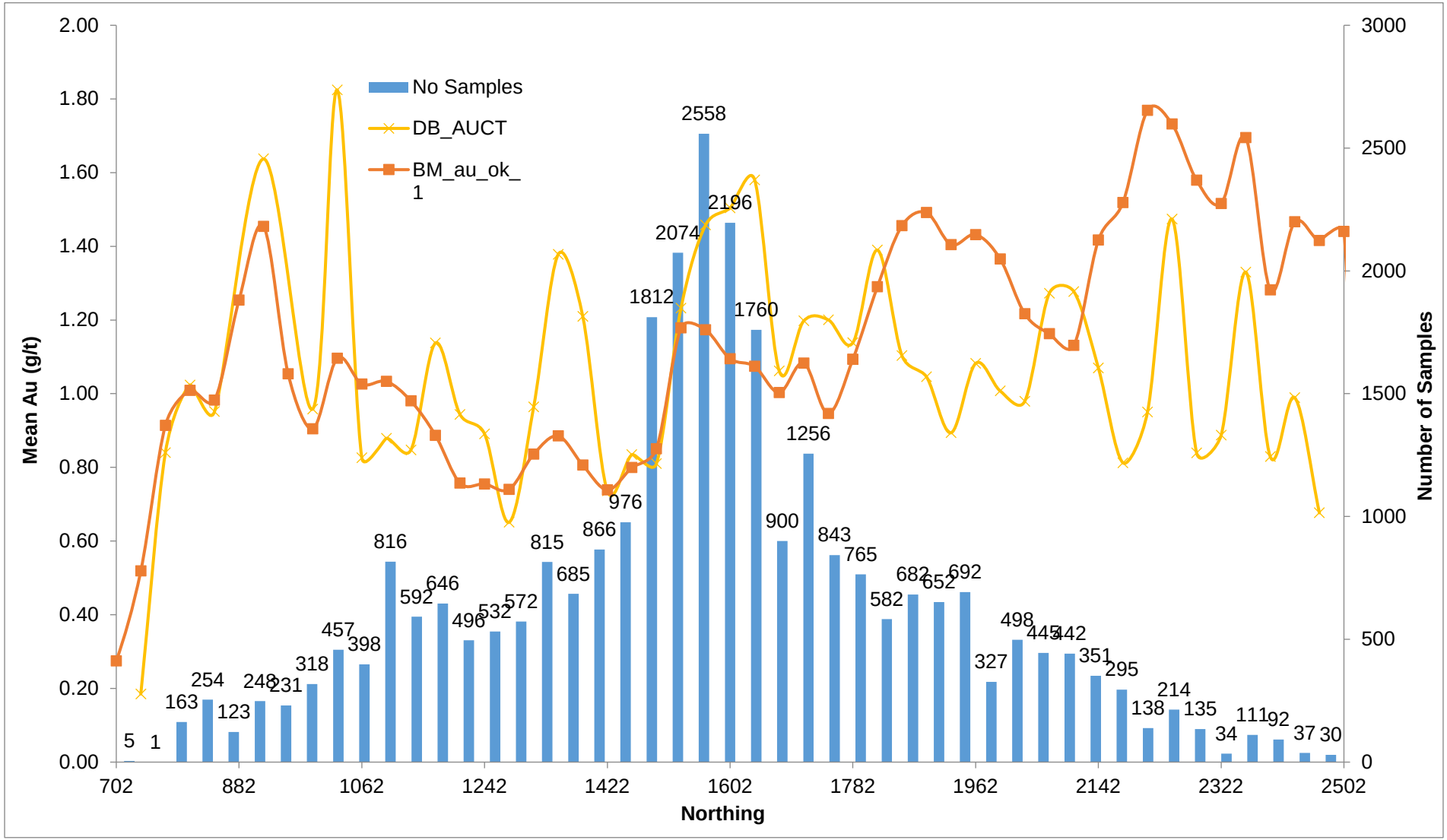


Figure 14-47 Massawa Central Zone Low-Grade Swath Plot (Y) Northing

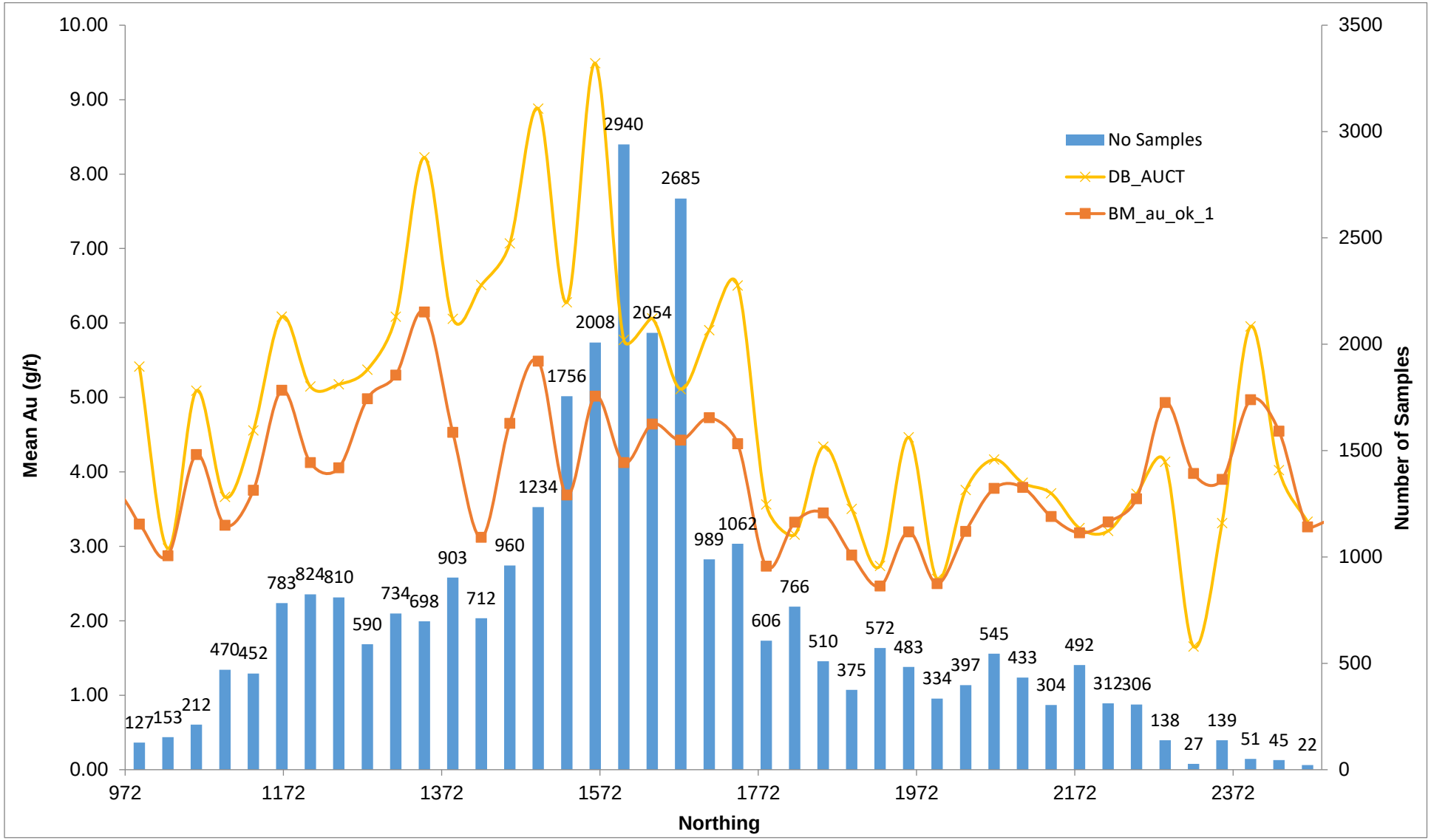


Figure 14-48 Massawa Central Zone High-Grade Swath Plot (Y) Northing

14.12. Resource Cut-off Grades

Massawa Mineral Resource cut-off grades use an input price of $1,500/oz Au.

14.12.1. Massawa North Zone Open Pit

The cut-off grade calculation for Massawa NZ and all associated costs are broken down in Table 14-34.

Table 14-34 Massawa North Zone Open Pit Mineral Resource Cut-off Grade Calculation at $1,500/oz Gold Price

Parameter	Unit	Symbol	Value
Gold Price	$/oz	Gp	1,500
Royalty	%	R	3%
Selling cost	%	S	0%
Net Gold Price	$/oz	Ng	1,455
Met Recovery	%	REC	83%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$/t mined	MCC	3.35
Mining Cost - Owner's	$/t mined	MCO	0.06
Mining Cost - Grade Control	$/t mined	MCGC	0.14
Total Mining Cost	$/t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	14.32
G&A	$/t milled	G_A	8.00
Ore Crushing & Hauling	$/t milled	rd	-
Mining	$/t milled	Cr	30.30
Process Plant	$/t milled	CP	32.12
Maintenance/Engineering	$/t milled	Mp	-
Total Operating Costs	**$/t**		**70.42**
Full Grade Cut-off	**g/t**	**FGO**	**1.87**
Marginal Cut-off Grade	**g/t**	**MO**	**1.07**

The processing costs for Massawa NZ Mineral Resources are set materially higher than in other Mineral Resources to reflect the BIOX process treatment costs.

14.12.2. Massawa North Zone Underground

The cut-off grade calculation for Massawa NZ underground Mineral Resources and all associated costs are broken down in Table 14-35.

Table 14-35 Massawa North Zone Underground Mineral Resource Cut-off Grade Calculation at $1,500/oz Gold Price

Parameter	Unit	Symbol	Value
Gold Price	$/oz	Gp	1,500
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	1,455
Met Recovery	%	Rec	86%
Dilution	%	Dil	12%
Ore Loss	%	Loss	4%
OPEX Development	$/t mined	MCC	11.74
OPEX Stoping	$/t mined	MCO	15.05
Backfill	$/t mined	MCGC	12.15
Fixed Cost	$/t mined	TMC	15.25
Grade Control	$/t mined	SR	3.38
Total Operating Cost	**$/t**	**G_A**	**57.57**
Mine Sustaining Capital	**$/t**	**Rd**	**166.67**
All in Cost	**$/t**	**Cr**	**224.24**
Full Grade Cut-off*	**g/t Au**	**FGO**	**7.70***
Marginal Cut-off Grade	**g/t Au**	**MO**	**2.50**

*FGO cut-off grades for UG include a capital cost of $180 million, – which is excluded from MO.

The underground mining costs are actual physical costs taken from Barrick's Loulo UG mining operations. The processing costs for Massawa NZ Mineral Resources are set materially higher than in other Mineral Resources to reflect the BIOX process treatment costs. Consequently, all Massawa NZ underground Mineral Resources are reported at a 2.50 g/t Au cut-off grade.

14.12.3. Massawa Central Zone

The cut-off grade calculation for Massawa CZ and all associated costs are broken down in the Table 14-36.

Table 14-36 Massawa Central Zone Cut-off Grade Calculation at $1,500/oz Gold Price

Parameter	Unit	Symbol	Value
Gold Price	$/oz	Gp	1,500
Royalty	%	R	3%
Selling cost	%	S	0%
Net Gold Price	$/oz	Ng	1,455
Met Recovery	%	REC	88%
Dilution	%	Dil	36%
Ore Loss	%	Loss	8%
Mining Cost - Contractor	$/t mined	MCC	3.35
Mining Cost - Owner's	$/t mined	MCO	0.06
Mining Cost - Grade Control	$/t mined	MCGC	0.14
Total Mining Cost	$/t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	6.34
G&A	$/t milled	G_A	8.00
Ore Crushing & Hauling	$/t milled	rd	0
Mining	$/t milled	Cr	24.15
Process Plant	$/t milled	CP	15.93
Maintenance/Engineering	$/t milled	Mp	-
Total Operating Costs	**$/t**		**48.08**
Full Grade Cut-off	**g/t**	**FGO**	**1.47**
Marginal Cut-off Grade	**g/t**	**MO**	**0.73**

The mining dilution and ore loss for Massawa CZ open pit are set materially higher than in other Mineral Resources to most appropriately reflect the dilution and ore loss that will be incurred during mining of the high-grade ore. Only the ore loss is taken into consideration for the insitu marginal resource cut-off grade. Consequently, all Massawa CZ open pit Mineral Resources are reported at a 0.73 g/t Au cut-off grade.

14.12.4. Sofia

The cut-off grade calculation for all Sofia Mineral Resources (including Sofia Main and Sofia North) and all associated costs are broken down in the Table 14-37. Consequently, all Sofia open pit Mineral Resources are reported at a 0.70 g/t Au cut-off grade.

Table 14-37 Sofia Cut-off Grade Calculation at $1,500/oz Gold Price

Parameter	Unit	Symbol	Value
Gold Price	$/oz	Gp	1,500
Royalty	%	R	3%
Selling cost	%	S	0%
Net Gold Price	$/oz	Ng	1,455
Met Recovery	%	REC	89%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$/t mined	MCC	3.35
Mining Cost - Owner's	$/t mined	MCO	0.06
Mining Cost - Grade Control	$/t mined	MCGC	0.14
Total Mining Cost	$/t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	4.94
G&A	$/t milled	G_A	8.00
Ore Crushing & Hauling	$/t milled	rd	3.25
Mining	$/t milled	Cr	19.84
Process Plant	$/t milled	CP	16.93
Maintenance/Engineering	$/t milled	Mp	0
Total Operating Costs	**$/t**		**48.02**
Full Grade Cut-off	**g/t**	**FGO**	**1.19**
Marginal Cut-off Grade	**g/t**	**MO**	**0.70**

14.12.5. Delya

The cut-off grade calculation for all Delya Mineral Resources and all associated costs are broken down in Table 14-38.

Table 14-38 Delya Cut-off Grade Calculation at $1,500/oz Gold Price

Parameter	Unit	Symbol	Value
Gold Price	$/oz	Gp	1,500
Royalty	%	R	3%
Selling cost	%	S	0%
Net Gold Price	$/oz	Ng	1,455
Met Recovery	%	REC	89%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$/t mined	MCC	3.352
Mining Cost - Owner's	$/t mined	MCO	0.06
Mining Cost - Grade Control	$/t mined	MCGC	0.14
Total Mining Cost	$/t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	9.56
G&A	$/t milled	G_A	8.00
Ore Crushing & Hauling	$/t milled	rd	4.20
Mining	$/t milled	Cr	30.30
Process Plant	$/t milled	CP	27.67
Maintenance/Engineering	$/t milled	Mp	-
Total Operating Costs	**$**		**70.17**
Full Grade Cut-off	**g/t Au**	**FGO**	**1.74**
Marginal Cut-off Grade	**g/t Au**	**MO**	**0.99**

14.12.6. Tina and Bambaraya

Tina and Bambaraya Inferred OP Mineral Resources are reported at a 0.50 g/t Au flat cut-off grade, because the modifying factors affecting the marginal gold cut-off have not yet been defined.

14.13. Massawa Gold Project Mineral Resource Summary by Deposit

The Massawa Mineral Resource estimate has been prepared according to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves standards and guidelines published and maintained by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (the JORC (2012) Code). Barrick has reconciled the Mineral Resources to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated 10th May 2014 as incorporated in NI 43-101, and there are no material differences.

The Massawa Gold Project Mineral Resources are listed in Table 14-39 with the open pit resources declared at an average cut-off grade of 1.0 g/t Au, within a $1,500/oz pit shell. The NZ Underground resources are estimated below the $1,500 pit at a 2.50 g/t Au cut-off grade.

The Massawa Gold Project total Indicated Mineral Resources are estimated to be 23 Mt at 4.00 g/t Au for 3.0 Moz on a total basis with an additional Inferred Mineral Resource of 6.3 Mt at 3.0 g/t Au for 0.61 Moz.

Barrick is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

Table 14-39 Massawa Gold Project Total Mineral Resources as of 31st December 2018, by Deposit

Deposit	Au Cut - Off Grade	Measured			Indicated			Measured + Indicated			Inferred		
		Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)
OPEN PIT													
Massawa Central Zone	0.73	-	-	-	8.7	4.67	1.3	**8.7**	**4.67**	**1.3**	0.87	3.6	0.1
Massawa North Zone 2	1.01	-	-	-	4.7	5.19	0.78	**4.7**	**5.19**	**0.78**	0.19	5.7	0.034
Massawa North Zone 1	1.01	-	-	-	0.74	3.10	0.074	**0.74**	**3.1**	**0.074**	0.042	3.5	0.0047
Sofia Main	0.70	-	-	-	4.75	3.13	0.48	**4.75**	**3.1**	**0.48**	-	-	-
Sofia North	0.70	-	-	-	3.5	2.02	0.23	**3.5**	**2.0**	**0.22**	0.75	1.8	0.44
Delya	0.77	-	-	-	0.74	4.51	0.11	**0.74**	**4.6**	**0.11**	0.08	3.9	0.01
Tina	0.50	-	-	-	-	-	-	-	-	-	1.48	1.1	0.05
Bambaraya	0.50	-	-	-	-	-	-	-	-	-	0.28	1.2	0.02
OP Total		**-**	**-**	**-**	**23**	**4.00**	**3.0**	**23**	**4.00**	**3.0**	**3.7**	**2.2**	**0.26**
UNDERGROUND													
Massawa North Zone UG	2.50	-	-	-	-	-	-	-	-	-	2.6	4.1	0.35
UG Total		-	-	-	**-**	**-**	**-**	**-**	**-**	**-**	**2.6**	**4.1**	**0.35**
TOTAL													
Massawa Total		**-**	**-**	**-**	**23**	**4.00**	**3.0**	**23**	**4.00**	**3.0**	**6.3**	**3.0**	**0.61**

Open pit Mineral Resources are reported as the insitu mineral resources falling within the $1,500/oz pit shell reported at an average cut-off grade of 0.8 g/t Au.
Underground Mineral Resources are those insitu mineral resources below the $1,500/oz pit shell of the North Zone 2 deposit reported at a 2.5 g/t Au cut-off grade.
Mineral Resources for Massawa were generated by Simon Bottoms, MGeol, FGS CGeol, FAusIMM, an employee of the company and Qualified Person.

15. Ore Reserve Estimate

15.1. Ore Reserve Summary

The Massawa Ore Reserves have been estimated in accordance with the JORC (2012) Code. Barrick has reconciled the Ore Reserve to CIM Definition Standards for Mineral Resources and Mineral Reserves dated 10th May 2014 as incorporated in NI 43-101, and there are no material differences.

Only Measured and Indicated Mineral Resources have been used for the conversion to Ore Reserves and thus the Massawa, Sofia, and Delya deposits have been incorporated into reserves. Tina, Bambaraya, and Massawa North Zone Underground only contain Inferred Resources and, therefore, have not been converted to Ore Reserves.

The 2018 Ore Reserve estimate includes an open pit (OP) Probable Ore Reserve of 7.1 Mt at 4.69 g/t Au for 1.1 Moz Au from the Massawa CZ; 4.6 Mt at 4.89 g/t Au for 0.72 Moz Au from the Massawa NZ; 5.7 Mt at 2.91 g/t Au for 0.54 Moz Au for Sofia; and 0.66 Mt at 4.40 g/t Au for 0.092 Moz for Delya. The OP Ore Reserves are those reserves occurring within a $1,000/oz pit design.

Total Massawa, Sofia, and Delya Ore Reserve estimates, as of 31st December 2018, are presented in Table 15-1 and a reserve reconciliation, due to model revisions, is detailed in Table 15-2.

Table 15-1 Massawa, Sofia, and Delya Ore Reserves as at 31st December 2018

Ore Reserve	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Attributable Gold (Moz)*
CZ Probable	7.1	4.69	1.1	0.89
NZ Probable	4.6	4.89	0.72	0.60
Sofia Probable	5.7	2.91	0.54	0.45
Delya Probable	0.66	4.40	0.092	0.077
Total OP Probable	**18**	**4.17**	**2.4**	**2.0**

*Attributable gold (Moz) refers to the quantity attributable to Barrick based on Barrick's 83.25% interest in the Massawa Project. Open pit Ore Reserves are reported at a gold price of $1,000/oz and include dilution and ore loss factors. Open pit Ore Reserves were generated by Shaun Gillespie, an employee of the company, under the supervision of Rodney Quick, MSc, Pr Sci Nat, an officer of the company and Qualified Person.

Table 15-2 Ore Reserve Reconciliation

Reconciliation Item	Contained Gold Ounces
2016 Declaration	2,607 koz
Model Revisions	-184 koz
2018 Declaration	2,423 Koz

Barrick is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Ore Reserve estimate.

15.1.1. Dilution

The mining dilution was assumed to be 10% of additional waste for each tonne of ore in Massawa NZ, Sofia, and Delya Main and 36% in Massawa CZ. The CZ dilution and ore loss applied are based on a dilution study carried out by Maptek taking into account the very narrow nature of the CZ ore body. The 10% figure is based on experience with similar steep tabular ore bodies at Barrick's Loulo operation.

Ore loss was set at 3% for NZ, Sofia, and Delya and 8% for CZ, which is also based on historical information from nearby operations with similar geology.

15.1.2. Grade Control Optimiser Study

Grade Control Optimiser (GCO) work was conducted for Randgold by Maptek consultants. The ultimate aim was to produce 'mineable' solids, with an assumed Standard Mining Unit (SMU) size of 4.5 m. Blocks are flagged with their preferred destination based on cost and revenue of processing individual block model blocks as well as any restriction on destination. These block model blocks are grouped together up to the chosen SMU size. If it is not possible to send a block to its preferred destination, it is grouped with other blocks in such a way as to minimise the cost of sending it to the wrong destination.

The results were flagged into another block model variable which recorded whether a block would be sent to an ore or waste destination. This variable was grade shelled to produce final solids of the mineable ore (Figure 15-1).

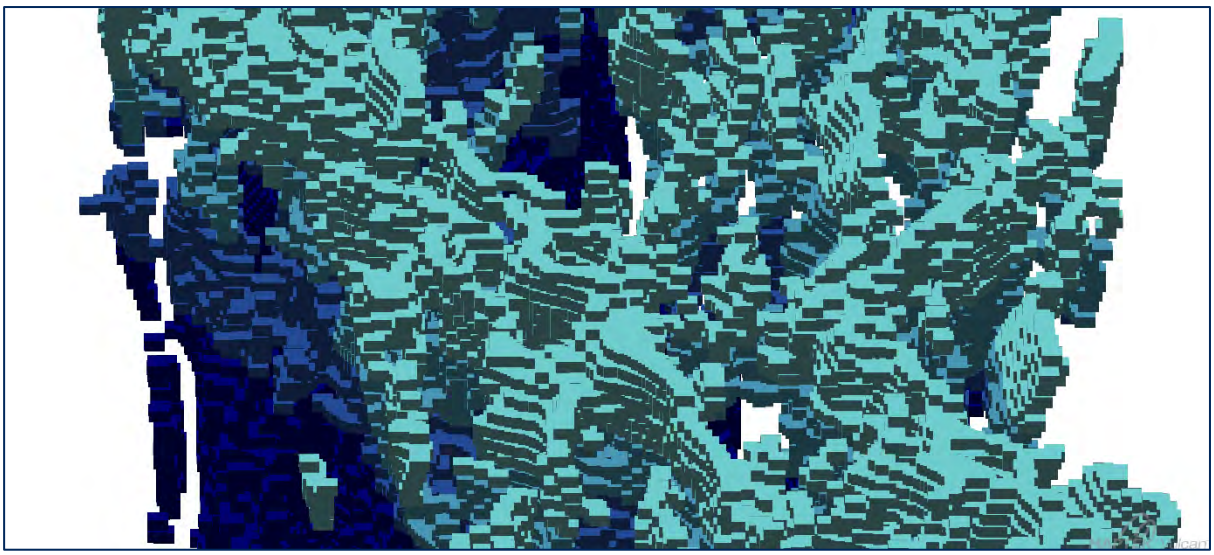


Figure 15-1 Final Grade Control Optimiser Solids

15.1.3. Grade Control Optimiser Method

The starting point of the GCO was a regularised block model. The block model that was used for this was 'MAS_CZ_SEP_18_TOTAL_MODEL_GC_BLOCK_AU_ONLY.bmf'.

Gold processing recovery was calculated into the original model as opposed to being calculated as a temporary variable in the GCO script. The script was updated to account for this. Three scripts were run on the original sub-blocked model before it was regularised.

- 01_lithox18_calc_2018.bcf – script to calculate the 'lithox18' variable code based on the group, oxidation, and redox variables.
- 02_rec_calc_2018.bcf – script to calculate metallurgical recovery as a decimal percentage based on geometallurgical recovery domain (metdom18), group, oxidation, and redox.
- 03_contained_met_calc_2018.bcf – script to calculate the gold mass in grams in a block and the recovered gold grams in each block when taking into account processing recovery.

The original aim, based on experience from the previous work, was to regularise and run the GCO process on a 0.5 m regular model. It was believed that this would best honour the original data from the sub-blocked model. Reducing the block size increases the number of blocks and corresponding processing time but gives better definition of the ore zone.

Attempts were made to run the GCO using several machines, including tests using two super-computers in the United Kingdom, but it was ultimately found to be impractical to run the GCO on a 0.5 m sized regular model with the lack of block restriction. It was decided to use a 0.75 m regularised model to estimate the 4.5 m SMU. The Z height of the blocks was made 2.5 m to match the proposed flitch height.

The block model was run on the scripts described above and the model was regularised to 0.75 m by Randgold, then provided as the block model 'MAS_SEP18_SMU_0_75m_2_5m.bmf', on which further scripts were conducted. These two scripts were:

- 00_add_variables_to_model.csh – script to add in the requisite variables to run the GCO costing script 04.
- 04_GCO_Calc_for_Values_2018.bcf – script to calculate the relative cost/profit of sending each block to each possible destination. Used in the GCO to decide on the optimum destinations for blocks.

These scripts prepared the regular block model to be run through the Vulcan GCO process (Figure 15-2). The '.csh' script added the required variables for the values and destination restrictions referenced in the GCO script (04_GCO_Calc_for_Values_2018). This was scripted to make the process of adding the variables easier and quicker, whilst ensuring that the variable types and defaults were correct for later parts of the process.

In Vulcan, the option 'Block > Grade Control Optimiser > Grade Control Optimiser' was used to set up a single run of 4.5 m at one elevation. The middle of the flitch elevation was used and it was told to use existing economic block value variables. Under economic value, five destinations were defined. These were 'dump', 'fresh', 'oxide', 'trans_oxide', and 'red_trans', with the corresponding constraint variables 'cons_dump', 'cons_fresh', 'cons_oxide', 'cons_oxide_trans', and 'cons_reduced_trans'. The process was told to use a mining unit of 6 x 6 blocks, this defining a 4.5 m mining unit. An edge buffer of one block was used. The destinations assigned were

written to ‘dest_id_45m’ and ‘dest_name_45m’. Optimisation took place from feasibility and 500 maximum iterations were used under the advanced settings.

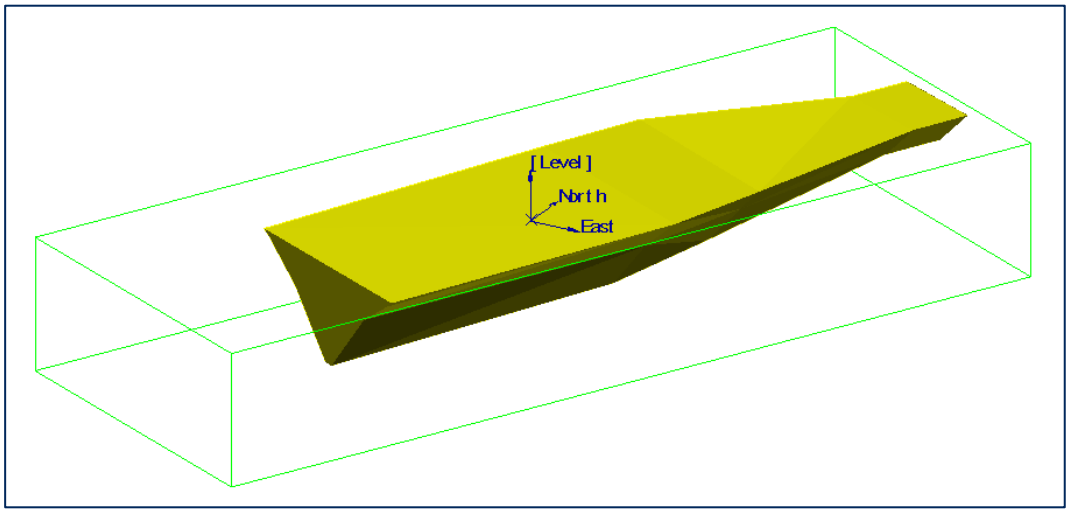


Figure 15-2 Solid Used to Limit the GCO Process and Reduce Processing Time to the Area of Interest

Once one spec file was set up, a custom lava script ‘GCO_script_v10_B.lava’ was run in order to produce and run spec files for all required benches. Due to time constraints, the data was copied to three different machines which all worked on different elevations of the model. Finally, another script was run which flagged blocks that had been sent to ore destinations with a 1.

- 05_flag_for_grade_shell.bcf – script to flag a variable called ‘flag_for_grade_shell_45m’ with a value of 1 for ore destinations and 0 for waste destination.

This flagged variable was grade shelled in Vulcan, honouring the exact block boundary to produce a solid triangulation. As this process was being run across three different machines on different elevations of the block model, each was grade shelled separately and then clipped such that the final solids did not overlap. These solids were sent back to Randgold for use in flagging back into the block model; those blocks which would be considered ore.

The number of blocks used in the process should be restricted as much as possible. A pit design or economic pit shell solid is the best thing to use for this task, although at the time of running the process, these restricting solids were not available. A very basic solid was therefore made for this task to restrict as much as possible the area to be optimised while still easily covering any reasonable economic pit shell that may be produced. Even with these restrictions, the GCO process would take days to run for a single SMU simulation.

The data therefore had to be used on several computers, each running simulations on different elevations of the pit and the results combined afterwards. The number of simulation iterations had to be set to 500 to produce a result in a reasonable timeframe. In the deeper parts of the pit, a single GCO run may take a few minutes as there are fewer blocks to simulate. Higher up the pit, a GCO run with the same settings on a bench could take a number of hours as there are so many more blocks to run simulations on.

The time and technical restrictions of the Project meant that only the 4.5 m SMU could be simulated. In the previous work on Massawa, three or four different SMU sizes were run to compare the results and ensure consistency.

It is recommended that for any future GCO runs, a pit shell or pit design be used to restrict the number of blocks for the simulations. Time should be allowed for single runs to be performed on individual benches at different elevations of the pit to ensure that results and scripts are correct for all of the various rock types and destinations throughout the pit. A single machine should be used to perform the optimisation work to ensure consistency between different versions of block models and scripts.

15.1.4. Optimiser Results

The final flagged variable 'flag_for_grade_shell_45m' was grade shelled with the grade shell honouring the exact block boundary. These four solids were clipped so that each 2.5 m flitch was only covered once, i.e. the triangulations were mutually exclusive. The results are shown below in Figure 15-3 and Figure 15-4. The solids were then used to flag into the block model. As they were used for flagging, it was necessary to combine them into a single closed valid solid. It may be preferable to combine them into one solid and clip by the final pit design, but in this case, it was not required to combine the solids into one.

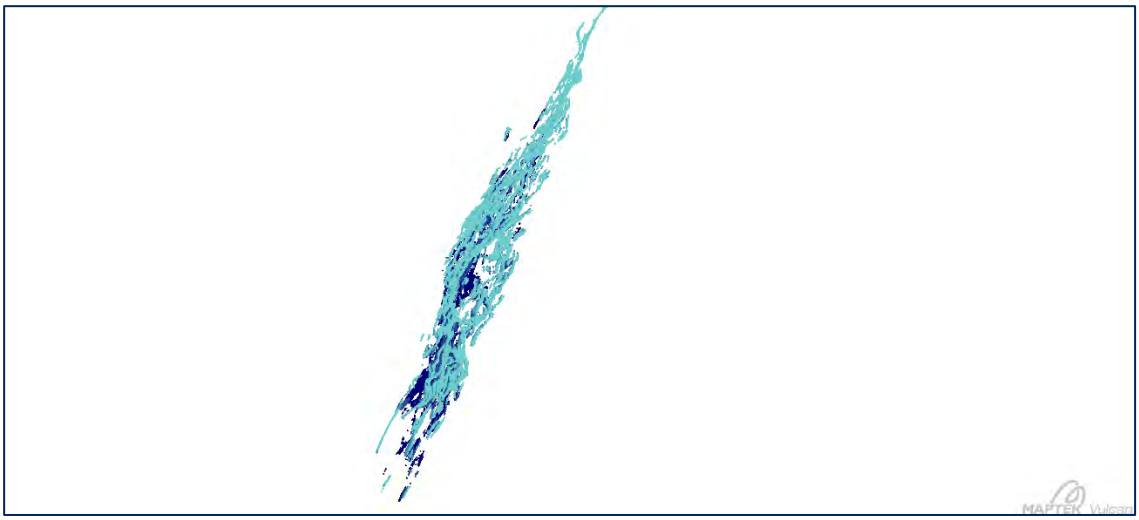


Figure 15-3 Final Grade Shelled Solids of Blocks Sent to Ore Destinations (Plan View)

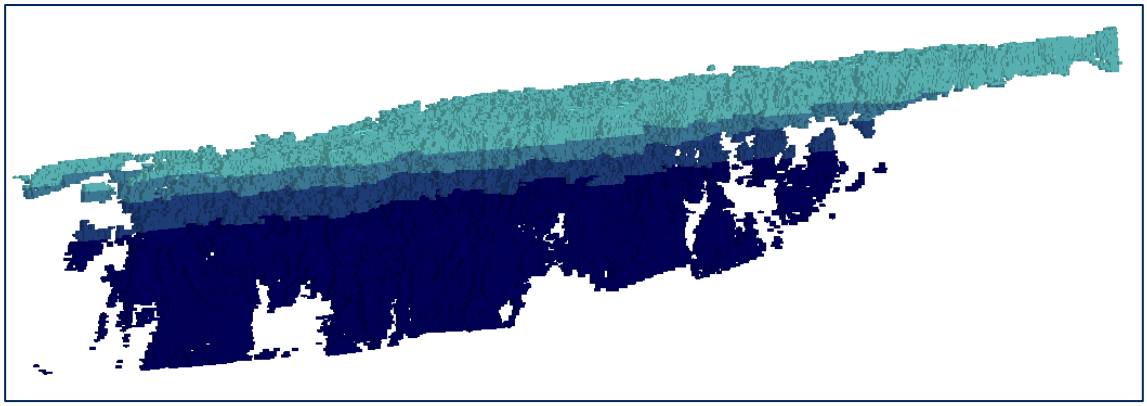


Figure 15-4 Final Grade Shelled Solids of Blocks Sent to Ore Destinations (Longitudinal View)

In order to verify the automated method described above, a manual exercise was carried out, whereby the mineable polygons were manually digitised, on a 4.5 m flitch, using the ore body wireframes and gold grade block model values as a visual guide. Figure 15-5 illustrates the results of this exercise which gave an overall 40% dilution and 10% ore loss. This compares favourably with the 36% dilution and 8% as determined by the GCO study.

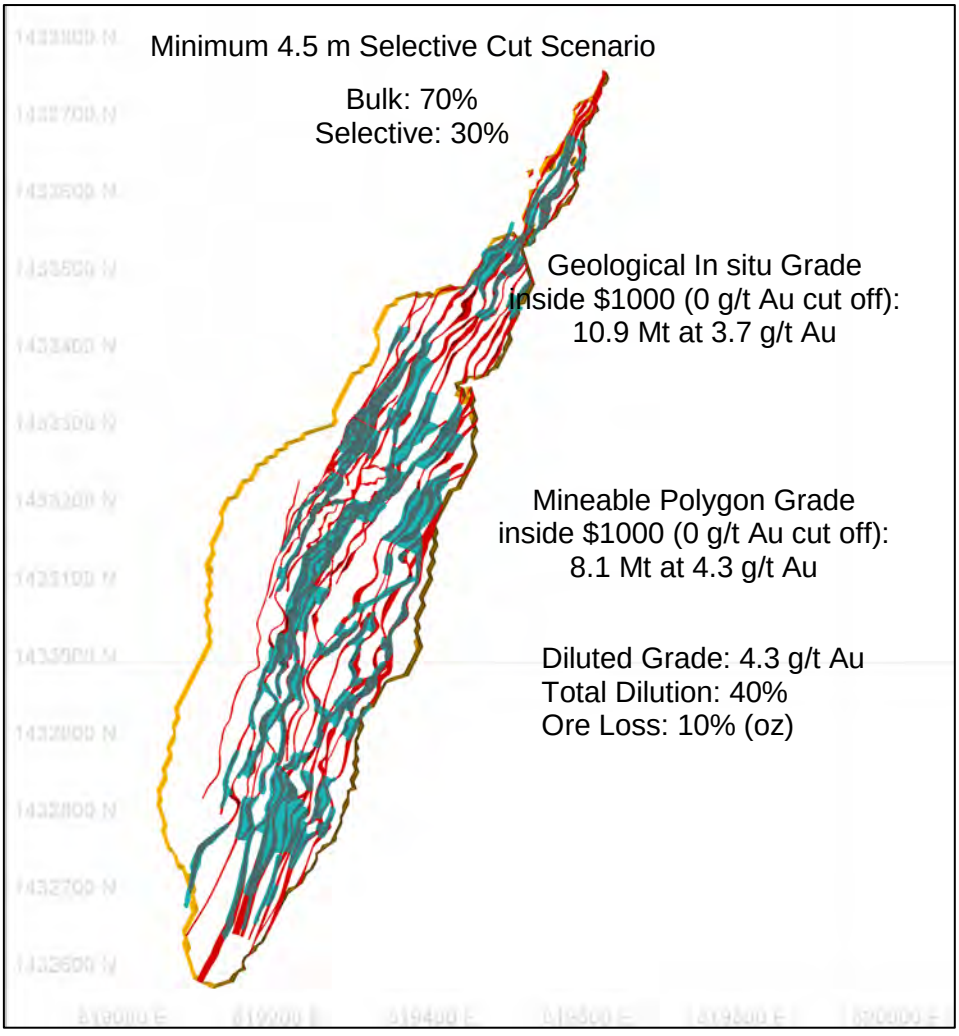


Figure 15-5 Flitch Showing Manually Digitised Mineable Polygons

The manual exercise also confirms the ratio of bulk versus selective mining that will be required in the CZ pit being 70% bulk mining and 30% selective mining.

15.2. Cut-Off Grade

To enable an improved cash flow, a full grade ore classification was implemented, with preferential treatment for high-grade ore. Marginal ore is treated only at the end of the mine life after being stockpiled, when mining has stopped.

The cut-off grade, g, is where revenue from the gold produced equals the cost of treating ore that includes the cost of refinery and shipping. Cut-off grades are calculated below as:

$$g = c \div r \times p$$

Where; **g** = cut-off grade

c = total operating costs $ /t treated = mining cost + metallurgical cost + G&A costs

r = metallurgical recovery

p = selling price $ /g = (gold price – (gold price x royalty)) ÷ 31.10348.

15.2.1. Massawa North Zone

The cut-off grade calculation for Massawa NZ Ore Reserve and all associated costs are broken down in Table 15-3.

Table 15-3 Massawa North Zone Ore Reserve Cut-Off Grade Calculation at $1,000/oz Gold Price

Parameter	Unit	Symbol	2018 Reserves
Gold Price	$/oz	Gp	1,000
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	970
Met Recovery	%	Rec	83%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	11.48
G&A	$ /t milled	G_A	8.60
Mining	$ /t milled	Cr	40.56
Process Plant	$ /t milled	Cp	31.35
Maintenance/Engineering	$ /t milled	Mp	-
Total Operating Costs	**$/t**		**79.91**
Full Grade Cut-off	**g/t Au**	**FGO**	**3.18**
Marginal Cut-off Grade	**g/t Au**	**MO**	**1.57**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**2.89**
		MO	**1.42**

The processing costs for Massawa NZ Mineral Resources are set materially higher than in other Mineral Resources to reflect the BIOX process treatment costs.

15.2.2. Massawa Central Zone

The cut-off grade calculation for Massawa CZ Ore Reserve and all associated costs are broken down in Table 15-4.

Table 15-4 Massawa Central Zone Ore Reserve Cut-off Grade Calculation for $1,000/oz Gold Price

Parameter	Unit	Symbol	2018 Reserves
Gold Price	$/oz	Gp	1,000
Royalty		R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	970
Met Recovery	%	Rec	76%
Dilution	%	Dil	36%
Ore Loss	%	Loss	8%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	5.57
G&A	$ /t milled	G_A	8.60
Mining	$ /t milled	Cr	21.35
Process Plant	$ /t milled	Cp	15.63
Maintenance/Engineering	$ /t milled	Mp	0.00
Total Operating Costs	**$/t**		**44.98**
Full Grade Cut-off	**g/t Au**	**FGO**	**2.06**
Marginal Cut-off Grade	**g/t Au**	**MO**	**1.08**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**1.69**
		MO	**0.89**

15.2.3. Sofia

The cut-off grade calculation for Sofia Ore Reserve and all associated costs are broken down in the Table 15-5.

Table 15-5 Sofia Main Ore Reserve Cut-off Grade Calculation for $1,000/oz Gold Price

Parameter	Unit	Symbol	2018 Reserves
Gold Price	$/oz	Gp	1,000
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	970
Met Recovery	%	Rec	89%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	4.16
G&A	$ /t milled	G_A	8.60
Mining	$ /t milled	Cr	18.63
Process Plant	$ /t milled	Cp	18.90
Maintenance/Engineering	$ /t milled	Mp	0.00
Total Operating Costs	**$/t**		**45.53**
Full Grade Cut-off	**g/t Au**	**FGO**	**1.69**
Marginal Cut-off Grade	**g/t Au**	**MO**	**1.00**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**1.54**
		MO	**0.91**

15.2.4. Delya

The cut-off grade calculation for Delya Ore Reserve and all associated costs are broken down in the Table 15-6.

Table 15-6 Delya Reserve Cut-off Grade Calculation for $1,000/oz Gold Price

Parameter	Unit	Symbol	2018 Reserves
Gold Price	$/oz	Gp	1,000
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	970
Met Recovery	%	Rec	91%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	7.02
G&A	$ /t milled	G_A	8.60
Ore Crushing & Hauling	$ /t milled	Rd	4.20
Mining	$ /t milled	Cr	26.07
Process Plant	$ /t milled	Cp	25.83
Maintenance/Engineering	$ /t milled	Mp	-
Total Operating Costs	**$/t**		**64.10**
Full Grade Cut-off	**g/t Au**	**FGO**	**2.33**
Marginal Cut-off Grade	**g/t Au**	**MO**	**1.38**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**2.12**
		MO	**1.26**

16. Mining Methods

The proposed mining method is conventional 90 t truck and excavator open pit mining method. Concepts considered when making this decision included:

- Ore bodies at the four deposits are long, narrow, and exposed on surface.
- Length of Project life - the Project has a current mining life of approximately 10 years.
- Mill feed, waste, and low-grade material all require storage in separate facilities requiring flexibility in haulage options.
- Loading units have been nominated as excavators in backhoe configuration rather than face shovels, due to the selective nature of the mining required together with the dimensions of the ore lodes.

16.1. Mine Design

16.1.1. Geotechnical Investigations and Implications for Mine Design

Mine design geotechnical investigations for Massawa were carried out by Dr Peter JS Gash of MineNet, an independent Qualified Person, in December 2018 and the work was assessed and approved by Barrick.

The average depth (Table 16-1) of the regolith categories (excepting the NE – holes 215 and 217 where the carbonaceous schist (CS) sub-outcrop results in deep penetration of broken rock and changes the picture) is as follows (note depths are approximate vertical equivalents to the down hole depths):

- Base of SAProlite = 25 m.
- Base of SAPRock = 40 m.
- Base of Weathered Rock = 50 m.
- Base of Broken Rock = 60 m.
- Base of Broken CS (locally in NE) = 90 m.

Table 16-1 Characterisation of Surface Material from Logs and Photos with Test Support

Hole	133	145	215	217	247	248	251	256	282	283
Zone	N2	N2	N2	N2	C2	C2	C2	C2	C1	C1
Wall	E	W	E	E	E	W	W	W	E	W
Base of Sap (m)	27	6	37	49	12	15	36	30	12	30
Base of SAPR (m)	39	51	56	77	22	41	54	43	42	39
Base of Weath (m)	46	62	58	81	30	52	68	71	48	48
Base of Broken (m)	59	65	148	115	30	71	72	72	54	50

Sufficient rock testing was carried out in the Massawa investigations to establish the rock strength and modulus characteristics for slope design purposes. The uniaxial strengths of intact rocks at the site varied between 70 MPa and 170 MPa. The key observation related to the CS, which were too broken for testing from depths down to 80 m below surface, but at depths were

competent with strength of 70 MPa. The density of all rock types was 2.75 t/m^3 with a very tight variation.

The rock types are different at the other sites across the lease area but display a similar range of strengths.

Slope designs for Massawa CZ and NZ, Sofia Main and North, and Delya are based on the following assumptions (Table 16-2):

Table 16-2 Slope Design Assumptions

Deposit	Soil	Weathered and Broken Rock	Fresh Rock	
			Hanging Wall	Footwall
Massawa CZ and NZ	10 m lifts 60° bench faces Inter-ramp angle (crest to crest) (IRA): 42° Berm width: 5.3 m	10 m lifts 75° bench faces IRA 45.9° Berm width: 7 m	Twin 10 m lifts 75° bench faces IRA 52.5° Operational Offset: 2 m Alternate catch berm width: 8 m	Twin 10 m lifts 75° bench faces IRA 54.3° Operational Offset: 2 m Alternate catch berm width: 7 m
Sofia Main	10 m lifts 60° bench faces Bench stacks for walls <40 m high IRA: 44° Berm width: 4.5 m Bench stacks for walls >40 m high IRA: 40° Berm width: 6 m	10 m lifts 75° bench faces IRA 47.4° Berm width: 6.5 m	Twin 10 m lifts 80° bench faces IRA 55° Operational Offset: 2 m Alternate catch berm width: 8.5 m	Twin 10 m lifts 75° bench faces IRA 51.6° Operational Offset: 2 m Alternate catch berm width: 8.5 m
Sofia North	10 m lifts 60° bench faces IRA: 40° Berm width: 6 m	10 m lifts 75° bench faces IRA 45.9° Berm width: 7 m	Twin 10 m lifts 80° bench faces IRA 52.2° Operational Offset: 2 m Alternate catch berm width: 10 m	Twin 10 m lifts 75° bench faces IRA 54.3° Operational Offset: 2 m Alternate catch berm width: 7 m
Delya	10 m lifts 60° bench faces IRA: 40° Berm width: 6 m	10 m lifts 75° bench faces IRA 47° Berm width: 6 m	Twin 10 m lifts 80° bench faces IRA 56° Operational Offset: 2 m Alternate catch berm width: 8 m	Twin 10 m lifts 80° bench faces IRA 54° Operational Offset: 2 m Alternate catch berm width: 9 m

Slopes in soil are based on the following groundwater provisions:

- Massawa Central and North Zone, Sofia Main
    - The slopes must be depressurised. The dewatering plan for the Massawa pit has been derived separately in the report by Artois Consulting (Artois) of December 2018 'Hydrogeology Assessment for Pit Dewatering'. This report has had input from, and has been approved by, Callistus Vog-Enga. This therefore becomes an integral part of the slope design.
    - Piezometers around the pit are needed to monitor the effectiveness of depressurisation measures. These are additional to the pumping provisions given in the Artois hydrogeology report.
    - Prioritise dry season working in soils, which is particularly relevant for the first drop cut and integral with the sump strategy.
    - Perimeter drainage and rainy season protection is still needed; sump strategy, run-off diversions, and ramp cross-falls.
    - Limitation on rate of deepening – not a precise science but an average of one bench lift (10 m) per month in order to allow water pressure dissipation under gravity.

- Sofia North
    - The slopes must be depressurised.
    - The NE sector of the FW will require advance dewatering using borehole wells. The requirement for any special measures for the outlying north extension pit should be assessed in the light of future investigations.
    - Water retaining structure(s) and diversion(s) will be needed upstream of the river that crosses the extreme north of the current target ($1,000/oz) pit. These projects will be in the remit of the Construction Team, but design collaboration with geotechnics/planning is essential throughout the process.
    - Prioritise dry season working in soils, which is particularly relevant for the first drop cut and integral with the sump strategy.
    - Perimeter drainage and rainy season protection is still needed; sump strategy, run-off diversions, and ramp cross-falls
    - Limitation on rate of deepening – not a precise science but an average of one bench lift (10 m) per month in order to allow water pressure dissipation under gravity.

- Delya
    - The slopes must be depressurised.
    - The SE sector of the FW will require advance dewatering using borehole wells
    - Prioritise dry season working in soils, which is particularly relevant for the first drop cut and integral with the sump strategy.
    - Perimeter drainage and rainy season protection is still needed; sump strategy, run-off diversions, and ramp cross-falls
    - Limitation on rate of deepening – not a precise science but an average of one bench lift (10 m) per month in order to allow water pressure dissipation under gravity.

16.1.2. Hydrogeology

The hydrogeological assessment for the Massawa mining feasibility evaluation concludes the following:

- Due to the naturally shallow groundwater levels across the site (typically 10 m to 30 m below surface), groundwater inflow to the excavations will occur throughout mining. The groundwater inflow horizons are associated with the near surface weathered horizons, and the NNE-SSW fracture zones and lithological contacts at depth. In general, the sedimentary units are more permeable than the igneous units.
- The inflows depend on the hydraulic characteristics of the units, the pit size, and the mining advance rate. Due to the accelerated mining rate of each pit (up to 40 m/year), high groundwater drawdown rates need to be achieved to keep pace with the operation. The likely range of pumping rates per pit during their Life of Mine (LOM) are as follows:
    - Massawa CZ Pit: average 70 L/s (range: 19 – 96 L/s)
    - Massawa NZ Satellite Pit: average 43 L/s (range: 32 – 72 L/s)
    - Massawa NZ Pit: average 18 L/s (range: 7 – 21 L/s)
    - Sofia Main Pit: average 10 L/s (range: 1 – 14 L/s)
    - Sofia Main Satellite Pit: average 3 L/s (range: 2 – 4 L/s)
    - Sofia North Pit: average 17 L/s (range: 12 – 22 L/s)
    - Delya North Pit: average 3 L/s (range 1 – 3 L/s)
    - Delya South Pit: <1 L/s

According to the geophysical surveys, the monitoring data, the insitu hydraulic testing, and the calibrated numerical forecast models, the majority of the groundwater can be intercepted by perimeter and in-pit dewatering wells (approximately 80% of groundwater inflow). The remaining water will be drained using sub-horizontal drains drilled into the pit wall. This groundwater seepage will collect in the in-pit sumps.

The geotechnical slope stability analysis can therefore be based on ‘drained’ or ‘dry’ pit slopes. In addition to seepage water, the in-pit sumps’ storage volume and pumping capacity are designed to retain and evacuate a 1:50 year return period storm event (183 mm/24 hr). The size of each sump varies between less than 1,000 m^3 and 40,000 m^3 and can be accommodated along the pit floor.

The capital and recurring capital investment for the dewatering system is estimated to amount to a minimum of $16.5 million for all pits combined. This investment is spread across the LOM in accordance with the mine advance rates. The operating costs are estimated to be in the order of $3.7 million. An additional contingency should be built into the final cost calculations as adjustments and modifications to mine dewatering plans commonly occur.

16.1.3. Mine Design and Road Design

The Whittle pit optimisation shells were imported into the Vulcan software suite for generating practical mine designs. The designs strictly adhered to the geotechnical regimes described in the previous section and were applied on the $1,000/oz optimal pit.

Typical ramp formations used are 25 m width (inclusive of berm) for double-lane ramps. At great depth, ramps are reduced to 14 m width (inclusive of berm), to reduce the strip ratio. This reduction of ramp width impacts production and has only been applied for the last five benches. Passing bays have been provided for at strategic positions. Ramp gradients are 10%. Figure 16-1 to Figure 16-4 show the design pits for the CZ, NZ, Sofia, and Delya pits.

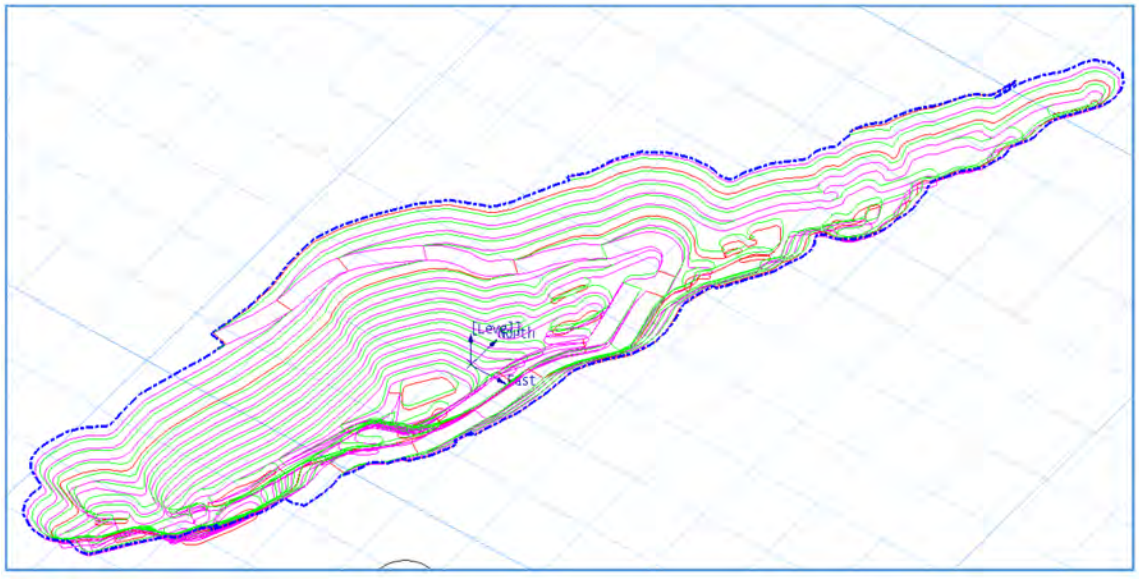


Figure 16-1 Central Zone Design Pit

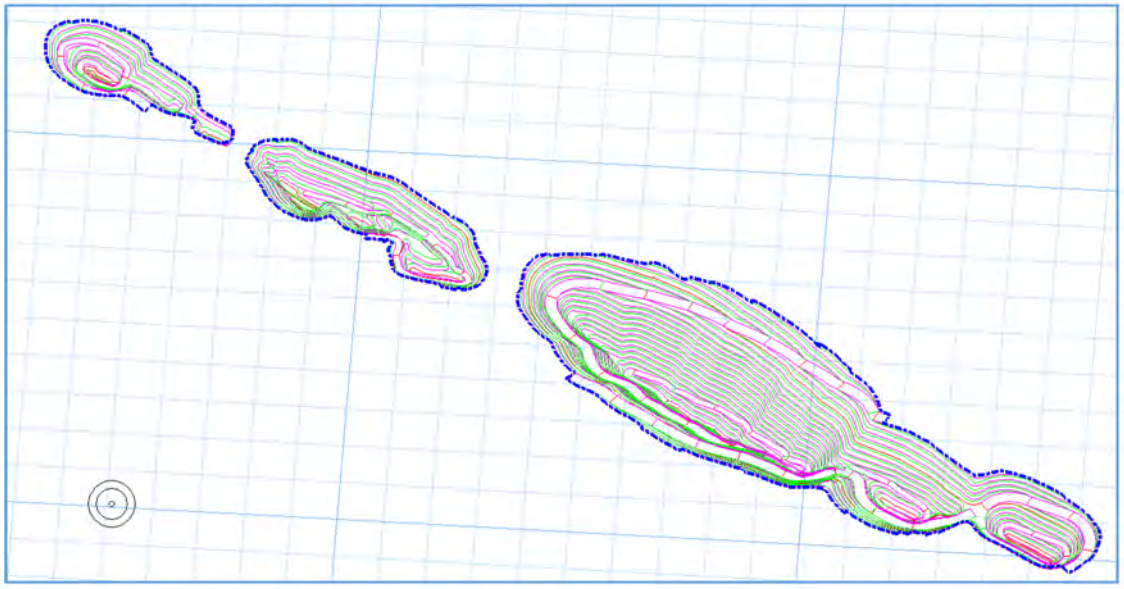


Figure 16-2 North Zone Design Pit

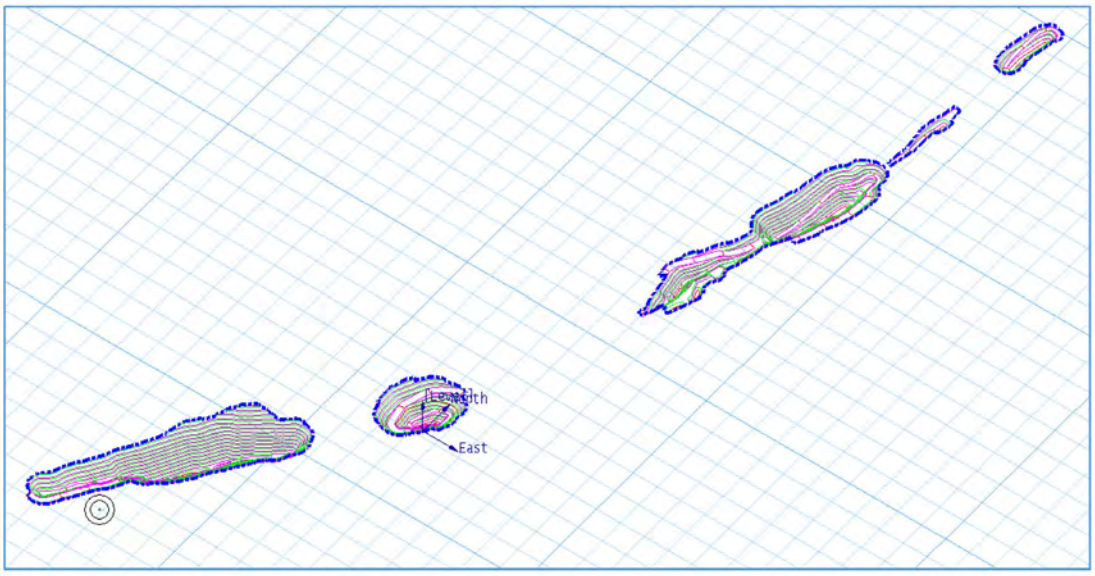


Figure 16-3 Sofia Design Pit

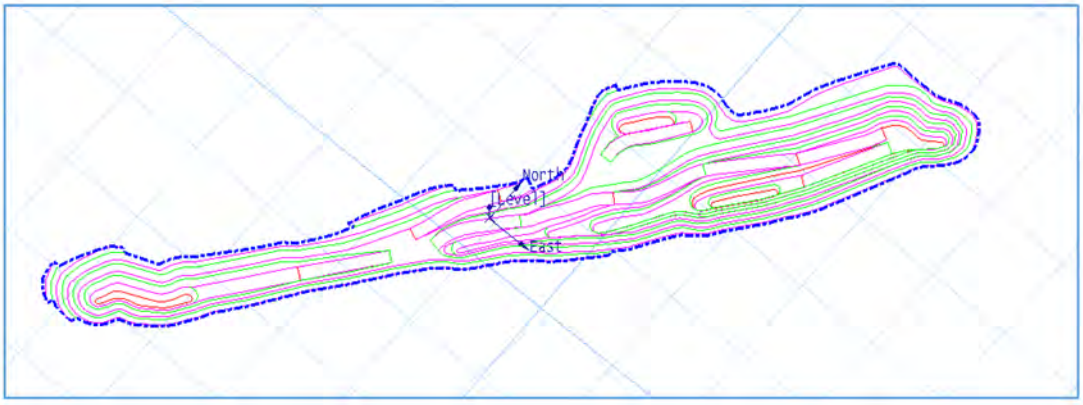


Figure 16-4 Delya Design Pit

16.1.4. ROM Pad Design and Mine Rock Dumps

The ROM pad was designed with a total capacity of 4 Mt, which will allow for 3 Mt of refractory ore to be mined and stockpiled for processing later in the mine life (Figure 16-5). The remaining 1 Mt capacity allows for the blending of WOL ore to maximise the efficiency of the processing. This calls for a footprint of 650 m by 500 m, on an assumption that 70% of the floor can be used for storage of material up to a maximum height of 20 m. The waste material used for constructing the ROM pad will come from the Main Zone pit and will be capitalised. A total of 4.1 MBCM is required for construction of the ROM pad.

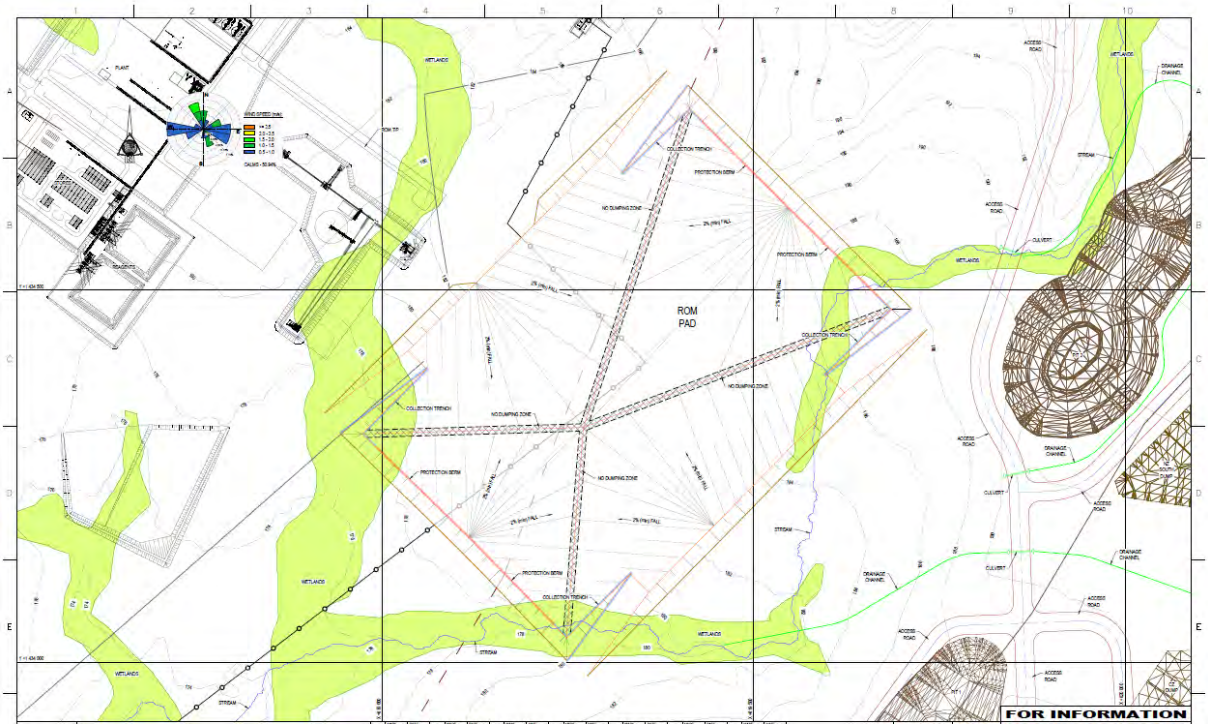


Figure 16-5 Image of ROM Pad Design

It should be noted that since the Massawa study in 2010, the waste dump design and environmental provisions have developed. There is now a company standard design across the region, which has been approved by the Environmental Department.

The objective of this exercise is to maximise the allowable dump stack geometry and the contained volume capacity whilst ensuring that the design will be stable in the long term and will respect the environmental commitments for rehabilitation.

There are two governing parameters: -

- Angle of repose of dumped materials – 37°. This angle is remarkably constant across a wide range of materials and conditions (always between 35° and 40°).
- Angle of battering for rehabilitation and revegetation – 30°. This is the angle defined by Environmental Department and is a governance issue. Historically varied from a maximum slope of 1-in-1.5 to below 20° (with many dumps not rehabilitated at all).

Thereafter, the design dimensions are a function of the operability and the method statement for how the rehabilitation will be carried out and what equipment will be trafficking. There is no environmental constraint on the lifts and offsets, only the battering angle. The battering of the slopes will be performed by bulldozers working on the end-tipped slope gradient from top to bottom. There is a limiting vertical lift over which this can be safely and efficiently performed. The size of bulldozer is taken as D9 which has a footprint of approximately 3 m wide.

The dimensions of the finished rehabilitated dumps should be:

- Dumping operating vertical lift: 20 m
- Minimum offset (construction berm): 5 m - with the finished slope profiled and battered to 30°.

In order to have sufficient width to carry out the battering, the dimensions of the stacks when they are built by tip and push dumping should be:

- Dumping stack lift: 20 m.
- Angle of end-tipped dump: 37° – angle of repose.
- Offset for each stack: 13 m.

Note that the depositional offset of 13 m narrows down to 5 m as the slope is battered by re-distributing material from a 37° slope down to a 30° slope.

The resultant slope angle (equivalent to mid-bench slope angles of the pit walls) for this geometry is:

- 27°.
- Factor of Safety (FoS) = 1.37.

This represents an increase in potential volume capacity of approximately 4° compared to the previous design. This design angle is shallower than the maximum allowable slope for long-term stability, which would be a little over 30° overall. The limiting design angle is a function of the requirement for site rehabilitation not dump stability.

In the case of the Massawa plan, the dump plans have not been referred for geotechnical review. The above design has been specified as a reference for any appraisal.

The locations of both the dumps and ROM pad were selected with the aim of minimising the haul distances, given the topography constraints of the area. The ROM pad is located adjacent to the plant position.

Figure 16-6 to Figure 16-8 are 3D images of the CZ, NZ, Sofia, and Delya pits and dump designs. Figure 16-9 shows the location of the Massawa CZ and NZ pits and dumps relative to the ROM pad.

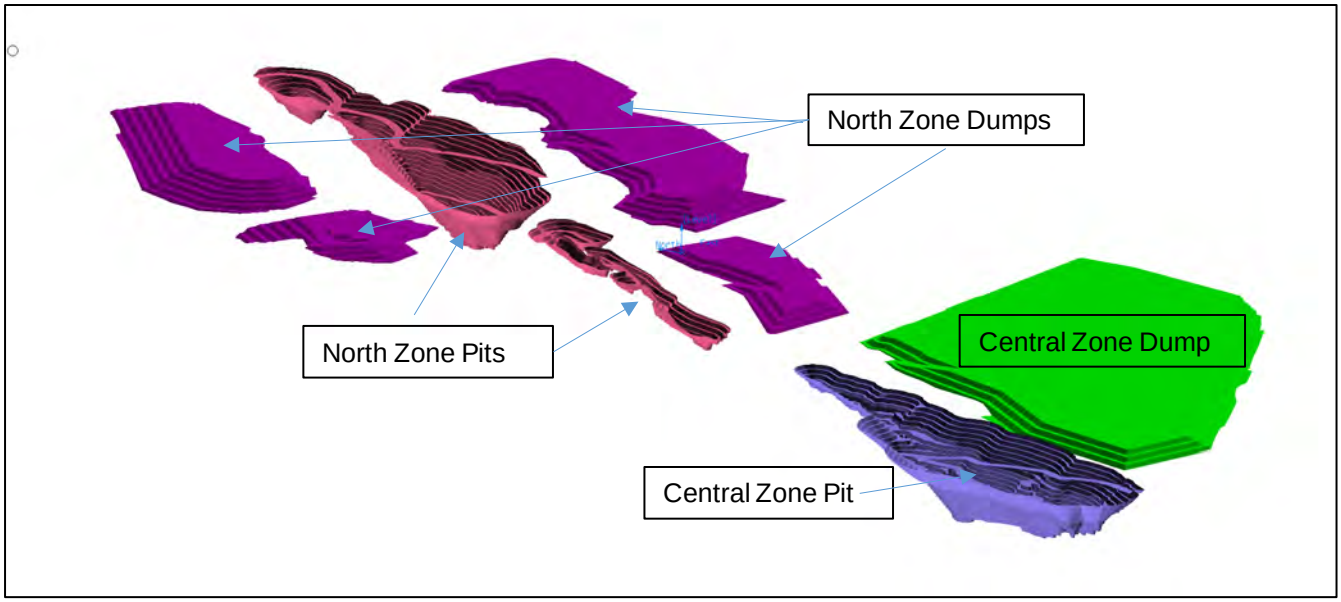


Figure 16-6 3D Image of Central and North Zone Pit and Dump Designs

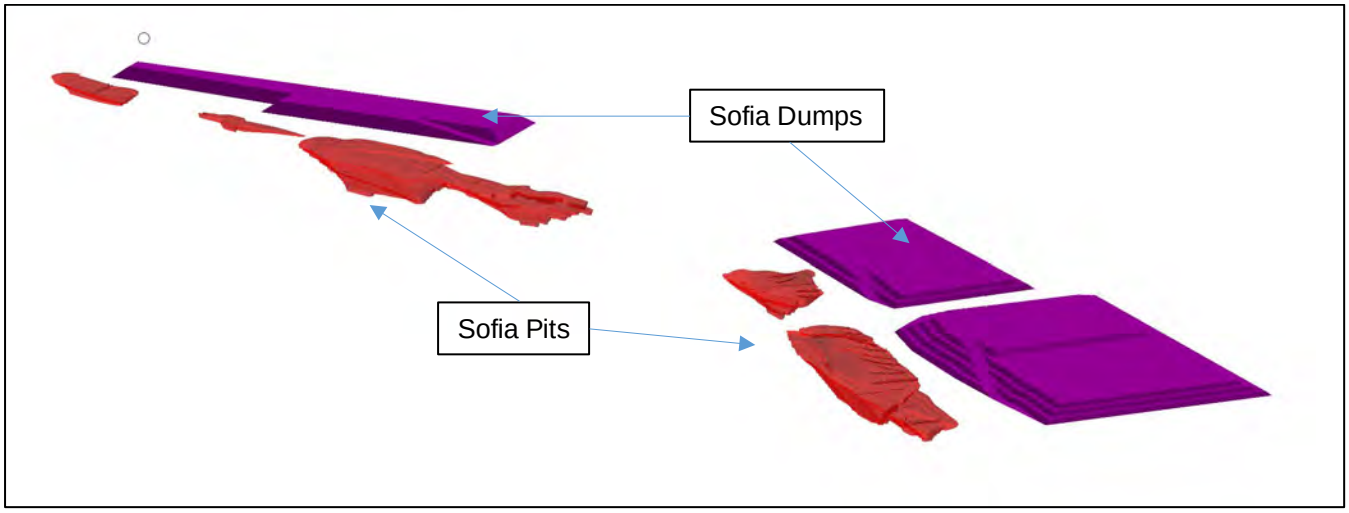


Figure 16-7 3D Image of Sofia Pit and Dump Designs

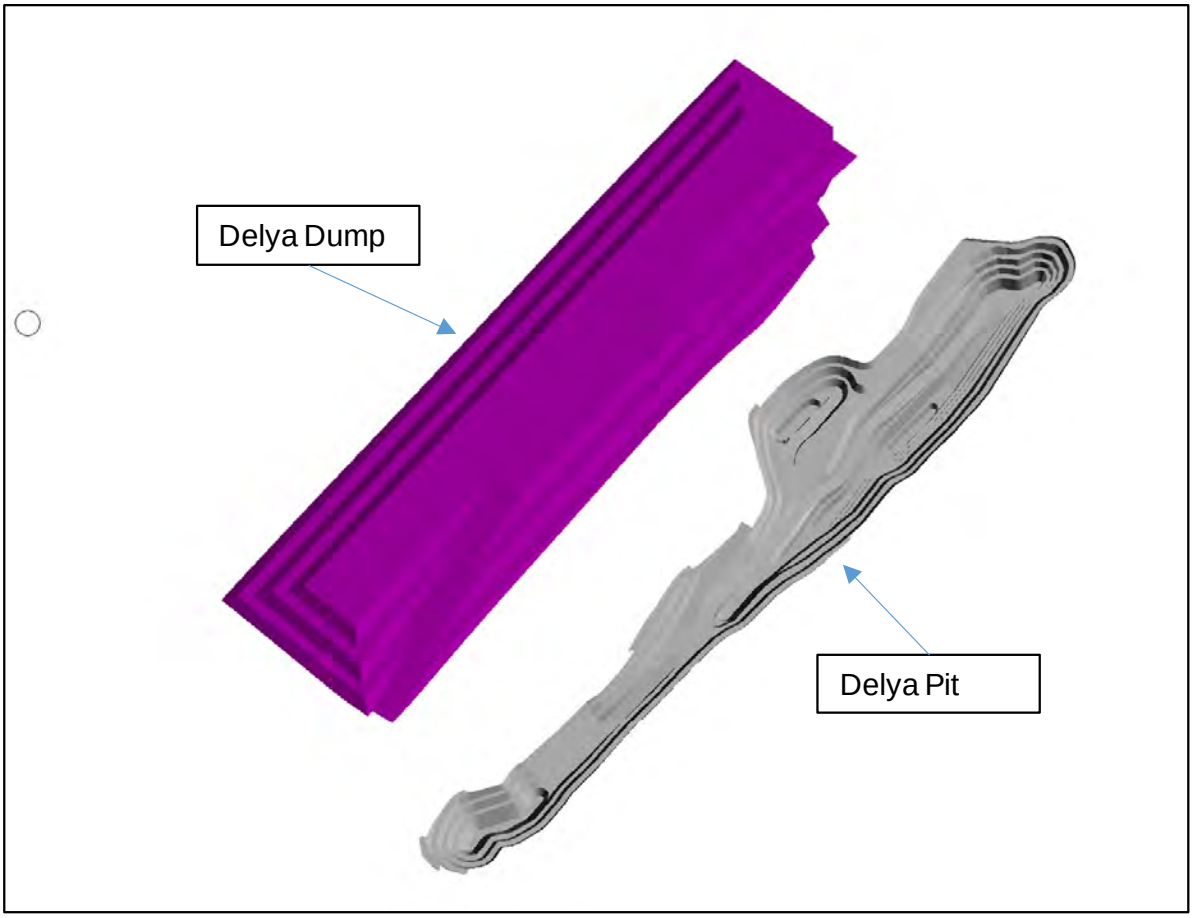


Figure 16-8 3D Image of Delya Pit and Dump Designs

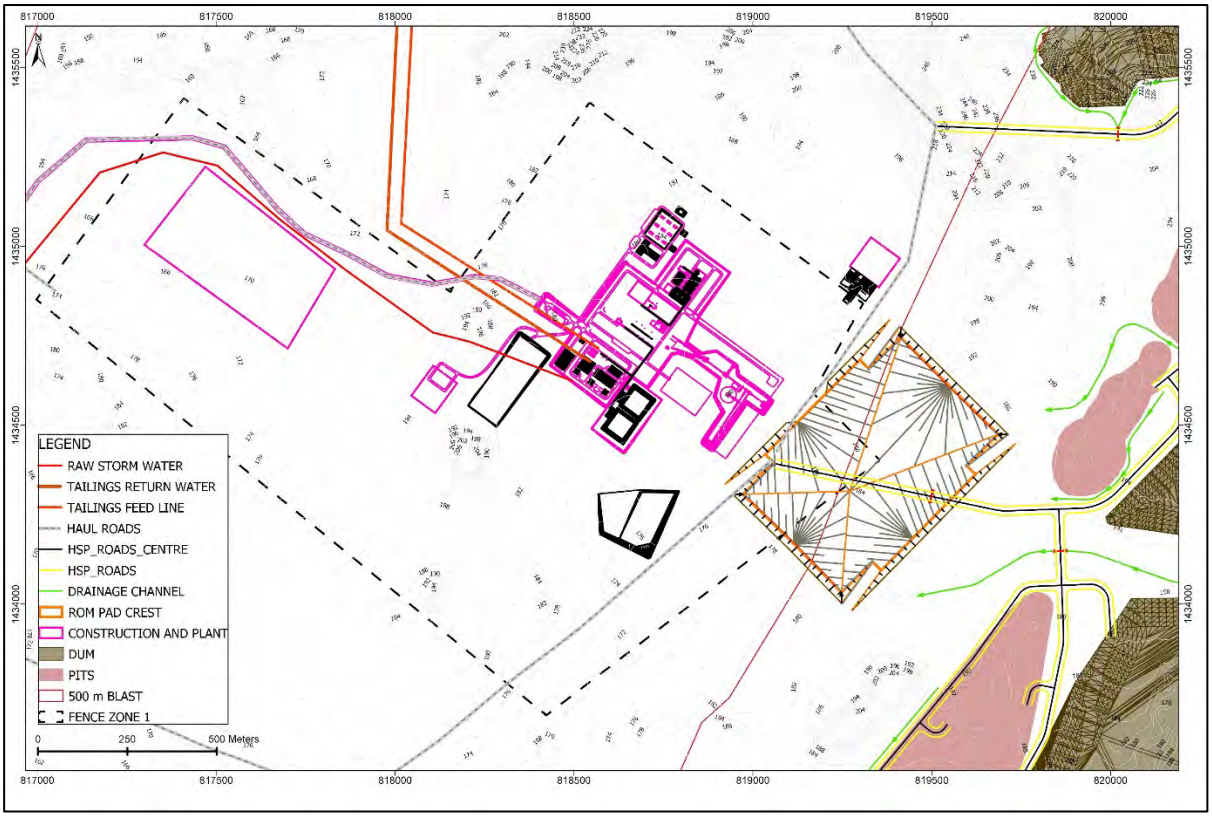


Figure 16-9 Massawa Central and North Pits and Dumps Relative to the ROM Pad

16.1. Open Pit Mining and Ore Feed Schedules

The Massawa CZ and NZ, Sofia, and Delya ore mining and feed schedules are shown in Table 16-3, Table 16-4, Table 16-5, and Table 16-6, respectively. The waste mining schedule for all four deposits is shown in Table 16-7. The overall open pit mining schedule is shown in Table 16-8.

Table 16-3 Massawa CZ LOM Ore Mining and Feed Schedule

Massawa Central Zone		Year										
		1	2	3	4	5	6	7	8	9	10	LOM
Oxide	'000 tonnes	0	362	902	288	213	211	129	0	0	0	2,105
	Grade (g/t Au)	0	3.50	3.36	2.64	2.90	3.20	2.72	0	0	0	3.18
	Gold koz	0	41	97	24	20	22	11	0	0	0	215
Transitional	'000 tonnes	0	79	172	73	232	111	182	0	0	0	848
	Grade (g/t Au)	0	5.64	4.05	3.87	3.68	3.87	4.09	0	0	0	4
	Gold koz	0	14	22	9	27	14	24	0	0	0	111
Fresh	'000 tonnes	0	136	56	734	895	1,215	1,081	0	0	0	4,118
	Grade (g/t Au)	0	10.48	17.04	9.00	4.41	4.32	4.50	0	0	0	5.60
	Gold koz	0	46	31	213	127	169	156	0	0	0	742

Table 16-4 Massawa NZ LOM Ore Mining and Feed Schedule

Massawa North Zone		Year										
		1	2	3	4	5	6	7	8	9	10	LOM
Oxide	'000 tonnes	0	99	215	226	327	318	65	0	0	0	1,250
	Grade (g/t Au)	0	3.76	2.85	2.58	4.13	5.34	4.06	0	0	0	3.90
	Gold koz	0	12	20	19	43	55	8	0	0	0	157
Transitional	'000 tonnes	0	33	3	55	35	26	46	34	90	173	495
	Grade (g/t Au)	0	3.38	2.68	2.08	2.48	4.67	4.51	4.88	4.88	4.88	4.24
	Gold koz	0	4	0	4	3	4	7	5	14	27	67
Fresh	'000 tonnes	0	0	0	0	0	0	0	1,146	1,055	664	2,864
	Grade (g/t Au)	0	0	0	0	0	0	0	4.69	5.25	6.98	5.43
	Gold koz	0	0	0	0	0	0	0	173	178	149	500

Table 16-5 Sofia LOM Ore Mining and Feed Schedule

Sofia		Year										
		1	2	3	4	5	6	7	8	9	10	LOM
Oxide	'000 tonnes	0	674	342	0	35	50	0	0	0	0	1,101
	Grade (g/t Au)	0	2.32	2.16	0.00	2.15	2.15	0	0	0	0	2.26
	Gold koz	0	50	24	0	2	3	0	0	0	0	80
Transitional	'000 tonnes	0	827	110	0	11	16	0	0	0	0	963
	Grade (g/t Au)	0	2.37	2.03	0.00	2.03	2.03	0	0	0	0	2.32
	Gold koz	0	63	7	0	1	1	0	0	0	0	72
Fresh	'000 tonnes	0	221	529	999	625	374	864	0	0	0	3,612
	Grade (g/t Au)	0	2.29	2.90	3.98	3.05	3.14	3.17	0	0	0	3.28
	Gold koz	0	16	49	128	61	38	88	0	0	0	381

Table 16-6 Delya LOM Ore Mining and Feed Schedule

Delya		Year										
		1	2	3	4	5	6	7	8	9	10	LOM
Oxide	'000 tonnes	0	101	59	0	2	3	0	0	0	0	164
	Grade (g/t Au)	0	4.58	5.23	0.00	5.68	5.68	0	0	0	0	4.84
	Gold koz	0	15	10	0	0	0	0	0	0	0	26
Transitional	'000 tonnes	0	17	178	0	0	0	0	4	12	32	243
	Grade (g/t Au)	0	2.99	4.52	0.00	1.16	0.00	0.00	5.00	5.00	5.00	4.50
	Gold koz	0	2	26	0	0	0	0	1	2	5	35
Fresh	'000 tonnes	0	0	0	0	0	0	0	16	44	121	182
	Grade (g/t Au)	0	0.00	0.00	0.00	0.00	0.00	0.00	5.10	5.10	5.10	5.10
	Gold koz	0	0	0	0	0	0	0	3	7	20	30

Table 16-7 Massawa, Sofia, and Delya Waste Mining Schedule

Deposit		Year										
		1	2	3	4	5	6	7	8	9	10	LOM
Massawa Central Zone	kt	8,046	1,026	6,766	4,342	5,211	4,530	675	0	9	0	30,595
Massawa North Zone	kt	3,039	1,495	2,091	9,605	12,341	11,834	14,577	7,358	4,477	0	66,817
Sofia	kt	8,714	16,142	7,857	2,116	260	56	0	0	0	0	35,144
Delya	kt	378	1,512	2,617	752	89	0	0	0	0	0	5,348
Total	**kt**	**20,176**	**20,175**	**19,331**	**16,814**	**17,901**	**16,419**	**15,253**	**7,358**	**4,477**	**0**	**137,904**

Table 16-8 Open Pit Mining Schedule

Schedule		Year										
		1	2	3	4	5	6	7	8	9	10	LOM
Ore Tonnes Mined - Oxide	kt	2,060	814	375	446	283	255	78	0	0	0	4,311
Ore Tonnes Mined - Transitional	kt	274	942	639	155	208	225	189	0	0	0	2,632
Ore Tonnes Mined - Fresh	kt	29	994	1,805	1,841	1,071	1,415	2,178	1,021	703	0	11,056
Total Ore Grade (g/t Au)		**2.87**	**3.23**	**3.84**	**5.31**	**4.08**	**4.57**	**4.56**	**4.71**	**6.86**	**0.00**	**4.18**
Total Waste Mined	kt	**20,176**	**20,175**	**19,331**	**16,814**	**17,901**	**16,419**	**15,253**	**7,358**	**4,477**	**0**	**137,904**
Total tonnes Mined	kt	**22,538**	**22,925**	**22,149**	**19,256**	**19,463**	**18,315**	**17,698**	**8,379**	**5,180**	**0**	**155,903**
Strip Ratio (Waste/Ore)		8.54	7.34	6.86	6.89	11.46	8.66	6.24	7.21	6.37	0.00	7.66

The grade profile increases as the depth increases due to the plunging nature of the high grade in the NZ. Initial years are dominated by oxide material followed by significant hard rock mining from year 3 onwards (Figure 16-10 and Figure 16-11).

Figure 16-10 Graph of Ore Mining Schedule and Material Type

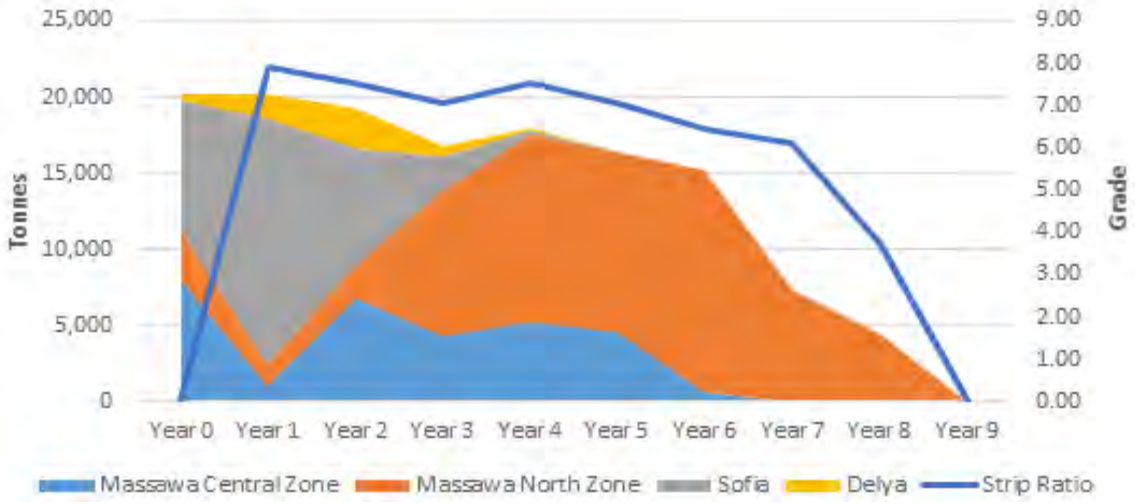


Figure 16-11 Graph of Waste Mining Schedule by Pit

The potential of owner mining is not excluded for consideration at a later date, however, for the purposes of the FS, it has been assumed that a mining contractor will be responsible for the mining at Massawa.

The impact of the owner mining approach is that the open pit operation incurs higher capital costs and reduced operating costs primarily due to the outright purchase of mobile mining equipment and absence of contractor's profit margin and financing cost. Randgold has made a successful transition to owner mining at the Loulo underground operations and therefore has the knowledge and experience to transition to operate owner mining at Massawa, if and when appropriate.

A tender process is in progress for the contract mining of material at Massawa. The first round of the tender process has been completed and four companies have been selected to proceed through to the second and final round. The tenders received have given a good indication of the expected mining cost as well the size of equipment and fleet to be implemented at Massawa.

The contractor mining activities at Massawa will include GC drilling, drill and blast, load and haul, and crusher feeding.

Grade control will be managed through the use of dedicated GC rigs, directed by the Mineral Resource management team using standard procedures from existing Barrick operating mines in West Africa. As part of the feasibility programme, Massawa CZ and NZ, Sofia, and Delya will be drilled to AdvGC drill spacing detailed in the resources section prior to the commencement of mining.

The supply of explosives emulsion and accessories will be provided by the explosives contractor which can deliver service from 'on the bench service' to a complete 'down the hole service'.

The Project objectives are to:

- Extract the ore and waste in an optimal manner.
- Separate the ore based on type and grade.
- Maximise the recovery of the in-pit ore.

- Minimise dilution of the ore.
- Minimise mining costs through improved mining efficiency of waste and ore.

16.2. Open Pit Mining Operating Costs

The Massawa OP mining operating costs have been calculated based on tenders received by selected reputable mining contractors.

All mine activity costs are treated as operating costs.

Crusher feed will be completed by direct tipping and by a Caterpillar 988 front end loader. A re-handle of 100% is envisaged based on the requirement to blend the ore for the optimal feed ratio required by the plant.

The contractor will provide its own mining equipment and therefore no capital expenditure will be required from Massawa.

Mining costs are based on contactor mining quotations as submitted through a mining tender process as shown in Table 16-11, Table 16-12, and Table 16-11. Table 16-12 shows the LOM mining operating costs which total $3.55/t mined.

Table 16-9 Expected Mining Fleet

Item No	Equipment Type	Make	Model	Total Number
Load & Haul Equipment				
	Excavator 01	CAT	6020B	2
	Excavator 02	CAT	6015B	2
	Dump Truck 01	CAT	777	36
	Subtotal			
L&H Support Equipment				
	Rock Breaker	CAT	365	1
	Wheel Dozer			5
	Motor grader	CAT	16M	2
	Water truck	CAT	777D	3
	Subtotal			
Drill and Blast				
	Blast hole Drill 01	Sandvik	DP1500i	8
	Stemming loader	CAT	966	1
	Trailers			1
	Pressure Washer			1
	Subtotal			
Grade Control				
	Grade Control Drill	DRA	GC600	1
	Subtotal			
Crusher Feed				
	Front End Loader 01	CAT	988	2
	Dump Truck 03	CAT	777	Allowed above
	Prime Mover	Scania	R580	3
	Subtotal			
Overhead Support Equipment				
	Light Vehicles	Toyota	Landcruiser	18
	Hiab Truck	MAN	Rigid	1
	Flat Deck	MAN	Rigid	1
	Service Truck	MAN	Rigid	3
	Integrated Tool Carrier	CAT	IT28	1
	Tyre handler			1
	Crane 01	Franna	20t	1
	100t Jack			2
	Lighting Plant			18
	Sleipner	Sleipner	250T	1
	Mobile Welder			2
	Welders			1
	Poly Welder			1
	Light Vehicles Trayback	Toyota	Landcruiser	8
	Subtotal			
Dewatering				
	Pit Pump 01			8

Table 16-10 Massawa Contract Mining Estimated Quantities

Item	Units	Total
Total Material Movement		
Ore	kt	17,999
Waste	kt	137,904
Total	**kt**	**155,903**
Ore Re-handle	kt	17,999

Table 16-11 Massawa Contractor and Owners Costs

Item Description	Total LOM Cost ($ '000)	Unit Cost $/t mined
Establishment of Contractor's Facilities	5,775	0.04
Mobilisation of Contractor's Equipment	3,360	0.02
Demobilisation of Contractor's Equipment	3,175	0.02
Monthly Management Fee	103,874	0.64
Preparatory Works	1,086	0.01
Load and Haul	281,574	1.72
Drilling and Blasting	134,028	0.82
RC Grade Control Drilling	5,301	0.03
Ore Re-handle	18,377	0.11
Dewatering	3,623	0.02
Owners Costs	20,000	0.12
Total	**580,173**	**3.55**

Table 16-12 LOM Mine Operating Costs

Mining Activity	Unit	Cost
Variable	$ '000	463,990
Fixed	$ '000	116,184
Total	**$ '000**	**580,173**
Variable	$/t	2.84
Fixed	$/t	0.71
Total	**$/t**	**3.55**

17. Recovery Methods

17.1. Processing Plant Summary

The proposed WOL and refractory process plant design is based on well-known and established gravity/CIL technology, which consists of crushing, milling, and gravity recovery of free gold followed by leaching/adsorption of gravity tailings, elution, gold smelting, and tailings disposal. The refractory process, which includes sulphide flotation, regrind, and BIOX process, is also a well-known technology that will be supplied by Outotec at a later stage. Services to the process plant will include reagent mixing, storage and distribution, water, and air services.

Ore will be processed in three phases:

- Phase 1 – Oxide ore will be processed through primary crushing (mineral sizer), milling, and gravity recovery, CIL, gold recovery stages (acid wash, elution, electrowinning, regeneration), and detoxification of tails prior to disposal.
- Phase 2 – Fresh (sulphide) ore will be processed through primary (jaw crusher) and secondary (cone crusher) crushing, mill feed bin, milling, and gravity recovery, CIL, gold recovery stages from carbon, and detoxification of solids prior to disposal.
- Phase 3 – Refractory ore will be processed through primary and secondary crushing, mill feed bin, milling, and gravity recovery, flotation, regrind mill, BIOX, CIL, gold recovery stages from carbon, and detoxification of tails prior to disposal.

During Phases 1 and 2, the plant will treat 2.4 Mtpa of oxide, transition, and WOL sulphide ore campaigned through the plant separately or in a combination, if required. For the refractory sulphide ore (Phase 3), the plant feed drops to 1.2 Mtpa. Oxide rock will be crushed through a mineral sizer during Phase 1, whereas during Phases 2 and 3, transition and sulphide (including refractory) rock will be crushed through a primary jaw crusher and stockpiled with bypass allowances directly to the milling circuit for the softer and stickier ores.

Milling will consist of a primary semi-autogenous grinding (SAG) mill, secondary ball mill, and a pebble crusher. The SAG mill will be operated in open circuit in the initial phase of the Project when treating predominantly oxide ores, while the ball mill will be in closed circuit with a hydrocyclone cluster which will be installed when the more competent transition and sulphide ores are mined. Pebbles generated from the SAG mill will be conveyed to a pebble crusher where they will be further reduced in size and re-circulated to the SAG mill, especially when treating competent ore. The discharge from both mills will be combined in a cyclone feed sump and will be pumped to the cyclone cluster for classification.

A proportion of the cyclone feed will be fed to the gravity circuit for recovery of gravity gold, with the cyclone underflow gravitating to the ball mill for further size reduction. Gold will be recovered from the gravity concentrates through a combination of intensive cyanidation and electrowinning facilities. The gravity recovery tailings will be transferred back to the mill feed for further liberation. Gold that is not gravity recoverable is recovered through the CIL process.

Depending on the phase, the overflow from the cyclone cluster will either be pumped to a pre-leach thickener or flotation circuit. The thickener underflow (Phase 1 and 2 material) will feed the seven-stage CIL circuit, where gold will be dissolved and adsorbed onto carbon. The resultant CIL tailings slurry will be subjected to a partial cyanide destruction process, prior to being pumped to the tailings disposal and storage facilities. The refractory (Phase 3) material that was pumped to the flotation circuit is concentrated in a single bank of five roughing stages before being sent for regrinding and fed to the concentrate thickening.

The thickened flotation concentrate is pumped to the BIOX stock tank. The BIOX reactors consist of five equidimensional reactors configured in parallel followed by four secondary reactors operating in series. The feed concentrate from the stock tank is diluted before being fed to the primary BIOX reactors.

The pulp temperature in the reactors is controlled via cooling coils and the excess heat is removed by evaporative cooling towers from the cooling water. Low-pressure air is injected into the BIOX reactors to supply oxygen for the oxidation reactions.

The oxidation of pyrite produces acid, while the oxidation of arsenopyrite and pyrrhotite and the dissolution of carbonate minerals consume acid.

The BIOX product contains high concentrations of dissolved ions and must be washed in a three-stage counter-current decantation (CCD) circuit before being sent to CIL for gold leaching. The CCD wash thickener overflow liquor is neutralised in a two-stage process to produce a stable precipitate containing all the iron and arsenic. The arsenic is precipitated as a stable ferric arsenate and is safe for disposal on a tailings dam.

The process plant design makes allowance for treating any contaminated water discharge from the Return Water Dams (RWD).

Loaded carbon from the CIL circuit will be acid washed prior to elution, followed by reactivation of the eluted carbon. The solution from the elution circuit will be subjected to electrowinning, where gold will be deposited onto cathodes as sludge. Periodically, the sludge will be washed off the cathodes and dried. The dried gold ‘sludge’ will then be smelted to produce gold bullion, which will be shipped to a refinery.

A simplified process flow diagram is shown in Figure 17-1.

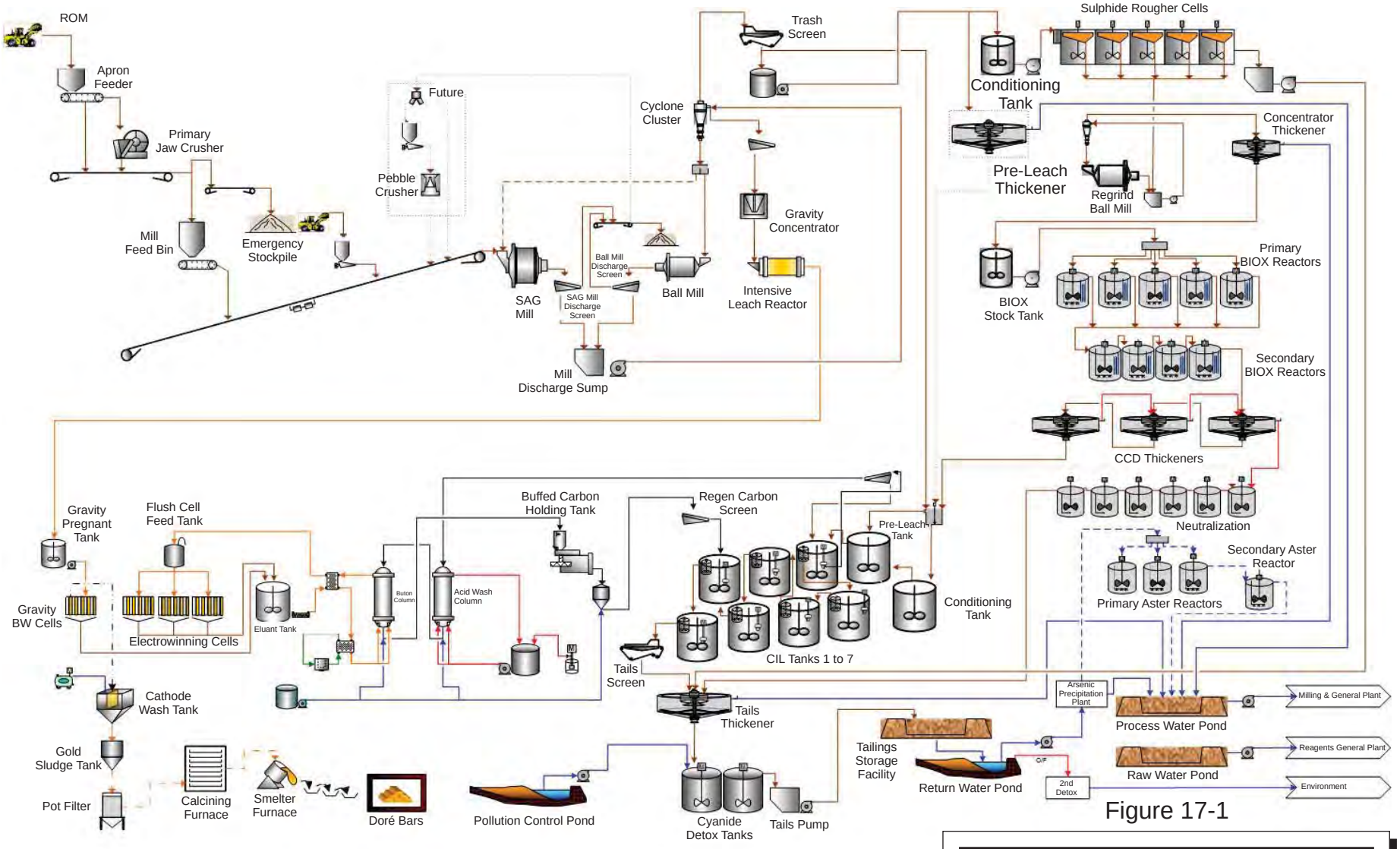


Figure 17-1

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Overall Process Flow Diagram

July 2019

Source: Randgold, 2018.

17.2. Process Plant Design Criteria

The process plant design criteria of the Massawa process facility were devised based on metallurgical testwork results obtained from MMSA, Hazen, SGS, and Outotec, and assumptions, where necessary. The plant design is based on the simultaneous treatment of ore from multiple pits ranging from oxides, fresh WOL, to refractory material over the LOM.

The throughput is based on the mining plan and the process plant will be designed to treat the following tonnage:

- Phase 1 – 2.7 Mtpa of a mixed feed consisting of oxide ore only for Phase 1 of the LOM.
- Phase 2 – 2.4 Mtpa of a mixed feed consisting of predominantly fresh ore blended with oxide transition and sulphide ore for Phase 2 of the LOM.
- Phase 3 – 1.2 Mtpa of a mixed feed consisting of predominantly NZ sulphide ore blended with Delya and CZ ores to lower acid requirements in the BIOX process for Phase 3 of the LOM.

Table 17-1 summarises the principal process design criteria established for the Project.

Table 17-1 Summary of Process Plant Design Criteria

Item	Unit	Oxides	Sofia Fresh	CZ - WOL	CZ - Slightly Refract	Refract Sulphides	Source
Ore Characteristics							
Ore Source		Open cut					Client
Ore type		Oxides	Sofia Fresh	CZ - WOL	CZ - Slightly Refract	Refract Sulphides	Client
Max Lump Size F_{100}	mm	1050					Client
Ore Head Grades:							
Ore Grade - Gold (Ave)	g/t Au	3.40	3.90	4.23	4.23	4.50	Client
Ore Grade - Gold (Design)	g/t Au	3.90	4.30	4.30	4.30	5.20	
Ore Grade - Arsenic	%As	0.11%	<10ppm	0.40%	0.56%	0.55%	Testwork
Ore Grade - Sulphide Sulphur	$\%S^{2-}$	0.02%	0.73%	1.15%	1.10%	1.18%	SGS
Ore Grade - Iron	%Fe	5.8%	5.22%	5.08%	4.90%	5.28%	SGS
Ore Grade - Organic Carbon	%C	<0.05	<0.05	0.063%	0.063%	0.16%	SGS
Mineralogical Characteristics:							
Quartz	%	63.4%	26.3%	41.52%	41.45%	40.55%	SGS
Muscovite	%	TBC	TBC	TBC	TBC	TBC	SGS
Alumina	%	TBC	TBC	TBC	TBC	TBC	SGS
Calcium Carbonate (Calcite)	%	0.72%	15.80%	3.11%	2.36%	1.54%	SGS
Dolomite / Ankerite	%	11.37%	TBC	10.47%	10.70%	13.67%	SGS
Pyrite	%	0.49%	1.73%	2.22%	2.19%	1.62%	SGS
Arsenopyrite	%	0.25%	0.00	0.95%	1.27%	1.12%	SGS
Moisture Content	%	10.0%	5.0%	5.0%	5.0%	5.0%	Client
Specific Gravity of Ore	t/m^3	2.50	2.84	2.75	2.75	2.75	Client
Bulk Density of Crushed Ore	t/m^3	1.50	1.70	1.65	1.65	1.65	Calculated
Angle of Repose	^{0}	35º	35º	35º	35º	35º	SENET
Angle of Surcharge	^{0}	20º	20º	20º	20º	20º	SENET
Angle of Withdrawal	^{0}	65º	65º	65º	65º	65º	SENET
Unconfined Comp Strength	MPa	121	121	121	121	121	Testwork/ OMC
Bond Crusher Work Index	kWh/t	10.5	15.5	30.4	30.4	27.5	Testwork/ OMC
Rod Mill Work Index	kWh/t	13.2	22.8	22.8	22.8	12.3	Testwork/ OMC
Ball Mill Work Index	kWh/t	11.0	18.9	28.4	28.4	18.9	Testwork/ OMC
Abrasion Index		0.100	0.195	0.345	0.345	0.246	Testwork/ OMC
JK Tech Parameters							
A x b		100.00	38.40	27.50	27.50	27.50	Testwork/ OMC

Item	Unit	Oxides	Sofia Fresh	CZ - WOL	CZ - Slightly Refract	Refract Sulphides	Source
Design Margins							
Mill Feed Conveyor	%	20%	SENET				
Ball Mill	%	10%	SENET				
Slurry Pumps	%	25%	SENET				
Gravity (Concentrator, ILR)	%	50%	SENET				
Interstage Screens	%	20%	SENET				
Water Pumps	%	15%	SENET				
Elution	%	20%	SENET				
General Screens	%	50%	SENET				
Operating Schedule							
General							
Annual Tonnage Treated	Mtpa	3.0	2.4	2.4	1.2	1.2	Client
Ore Processing tonnes per month	kt/month	250	200	200	100	100	Calculated
Primary Crushing							
Overall Utilisation	%	91%	69%	69%	69%	69%	Client
Crushing tonnes per hour	t/h	375	396	198	198	198	Calculated
Design Crushing Throughput	t/h	475	SENET				
Secondary Crushing							
Overall Utilisation	%	-	69%	69%	69%	69%	Client
Crushing tonnes per hour	t/h	-	396	396	198	198	Calculated
Design Crushing Throughput	t/h	-	475	SENET			
Milling, CIL & Tailings							
Overall Utilisation	%	91.3%	91.3%	91.3%	91.3%	91.3%	Client
Milling tonnes per hour	t/h	375	300	300	150	150	Calculated
Gold Recovery & Production							
Gold (Au) Recovery							
Au Head Grade	g/t head	3.90	4.30	4.30	4.30	5.20	Client
Gravity Recovery	% of Head Grade	40%	40.0%	60.0%	10%	10%	Testwork
Gravity Dissolution	%	98%	95%	95%	59%	98%	Assumption
Float Overall Recovery	**% of Head Grade**	**-**	**-**	**-**	**91.4%**	**91.2%**	**Testwork**
BIOX CIL Dissolution/Recovery	% of CIL Feed Grade	-	-	-	94.4%	97.0%	Testwork
BIOX CIL Overall Recovery	**% of Head Grade**	**-**	**-**	**-**	**81.0%**	**79.7%**	**Calculated**
WOL CIL Dissolution/Recovery	% of CIL Feed Grade	91.2%	87.7%	79.6%			Testwork
WOL CIL Overall Recovery	**% of Head Grade**	**54.9%**	**53.9%**	**33.8%**			Calculated
Total Overall Recovery (Incl. GRG)	**% of Head Grade**	**94.1%**	**91.8%**	**90.7%**	**86.9%**	**89.5%**	**Calculated**
Gold Production							
Annual Gold Production	oz	353339	304285	143920	143920	179303	Calculated
Annual Gold Production	kg	11007	9478	4483	4483	5585	Calculated

17.2.1. Plant Throughput and Operating Schedule

The plant has been designed for a throughput of 2.4 Mtpa of fresh ore which can increase to 3.0 Mtpa when treating the softer oxides. The average and LOM milling rates were determined for all the composites by using the operating schedule and the relative abundances of the type of ore to be treated as given in the mining schedule. Soft ore crushing will be via a mineral sizer and hard ore via a primary jaw crusher, the overall utilisation is assumed to be 69%, after deducting maintenance hours, and based on operating experience from similar plants. The overall utilisation of the comminution and extraction circuits is assumed to be 91.3%.

17.2.2. Crushing

The design of the crushing circuit is based on one of Barrick's operational plants, Gounkoto. The design also incorporated SENET's experience with previous projects when treating material with similar characteristics, together with recommendations obtained from Orway Metallurgical Consultants (OMC) and CMD.

The initial mine blend will consist of weathered material from various pits. Phase 1 of the LOM will have a very high oxide contribution. The competent ore becomes the majority of the feed during Phase 2. Crushing the oxides, however, may result in ore handling challenges that have to be managed upfront. There will be a single primary crushing station (Mineral sizer) to treat soft ore from both the Sofia Main and North pit with inclusion of the CZ and NZ pits. Both the crushing and milling circuit will be designed to cater for the variation in ore handling required during the LOM.

When treating the softer oxide material (Phase 1), the crusher product will be conveyed directly to the SAG mill feed with no stockpiling, as rat holing may result in a stockpile due to the presence of excessive amounts of fines. Soft rock can also potentially cause material handling problems when processed through a jaw crusher, therefore, a second-hand mineral sizer will be relocated from the Morila Mine in Mali. The soft ore can, at times, contain high quantities of clay and moisture that cause flowability constraints and, therefore, this change is considered prudent. An apron feeder will be installed for the withdrawal of ore from the ROM bin that is also used when treating fresh ore.

The balance of the ore in Phase 2, which will include medium and hard rock types (fresh ore), will be treated through a new jaw crusher (CJ815 Sandvik) to produce a secondary crushing feed.

The single-stage primary jaw crusher (single toggle) is easy to operate and to maintain, and it can withstand heavily abrasive ores when compared to double-toggle jaw crushers. Primary crushed material will discharge on a sacrificial conveyor followed by a longer secondary crusher feed conveyor which feeds directly into the secondary cone crusher. The secondary cone crusher product is conveyed into the mill feed bin with a 90 min live capacity, from which the fine ore will be withdrawn at a controlled rate using the mill feed apron feeder onto the mill feed conveyor feeding the SAG mill.

The mill feed bin ensures a constant feed to the mill. When the mill feed bin is full or if maintenance work is being carried out on the bin, a provision has been made for the crushed ore to be diverted to an emergency stockpile with a minimum of 24 hr emergency storage capacity.

Conveyors will be designed to transport the wet equivalent of the rated dry capacity at the design bulk density.

Dust suppression systems are included in the design as a means of containing the dust produced by the crushing circuit. The dust suppression system uses fine water sprays, which will be used to suppress dust at the main dust-generating points.

17.2.3. Milling and Classification

The milling and classification circuit was designed by considering the following:

- Variability testwork conducted on the ore bodies;
- Rheological characteristics and comminution testwork results;
- OMC Circuit Modelling (Report No. 7860-02 Rev 0 October 2017);

- CMD Circuit Modelling - Second DRAFT report for Massawa 2CSAB 30 May 2018.

The initial comminution modelling that OMC completed focussed on the best flow sheet selection for the three distinct processing phases (oxide (Phase 1), fresh WOL (Phase 2), and fresh refractory (Phase 3) material) that was identified in the PFS phase. OMC was requested to consider two mill sizes, one sized as new and one sized identical to that which is in operation at numerous Barrick sites in West Africa. The mill sizes are 5.5 m by 8.0 m EGL, 4,500 kW installed and 6.1 m by 9.5 m EGL, 7,000 kW installed, both in overflow configuration.

Table 17-2 summarises the selected mills and phases simulated after the first round of modelling.

Table 17-2 Mill Simulation Summary

Parameter	Units	Phase 1	Phase 2	Phase 3
Throughput	Mtpa	3	2.4	1.2
Material		Oxide Only	Fresh (CZ) WOL	Fresh (NZ refractory)
Crushing Equipment		1 x Primary Jaw	1 x Primary, 1 x Secondary Cone	1 x Primary, 1 x Secondary Cone
Comminution Equipment		1x6.1 m ø x 9.5 m EGL Scrubber (4.5 MW), 1x6.1 m ø by 9.5 m EGL Ball Mill (8.0 MW)	1x6.1 m ø x 9.5 m EGL Scrubber (4.5 MW), 1x6.1 m ø by 9.5 m EGL Ball Mill (8.0 MW)	1x5.5 m ø by 5.5 m EGL Ball Mill (4.5 MW)

Comminution circuit modelling was completed using client supplied information that higher than normal power draws may be achieved with the nominated mill sizes (6.1 m Ø by 9.5 m EGL ball mills), which is in line with observations at Tongon after modifications. Table 17-3 summarises the phase and associated mills that were evaluated.

Table 17-3 Mill Duty Achievable Commentary

Phases	Duty Achievable/Commentary
1	Yes
1 - Alternative	Yes - where scrubber is omitted and only a single stage ball mill is considered
2	No - ball mill limited. Duty achievable by sending a portion of primary cyclone underflow back to the primary mill
3	0.6 Mtpa - Yes, 1.2 Mtpa - No – A larger mill will be required

Power modelling of the circuit furthermore indicated that the circuit will be secondary ball limited. To balance the circuit power draw, recycling a portion of the primary mill cyclone underflow to primary ball mill feed will be required.

The Phase 3 evaluation indicated that the nominated 5.5 m Ø by 5.5 m EGL ball mill with 4.5 MW motor is suitable to achieve the lower 0.6 Mtpa throughput duty but not the planned 1.2 Mtpa target. It was found that a 6.1 m Ø by 6.1 m EGL grate discharge ball mill with a 5 MW motor, operating in a closed-circuit configuration, is suitable.

To further assist the development of the new comminution circuit, another consultant, CMD, was approached to evaluate the now dormant Morila SAG and ball mills in terms of being refurbished and made available for the Project.

Two circuit configurations using mill sizes that are common within the Barrick mines in West Africa were evaluated. These are:

OPTION 1a

- A 300 tph, 2C/PM/BM circuit configuration with primary and closed-circuit secondary crushing followed by a new low aspect ratio primary mill (20 ft by 30.7 ft EGL) and ball mill in series (similar to the Loulo circuit). Common pebble/secondary crushing.

OPTION 1b

- A 300 tph, 2C/PM/BM circuit configuration with primary and open circuit secondary crushing followed by a new low aspect ratio primary mill (20 ft by 30.7 ft EGL) and ball mill in series (similar to the Loulo circuit).

OPTION 1c

- A 150 tph, 2CB (or 2C/SS SAG +PC) circuit configuration with primary and closed-circuit secondary crushing followed by a two trains of low aspect ratio ThyssenKrupp (20 ft by 30.7 ft EGL) mills in parallel with common pebble crushing (similar to the Kibali sulphide circuit with secondary crushing and coarse feed to the overflow discharge ball mills).

OPTION 1d

- A 150 tph, 3CB+scatting PC circuit configuration with primary and open circuit secondary crushing followed closed circuit tertiary crushing and a low aspect ratio ThyssenKrupp (20 ft by 30.7 ft EGL) mills (similar to the Tongon sulphide circuit with quaternary crushing and fine feed to the grate discharge ball mills).

OPTION 2

- A 300 tph, 2C/SAB(C) circuit configuration with primary and open circuit secondary crushing followed by a relocated and refurbished Morila SAG mill (26 by 18 ft EGL) and followed by the Morila ball mill (20 ft by 29.5 ft EGL). HPGR was also considered in the flowsheet as the SAG pebble crusher (2C/SABC) (similar to Morila/Geita circuit configurations).

The analysis of the available data within the context of the above-mentioned flow sheet options, CMD confirmed that either option will meet the 300 tph throughput rate during year 2025 which appears to be the most difficult year in terms of ore blending and processing of hard ore.

CMD also concluded that either of the 300 tph circuits will work, however, CMD indicated that Option 1b with a higher capital cost is the lower risk option and that Option 2 would require a proper investigation to quantify the extent of refurbishment that is required. This exercise was already completed by the Original Equipment Manufacturer (OEM) including another suitable Vendor to validate the intended refurbishment costs.

Initially, the circuit could cope without the need for a pebble crusher, however, CMD recommended that a pebble crusher be designed so that it can be installed at a later date if required. The idea of using an HPGR as a pebble crusher is not warranted due to the very low pebble rates that would be expected.

CMD also recommended that the ore blends and ore characteristics should be measured, and a forecast model developed that will enable appropriate design changes to the ball charge, ball

size distribution, and grate design to match the operating conditions over six monthly or quarterly intervals.

The proposed comminution circuit specifications for the mills are as follows:

Morila SAG Mill 26 by 18 ft EGL (19 ft F/F):

- The mill is 8.0 m in diameter (7.8 m inside liner) by 5.9 m long F/F (5.6 m EGL, measured effective grinding length).
- Feed trunnion diameter is 1.84 m, feed and discharge cone ends at 0°.
- Effective discharge cone end has zero degrees as it is a flat ended grate design that defines the volume of the grinding chamber within the mill.
- Current grate open area (OA) configuration is 12 segments with 15 mm by 25 mm slots. The calculated OA is 8.0%.
- Variable speed with 60% to 85% of critical.
- Pinion driven mill is fitted with a 6,000 kW motor.
- Ball size to initially be a blend of 105 mm and 70 mm.
- Ball filling range 9% to 12% for the harder ore types (year 2025).
- Total mill filling with this ore density to range from 22% to 32% filling average 25%.

Morila Ball Mill 20 by 29.5 ft EGL:

- The mill is 6.1 m in diameter (5.9 m inside liners) by 9.11 m long EGL. Ball mill is an overflow mill as is the case at Morila.
- Feed trunnion diameter is 1.4 m, feed and discharge cone ends at 0°.
- Effective discharge cone end has zero degrees as it is a flat ended grate design that defines the volume of the grinding chamber within the mill.
- Driven at a fixed speed of 75% of critical by a 6,000 kW motor.
- Ball size to initially be a blend of 70 mm and 50 mm.
- Ball filling 38% max for the harder ore types (year 2025) draws 6463 kW.
- Ball mill is too small for 2C/SABC circuit, unless the Morila mill motor is upgraded to 7 MW. Total mill filling with this ore density to range from 36% to 38%.

A variable speed drive is installed on the SAG mill to cater for variations in the hard and soft ore characteristics during the LOM. The SAG milling circuit, particularly when operating in a single-stage configuration, requires a reasonably consistent feed blend to ensure stable operation.

The installed pebble crusher will operate when needed and is required for hard ore handling. The pebble crusher will assist in returning the recycled pebbles that do not readily break down during SAG milling. This will result in improved power usage and throughput.

Calculations have demonstrated that CZ ore is the hardest, however, the Morila mills are capable of the task and design throughputs, although it is prudent for CZ ore to be fed as a blend with other ore types, with CZ making up a maximum of the order of 75% of the feed.

Classification of the milled product will be through hydrocyclones. A cyclone cluster will be designed, from which a 25% solids cyclone overflow product will be targeted to be fed to a pre-leach thickener from Phase 2 onwards. Based on the comminution circuit modelling exercise, an allowance has been made in the design for recycling the cyclone underflow to the SAG mill, required for certain periods during the LOM.

A linear trash screen will be included in the design prior to leach, i.e. to remove tramp material. The trash screen will be configured such that the undersize from the trash screen will gravitate to a sump and will be pumped to the pre-leach thickener from Phase 2 onwards and directly to the leach tank train for oxides.

17.2.4. Gravity Concentration

The results of the GRG batch laboratory tests indicated that free gold is available in all the ore types in significant quantities. In addition, intensive cyanidation of the resultant gravity concentrates from most of the ore types showed favourable leach kinetics and gold extraction. As a result, a gravity circuit consisting of a centrifugal concentrator, intensive cyanidation reactor, and a dedicated electrowinning cell for handling the pregnant leach solution will be designed to recover free gold from a portion of the cyclone underflow. This circuit will improve the overall gold recovery and reduce the residence time in the CIL circuit. Simulation results from the GRG tests and the concentrate leach extraction parameters obtained from the tests were used as the basis of the design.

17.2.5. Trash Removal and Pre-Leach Thickening

The variable operation of the milling circuit due to the varying competency of the materials being treated will require the classification circuit to cater for a range of overflow densities expected during operation as mill re-circulating loads and feed water dilution are adjusted. To stabilise the control of the SAG and ball milling circuit when treating the more competent ore, a dilute overflow solids density from the cyclone cluster (25% to 30%) will be required during operation. This is much lower than the optimum CIL tank solids density (40%). Pre-leach thickening will, therefore, be required to densify the milled product to the optimum solids concentration required in the CIL tanks. The basis of design for the 35 m pre-leach thickener and associated flocculant make-up system includes settling rates obtained from thickening and rheology testwork conducted by specialist thickening consultants Vietti Slurrytec (2018).

17.2.6. Flotation and Concentrate Production

<u>Rougher Flotation</u>

The flotation design is based on results obtained from three campaigns, 22 and 19/20, including testwork on NZ and, later, CZ material. The testwork results indicated that a much longer residence time was required for NZ material than the faster floating CZ material. A single bank of rougher flotation cells was chosen because of the initial low mass pull and high recoveries achieved; the minimum number of cells required was four, and five cells were chosen for the design. The cell sizes were based on NZ material with a residence time of 120 minutes.

The concentrates from the different cells were combined to form the final concentrate product, although allowance have been made to combine the tail end rougher concentrates for possible recycling at a later stage when treating the higher mass pull CZ material.

Concentrate Regrind Circuit

The regrind circuit design was based on a concentrate mass pull of 20% and an estimate Bond Ball Mill Work Index of 28.4 kWh/t.

Concentrate Thickening

The basis of design for the 11 m concentrate thickener and associated flocculant make-up system includes settling rates obtained from thickening and rheology testwork conducted by specialist thickening consultants Vietti Slurrytec (2017).

17.2.7. BIOX

Primary and Secondary Reactors

The proposed flow sheet for the BIOX plant includes the treatment of a blend of Massawa NZ and refractory CZ sulphide concentrate at an ore feed rate of 1.2 Mtpa to the plant. The BIOX plant will also be able to treat concentrates generated from various ratios of NZ, CZ, and Delya ores as listed in Table 17-4.

Table 17-4 Massawa Process Plant Ore Feed Scenarios

Parameter	Design	Scenario 1	Scenario 2	Scenario 3	Scenario 4
	NZ-CZ blend	NZ only	NZ-CZ blend	CZ only	NZ-CZ blend
Ore Feed Tonnage (tpa)	1 200 000	1 200 000	1 200 000	830 000	1 200 000
NZ ore in feed mix (%)	75	100	75	0	76
CZ ore in feed mix (%)	25	0	25	0	0
Delya ore in feed mix (%)	0	0	0	100	24

The main considerations for developing the flow sheet for the BIOX plant were to consistently achieve high levels of sulphide sulphur oxidation in the most cost-efficient way. This will be achieved by:

- Optimally configuring the BIOX reactors, internals, and utilities.
- Minimising limestone requirement for BIOX pH control through concentrate blend optimisation.
- Minimising limestone requirement for neutralisation pH control through the optimal use of available carbonates in the flotation tailings.
- Minimising gold losses over the CCD circuit.
- Minimising gravity gold lock-up in the BIOX reactors.
- Ensuring the disposal of environmentally acceptable process effluent streams on a constant basis/
- Using ASTER to allow for the recycling of cyanide contaminated tailings water.

Based on the current design criteria, the BIOX circuit will consist of one BIOX module, configured as five primary reactors operating in parallel followed by four secondary reactors operating in series.

CCD

The BIOX product will be washed in a series of three CCD thickeners to remove Fe(T), As(T), and sulphate solubilised during bio-oxidation. Test results indicated that the settling rate of the BIOX product is enhanced by the dilution of the feed slurry to a targeted 8% solids. It is therefore recommended that the CCD thickeners be equipped with auto-dilution systems to maintain a low slurry density in the feed wells.

The BIOX liquor will report to the neutralisation circuit and the solids will report to the CIL circuit.

Neutralisation

Flotation tailings slurry, limestone, and lime will be used for neutralising the BIOX liquor. The solution will flow through a series of six neutralisation tanks. Testwork has shown that the use of flotation tailings as a neutralising agent can reduce neutralisation reagent consumptions by as much as 62%, depending on the percentage of available flotation tailings slurry used. Thickened flotation tailings will be added to the first, second, and/or third neutralisation tank.

Limestone slurry will be added to the second and/or third neutralisation tank to raise the pH to 4.5. Milk of lime slurry will be added to the fifth and/or sixth neutralisation tank to raise the pH to 7.

Aster Circuit

When water from the Phase 3 TSF is to be returned to the plant, a secondary cyanide detoxification treatment using technologies such as the ASTER process will be required for the removal of thiocyanate thus enabling the water to be recycled upstream of the BIOX circuit.

It is important to ensure that no cyanide contaminated water ever reaches the BIOX plant since the toxic effect of cyanide on the bacteria used in the BIOX process will be severe. The detoxified water returning to the process plant via the ASTER circuit should not exceed the following maximum residual concentrations:

- SCN- < 1 mg/L.
- CNO < 2.5 mg/L.
- Metal cyanides < 5 mg/L.

BIOX Services

The correct distribution and supply of air is critical for optimum BIOX performance and to ensure that the required sulphide oxidation is achieved in each reactor. The design flow rate for the blowers is based on 10,709 Nm^3/hr per primary reactor and 6,622 Nm^3/hr per secondary reactor and another 4,236 Nm^3/hr for the neutralisation circuit.

The cooling circuit is designed to remove the heat produced by the exothermic sulphide oxidation reactions. The upper-temperature limit at which the BIOX culture can operate is approximately 45°C, and the slurry temperatures should not be allowed to exceed this temperature for extended periods. Also, it is critical to maintain the reactor operating temperatures within a narrow range in order for a robust microbial culture to be developed and sustained. The slurry temperature in the BIOX reactors will be maintained at 42°C by cooling water circulating through the cooling coils. The cooling tower flowrate was designed for 2,133 m^3/hr based on the maximum heat load that would need to be catered for.

17.2.8. Carbon in Leach

Gold dissolution and adsorption on the milled ROM and BIOX products (CCD underflow) will be done via a seven-stage CIL circuit. Conditioning and pre-leach tanks are included before carbon is in contact with the slurry.

The design will allow for space to include additional CIL tanks if needed during the Project. Cyanidation tests conducted on all the composites indicate reasonable gold dissolution within a leaching time of 24 hours to 32 hours on gravity middlings and tailings of most of the oxide ores under normal CIL conditions. The leaching residence time for the Phase 3 (refractory ores) requires extended hours of up to 42 which is offset by the lower design throughput. The leach circuit will be designed to treat feed from the thickener underflow at a solids concentration of 40% for Phase 2 material and 30% for the Phase 3 refractory material. Tests were conducted at natural oxygen levels and the introduction of oxygen will potentially enhance leach kinetics on the fresh material. In addition, oxygen uptake rate tests have indicated that aeration is sufficient to meet the demand for oxide treatment only, but that oxygen sparging will be required when treating non-refractory sulphides. The design, therefore, allows for oxygen sparging via two Aachen reactors into the conditioning tank and normal sparging into the pre-leach and first seven CIL tanks from Phase 1 onwards. A carbon addition of 10 g/L to 12 g/L will be sufficient for the gold adsorption in the CIL circuit.

The SENET internal CIL modelling confirmed the tank volume of 2,500 m^3 each, as well as the carbon to gold loading ratios (1,000), are achievable. These results were then used to confirm the selected daily elution capacity based on the expected carbon movement per day. The modelling indicated that adequate gold adsorption could be achieved across the seven stages for all the Massawa composites tested. The resulting carbon movement per day was confirmed to be not larger than 12 t.

During Phase 3 the BIOX slurry will be subjected to two stages of pH correction prior to the CIL. The pH of the thickened BIOX slurry will first be raised to pH 5, in order to help stabilise and precipitate any residual arsenic in solution and then raised to a pH to 10.5 to make it acceptable for cyanidation.

The flow from one CIL tank to another will be through intertank launders, and all tanks will have a bypass facility to ensure continuity in production during periods when a tank is taken offline for maintenance.

The design of the CIL area will also incorporate a tower crane on rails, which will be used during the construction and operating phases. During the operating phase, it will be used to facilitate cleaning of the interstage screens and general maintenance.

The results from optimised extraction and rheological tests were used as the basis for the CIL design. The feed density to the CIL tanks will be designed at 40% solids from Phase 2 onwards to cater for flow through the interstage screens. During the oxide ore phase, which tends to be associated with higher viscosities, the feed density will be lower being directly from the overflow of the classification cyclones.

Interstage Screens

The interstage screens will be designed to accommodate the flow that can be expected to arise when treating the ores in the CIL tanks. Mineral processing separating pumping (MPS (P)) screens were considered for the design as these would maintain the same pulp level in the CIL tanks, as the pumping action of these screens would allow pulp movement from one tank to the another while carbon is retained in tanks.

Loaded Carbon Screen

Loaded carbon from the first CIL tank will be pumped onto a screen, where the screen oversize (washed loaded carbon) will gravitate to the acid wash cone, and the undersize (slurry) will report to the first CIL tank. A linear screen will be used for this function.

17.2.9. Carbon Safety and Tails Thickening

A linear screen will be incorporated into the design to recover any fugitive carbon.

The basis of design for the 35 m tails thickener and associated flocculant make-up system includes settling rates obtained from thickening and rheology testwork conducted by specialist thickening consultants Vietti Slurrytec (2018).

17.2.10. Cyanide Detoxification

The results of the batch and continuous laboratory tests indicated that cyanide destruction could be effected by using SMBS and copper sulphate as catalysts for a period of one hour, reducing the WAD cyanide in the final tailings to less than 50 ppm. The cyanide detoxification process will thus be designed as two-stage agitated reactors into which compressed air will be blown.

Using these results, the cyanide detoxification facility in the process plant was designed to achieve a nominal WAD cyanide concentration limit of 50 ppm or less in the final tailings to render the slurry suitable to be stored in a lined tailings facility according to environmental guidelines. If there is a need to detoxify the slurry to a lower WAD cyanide concentration, then additional SMBS, copper sulphate, and lime will be required. The design of the reagents storage area, therefore, makes provision for the storage of enough reagents to detoxify the tailings slurry to a lower discharge concentration of 50 ppm WAD cyanide.

Allowance has been made for a secondary emergency cyanide detoxification facility at the TSF to cater for the detoxification of possible higher than above international standard residual cyanide present in the discharge water to the environment.

17.2.11. Arsenic Precipitation

The Arsenic Precipitation circuit will receive environmental release quality cyanide solution and remove the arsenic in a two-stage precipitation stage. The Arsenic Precipitation plant will be a vendor-supplied plant. The design of this facility is based on preliminary testwork, which will need to be validated by further testwork during the detailed design phase.

17.2.12. Acid Wash

A cold acid wash circuit capable of handling the full 12 t loaded carbon batch will be included to remove any carbonates that might otherwise foul the carbon. The circuit will be designed to acid wash every batch of carbon before it is eluted by circulating 3% hydrochloric acid through the carbon in the acid wash column. The acid wash cone will be designed with an overflow weir to facilitate elutriation of the loaded carbon prior to the acid wash step. The elutriation process will remove trash from the carbon (such as wood chips and plastics), which might otherwise interfere with the flow through the strainers during elution.

17.2.13. Elution

Based on the carbon loadings and the amount of gold to be produced per month, the number of elutions has been estimated to be 54 per month and a pressure ZADRA elution method was selected to strip gold from the loaded carbon in 12 t carbon batches. Heating of the eluant will be achieved through diesel-fired thermic oil heaters.

17.2.14. Electrowinning

Electrowinning will be affected through sludging-type electrowinning cells. Pregnant solution exiting the column will be directed to three double electrowinning cells, operating in parallel for CIL, via a flash/header tank where gold will be deposited on the cathodes as sludge and the barren solution will be circulated back to the elution tank. A dedicated header tank and electrowinning cell will be used to recover gold from the gravity pregnant solution. Electrowinning will be affected through sludging-type electrowinning cells.

17.2.15. Reactivation

A carbon reactivation facility will be designed to treat the entire eluted carbon batch in a period of 11 h. The reactivation kiln will be diesel fired.

17.2.16. Calcining and Smelting

Fully loaded cathodes will be periodically removed from the cells, and the gold sludge will be washed off using a high-pressure washer and the solution decanted. The gold sludge will be calcined (dried) in an electric calcination furnace. The calcined sludge will then be mixed with

fluxes and loaded into an induction smelting furnace. During smelting, metal oxides will form slag and, once the furnace crucible contents are poured into cascading moulds, gold will solidify at the bottom while slag separates easily from the gold. The gold bullion bar(s) will be cleaned, labelled, assayed, and readied for shipping.

17.2.17. Reagents

Facilities to mix, store, and distribute reagents and consumables will be allowed for in the design. These reagents and consumables will include grinding media, cyanide, caustic, lime, limestone, flocculant, SMBS, copper sulphate, potassium amyl xanthate (PAX), frother, collector, hydrogen peroxide, nutrients, water treatment reagents, diesel (for plant use only), hydrochloric acid, and smelting fluxes. The reagent consumptions obtained during the testwork for each oxide and fresh composite were used to estimate the size of the equipment associated with mixing, storage, and distribution. The storage area will allow for minimum of three-month stock holding capacity for grinding media, carbon, caustic, flocculant, copper sulphate, PAX, frother, collector, hydrogen peroxide, hydrochloric acid, and smelting fluxes.

Table 17-5 shows the lime and sodium cyanide consumptions based on optimal conditions as determined from testwork.

Table 17-5 CIL Sodium Cyanide and Lime Consumptions per Ore Type

WOL 2.7, 2.4 and BIOX 1.2 Mtpa	Central Zone			North Zone		Sofia Main	Sofia North	Delya		
	WOL	BIOX - 100% CZ	BIOX - 25% CZ / 75% NZ	WOL 100% NZ	BIOX 100% NZ	WOL	WOL	WOL 100% Delya	BIOX 100% Delya	BIOX 20% Delya 80% NZ
Oxide NaCN kg/t	0.49	na	na	0.40		0.45	0.72	0.41	na	na
Oxide Lime CaO kg/t	0.99	na	na	1.13		0.99	1.74	1.35	na	na
Oxide Trans NaCN kg/t	0.77	na	na	0.62		0.62	0.67	0.60	na	na
Oxide Trans CaO kg/t	1.21	na	na	1.72		0.99	0.93	1.62	na	na
Reduced Trans NaCN kg/t	0.90	na	na	0.90		0.64	0.64	0.69	na	na
Reduced Trans CaO kg/t	1.31	na	na	1.31		0.52	0.52	1.76	na	na
Fresh NaCN kg/t ore (WOL), kg/tBIOX product (BIOX)	1.04	17.50	17.50	0.84	20.00	0.79	0.61	0.79	20.00	20.00
Fresh CaO kg/t	1.42	13.18	13.18	2.30	13.18	0.99	0.11	1.90	13.18	13.18

17.2.18. Plant Diesel

Plant diesel will be distributed from the storage vessel (20 m^3) to elution and regeneration where there will be a header tank (5 m^3) for general plant use.

17.2.19. Smelting Fluxes

The consumptions of smelting fluxes: borax, sodium carbonate, and silica were estimated by assuming a flux ratio of 50%, 25%, and 25%, respectively, to the weight of calcine, as per standard industry practice.

17.2.20. Grinding Media

The SAG and ball mill grinding media consumptions were based on Bond's estimating method using abrasion index results obtained from laboratory tests. The consumptions for grinding media were estimated by SENET.

17.2.21. Mill Liners

The SAG and ball mill liner consumption was based on Bond's estimating method using abrasion index results obtained from laboratory tests. Consumptions for liners were estimated by SENET.

17.2.22. Jaw and Pebble Crusher Liners

The estimated number of liner changes per annum was determined using the abrasion indices obtained from laboratory tests and the expected liner life for the individual composites. The expected liner life was supplied by Sandvik.

17.2.23. Air and Oxygen Services

Oxygen uptake results obtained through testwork for the CIL circuit will be used as the basis for the sizing of the pressure swing adsorption (PSA) unit oxygen plant. Two 10 ton units are allowed for in the design with the possibility to increase to three may the need arise.

The plant air high-pressure compressors, one duty and one standby, supply the plant air required (400 Nm^3/hr) for the lime plant, workshops and for plant-wide general use.

Air requirements for the detoxification process needs will be supplied by low-pressure blowers (operating and standby) with design based on Phase 3 air requirements (2,748 Nm^3/hr).

Instrument air requirements (250 Nm^3/hr) for the entire plant will be supplied by a dedicated compressor and air dryer.

The flotation air blowers (Phase 3), one duty and one standby, will supply the low-pressure air required for flotation based on 1,200 Nm^3/hr/cell.

17.2.24. Water Services

The process water and raw water ponds were sized based on accommodating a 24 hr residence time in each.

Two pontoon pumps will be installed in the diversion dam to enable pumping of water to the process and raw water ponds located in the plant. These pumps will be sized to cater for commissioning, and dry and wet seasons, where raw water demands vary significantly. During the wet season, raw water supply will be supplemented from the RWD. A water balance for these three scenarios has been used as the basis to size the transfer pumps, the raw and process water ponds, the gland water tank, and all the water pumps including a fire water facility.

The sizes of the raw and process water ponds are shown in Table 17-6 and Table 17-7, respectively. The size for the process water pond was based on the average water requirements during Phase 1 (Oxides), for the raw water, it was decided to design a similar size as the process water pond. This would cater for maximum requirement during commissioning and ample supply during Phase 3.

Table 17-6 Raw Water Pond

Raw Water	Unit	WOL (P1/2)	Refractory (P3)	Maximum (Commissioning)
Type of Reservoir		Pond	Pond	Pond
Process Water Top-Up	m^3/h	458	685	757
Raw Water Demand	m^3/h	123	153	123
Total	m^3/h	581	675	881
Residence Time	hr	24	24	24
Raw Water Reservoir Volume	m^3	13,954	16,203	21,139
Selected Raw Water Storage	m^3	20,000		

Table 17-7 Process Water Pond

Process Water	Unit	Average
Type of Reservoir		Pond
Source of Water		Return Water and Raw Water Top-Up
Total Process Water – Phase 1	m^3/h	852
Residence Time	h	24.00
Process Reservoir Volume	m^3	20,451
Selected Process Water Storage	m^3	20,000

17.2.25. External Water Treatment Plants

Arsenic Precipitation Plant

According to the mining plan, the oxides from either pit will be the predominant feed to the plant during the first two years. Thereafter, the CZ sulphides will be blended in significant quantities, resulting in higher concentrations of arsenic in the plant tails reporting to the TSF. Consequently, the resulting water in the RWD will contain arsenic contaminant.

The excess water contained in the RWD will discharge into the environment during the wet season. The water balance modelling exercise conducted by the environmental team (Digby Wells) has stated that arsenic concentrations in the discharge from the RWD will be above the IFC guideline limitation during the course of the Project. This is supported by the geochemical testwork conducted on the tailings material by Digby Wells. The water will be contaminated with arsenic and antimony when the CZ sulphide ore is treated in significant quantities after the first two years.

The treatment plant will be installed to operate from Phase 2 when CZ material is fed and will have the capability for arsenic and antimony removal to render the discharge from the RWD environmentally compliant. The treatment plant is a vendor-packaged plant provided complete by the selected equipment supplier.

Clarified effluent is collected in the Stage 1 rapid mixing tank 4.5 m (L) by 4.5 m (W) and dosed with ferric chloride. The rapid mixing tank will provide for the rapid mixing of the chemicals before the water overflows to a concrete flocculation tank. The large flocculated clusters produced will settle out in the Stage 1 two lamella clarifiers 12 m (L) by 5 m (W).

Stage 1 clarified effluent overflows into the Stage 2 rapid mixing tank and follows the same treatment process with rapid mixing, flocculation, and settling in final two lamella clarifiers. The collected precipitate can either be pumped back to the TSF or solids collected in an impermeable lined pond or containers.

Detailed testwork will be required during the detailed design phase to confirm vendor selection, plant sizing, as well as the interfacing with the process plant.

At this level of study, the arsenic precipitation plant was sized based on a maximum yearly discharge volume of 770,268 m^3 supplied by Digby Wells; see Table 17-8 for design parameters used.

Table 17-8 Arsenic Removal Design Parameters

Parameter	Unit	Value	Source
RWD Excess Water Treatment Rate	m^3/yr	770,268	Digby Wells
Operating Months	months/a	4-5	Digby Wells
Feed Arsenic Concentration	mg/L	14	Digby Wells
Product Arsenic Concentration	mg/L	0.1	IFC Guideline

Secondary Detoxification Plant

The secondary detoxification plant will be installed and operable from the first year of the Project and will have the capability of secondary cyanide detoxification to render the discharge from the RWD environmentally compliant. The secondary detoxification plant has, therefore, been included as a solution to managing the excess water from the RWD. The design was based on a maximum yearly discharge volume of 770,268 m^3 supplied by Digby Wells (Table 17-9).

Table 17-9 Secondary Cyanide Removal Design Parameters

Parameter	Unit	Value	Source
RWD Excess Water Treatment Rate	m^3/yr	770,268	Digby Wells
Operating Months	months/a	4-5	Digby Wells
Feed Cyanide Concentration	mg/L	2	Digby Wells
Product Cyanide Concentration	mg/L	0.1	IFC Guideline

17.3. Block Flow Diagram

The block flow diagrams of the WOL and refractory process plants as shown in Figure 17-2 and Figure 17-3, respectively, provide an outline of the unit operations in the plant.

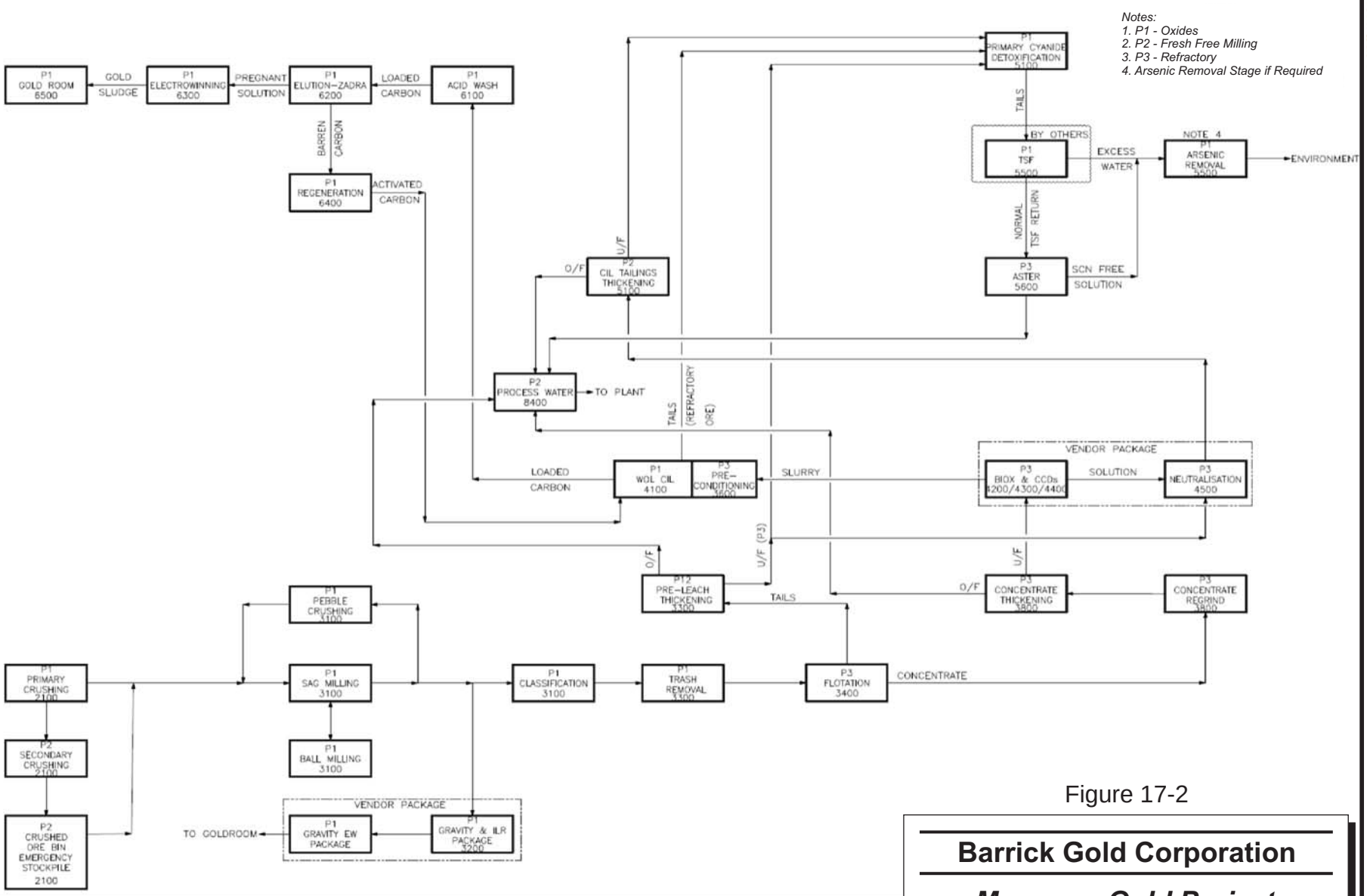


Figure 17-2

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Whole Ore Leach Process Plant Block Flow Diagram

July 2019

Source: Randgold, 2018.

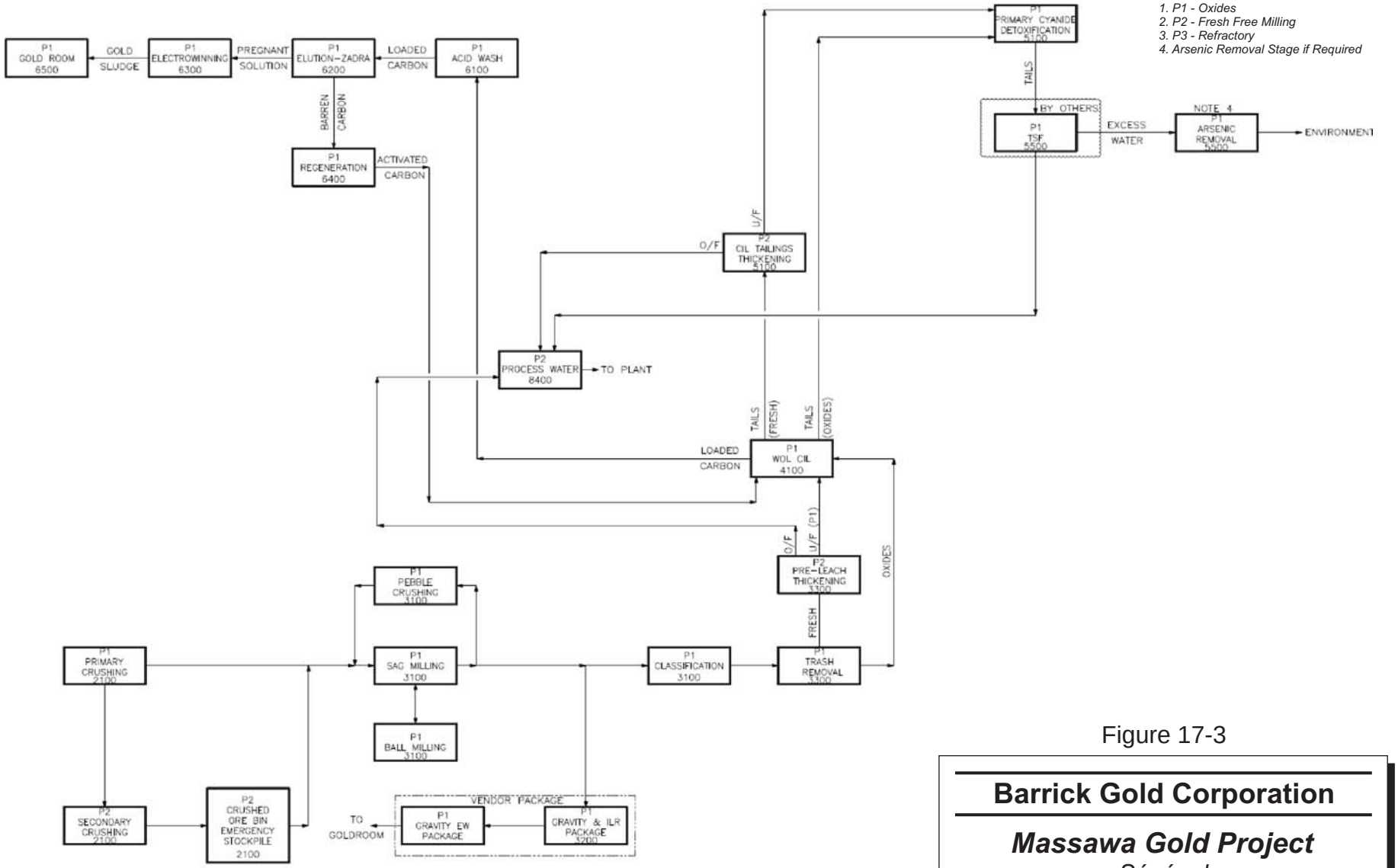


Figure 17-3

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Refractory Process Plant Block Flow Diagram

July 2019

Source: Randgold, 2018.

17.4. Tailings Storage Facility and Diversion Dam

17.4.1. Introduction

As part of the FS, Epoch Resources (Pty) Ltd (Epoch) undertook the FS design of the TSF and associated Diversion Dam System (DD), as summarised in this section. The associated capital costs were estimated to a Class 3 AACE accuracy (+25%, -15%) and the TSF operational costs, to an accuracy of ±25%.

The design of the TSF comprises:

- An HDPE lined self-raising Tailings Dam (TD).
- A HDPE lined RWD associated with the TD.
- A DD.
- The storm water management and associated infrastructure for the facilities as mentioned above (i.e. perimeter slurry deposition pipeline, storm water diversion trenches, perimeter access road, solution trenches, catchment paddocks etc).

The design criteria associated with the Massawa TSF and DD are summarised in Table 17-10. Two types of mineralisation will be mined, namely oxide and sulphide, with gold being extracted by three distinctly different metallurgical processes. The oxides and certain of the sulphides will be treated by means of the Whole Ore Leach process, while other sulphides will be treated by means of the Float Tails process. A small portion of the ore will be treated by means of the BIOX process towards the later stages of the mine life.

In addition, the following legislation, regulations, and design standards have been taken into account during the FS design of the TSF and DD:

- International Cyanide Code Standard of Practice;
- International Financial Corporation Guidelines; and
- Since Senegal does not have legislation pertaining specifically to the design of TSFs, ‘Appropriate Best Practice Measures’ have been adopted.

Table 17-10 Design Criteria for the Massawa TSF and DD

Item	Design Criteria	Value	Source/Comment
1	Tailings material	Gold	Massawa Specific
2	Design life of facility	10 years	Randgold
3	Ore body composition	The tailings product produced at Massawa consists of two materials namely oxides, originating from the CZ and NZ pits located in the SE of the perimeter, and sulphides, originating from the main Sofia ore body located in the north-west of the perimeter	
3	Tailings deposition rate	Year 1 - 2.39 Mtpa (Whole Ore Leach & Float tails) Year 2 to 6 - 2.4 Mtpa (Whole Ore Leach & Float tails) Year 7 - 2.32 Mtpa (Whole Ore Leach & Float tails) Year 8 to 9 - 1.2 Mtpa (Float tails & BIOX) Year 10 - 0.84 Mtpa (Float tails & BIOX)	Randgold / SENET
4	Total tonnage	19.9 Million Tonnes	Randgold / SENET
5	Tailings SG	Oxides and Sulphides WOL (Sofia and CZ) = 2.84 NZ Sulphides (Float Tails) = 2.84	SENET - Confirmed
6	Slurry % solids by mass	Oxides and Sulphides WOL (Sofia and CZ) = 45.7% NZ Sulphides (Float Tails) = 48.9%	SENET- Assumed
7	Placed dry density of tailings	1.42 t/m^3	Epoch - Calculated from assumed void ratio and provided SG
8	Tailings Insitu Deposited Void Ratio	1	Epoch - Assumed
9	Terminating Rate of Rise Limit of the TSF	1.5 m/yr	Epoch
10	Rate of Rise Limit of the Starter Wall	2.5 m/yr	Epoch
11	Diversion Dam Required Storage Capacity	± 5 million m^3	Randgold

17.4.2. Geochemical and Geotechnical Investigations

Geotechnical Site Investigation

A geotechnical investigation was conducted by Inroads Consulting LLC (Inroads) during May 2018.

Due to the laboratory test results showing very high cohesions under undrained effective stress conditions, Inroads recommended shear strength parameters for the remoulded soil types as per Table 17-11, which have been utilised in the design of the TSF.

Table 17-11 Inroads Recommended Design Parameters

Material	Effective Cohesion - c' (kPa)	Effective Friction Angle - ϕ' (°)	Permeability (m/sec)
Clay and clayey soil	6	25	10^{-9}
Silt and sandy silt	5	30	10^{-8}
Laterite gravels	8	30	10^{-6}

Geotechnical Assessment of the Tailings Residue

Laboratory testwork was not conducted on the Massawa tailings as representative tailings samples could not be provided in time for the completion of the FS. Tailings geotechnical parameters were therefore adopted from other projects undertaken in West Africa with similar tailings characteristics. During the next phase of the Project, tailings laboratory testwork will be required to validate the current assumed tailings geotechnical parameters adopted and presented in Table 17-12.

The indicative particle size distribution (PSD) curves for the Massawa tailings are expected to be similar to the PSD curves for other typical West African gold tailings as depicted in Figure 17-4. The variation in PSD between the sulphide and oxide feed is also not expected to be significant different from each other.

Table 17-12 Tailings Geotechnical Parameters Adopted for the Massawa Tailings

Material	Unit Weight (kN/m^3)	Effective Friction Angle (φ') (degrees)	Effective Cohesion (c') (kPa)	Saturated Permeability (m/s)
Gold Tailings	18.6	27	0	2.5 by 10^{-8}

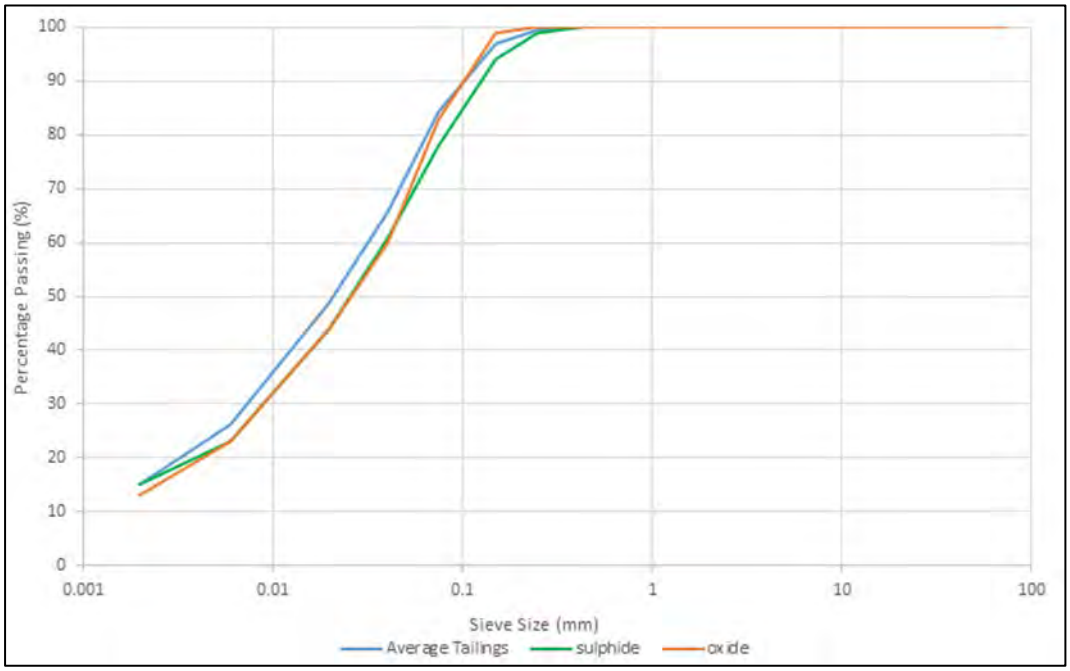


Figure 17-4 Tailings Particle Size Distribution for Typical Gold Tailings Made up of Oxides and Sulphides

Tailing Geochemistry

The tailings product produced at Massawa consists of two materials, namely: oxides, originating from the CZ and NZ pits located in the SE of the perimeter, and sulphides, originating from the Sofia Main ore body located in the NW of the perimeter. The tailings geochemistry testwork and evaluation study was completed by Digby Wells; the following results/observations were relayed:

- The sulphides are found to be benign and thus present no risk to the ground water system in the area.
- The oxides are found to contain cyanide and arsenic. These chemicals present a risk in polluting the ground water system should it ingress the soil below the TD and RWD.

Due to tailings geochemical characteristics both the TD and RWD shall be HDPE lined and the discharge water stemming from the RWD into the DD and downstream environment is treated by means of an arsenic and cyanide treatment plant located at the RWD.

17.4.3. TSF and DD Design

The key design features of the TSF and DD are:

- A HDPE lined valley dam ring-dyke TD, with an initial starter wall embankment (14 m high), to be compacted with remoulded lateritic/saprolitic material from a suitable borrow pit within the allocated areas. The TD converts to an upstream self-raising facility above the starter wall elevation to a final height of 27 m. This facility has been designed to store 20 Mt over the LOM at a varying depositional rate and an average insitu placed dry density of 1.35 t/m^3.
- The TD has an average rate of rise of 1.5 m/year and an overall upstream slope of 1V:3H.

- A concrete encased penstock decant system, that decants supernatant water into the energy dissipator and silt trap system before reporting to RWD and DD complex.
- An HDPE-lined RWD, which consists of a maximum wall height of 8.5 m above Natural Ground Level (NGL), has been sized to cater for 7 days of slurry water with a storage capacity of 189,200 m^3 it has a footprint area of 12.3 ha.
- An unlined DD, with a storage capacity of 4.7 million m^3 and an area of 177.24 ha, consists of a maximum wall height of 10 m above NGL.
- Associated infrastructure (i.e. slurry delivery line, storm water diversion trenches, toe drains, solution pipeline, silt traps, energy dissipator, and spillways).

Figure 17-5 displays the general layout of the TSF.

The zone of influence of a TD may be described as the extent of the area around it that may be affected with time taking into consideration possible impacts that may arise from the TD, e.g. flow slide, dust, surface and groundwater contamination, sterilisation of arable land etc. Based on the zone of influence assessment, the Massawa TD has been classified as a medium-hazard TD.

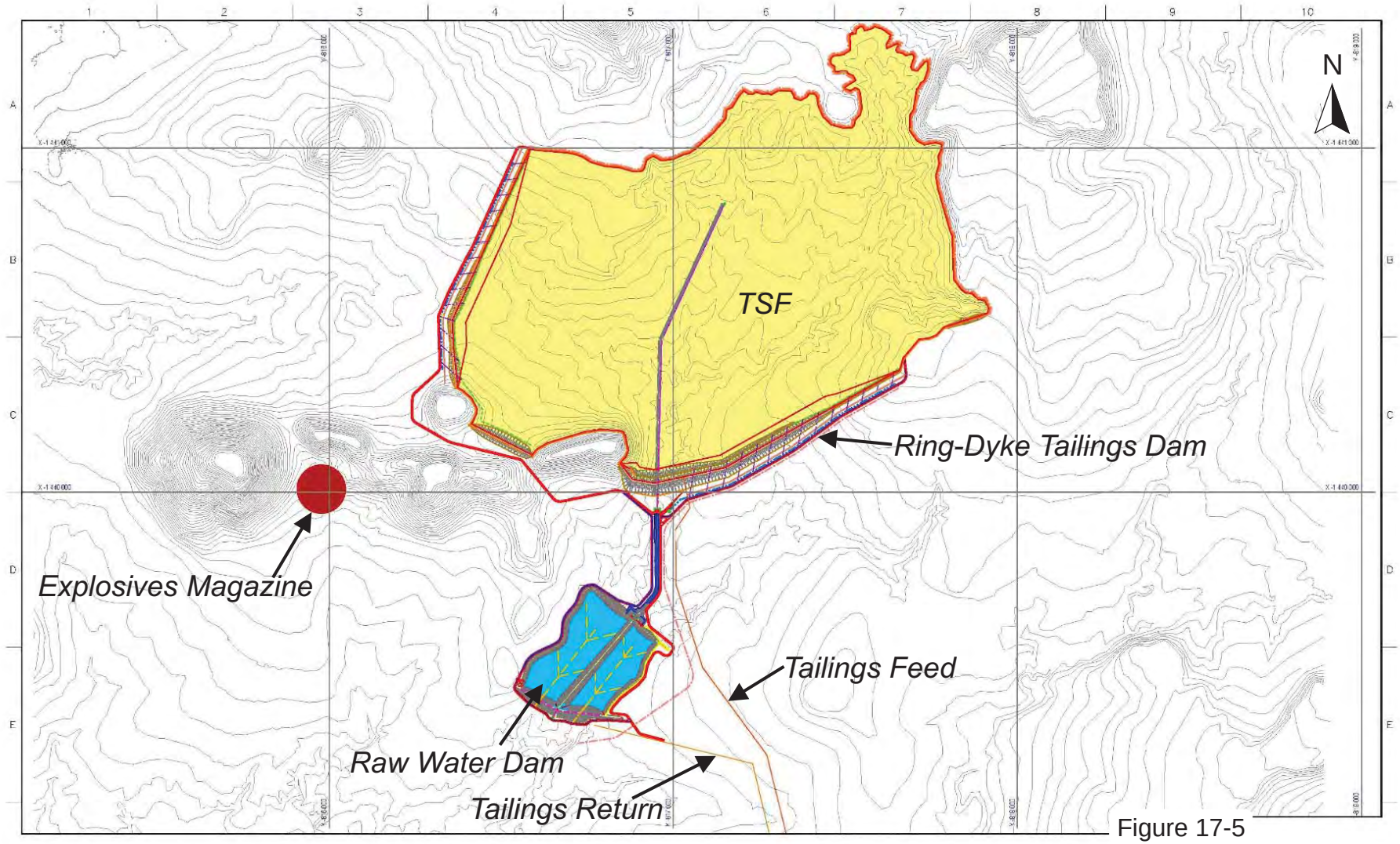


Figure 17-5

Barrick Gold Corporation

Massawa Gold Project

Sénégal

General Arrangement of the Massawa TSF

July 2019

Source: Randgold, 2019.

17.4.4. Seepage and Slope Stability Assessments

Seepage and slope stability analysis was conducted on the TSF using the material parameters determined from the geotechnical investigations of the insitu and assumed assigned tailings parameters. The results of the seepage and slope stability analyses indicate that:

- Operational drains and an operational liner have a significant impact on the position of the phreatic surface.
- Non-functional drains or a punctured/torn HDPE liner decrease the stability of the TSF.
- The slope stability FoS, probability of slope failure and reliability index, for the various departments meet or exceed the minimum requirements, under designed operational and upset conditions, namely:
    - A probability of failure and reliability index values less than 0.0007% (< 1:143,000) and greater than 4.35, respectively;
    - Deterministic minimum FoS as outlined in Table 17-13.

Table 17-13 Minimum FoS Requirements

Failure Mode	Loading Scenario	Required Minimum FoS
Upstream and Downstream	End of Construction (including staged construction)	1.3
Downstream	Long Term (1)	1.5
Downstream	Maximum Surcharge Pool	1.4
Upstream	Rapid Drawdown	1.1 (2) – 1.3 (3)
Downstream (high pool) or Upstream (Intermediate pool)	Earthquake (Pseudo-Static Analysis)	1.1
	Post-Earthquake (Post liquefaction strength)	1.1

Notes:
1 – Long term seepage includes steady seepage, maximum storage pool and spill way crest.
2 – FoS = 1.1 for drawdown from the maximum surcharge pool.
3 – FoS = 1.3 for drawdown from the maximum storage pool.

17.4.5. Deterministic TSF Water Balance

A TSF deterministic monthly water balance was developed in Microsoft Excel, based on average monthly, wettest year monthly, driest year monthly rainfall figures and average monthly evaporation figures and the simulated flow of water between the various facilities within the TSF over the operational LOM. The outcomes of the balance indicate the following:

- The TSF water balance is both a positive and negative water balance in the sense that during the wet months excess water is discharged, whilst in the dry season water is required from alternative water sources to meet the plant's water requirements.
- Overall, the balance indicates that there is sufficient water within the circuit and from the various sources to meet the plant's water requirements except for the month of May when approximately 120,000 m^3 additional water needs to be sourced from a fresh water source, e.g. the open pit ground dewatering etc. This value is over and above the fixed fresh water requirement for the plant, i.e. 11%/23%/38% of the slurry water sent to the TSF, depending on which metallurgical process is operating at that time.
- The DD discharges under steady state conditions annually but also runs dry/empty in the month of May of every year.

17.4.6. Estimation of Capital and Operational Cost

The capital expenditure associated with the TSF has been determined to a Class 3 AACE accuracy of (+25%, -15%). The rates used were sourced from a recent project located in Guinea, which was on a tendered basis, undertaken in June 2018.

- The total capital cost associated with the Massawa TSF has been determined as $31.16 million.
- For the duration of the construction of the TD and DD, a total capital cost of $0.6 million has been allowed for; this covers full time supervision and office support from the design engineer. This cost has not been incorporated in the total capital cost quoted above.
- The total capital cost associated with the Massawa DD has been determined as $6.0 million.

Table 17-14 TSF Capital Estimate

Item	Description	$ Millions
1	Preliminary and General (As provided by BCM and Liner Contractor)	7.55
2	TD Capital Expenditure	17.40
3	RWD Capital Expenditure	3.13
4	Unmeasured Works (15%)	3.08
4	Contingencies (0% as has been included in overall project contingency)	0
	Total Capital Cost	**31.16**

Table 17-15 Diversion Dam Capital Estimate

Item	Description	$ Millions
1	Diversion Dam	3.63
2	Spillway	2.27
4	Contingencies (0% as has been included in overall project contingency)	0
	Total Capital Cost	**6.0**

The operating costs associated with the TSF have been estimated at $7.6 million over the LOM, or a total of $0.76 million per annum. These operational costs include:

- $0.7 million per annum for operational management, comprising a team leader/manager, a supervisor, unskilled labour, and a spares workshop. This team will be responsible for:
  - Depositional management.
  - Upstream day wall paddock packing and control of raising the TD.
  - Penstock management (increasing the rings, ensuring the sump does not discharge, sleeving of the penstock).
  - Maintaining the pool wall, catwalk, and access to the penstocks.
  - Maintenance and repairs to the slurry delivery pipeline and valves.
  - Monitoring and cleaning of the toe drains, leakage detection.
  - Return water pump monitoring/manhole drainage pump monitoring.
  - General maintenance (cleaning trenches, silt traps and the energy dissipator).
  - Monitoring various components (freeboard, drain flows, water returns, rainfall, tonnes deposited etc).
- $0.06 million per annum for quarterly inspections, monitoring, and quarterly reports by the design engineer.

The TSF closure, rehabilitation, and aftercare of the TSF has been undertaken and assessed by Digby Wells.

The total LOM cost associated with the TSF over the duration of the Project life has been estimated, at a net present value (NPV) of zero-discount rate, as $39.66 million. This includes detailed design of the TSF and construction supervision during the construction of the facility, a total of $0.9 million, but excludes the rehabilitation, closure, and post closure monitoring and of the TSF.

17.4.7. Recommendations

The following recommendations are proposed for consideration and evaluation during the detailed design of the TSF:

- As the location of the RWD main wall was relocated post the test pits being finalised, additional test pits should be excavated and profiled prior to construction commencing to confirm the design depth of this facility etc.
- The filter design of the elevated and NGL toe drain systems should be undertaken.
- A geotechnical laboratory testing on a representative tailings sample is required to verify the strength parameters adopted during the FS design.
- A more comprehensive bill of quantities (BOQ) should be developed and tendered Senegalese construction rates be obtained during the next phase of the Project to improve the confidence and accuracy of the capital cost estimate.
- Various construction and operational QA/QC documentation, particular project specifications document, operational and inspection manuals for the TSF and DD should be developed.

18. Infrastructure

The selected Massawa site is a greenfield site without any existing infrastructure although some laterite roads do exist.

The on-site infrastructure required will be related to the processing plant and the supporting facilities as follows:

- In-plant access roads.
- Plant buildings.
- Plant reagents and consumables stores.
- Process plant site drainage.
- Sewerage disposal.
- Security and Fencing.
- Water supply.
- Communications.
- Power supply.

The proposed off-site infrastructure will support the mining and plant operations. Camp accommodation will also be provided at the site for the mine and plant site personnel. The main off-site infrastructure required for the development of the Project will be the following:

- Airstrip.
- Road network.
- Office complex.
- Accommodation.
- Diesel fuel storage.
- HFO storage.
- Off-site communications.
- Waste management.
- Raw water supply system.
- Sewage treatment.
- Fencing.
- Transport.

The general layout of the infrastructure may be seen in Figure 18-1.

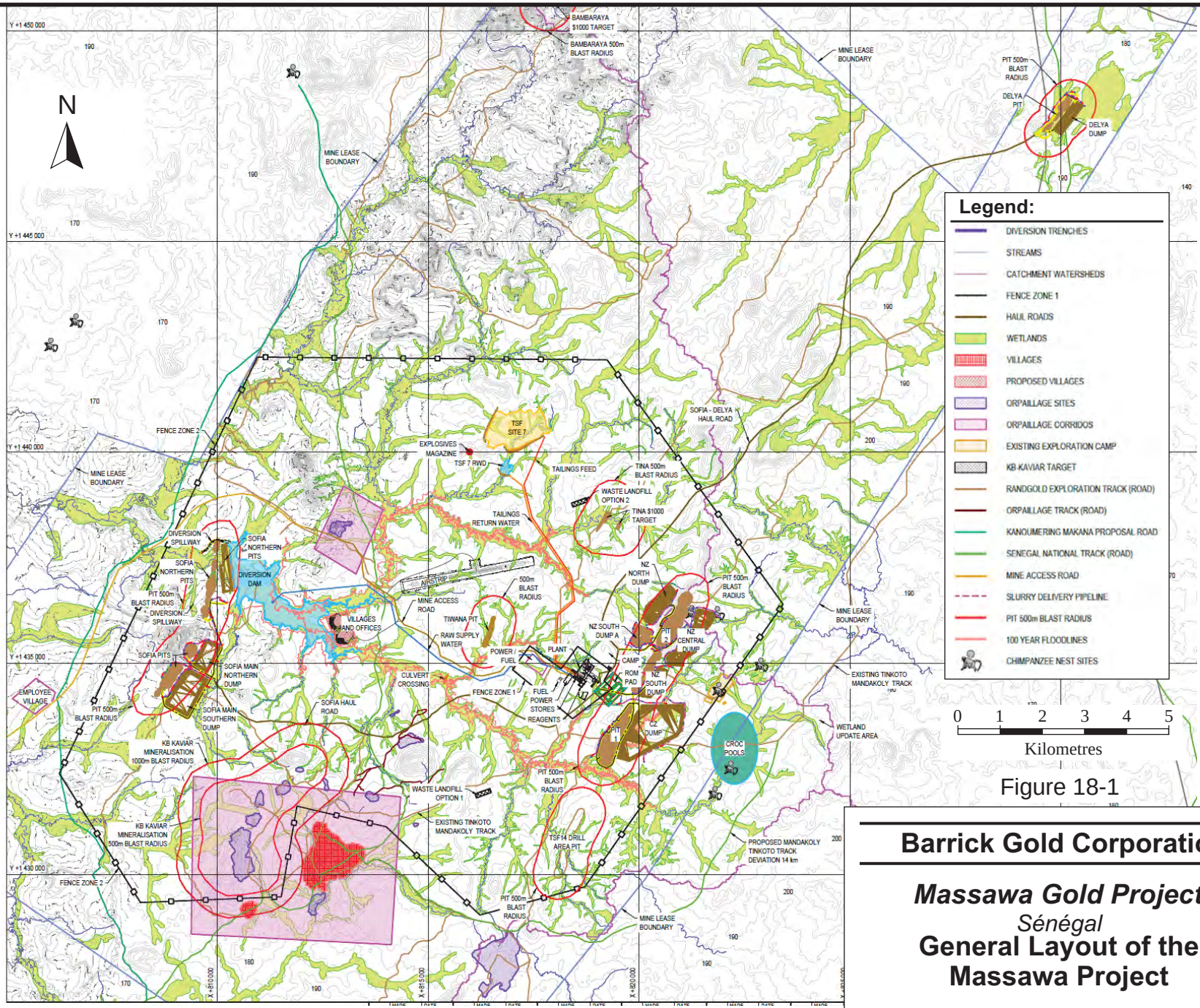


Figure 18-1

Barrick Gold Corporation

Massawa Gold Project

Sénégal

General Layout of the Massawa Project

July 2019

Source: EPOCH, 2019.

18.1. Project On-Site Infrastructure

The on-site infrastructure required will be related to the processing plant and the supporting facilities

18.1.1. In-Plant Access Roads

Access for the purposes of daily operations to respective in-plant functional areas, will be provided by utilising the bulk earthworks terracing.

18.1.2. Plant Buildings

The plant buildings will consist of the following:

- Store office (140 m^2) - A fully furnished, prefabricated, Chromadek panel store office building (14 m by 10 m) will be provided for the store management personnel.
- Security office and change house (408 m^2) - A prefabricated, Chromadek panel security office/change house block (60 m by 10 m) will be constructed at the main access to the plant and will include a security office, change house, laundry, and training room.
- Plant control room (180 m^2) - A fully furnished, prefabricated, Chromadek panel constructed building (18 by 10 m) will be provided for the process plant control personnel. The control room will house the supervisory control and data acquisition (SCADA) system.
- First aid station (35 m^2) - A fully furnished, prefabricated, Chromadek panel constructed building (7 m by 5 m) will be provided for the treatment of minor injuries sustained within the process plant, or as a holding room before evacuation from the process plant site.
- Reagents change house (50 m^2) – prefabricated.
- 2 x Plant offices (168 m^2) - Initially, these two prefabricated Chromadek panel constructed plant office buildings (28 m by 6 m) will be used as construction offices. Thereafter, they will be converted to fully furnished plant office buildings for the process plant management personnel.
- Metallurgical laboratory (506 m^2) - A single-storey, prefabricated, Chromadek panel construction metallurgical laboratory building (46 m by 11 m) will be provided, and it will be fully equipped to perform a daily analysis of process samples.
- 4 x Gate houses (25 m^2) – prefabricated.
- Plant workshop (780 m^2) – The workshop will consist of a steel pre-engineered portal frame building enclosed by Chromadek roof and side wall sheeting. The workshop floor area will be 39 m by 20 m, which will incorporate a mechanical repair shop of 300 m^2 and an electrical repair shop of 150 m^2, both of which will be serviced by a 5 t overhead gantry crane.
- Main store (1,500 m^2) – An enclosed steel structure warehouse (50 m by 30 m) will be provided to the south of the process plant for the purpose of storing spares and equipment for the process plant. Suitable shelving and racking will be incorporated in this building to store and manage the stored items.

18.1.3. Plant Reagents and Consumables Stores

The reagents stores are located to the south of the process plant, in close proximity to the reagents make-up area to facilitate unrestricted access to the make-up area. Reagents will be

offloaded to the north of the stores and a ring road has been provided around the storage area to facilitate the safe exit of the delivery trucks. The layout of the storage area has been designed to facilitate the first-in last-out principle, with roller shutter doors provided on both the north and south sides of the stores and a central 5 m wide alley that will allow forklift access.

The reagents will be contained within the following storage areas:

- Cyanide Shed (595 m^2) - The store is 35 m by 17 m and 4 m in height to eave level to accommodate 1,332 bulk boxes of cyanide, which will be stacked three boxes high.
- Senfroth Shed (78 m^2) - The store is 6 m by 13 m and 4 m in height to eave level. The Senfroth Shed will accommodate the frother whereby provision have been made for 180 intermediate bulk containers (IBC).
- Hydrochloric Acid Shed (156 m^2) – The shed will accommodate HCl (supplied in 210 L drums), with provision for 360 drums. It will also accommodate activated carbon (supplied in 500 kg bags), with provision for 48 bags. The activated carbon will be separated from the HCI reagent by a brick wall.
- Reagents Shed (1,225 m^2) - This shed will accommodate SMBS (supplied in 1 t bulk bags) with provision for a total of 476 bags. It will also accommodate quicklime (supplied in 1 t bulk bags) with provision for 2,196 bags.
- PAX Shed (156 m^2). The shed will accommodate the following reagents:
    - PAX (supplied in 25 kg bags) with provision for 42 pallets
    - Antiscalant with provision for 2 pallets
    - Corrosion Inhibitor with provision for 2 pallets
    - Bioxide with provision for 2 pallets
    - Flocculant (supplied in 25 kg bags) with provision for 120 pallets
    - Borax (Flux) (supplied in 25 kg bags) with provision for 4 pallets
- Copper Sulphate Shed (156 m^2) - The shed will accommodate copper sulphate (supplied in 25 kg bags) with provision for a total of 120 pallets. It will also accommodate caustic soda (supplied in 1 t bulk bags) with provision for 70 bags.
- Hydrogen Peroxide - Hydrogen peroxide solution will be supplied in 1,200 kg IBCs and will be stored in a 12 m container, which will be fenced off.

The reagents stores will comprise a steel-clad structure and concrete floor slab, complete with drainage and spillage handling facilities.

A concrete, paved area will be provided on both sides of the reagents store to facilitate loading into the store and movement from the store to the reagents make-up area.

18.1.4. Process Plant Site Drainage

The process plant area will be elevated (± 500-mm minimum above the remaining natural ground levels at the highest elevation) to divert storm water around the plant area. Drainage of the terrace will be into a drainage channel discharging to a silt trap prior to discharge into the Pollution Control Dam.

18.1.5. Sewerage Disposal

A containerised sewage treatment plants will be provided for sewage treatment of the plant area as indicated below. The unit will service the plant area and construction camp directly adjacent to the area:

- Plant area 120 m³/day.
- Metallurgy staff: 248 (say 250) persons per shift.
- Engineering staff: 129 (say 130) persons.
- Total persons serviced 380 maximum.

18.1.6. Security and Fencing

The plant site will be enclosed by a 2.1 m razor mesh fence in high density option (150 mm by 75 mm aperture). Mesh wire diameter will be 2.4 mm high strain wire. The razor mesh type will be a ripper blade profile.

The reagents and storage areas will be equipped with separate access control gates and gate houses for different functional areas. This area will be operated independently from the general process plant areas.

A 2.4 m high anti-climb, cut resistant weld mesh ClearVu security fence, including shark tooth spikes, will be erected around the gold room.

Furthermore, the plant will be fitted with CCTV cameras installed at strategic locations.

18.1.7. Water Supply

A DD will be constructed to ensure uninterrupted supply of water to the plant. Water stored in the DD will be pumped to the process plant for make-up operations. These pumps will be sized to cater for commissioning and during the dry and wet seasons, where raw water demands vary significantly.

Pumps will be installed in the RWD to enable pumping of process water to the process water ponds located in the plant.

Potable Water Distribution

On-site Potable Water Distribution

Raw water will be supplied from DD to the plant raw water pond and subsequently to the potable water storage tank via the plant potable water treatment plant. The potable water plant will be a containerised unit capable of producing 5 m³/h of potable water.

Off-site Potable Water Distribution

Raw water will be supplied from DD directly to the staff village/office complex potable water treatment plant.

Fire Water Distribution

There will be an electric and a diesel-powered fire water pumping system. The fire water system will consist of a buried fire water ring main and fire hydrants at the plant site, ancillary buildings, and the process plant. Hose cabinets will be placed at the fire hydrant locations and the system will be supplemented with portable fire extinguishers placed within the process plant facilities.

18.1.8. Communications

Communication will be catered for by means of both a telephone system as well as a two-way radio system

18.1.9. Power Supply

Power Demand

Three operational power demands have been established for the Project:

- During the first two years of operation (Phase 1 - oxide ore) - Steady-state demand was calculated at 11,349 kW with maximum demand at 19,609 kW.
- From Year 2 to Year 6 (Phase 2 - fresh ore) - Steady-state demand was calculated at 17,720 kW with maximum demand at 23,180 kW.
- From Years 6 onwards (Phase 3 - BIOX) - Steady-state demand was calculated at 19,164 kW with maximum demand at 27,074 kW.

HFO and LFO Power Generation Plants

The power generation plant will be financed, owned, and operated by Barrick.

Table 18-1 provides details on HFO generator sets and Table 18-2, on LFO generator sets.

Table 18-1 HFO Generator Sets Details

Item	Description
Model	12CM32C
Engine Manufacturer	CATERPILLAR
Continuous Rated Power	5,358 kW
Sound Level (dB)	75 dB at 7 m
Rated Voltage	11 kV
Optimal Fuel Consumption	199 L/h

Table 18-2 LFO Generator Sets Details

Item	Description
Model	3516BH
Engine Manufacturer	CATERPILLAR
Continuous Rated Power	1,600 kW
Sound Level (dB)	75 dB at 7 m
Rated Voltage	11 kV
Optimal Fuel Consumption	262 L/h

Electrical Infrastructure

The Massawa mine will include various electrical infrastructure items required for operation. In addition to the process plant requirements, the electrical supply facilities will feed the mine accommodation camps, mining infrastructure, fuel storage, administration building, metallurgical building, clinic building, storehouses, workshops, TSF, and arsenic treatment plants and DD.

Medium Voltage Switchgear

The Massawa plant will have three medium voltage switchboards rated at 11,000 V AC.

The switchgear will be rated using the following details:

- Medium voltage: 11 kV.
- Frequency: 50 Hz.
- Main bus bar rating: 1,600 A.
- Basic insulation level: 95 kV.
- Short circuit rating: 25 kA – 3 s.

Transformers

Distribution transformers will be manufactured in accordance with IEC 60076.

Low Voltage Distribution

The maximum transformer rating for low voltage supplies will be 2,500 kVA. Each transformer will feed a 525 V motor control centre (MCC) that supplies power to a dedicated section of the plant.

Motor Control Centres

Multiple containerised MCCs have been allocated for the process plant.

MCCs will be of the compartmentalised, non-withdrawable type with moulded case circuit breakers, magnetic contactors, and earth bus, and will comply with IEC 61439-2

Earthing Protection

Provision has been made for earthing of all electrical equipment and buildings where applicable.

Provision has been made for earth resistivity testing prior to the installation. Earth mats shall be installed at all medium voltage substations, ring main units, and transformers.

Lighting Protection

Provision has been made for high pressure sodium lighting that will be structure-mounted to ensure safe working conditions. Lighting will also be installed to ensure that visual security monitoring can be conducted at all times in and around the process plant and associated infrastructure to maintain a safe work environment.

Fire Detection and Suppression Systems

All of the medium voltage switchgear, transformer buildings, MCCs, server room and control room are equipped with a Pyrogen fire detection and suppression system.

Overhead Power Lines

Overhead Power Lines (OHLs) have been provided for all off-site infrastructure power requirements. The OHL reticulation will originate from 9000-MVSG-01 and will be routed to the following off-site infrastructure at 11,000 V AC:

18.2. Project Off-Site Infrastructure

18.2.1. Airstrip

As part of the Project, a dedicated private airstrip will be constructed, with the primary function being the safe transport of gold exports from site, and expat staff transport.

An allowance has been made for a 400 mm thick unsuitable layer of material to be removed to spoil sites. Same will be replaced with suitable material (minimum G6 quality) from borrow pits, compacted to specification.

18.2.2. Road Network

Two types of roads have been included as part of the study, site roads and plant main access road.

Site Roads

All site roads are assumed to be 6 m wide, with a 500 mm wide road shoulder on each side.

The site roads will provide access to all the Project areas including the airstrip, process plant and construction camp, staff village (and internal distribution access), and TSF.

Plant Main Access Road

The total distance from the junction with the national road to the proposed plant is approximately 28 km. The level of service provided for the specific road would typically be similar to that of the national road network.

18.2.3. Office Complex

The office complex buildings will be constructed of prefabricated panels, be located approximately 6 km from the plant and near the staff village, and will consist of the following:

- One Exploration Office building (210 m^2).
- One Mineral Resources Office building (140 m^2).
- One Mining Office building (140 m^2).
- One Finances Office building (140 m^2).
- One Engineering Office building (140 m^2).
- One Plant Administration Offices building (1,026 m^2).

18.2.4. Accommodation Facilities

Staff Village

The staff village houses will be constructed of prefabricated panels, be located approximately 7 km from the plant, and will consist of the following:

- Eight Management houses – three bedroom (169 m^2).
- 21 Heads of Department houses – two bedroom (72 m^2).
- Five guest houses – five bedroom (140 m^2).
- 25 single quarters - five bedroom (114 m^2).
- Entertainment club (490 m^2).
- Kitchen and dining room (477 m^2).
- Two laundry rooms (99 m^2).
- Gymnasium (169 m^2).
- Office (140 m^2).

Construction/Staff Camp

The construction/staff camp will be constructed of prefabricated panels, be located directly alongside the process plant, and will consist of the following:

- 18 single quarters - 10 bed (114 m^2).
- Entertainment club (276 m^2).
- Kitchen and dining room (590 m^2).
- Two laundry rooms (99 m^2).

- Office (140 m^2).
- Gate and gatehouse for security control.

Massawa Mine Employee Village

This village will be built by the mine and will house staff and families.

18.2.5. Diesel Fuel Storage and Refuelling Facility

A fuel storage and refuelling facility (i.e. light furnace oil – LFO/Diesel) will be constructed by the nominated fuel supply subcontractor including the following:

- Storage capacity of 1,150 m^3.
- Unloading and dispensing equipment/
- Civil and structural works.
- Electrical works.
- Firefighting system.
- Fuel management system.

Day tanks will hold capacity for daily consumption of the following dispensing stations:

- Heavy mining equipment dispensing station: This station will use two self-priming positive displacement pumps 30 m^3/h each (one being standby) via a metering unit. The heavy vehicle refuelling will be done using fast fuelling nozzles (Wiggins or equivalent).
- Light vehicle dispensing station: This station will use two suction dispensers of 80 L/min each (one being standby). The maximum distance between the day tanks and the light vehicle station shall be 40 m to avoid priming issues with the pump.

HFO/LFO Storage Facility

The annual fuel consumption for the electrical power plant is estimated at 100,800 m^3.

A fuel storage facility (i.e. HFO and LFO) will be constructed by the nominated fuel supply subcontractor includes for the following:

- Storage capacity of 4,200 m^3;
- Provision for LFO storage tanks to ensure HFO generators start and shut down;
- Unloading and dispensing equipment;
- Civil and structural works;
- Electrical works;
- Firefighting system;
- Heating system for HFO tanks.

18.2.6. Off-Site Communications

Telephone System

The telephone system includes the UTP cabling, switches, and fibre cabling to off-site telephones located in the offices at the airfield, staff village and office complex.

PLC Panels

Small Remote I/O panels will be located off site where required. Overhead fibre lines will be used to connect remote I/O panels and MCCs to the plant control system.

18.2.7. Waste Management

Solid waste generated from the mine plant site, including ancillary buildings, will primarily be domestic and industrial non-hazardous waste. A comprehensive waste management plan should be developed for the Project by Barrick as part of the environmental impact assessment.

18.2.8. Raw Water Supply and Potable Water Distribution

Raw Water Supply

A water management model was developed for the site. The model was used to size the DD to supply the mine's raw water demands.

The DD, located on the NE side of the Sofia North pit, is proposed for the supply of plant raw water and potable water. The maximum storage capacity required is estimated to be 706,264 m^3.

A pipeline conveying fourteen million litres of water per day from the raw water supply dam to the plant was designed. The pipeline length is approximately 9,000 m. The water is abstracted by means of a barge from the DD and discharged into the raw water pond at the plant.

Potable Water Supply and Water Distribution in the Camps

Staff Village and Office Complex Potable Water

The staff village water treatment plant will supply potable water to the village and office complex buildings.

Construction Camp Potable Water

The potable water supply for the construction camp will consist of a supply pipeline from the potable water treatment plant in the process plant site to storage tanks located at the construction camp.

18.2.9. Camp Sewage Treatment

The camp will have a network of sewer lines transferring the sewage to a containerised sewage treatment plant, which will consist of separate chambers for anaerobic/anoxic/aerobic treatment with chlorine injection, and the discharge to the environment will be according to WHO standards.

18.2.10. Fencing

The perimeter of the areas listed below will be secured by a 2.1 m razor mesh fence in high density option (150 mm by 75 mm aperture). Mesh wire diameter will be 2.4 mm high strain wire. The razor mesh type will be a ripper blade profile.

18.2.11. Transport

Light Vehicles

The selection of vehicles is based on the Barrick organogram for process plant, operational and administration staff. The resultant fleet amounts to 26 vehicles.

Plant Operational Vehicles

The fleet shown in Table 18-3 will form part of the plant operational vehicles.

Table 18-3 Plant Operational Vehicles

Designation	Quantity
XJCM RT30 Rough Terrain Crane 30 t	1
ZOOMLION ZTC700V552 Hydraulic Truck Crane 70 t	1
ZOOMLION QY95V643 Hydraulic Truck Crane 95 t	1
MX50-2; Manitou Forklift 5 t	1
MT-X1440; Manitou Forklift 4 t	1
TFM 2628 Mercedes-Benz V-Series; 16,000 litre Water Bowser	1
Isuzu; 3000 litre Water; 350 litre Foam; Fire Fighting Truck	1

Personnel Transport Vehicles

The fleet shown in Table 18-4 will form part of the personnel transport vehicles.

Table 18-4 Personnel Transport Vehicles

Designation	Quantity
Small Bus - Toyota Coaster 30 seater	4
Large Bus - Tata LPO 1823 56 seater	2

18.3. Logistics and Transport

Logistics and transport studies were carried out by Multilog, which was nominated by Barrick as the freighter forwarder due to its proven track record and experience in West Africa.

18.4. Potential National Grid Power

Although this FS assumed self-generation using HFO generator sets, there is a potential upside, where one could forego the capital investment associated with a local power plant and take advantage of Senegal's electrification expansion programme. Realistically this will only be available for Massawa from 2020 onwards; nevertheless, this warranted greater investigation and monitoring of the progress, since the potential beneficial effects for the Project are significant.

Senegal's total installed energy generating capacity is approximately 1,000 MW of which the bulk is supplied via the national grid. La Société Nationale d'Électricité du Sénégal (SENELEC) owns most of the power stations while the remainder are private, either GTI-Dakar or ESKOM-Manantali. SENELEC is a Senegalese company created in 1983 and is responsible for the generation, transmission, distribution, and sale of electrical energy but also, identifies and finances new projects, where current initiatives could have a direct bearing on the Massawa Project.

This section gives a brief description of the current status but also then provides some insight into the future anticipated projects to expand the national grid.

18.4.1. Existing Power

SENELEC

As indicated below, most of the power is generated by diesel plants, while there are also gas, steam, and wind and solar power plants.

A significant portion of the total installed capacity comes from other countries, such as the 60 MW that Senegal uses of the 200 MW Manantali hydroelectric power plant across the border in Mali.

The list and split of power attributed to SENELEC and to the other private providers is seen in Table 18-5.

Table 18-5 Total Split of SENELEC Power

Description	Power (MW)	%
Inter-connected network	968.58	92.53
Isolated	78.25	7.47

Private Power Stations (IP)

These consist of power plants belonging to private independent producers, industrialists, or entities providing energy to SENELEC through legal purchase contracts.

18.4.2. Future Projects

According to a presentation made by the Senegalese Energy authorities in May 2018, there is an already underway ambitious expansion of the grid network, such that more than 2,000 MW of

generating capacity is envisaged being installed between 2017 and 2035. This is summarised in Table 18-6.

Table 18-6 Proposed Future Power Installations

Power Generation Type	Installed Power (MW)
Thermal (HFO & LFO)	240
CCGT (Combined Cycle Gas Turbine)	960
Hydro-Electric	435
Wind Farm	265
Solar	265
TOTAL	**2,165**

Of special interest to Massawa is the plan for the Kedougou distribution which will reportedly be ready to tie-in to the national grid in Q1 2020, with power transmission lines already being installed. It is not yet known whether the closest or earliest grid connection will tie in to the OMVS grid from Manantali, which is waiting for a SENELEC interconnection to Tambacounda, or the OMVG interconnection from Kaolack to Kedougou, or the supply from Sambangalou to Tambacounda, presumably via Kedougou.

18.4.3. Rates

Massawa was provided with a generic tariff which is seen in Table 18-7, denoting pricing in both West African CFA franc (fcfa) plus expected resultant unit power costs in $. Of particular interest to Massawa is the medium-tension (MT) long-term rates or the high-tension (HT) general rates, which offer a distinct advantage over thermal power generation.

Table 18-7 Proposed Senegalese Grid Power Rates

Tension Tariffs	Off-Peak	Peak	Monthly Fixed Fee	Final Expected Unit Cost
MT	**(fcfa/kWh)**	**(fcfa/kWh)**	**(fcfa/kW)**	**($/kWh)**
Tariff Short	118.51	183.48	907.32	$0.219
Tariff General	85.29	136.46	3,861.89	$0.166
Tariff Long	70.07	112.12	9,321.26	$0.151
HT				
Tariff General	55.69	80.2	9,461.23	$0.121
Tariff Special	74.16	106.78	4,206.24	$0.142

18.4.4. Conclusion

Further dialogue with the Senegalese authorities is anticipated to occur early in 2019, with Massawa intending to have direct representation. This should provide Massawa with a greater degree of confidence as to the positioning (offtake), source and cost of the power, including most importantly, the timing. Of additional pertinent interest is to understand the risk associated with the ability to receive a stable and consistent supply, which is critical for steady operational performance.

19. Market Studies and Contracts

19.1 Markets

The gold market is highly liquid and benefits from terminal markets (London, New York, Tokyo, and Hong Kong) on almost a continuous basis. Gold prices were in general downward trend from 1980 to 2000 where gold traded down to approximately $250/oz. From 2000 the price increased annually until 2011 and 2012 where the price peaked at just under $1,800/oz. There was a sharp correction to the gold price in 2013 with the end of Quantitative Easing monetary policy by the US Reserve Bank. Since then god has remained range bound between $1,350/oz and $1,050/oz.

Gold produced at the mine site will be shipped from site, under secured conditions, to a refining company. Under proposed contractual conditions, the refiner purchases the gold from the mine with the proceeds automatically credited to the mine's bank account.

Gold production will be sold on the spot market, with no plan currently to hedge any sales.

19.2 Contracts

It is Barrick strategy to outsource mining activities to contractors and in all instances ensure that the mining operation can purchase the equipment at the end of the contract period at its depreciated price or, should the contractor default, at a pre-determined pricing mechanism. Prior to start-up, all major mining contractors are requested to tender and the most appropriate tender is accepted, thereby ensuring that the best competitive current pricing is achieved. Care is taken at the time of finalizing contracts to ensure that the rise and fall formula is totally representative of the build-up of the quoted price per unit. At the time of award, prices quoted are compared to benchmark prices of other owner mine operations.

The contract mining costs are highly dependent on when tenders are issued as the price of major equipment varies dependent on demand as well as the cost of finance. Rise and fall can be negatively affected by currency fluctuations as well as price squeeze due to scarcity.

The mine will produce doré bars which will be sent to an accredited gold refinery for refining. For other operations, Barrick Africa and Middle East presently utilises the Rand Refinery in South Africa is used as it has been found to be the most competitively priced.

Refining prices are subject to fluctuations in the cost of transport as well as insurance costs.

20. Environmental and Social Impact Assessment

To exploit the mineralisation identified on the Project, Barrick is required to undertake an environmental impact study in accordance with Chapter 5 (Article 48) of the Environmental Act (Code de l'Environnement - Law (N° 2001-01 of 15th January 2001) and its implementing decree (Décret no 2001 – 282 du 12 avril 2001 portant application du code de l'environnement). For the FS, Randgold appointed Digby Wells Environmental (Jersey) Limited (Digby Wells) as the independent consultant to undertake an Environmental and Social Impact Assessment (ESIA) process for the Project in terms of the national Environmental Act and in accordance with international standards including the International Finance Corporation (IFC) Performance Standards (PS). Digby Wells has undertaken the ESIA process with in-country partners, Tropica Environmental Consultants (Tropica). The partnership between Digby Wells and Tropica ensures that there is two-way knowledge transfer during the ESIA in terms of both international good practice and local expertise and knowledge.

The main aim of the ESIA is to outline the Project activities and examine positive and negative environmental and social impacts that will be caused by the implementation of the Project. From this, the ESIA proposes mitigation/enhancement and management measures for such impacts to ensure that the Project is conducted in a manner that allows for future productive use of the land and a sustainable social environment following the conclusion of the mining operations.

20.1. Description of the Project Areas Affected

The Project will involve the development of open pits, predominantly known as Massawa NZ and Massawa CZ located to the east of the Project area as well as Sofia Main and Sofia North located to the west of the Project area, and the Delya satellite to the north. Additional satellite deposits are located throughout the permit area that may be exploited in the future. The depth of the open pits ranges between 70 m and 150 m. Holes will be drilled in targeted areas, and once drilled, the drill holes will be filled with explosives to blast overburden rock which lies above the gold ore as well as the ore itself. Once blasted, the overburden rock (waste rock) and subsequently gold ore can be removed. The waste rock will be transported to designated areas known as Waste Rock Dumps (WRDs) for storage. The gold ore removed will similarly be transported by truck to a designated area known as the ROM pad where it is temporarily stored before being placed into a primary crusher which breaks the ore into reduced pieces before the gold mineral can be extracted from the ore in the processing plant utilising various mechanical and chemical processes. Various supporting infrastructure including roads, offices, workshops, processing plant, and power infrastructure will be required to operate the mine. The mining operation will be carried out in three main phases to ensure that all required infrastructure, operational components, as well as the activities required to close-off the operation are in place. The phases consist of, but are not limited to, the following main activities:

Construction Phase:

- Site clearance and topsoil removal – the preparation of the site will include the removing of the vegetation cover and the top layer of soil on the direct footprint areas.
- Topsoil stockpiling – the topsoil removed will be preserved in stockpiles for use during rehabilitation once mining is concluded.
- Construction of mine related infrastructure – this includes mine housing camp, office buildings, haul roads transport people and material between the various areas, pipelines, dams which are required to manage water coming into and leaving the mining area, the processing plant where gold will be extracted from the ore removed from the open pits, as well as storage, workshop, and power facilities.

Operational Phase:

- Removal of waste rock - once overburden rock is reached, drilling and blasting will need to be undertaken to break the rock material. The waste rock will then be removed from the area to expose the targeted ore.
- Loading, hauling, and stockpiling of waste rock – waste rock removed will be transported on the haul roads via truck to designated WRDs.
- Mining gold ore – similar to the removal of waste rock, this consists of drilling and blasting of the ore material for its removal.
- Load, haul, and stockpiling of ROM ore – the ore will be removed from the pits and transported on the haul roads via truck to the ROM pad where it will be temporarily stored and subsequently fed into a crusher and mill to break the ore into smaller pieces that can be put through the processing plant.
- Processing plant – once crushed the ore will be transported to the processing plant where mechanical and chemical processes will be utilised to extract gold from the ore.
- Disposal of residue tailings material from the processing plant onto the TSF via pipeline – once the gold is extracted from the ore, the remaining material, which will be in slurry (semi-liquid mixture) form known as residue tailings, will need to be disposed of. A TSF will be constructed as gold processing progresses. The residue tailings will be transported via pipeline in their slurry form from the processing plant to the TSF. Once disposed of, the slurry will dry and eventually result in a residue tailings stockpile.
- Water use and storage on site – during the operation, water will be required for various domestic and industrial uses. Dams will be constructed that capture water from the mining area, which will be stored and used accordingly.
- Storage, handling, and treatment of hazardous products (including fuel, explosives, and oil) and waste – hazardous products will be utilised for machinery and power generation for the various activities during operations. Furthermore, domestic and hazardous waste will be generated which together with the hazardous products must be handled carefully and strategically to prevent pollution.
- Maintenance activities – through the operations, maintenance will need to be undertaken to ensure that all infrastructure is operating optimally and does not pose a threat to human or environmental health. Maintenance will include haul roads, pipelines, processing plant, machinery, water and storm water management infrastructure, stockpile areas, and the TSF.

Decommissioning and Rehabilitation Phase:

- Demolition and removal of infrastructure – once mining activities have been concluded, infrastructure will be demolished in preparation of the final land rehabilitation. It should be

noted that some infrastructure such as buildings may remain in agreement for future land users. Furthermore, the TSF, WRDs, and open pits will remain but will be rehabilitated to ensure a stable and safe state to prevent future negative impacts as much as possible.

- Rehabilitation – rehabilitation mainly consists of spreading of the preserved subsoil and topsoil, profiling of the land, and re-vegetation.
- Post-closure monitoring and rehabilitation – given the extent of disturbance that will be caused by the operation, monitoring of specific environmental aspects such as surface and groundwater water that could potentially be contaminated from the open pits, WRDs and seepage from the TSF, the success of vegetation establishment and the general site status will need to be undertaken to ensure that no further impacts to the natural and social environment will be experienced.

The operational phase of the Project is expected to run over a 10-year period with a total of 18.8 Mt ore and 137 Mt of waste mined during this time.

20.2. Relevant Policies, Legislation, and Institutional Framework

Similar to many other industrial projects, the extraction of minerals results in a number of environmental impacts that must be addressed during the life of the Project. The actions associated with construction, installation, operational, and decommissioning phases of the Project, have various positive and negative environmental impacts. While the resulting negative impacts must be addressed so as to maintain the integrity of the environment, Barrick must also ensure compliance to various policies and/or legal frameworks that govern the management of the environment.

As indicated above, the ESIA has been completed in accordance with the Environmental Act (Code de l'Environnement - Law (N° 2001-01 of 15th January 2001) and its implementing decree (Décret no 2001 – 282 du 12 avril 2001 portant application du code de l'environnement). Other applicable legislation includes the following:

- Mining Code Law No. 2003-36 of 24th November 2003 and updated with Mining Code Law No. 27/2016 of 30th October 2016.
- Water Code Law No. 81-13 of 4th March 1981.
- The Forestry Code Law No. 98-03 of 8th January 1998.
- Hunting and Protection of Fauna Code Law No. 86-04 of 24th January 1986.
- Water Treatment Code Law No. 2009-24 of 8th July 2009.
- Law n° 96-06 of 22nd March 1996 Local Authorities (full jurisdiction in terms of waste management).
- Atmospheric Emissions standard NS 05-062 of December 2004.

Furthermore, the Project aims to also demonstrate compliance with international standards such as the IFS PS on Environmental and Social Sustainability. The IFC PS comprise eight quality standards which the project developer is required to meet throughout the project life which pertain to various environmental and social aspects. International best practice will be utilised to complement Senegalese legislation and not replace it.

20.3. Project Baseline Conditions

The Project is located in the SE portion of Senegal within the Kédougou Region between the villages of Kédougou and Bembou, Mako and Khossanto. Dakar, the capital of Senegal, is approximately 700 km NW of the Project area. Regionally, the Sofia deposit is located approximately 45 km north of the town of Kédougou, with the nearest town of Tinkoto located 6 km south of the Massawa deposit. The Peulh and Malinke villages of Mandakholé and Kanoumering are both situated 15 km to the northeast and southwest of the Massawa deposit respectively. The Faleme River marks the international border with Mali and is located 75 km east of the Project area. Downstream of the proposed Project area, the Niokolo-Koba River flows through the Niokolo-Koba National Park, a United Nations Educational, Scientific and Cultural Organisation (UNESCO) World Heritage Site (approximately 15 km west of the Project area).

The site is in a largely undeveloped rural area surrounded by informal (artisanal) mining activities, communities in small towns and villages founded on the mining legacy of the area. The common land uses in the Project surroundings are subsistence agriculture, animal rearing, vegetable gardening, and artisanal mining.

The soils found in the Project area are shallow resulting in low agricultural potential, which is best suited for grazing practices. The land cover within the Project area is dominated by low shrubland, a mixture of low shrubland/savannah is found in the flatter, low-lying areas, while the savannah class dominates the hillier areas in the NE, as well as in the far south of the licence area. The low agricultural/land capability is echoed in the fertility status of the soils. The fertility status of the soils is acidic and, to achieve full cropping potential, the soils would require lime to counteract acidity, as well as supplemented by nitrate, potash, and phosphate fertiliser.

In terms of terrestrial biodiversity, the proposed Project is located in a largely natural undisturbed environment. The larger study area is currently under pressure from existing anthropogenic land use including grazing, wood collection, and artisanal mining from surrounding villages. The general health of the natural habitat, however, is diverse and of high value, therefore should be conserved. Several protected plant and animal species and Species of Special Concern (SSC) were identified within the Project area, including the endangered Western Chimpanzee. The Project area is in close proximity of the Niokola-Koba National Park where many protected species may be recorded which extend into the general area. The Project area is also characterised by sensitive wetland ecosystem which currently has minimal disturbance. Further chimpanzee baseline studies are being undertaken to determine the home and core ranges of the respective communities.

The Project is located in the upper reaches of the Niokolo-Koba River catchment, approximately 34 km upstream of the Niokolo-Koba National Park. The status of the main streams within the Project area is largely natural with neutral, fast-flowing ephemeral streams, which are largely free of clear signs of contamination and/or notable anthropogenic impacts, which is believed to be driven by the limited surrounding land-use activities and the limited accessibility. Due to the highly dynamic flow regime observed within the area, the macroinvertebrate assemblages present within the study area were largely dominated by tolerant, pioneering taxa with a moderate tolerance to water quality impairment and a preference for submerged marginal vegetation

habitats. On the other hand, representative fish species were collected across a variety of habitats during the wet season, which suggested favourable conditions and a potential underestimation of the fish assemblage diversity, as demonstrated by the low abundances observed over a high-density river network.

In terms of groundwater, monitoring data shows that the baseline groundwater quality is generally suitable for drinking with limited parameters (namely pH, arsenic, and lead) exceeding the World Health Organisation (WHO). The elevated arsenic levels are suspected to be a result of natural dissolution of the host rocks. The water level in most of the boreholes was found to fluctuate seasonally. This indicates that the shallow weathered aquifer is mainly unconfined and is susceptible to climatic conditions.

An environmental noise impact assessment was conducted and baseline noise levels were benchmarked against the IFC guideline limits for disturbing noise. The baseline ambient day and night time soundscape generally exceeds IFC guidelines. The relatively excessive levels were mainly due to intermittent motorbikes passing through, with bird song from the various avifauna and socialising activities by the villagers during the daytime and insect noise during the night time.

In terms of air quality, the South African National Dust Control Regulation (NDCS), 2013 were utilised to evaluate the baseline air quality associated with the Project area. The dust deposition data from 2010 to 2015 shows that the ambient environment is not pristine, due to the levels of dust measured. During dry season months (November to May), monitoring records revealed some exceedance of the Residential Limit (600 µg/m^3).

A heritage survey and assessment was conducted through which several heritage resources were identified with a Cultural Significance range from negligible to very high. These heritage resources consist of sacred sites; archaeological metalworking sites; archaeological sites and features; and isolated finds – ceramic and lithic scatters.

20.4. Assessment of Impacts

The proposed Project is located in a largely natural undisturbed environment and the general health of the natural habitat is diverse and of high conservation value. The Project area is characterised by a sensitive wetland ecosystem which currently has minimal disturbance. The proposed Project will result in the direct loss of approximately 370 ha of wetland area as a result of infrastructure development, which has been deemed a major impact. Further indirect loss of wetlands is expected downstream of the Project site due to altered hydrology mainly as a result of the construction of the diversion dam on the Niokolo-Koba River. The diversion dam development will have a major impact on surface water, namely loss of water quantity required to sustain floodplains downstream the proposed diversion dam. The diversion dam inundation may impact on the wetland systems and ecological functions, aquatic ecology, and potential biodiversity between the Project area and the Niokolo-Koba National Park.

Further chimpanzee baseline studies are being undertaken in the Project areas to determine the number of chimpanzee groups, the sizes of the chimpanzee groups, and their respective home and core ranges. Potential impacts on chimpanzee groups, and the respective mitigation

measures will be determined as part of these studies and incorporated into a Biodiversity Action Plan.

Another major impact identified is the potential migration into the surrounding area. The proposed Project is likely to result in the expansion of the local population around the villages and artisanal mining sites; this induced impact will further expand the extent of disturbance and consequently direct impacts to the natural habitat.

In addition, the proximity of the Niokolo-Koba National Park is noteworthy. The national park is located more than 15 km from the proposed infrastructure area and as such no direct impacts are anticipated. However, indirect and/or induced adverse impacts may occur. This will mainly be a result of the necessity to divert and dam water from the main stem of the Niokolo-Koba River to facilitate access to the Sofia deposits which is likely to affect local streamflows and sediment regimes that may extend up to the Niokolo-Koba National Park. Furthermore, a limited potential exists for nuisance impacts (visual, air quality, and noise) as a result of the Project activities and the induced impact of influx on the surrounding area to occur.

Of the most significant potential impacts identified, the following require mitigation and constant monitoring:

- Habitat and biodiversity loss (for both terrestrial and aquatic) and the disturbances to fauna;
- Decreased volume of run-off reporting to downstream rivers and reduced quantity of groundwater as a water resource for communities and Niokolo-Koba National Park use and support to ecosystems;
- Loss of wetland areas (total wetland loss of 371.3 ha of which 7.2 ha is direct loss associated with the mine pits) as well as fragmentation of riverine corridors, erosion, sedimentation, and altered wetland Present Ecological State;
- Direct disturbance of heritage resources;
- Potential influx into the local area as a result of the Project's presence which will likely intensify the existing artisanal mining operations as well as social ills associated with increased economic and social activity in the local area;
- Pressure placed on existing socio-economic infrastructures which were found to be limited within the local area.

With respect to the existing socio-economic conditions, the establishment of the mine will necessitate the occupation of land which is mostly virgin land. Therefore, although a loss of livelihood is anticipated, this impact has not been deemed as highly significant.

As indicated above, the establishment of the mine is expected to result in an influx of people into the area. This comprises people who are seeking opportunities directly from the mining activities and those attracted to the area as a result of the socio-economic improvements that will occur as a result of the presence of the mine. The mine will have positive impacts on both local and the wider regional economy as the mine will empower local communities through employment, use of local goods and services, as well as the establishment of Community Development Plan which will result in improvements of socio-economic infrastructure and skills development. On the other hand, however, these improvements which will lead to migration into the area will potentially also

result in related negative socio-economic impacts including pressure on existing infrastructure, degradation of local customs and mixing of cultures which may have negative influences, health and safety related issues related to increased social ills. As such, the positive socio-economic aspects and improvements need to be closely managed to counter the potential offset of negative impacts associated with increased economic activity.

Furthermore, rehabilitation activities have also been prescribed which fundamentally aim to ensure that the natural and social environment can progressively be managed in a manner that allows sustainable productive land uses and communities following the completion of mining activities.

20.5. Mitigation Measures

Mitigation measures proposed are cognisant of the mitigation hierarchy which is international best practice for managing risks and impacts, and is listed by the IFC as the primary objective of in IFC PS 1 as follows:

> *To adopt a mitigation hierarchy to anticipate and avoid, or where avoidance is not possible, minimize, and, where residual impacts remain, compensate/offset for risks and impacts to workers, Affected Communities, and the environment.*

This mitigation hierarchy is as shown in Figure 20-1.

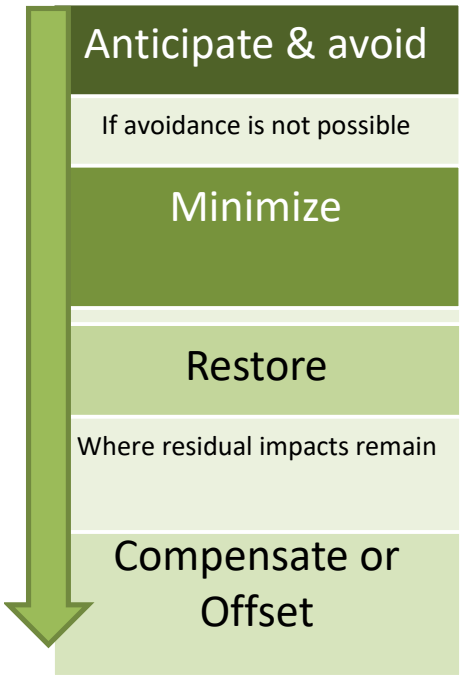


Figure 20-1 The Mitigation Hierarchy as Defined by the IFC

Not all of the potential negative impacts identified can be avoided, and as such mitigation measures have been provided to minimise the significance of the impacts as far as possible. This includes the development and implementation of various action, monitoring, and management plans/procedures which are to be utilised throughout the operations.

An Environmental and Social Management Plan (ESMP) as well as an associated monitoring programme has been developed as a tool which will be utilised to manage and mitigate as far as possible against the identified potential adverse environmental impacts associated with each phase of the proposed Project. The ESMP can also be used as a tool to enhance the possible benefit that can come as a result of the mine development.

Furthermore, it is recommended that the Sofia pit WRD be moved to the east of the pit if possible while the Massawa pit WRD be moved to the west of the pit and out of the natural drainage lines of the systems present. Where possible, remaining infrastructure locations such as the camp area, mine offices, and the plant area (all affecting category B wetlands) should be optimised.

20.6. Rehabilitation and Closure Plan

The preliminary Rehabilitation and Closure Plan for the Project is presented in the report and has been prepared with the aim of ensuring that the area is sustainable in the long term from an environmental and social point of view.

The following points outline the main objectives for rehabilitation and closure at the proposed Project:

- Return impacted land to a sustainable land use in agreement with the current landowner or end land user.
- Remove mining infrastructure that cannot be used by a subsequent land owner or a third party. Where buildings can be used by a third party, agreements must be put in place to ensure their long-term sustainable use.
- Manage the impact of physical effects and chemical contaminants on the environment such that the environmental quality is not adversely affected after closure.
- Follow a process of closure that is progressive and integrated into the short- and long-term plans and that will assess the closure impacts proactively at regular intervals throughout the Project life.
- Start implementing progressive/concurrent rehabilitation measures wherever possible as soon as the operational phase commences.
- Leave a safe and stable environment for both humans and animals and make their condition sustainable.
- Prevent soil, surface water, and groundwater contamination by managing water on site.
- Ensure monitoring and maintenance of vegetation on all rehabilitated areas.
- Comply with national closure and rehabilitation regulatory requirements.
- Follow an appropriate stakeholder engagement process with all interested and affected parties (I&APs) and authorities.

20.7. Project Stakeholders and their Involvement in ESIA

Forty-four meetings were organised with village communities, institutions, technical services, and NGOs between 30th July and 7th September 2018. The presentation was used as a script for the Stakeholder Engagement meetings; however, select meetings were altered to focus the

discussion based on the aspects that most affected the stakeholder. Key concerns raised are summarised as follows:

- The need for local employment and training, with focus on the youth and woman;
- Infrastructure and local economic development and improvement;
- Continued inclusive communication and fulfilment of commitments;
- Concern over potential economic and physical displacement and the impact on livelihoods, particularly orpaillage;
- Protection of the environment:
    - Potential contamination of water sources from mineral and hazardous waste;
    - Management and conservation of biodiversity.

20.8. Concluding Statement

From the information gathered during the ESIA study, it can be concluded that the Project will have adverse environmental and social impacts. The key impacts include direct loss of wetland habitat and the reduction of water quantity particularly downstream of the proposed diversion dam. Impacts associated with the diversion dam include loss of wetland systems (through desiccation and inundation) and ecological functions and loss of aquatic ecology and biodiversity between the Project area and the Niokolo-Koba National Park. In addition, the Project is likely to result in the direct loss and fragmentation of habitat of high conservation value, along with environmental and social impacts associated with potential population influx resulting in increased pressure on natural resources and increased associated social ills in the local area.

The endangered Western Chimpanzee was identified in the Project area. Continued studies are being undertaken to determine the chimpanzee communities, number of individuals and the respective core and home ranges. These baseline studies will determine the level of impact from the Project, as well as the mitigation and management measures required as part of a Biodiversity Action Plan.

The mitigation measures and management plans have been developed which, if correctly implemented and monitored, can reduce the significance of the impacts. Subsequently, successful rehabilitation and sustainable post-closure land uses are possible. Extensive stakeholder engagement has been undertaken and will continue throughout the Project life. The Project is expected to create job opportunities for local communities, assist in social upliftment through community based projects and skills development, as well as contribute to formalising mining activities in the area in a manner that exercises duty of care to the natural and social environment.

For this Project to be successful from an environmental and social perspective, the following key commitments must be made by Barrick:

- Ensure continuous monitoring as per the monitoring programme of the following:
    - Surface water.

  - Groundwater.
  - Dust and air quality.
  - Biodiversity.
  - Soils.
  - Wetlands.
  - Biomonitoring.

- Develop and continuously maintain storm water management structures to ensure effective clean and dirty water separation.
- Develop and implement environmental action plans to management potential adverse impacts and unplanned environmental incidents.
- Undertake training and environmental awareness of mine personnel, contractors, and surrounding communities.
- Manage soil erosion and establishment of alien species timeously.
- Undertake concurrent rehabilitation throughout the Project life where possible and maintain soil stockpiles for final rehabilitation.
- Continuously engage with stakeholders during operations and during closure planning as well as implement a suitable grievance mechanism.
- Report and record all monitoring data which should be utilised to identify areas of potential improvements. Furthermore, periodic internal and external audits of the ESMP should be undertaken, and amendment should be made where necessary in consultation with government authorities.

21. Capital and Operating Costs

21.1. Capital Costs

SENET was commissioned to conduct a capital cost estimate study for the proposed Project.

The capital programme is broken into a number of phases with each phase employing additional processing units and technology to optimally recover the gold content.

- Phase 1: Initial capital to construct the mine and treat oxide and oxidised transition is expended over the first two years of the Project.
- Phase 2: Fresh ore from Sofia and CZ deposits commences. The processing of Sofia fresh ore will continue as per the Phase 1 leach and detoxification process. CZ ore will be processed through a gravity circuit prior to CIL.
- Phase 3: Construction of the BIOX circuit takes place in year six which will be designed to process the sulphide material in the northern part of the CZ, NZ, and Delya pits.

Table 21-1 lists the capital cost estimate by phase.

Table 21-1 Capital Cost Estimate per Phase

Description	Phase 1	Phase 2	Phase 3	Total
Earthworks	13,126,536	1,627,105	2,404,342	17,157,984
Civil Works	9,932,608	2,119,608	5,573,247	17,625,463
Structural Steel	2,501,909	1,127,040	766,695	4,395,644
Platework	528,992	734,510	155,984	1,419,486
Tankage	2,156,710	19,471	6,140,885	8,317,066
Mechanicals	26,406,150	2,855,884	18,811,381	48,073,415
Plant Piping	2,518,523	128,446	1,515,887	4,162,856
Overland Piping	3,548,961		865,204	4,414,165
Valves	847,314	27,872	994,595	1,869,781
Electricals	7,604,341	1,264,083	4,837,523	13,705,946
Instrumentation	3,006,997	178,406	1,459,666	4,645,069
Plant First Fills	1,378,516			1,378,516
Spares - Commissioning	332,429	9,765	247,028	589,222
Spares - Operational	3,069,469	44,328	256,055	3,369,853
Spares - Insurance (Strategic)	2,248,931	43,653	251,158	2,543,742
Vendor Representation	1,430,610		32,697	1,463,307
Transport	8,973,145	752,927	3,246,791	12,972,863
Dismantle Front End at Morila (Excl Mills and Z2)	1,666,200			1,666,200
Sub - Total Direct Field Cost	**91,278,343**	**10,933,099**	**47,559,136**	**149,770,578**
SMPP Erection	16,030,765	3,624,453	10,042,819	29,698,038
Electrical & Instrument Erection	4,992,792	629,539	2,266,757	7,889,088
Total Indirect Field Costs	**21,023,557**	**4,253,992**	**12,309,577**	**187,357,703**
Home Office Costs (Project Management)	12,442,625	1,456,895	5,538,306	19,437,826
Plant Geotech	45,619			45,619
Total Home Office Costs	**12,488,244**	**1,456,895**	**5,538,306**	**19,483,445**
Total Plant Cost	**124,790,144**	**16,643,986**	**65,407,019**	**206,841,148**
Construction Camp	4,362,797			4,362,797
Fuel Depot, LFO Mine Fuel)	3,458,948			3,458,948
Plant Buildings & Infrastructure	7,343,144			7,343,144
Plant Buildings Offsite	726,047			726,047
Power Plant HFO	30,987,458		3,456,350	34,443,808
Fuel Depot (HFO - Power Plant)	5,273,112			5,273,112
Diversion of National Roads	4,770,000			4,770,000

Description	Phase 1	Phase 2	Phase 3	Total
Access Road; Plant Roads and Tails Pipe Trench	2,932,675			2,932,675
Cyanide Safety Equipment	220,357			220,357
Vehicles	2,199,256			2,199,256
Mobile Plant	1,186,869			1,186,869
Equipment and Tools	266,833			266,833
Air Strip	2,753,331			2,753,331
Pollution Control Dam	1,902,371			1,902,371
Perimeter Fencing	2,506,427			2,506,427
Staff Village Ex Pats	5,501,518			5,501,518
Employee Village	4,362,797			4,362,797
Tree Tax	734,552			734,552
Owners Relocation Costs	1,640,011			1,640,011
Preproduction Operation, P & G etc	3,263,495			3,263,495
Owners – IT	3,000,000			3,000,000
WC Lock up	10,000,000			10,000,000
Owners Team	2,662,000			2,662,000
Other Contractors	1,000,000			1,000,000
Tailings	30,464,476			30,464,476
Tailings On-site Soils Lab	823,933			823,933
Diversion Dam Wall	3,545,254			3,545,254
Main Sofia Diversion Channel	2,219,672			2,219,672
Clean Storm water Management around Pits and Waste Dumps	3,324,990			3,324,990
Clean Storm Water Management around Plant and ROM Pad	745,000			745,000
Dirt Storm Water Management	1,662,495			1,662,495
Project Insurances	1,963,309	181,009	744,017	2,888,335
BIOX Licence Fee Construction			800,000	800,000
Duties, Other Taxes on Later Capital			2,541,144	2,541,144
Total Other Costs	**147,803,128**	**181,009**	**7,541,511**	**155,525,648**
Mining Miscellaneous (Survey Equipment, Software, Vehicles etc)	4,647,650			4,647,650
Mining Geotech	1,056,150			1,056,150
Mining Office and Workshop	5,000,000			5,000,000
Mining Mobilisation	1,500,000			1,500,000
Mining Demobilisation			1,500,000	1,500,000
Explosive Magasine Infrastructure	1,500,000			1,500,000
Haul Roads	12,700,000			12,700,000
Haul and Access Road Culvert Crossing	5,748,765			5,748,765
Pit Dewatering	4,950,329	6,525,273	5,065,398	16,541,000
Total Mining Costs	**37,102,894**	**6,525,273**	**6,565,398**	**50,193,565**
Total Construction Capital	**309,696,166**	**23,350,267**	**79,513,928**	**412,560,361**

The estimated LOM capital expenditure is detailed in Table 21-2.

Table 21-2 LOM Estimated Capital Expenditure

Item	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Construction & Project Capital	89,017,300	166,523,217	77,505,917	-	-	-	-	53,009,285	26,504,643	-	-	412,560,361
On-going Capital	-	-	1,112,500	1,112,500	1,112,500	1,112,500	1,112,500	1,712,500	1,712,500	1,712,500	1,662,500	12,362,500
Pre-Production Capitalised	16,933,000	-	-	-	-	-	-	-	-	-	-	16,933,000
Rehabilitation Asset	-	-	-	-	-	-	-	-	-	-	-	23,000,000
Total	**105,950,300**	**166,523,217**	**78,618,417**	**1,112,500**	**1,112,500**	**1,112,500**	**1,112,500**	**54,721,785**	**28,217,143**	**1,712,500**	-	**464,855,861**

21.2. Operating Costs

21.2.1. Mining and Haul Operating Costs

It is planned to contract out the mining services for Massawa. Randgold has used various mining contractors over the last 20 years in West Africa and the first round of the tender process has been completed where four companies have been selected to proceed through to the second and final round. The tenders received have given a good indication of the expected mining cost as well the size of equipment and fleet to be implemented at Massawa. The contractor mining activities at Massawa will include GC drilling, drill and blast, load and haul, and crusher feeding.

The potential of owner mining is not excluded for consideration at a later date, however, for the purposes of the FS, it has been assumed that a mining contractor will do the mining at Massawa. The impact of the owner mining approach is that the open pit operation incurs higher capital costs and reduced operating costs primarily due to the outright purchase of mobile mining equipment and absence of contractor's profit margin and financing cost. Randgold has made a successful transition to owner mining at the Loulo underground operations and therefore has the knowledge and experience to transition to owner mining at Massawa if and when appropriate.

Mining operating costs are based on contractor mining as shown in Table 21-3, Table 21-4, and Table 21-5.

Table 21-3 LOM Material Movement

Item	Units	Total
Ore	kt	17,999
Waste	kt	137,904
Total	kt	155,903

Table 21-4 Breakdown of Contractor Operating Costs

Item Description	Total LOM Cost ($ '000)	Unit Cost ($ /t mined)
Establishment of Contractor's Facilities	5,775	0.04
Mobilisation of Contractor's Equipment	3,360	0.02
Demobilisation of Contractor's Equipment	3,175	0.02
Monthly Management Fee	103,874	0.64
Preparatory Works	1,086	0.01
Load and Haul	281,574	1.72
Drilling and Blasting	134,028	0.82
RC Grade Control Drilling	5,301	0.03
Ore Re-handle	18,377	0.11
Dewatering	3,623	0.02
Owners Costs	20,000	0.12
Total	**580,173**	**3.55**

Table 21-5 Breakdown of Fixed and Variable Operating Costs

Mining Activity	Unit	Cost
Variable	$ '000	463,990
Fixed	$ '000	116,184
Total	**$ '000**	**580,173**
Variable	$/t	2.84
Fixed	$/t	0.71
Total	**$/t**	**3.55**

A total of 3.55 Mt of ore will be hauled from the Sofia pit to the ROM pad at Massawa. The hauling cost has been calculated at $3.20/t hauled. This amounts to a total of $11.35 million over the LOM based upon existing contracts from other Barrick's operating mines in West Africa.

21.2.2. Process Operating Costs

SENET was commissioned to conduct a cost estimate study for the proposed Project. All the deposits were considered: Massawa CZ, Massawa MZ, Sofia Main, Sofia North, and Delya, thus the process operating cost estimate was split by the various ore types as shown in Table 21-6.

Table 21-6 Process Operating Cost Estimate Summary

Rock Type	Central Zone			North Zone		Sofia Main	Sofia North	Delya		
	WOL	BIOX – 100% CZ	BIOX – 25% CZ/ 75% NZ	WOL 100% NZ	BIOX 100% NZ	WOL	WOL	WOL 100% Delya	BIOX 100% Delya	BIOX 20% Delya 80% NZ
Oxide ($/t milled)	12.59	N/A	N/A	12.90	N/A	12.46	13.39	13.31	N/A	N/A
Oxide Trans ($/t milled)	16.44	N/A	N/A	15.50	N/A	14.97	14.86	15.88	N/A	N/A
Reduced Trans ($/t milled)	17.95	57.05	41.52	16.71	40.37	16.41	15.75	16.17	62.41	44.25
Fresh $/t milled)	19.58	57.05	41.52	17.93	40.37	17.84	16.48	17.81	62.41	44.25

The process operating costs were generated based on the various process routes possibly required.

21.2.3. G&A Costs

General and administration (G&A) costs cover all operational costs outside of mining and processing. They include administrative, finance, medical, environmental, and social costs, together with outside engineering which covers all engineering costs outside of the processing plant. Table 21-7 is a summary of the G&A costs estimated for the first six years, followed by the final three years of production at Massawa. The model equates to $8.60/t of ore processed over the LOM. This correlates well on a gross level to Barrick's Tongon Mine, which is also a West African coastal country operation.

Table 21-7 G&A Costs Over the Life of Mine

Item	Year 1 -6 ($)	Year 7 – 10 ($)
General Management	1,871,597	1,122,958
Site Administration	7,833,609	5,483,526
Finance	1,050,525	735,367
Supply Chain Management	940,452	705,339
Human Resources	402,940	322,352
Safety, Health and Environment	1,619,129	1,295,303
Social	446,812	446,812
Stores - Electrical & Mechanical	398,488	239,093
Labour	1,413,483	848,090
General Costs	2,212,068	1,548,448
Equipment Hire	206,196	144,337
Consultants	5,400	3,780
Power	371,798	223,079
Stores - Other	664,600	398,760
Total	**19,437,096**	**13,517,244**

22. Economic Analysis

22.1. Basis of the Economic Analysis

An economic assessment to confirm the reserve status of the Massawa, Sofia, and Delya pits was carried out based on the key parameters summarised below:

- Total ore mined from Massawa, Sofia Main, and Delya pits of 18 Mt of ore at an average grade of 4.2 g/t Au containing 2.4 Moz of gold.
- Strip ratio of 7.6:1 to give total tonnes mined of 156 Mt.
- Mining costs average of $3.55/t mined over the LOM.
- Haulage cost average of $1.20/t of ore milled over the LOM ($0.18/t km hauled).
- Plant cost average of $21.33/t ore but includes a range of costs dependant on ore feed and process route.
- G&A costs of $8.76/t ore milled over the LOM, including outside engineering costs.
- Pre-production related capital amounts $16.9 million. $80 million will be spent on pre-production mining and absorbed in Total Cash Cost as the ore is fed.
- Capital construction cost of $413 million.
- On-going capital of $12 million over the LOM.
- Rehabilitation cost of $23 million at the end of the LOM.

The financial model is based on annual cash flow projections, with technical and economic parameters stated above using constant money terms. No escalation or de-escalation has been applied. In generating the financial model for the operations and project, the following principles were adopted:

- Financial implication on the methods of funding was not evaluated, since it has been assumed the Project will be financed by Barrick.
- Annual figures are based on financial years 1st January to 31st December.
- Real term annual cash flows were used to calculate the internal rate of return (IRR), net present values (NPV), and simple and discounted payback periods in real after-tax terms.
- Costs up to start of construction are considered as sunk costs.
- No salvage value for plant and equipment on cessation of operations was included.
- Calculations are based in US dollars ($).

22.2. Production and Cash Flow Forecast

The estimated production and cash flow forecast are summarised in Table 22-1.

Table 22-1 Production and Cash Flow Forecast

Item	Year											
	-1	0	1	2	3	4	5	6	7	8	9	Total
Production (koz)	-	-	226	257	295	201	212	211	159	173	158	1,893
Cash Flow ($ Million)	(106)	(247)	(3)	109	154	62	80	20	31	79	79	258

22.3. Financial Analysis

A financial model was run using a $1,000/oz gold price feeding the reserve mining schedule, together with a 3% royalty on revenue, seven-year tax holiday (two year construction, five years for operation), followed by corporate tax at a 25% rate, which produced a total net cash flow after tax of $258 million, and IRR of 12%. Payback is five years from start of production. A sensitivity table on NPV, IRR, and payback against gold price is supplied in Table 22-2. The Project is profitable at $1,000/oz and thus justified to be reported as an Ore Reserve at $1,000/oz gold price.

Table 22-2 Project Financial Analysis

Discount Rate	Gold Price ($/oz)		
	1,000	1,200	1,400
0%	$258 million	$591 million	$925 million
5%	$114 million	$361 million	$608 million
10%	$24 million	$212 million	$400 million
IRR	12%	25%	37%
Payback	5 years	2.8 years	2.4 years

22.4. Government Revenue

The government revenue earnings from the Project are sourced from the following assumptions incorporated into the financial model:

- 3% royalty on revenue;
- Tax rate of 25%. Payments are made throughout the year after the seven-year tax holiday from issuance of the mining permit.
- Dividends from 10% free carry share of the Project, which are payable after capital has been redeemed.
- Other taxes which include withholding taxes on dividends and salaries.

Under the current Kounemba Convention, Barrick will be exempt from all taxes, levies, and duties for a period of seven years from issuance of the mining title.

A sensitivity of expected government revenue based on the cash flow is detailed in Table 22-3.

Table 22-3 Government Revenue Sensitivity

Revenue	Unit	Gold Price ($/oz)	
		1,000	1,250
Royalty	$ million	57	71
Company Tax	$ million	14	56
Dividends	$ million	21	59
Other Tax ($M)*	$ million	62	105
Total	**$ million**	**154**	**291**

*Other taxes include withholding taxes on dividends and salary taxes.

22.5. Sensitivity Analysis

22.5.1. Discount Rate and Gold Price

The proposed Project is profitable at current gold prices, but it is sensitive to gold price fluctuations and therefore becomes a marginal project at gold prices lower than $1,000/oz, but very attractive at current or higher gold prices (Table 22-4).

Table 22-4 NPV Sensitivity at Different Gold Prices and Discount Rates

US millions	Gold Price ($/oz)						
Discount	**900**	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
0%	84	258	425	591	758	925	1,091
5%	(14)	114	238	361	484	608	731
10%	(73)	24	118	212	306	400	493
15%	(108)	(33)	40	113	186	259	332
20%	(129)	(70)	(12)	46	104	162	220
25%	(141)	(93)	(46)	0	47	94	140

22.5.2. Grade and Gold Price

The Project is fairly resilient to grade changes and at current gold prices would be able to absorb a 10% reduction in grade (Table 22-5). The Project is sensitive to gold price fluctuations with a $1,000/oz and a 20% reduction in grade making the Project very marginal. Conversely, at higher gold prices than current, the Project is very resilient to a grade reduction.

Table 22-5 NPV at 20% Grade Variation at Different Gold Prices

Grade	$ millions	Gold Price ($/oz)						
		900	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
3.35	-20%	(246)	(100)	47	189	324	457	591
3.77	-10%	(81)	84	241	391	541	691	841
4.19	**0%**	**84**	**258**	**425**	**591**	**758**	**925**	**1,091**
4.60	10%	242	425	608	792	975	1,158	1,341
5.02	20%	392	592	792	992	1,192	1,392	1,592

22.5.3. Operating Cost and Gold Price

At current gold prices, the Project can absorb a 20% increase in operating costs and still remains largely profitable (Table 22-6). The Project will require a gold price of less than $1,000/oz and an increase of 20% in operating costs to become a marginal project.

Table 22-6 NPV at 20% Change in Operating Costs

Operating Cost		Gold Price ($/oz)						
Operating Cost ($/t)	**$ millions**	**900**	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
49.57	-20%	280	447	613	780	946	1,113	1,280
55.77	-10%	184	352	519	686	852	1,019	1,185
61.97	**0%**	**84**	**258**	**425**	**591**	**758**	**925**	**1,091**
68.16	10%	(19)	162	330	497	664	830	997
74.36	20%	(123)	61	235	403	569	736	903

22.5.4. Capital and Gold Price

The Project is sensitive to increases in capital cost (Table 22-7). At a $1,000/oz gold prices, an increase of 10% on capital reduces the NPV by $42 million, or 16% of the NPV.

Table 22-7 LOM Capital Cost Sensitivity

Capital $ millions	Gold Price ($/oz)						
	900	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
-15%	150	318	485	652	818	985	1,151
-10%	129	298	465	631	798	965	1,131
-5%	107	278	445	611	778	945	1,111
0%	84	258	425	591	758	925	1,091
10%	38	217	385	551	718	884	1,051
25%	(30)	153	324	491	658	824	991
50%	(143)	40	221	391	557	724	890

23. Adjacent Properties

23.1. Sabodala Gold Mine

The Sabodala Gold Mine, owned by Teranga, is located adjacent to the Kanoumba Permit, approximately 20 km NW of Massawa (Figure 23-1). Sabodala reported June 2017 Proven and Probable Open Pit Reserves, including stockpiles, of 59.5 Mt at 1.23 g/t Au for 2.36 Moz of contained Au, plus Probable Underground Reserves of 2.15 Mt at 5.01 g/t Au for 0.35 Moz. Measured and Indicated Resources were reported at 86.0 Mt at 1.59 g/t Au for 4.44 Moz of contained Au, and an Inferred Resource of 17.25 Mt at 1.81 g/t Au for 1.0 Moz (Teranga, RPA Report August 2017). Resources are inclusive of reserves. These resources and reserves mainly consist of open pit, although underground and stockpile material are also included in the total. The Sabodala mining concession consists of multiple ore bodies, although to date the majority of the treated ore has been sourced from the Sabodala, Masato, Gora, and Golo0075ma deposits.

The Sabodala deposits are located along the same first-order shear zone (Sabodala Shear Zone) that hosts the Sofia deposit approximately 20 km to the south. The shear zone is approximately 2 km wide and is identified by a subtle magnetic trend that extends through the 7 km long permit and transects the Mako belt stratigraphy. Similar to Sofia, deposit geology consists of andesitic volcanics and volcaniclastics, gabbro sills, and quartz-feldspar sills/dykes with gold mineralization spatially linked to the intrusive rocks.

23.2. Makabingui Gold Project

The Makabingui exploration project, owned by Bassari Resources Limited (Bassari), is located approximately 25 km NE of Massawa Figure 23-1. Makabingui currently hosts an open pit Mineral Resource which comprises 11.9 Mt at 2.6 g/t Au for a contained 1 Moz of gold, and a maiden Ore Reserve of 0.9 Mt at 5.7 g/t Au for 158 koz Au (Bassari, 2016). Regionally, Makabingui is located in the Diale-Dalema sedimentary basin to the east of the Main Transcurrent Shear Zone which hosts Massawa. The deposit is hosted in gabbros in a pressure shadow along the southern margin of the Sambarabougou Granite. Exploration is also focussed on a NE trending structural zone termed the Lafia Shear Zone which is situated both to the NE and SW of Makabingui.

23.3. Mako Project

The Mako project is located to the south of the Kanoumba Permit (Figure 23-1), and is owned by private exploration and development company Toro Gold Limited (Toro Gold). Toro Gold reported a 2016 Proven and Probable Mineral Reserve of 14.2 Mt at 2.25 g/t Au for 1 Moz and Measured, Indicated, and Inferred Mineral Resources of 22.93 Mt at 1.86 g/t Au for 1.37 Moz. Production started at the Mako project in January 2018.

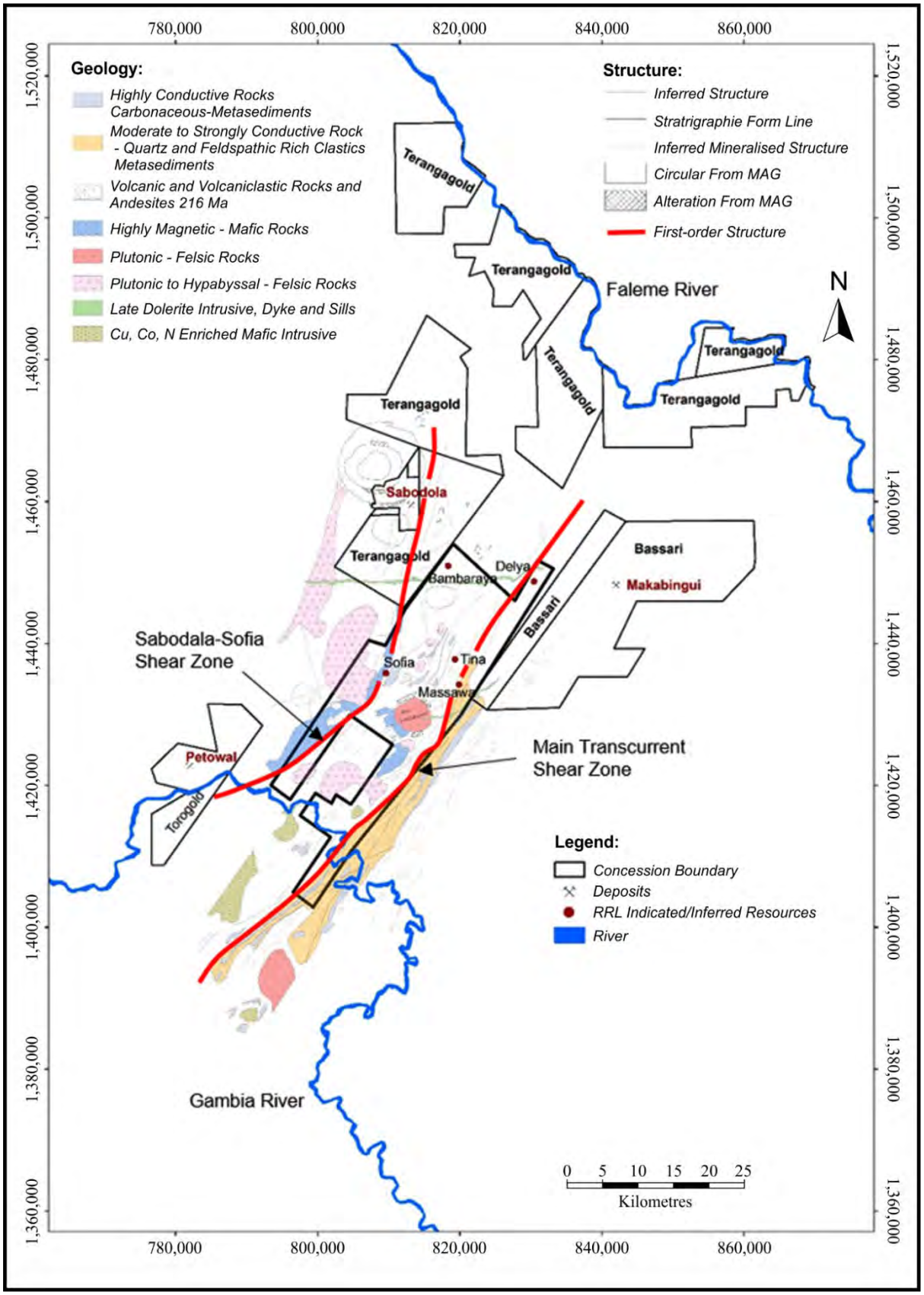


Figure 23-1 Regional Map Showing Location of Adjacent Properties

Barrick has not independently verified the information on adjacent properties and this information is not necessarily indicative of the mineralisation at Massawa.

24. Other Relevant Data and Information

24.1. Project Implementation

24.1.1. Implementation Strategy

The FS capital cost estimate has been compiled on the basis that Barrick will adopt an Engineering, Procurement, and Construction Management (EPCM) approach to implementing and executing the Project.

Upon the onset of the Project and following a formal EPCM Contractor tender, selection, and award process to be conducted by Barrick, some 1st Phase early project execution activities may take place such as front-end engineering and design (FEED) of critical deliverables, which can commence in the first quarter of 2019 as Project approval is expected from both the Senegal government and the Barrick Board before June 2019.

The implementation strategy to be adopted is generally structured into four broad stages undertaken by the successful EPCM Contractor under the direct auspices of Barrick:

- Detailed design of the process plant and infrastructure whereby the FS capital cost estimate (CCE) is further developed into a Barrick approved and EPCM controlled budget estimate (CBE) for the Project going forward during execution.
- Procurement comprising formal tender, adjudication, award and thereafter fabrication and expediting with logistics undertaken by the Barrick nominated logistics supplier.
- Construction management and cost control.
- Commissioning management and handover to operations.

24.1.2. Implementation Budget Control

Budget control will be performed by the nominated EPCM's Cost Controller with the EPCM Project Manager being ultimately responsible for the accuracy and control of the approved budget. Cost control at the construction site will be done by the site based Cost Controller supported by the Quantity Surveyors.

All orders will be signed off by the EPCM Cost Controller, before being signed off by the EPCM Project Manager and approved by Barrick before processing further.

24.1.3. Implementation Teams

The project implementation teams are sub-divided as follows:

- Head Office Based EP Team.
- Site Based Barrick and EPCM Team.

24.1.4. Head Office Based EP Contractor Team

The nominated EPCM Contractor will manage its own head office based engineering and procurement (EP) team, including contract management with direct input from Barrick Owners Team as required for the Project. The scope of services for the EP works shall include the following:

- Detailed engineering design.
- Project engineering and drawings.
- QA/QC.
- Cost management.
- Procurement, contracts administration and management of subcontractors.
- Quality inspections and expediting (logistics/shipping by others).
- Project planning and scheduling.
- Project reporting (to include progress and performance measurement and any scheduled or event-driven risks).
- Document control systems.
- Project standards, specifications, and systems.
- Recommendation and procurement of mechanical spare parts, consumables, reagents, and lubrication first fills.

24.1.5. Site Based Barrick and EPCM Contractor Team

The site based Barrick Owners Team and EPCM Contractor will jointly perform the construction and commissioning management as required for the Project. The scope of services for the CM works shall include the following:

- Site QA/QC;
- Site cost management;
- Site project planning and scheduling;
- Site project reporting (to include progress and performance measurement and any scheduled or event-driven risks);
- Site document control systems;
- Site materials control;
- Site construction management;
- Snagging, commissioning, and final handover to operations.

24.1.6. Implementation Schedule

The Project will be executed over three different phases at estimated durations apart from one another. The various project execution schedules reflect the work required from detailed design and engineering, procurement, construction, and commissioning of the following work packages:

Phase 1:

- Metallurgical process plant.
- Morila plant relocation and refurbishment.
- Raw and process water ponds.
- Airstrip.

Phase 2:

- Secondary crushing and stockpiling.
- Pre-leach thickening, tails thickening and process water clarifier.
- Morila plant relocation and refurbishment.

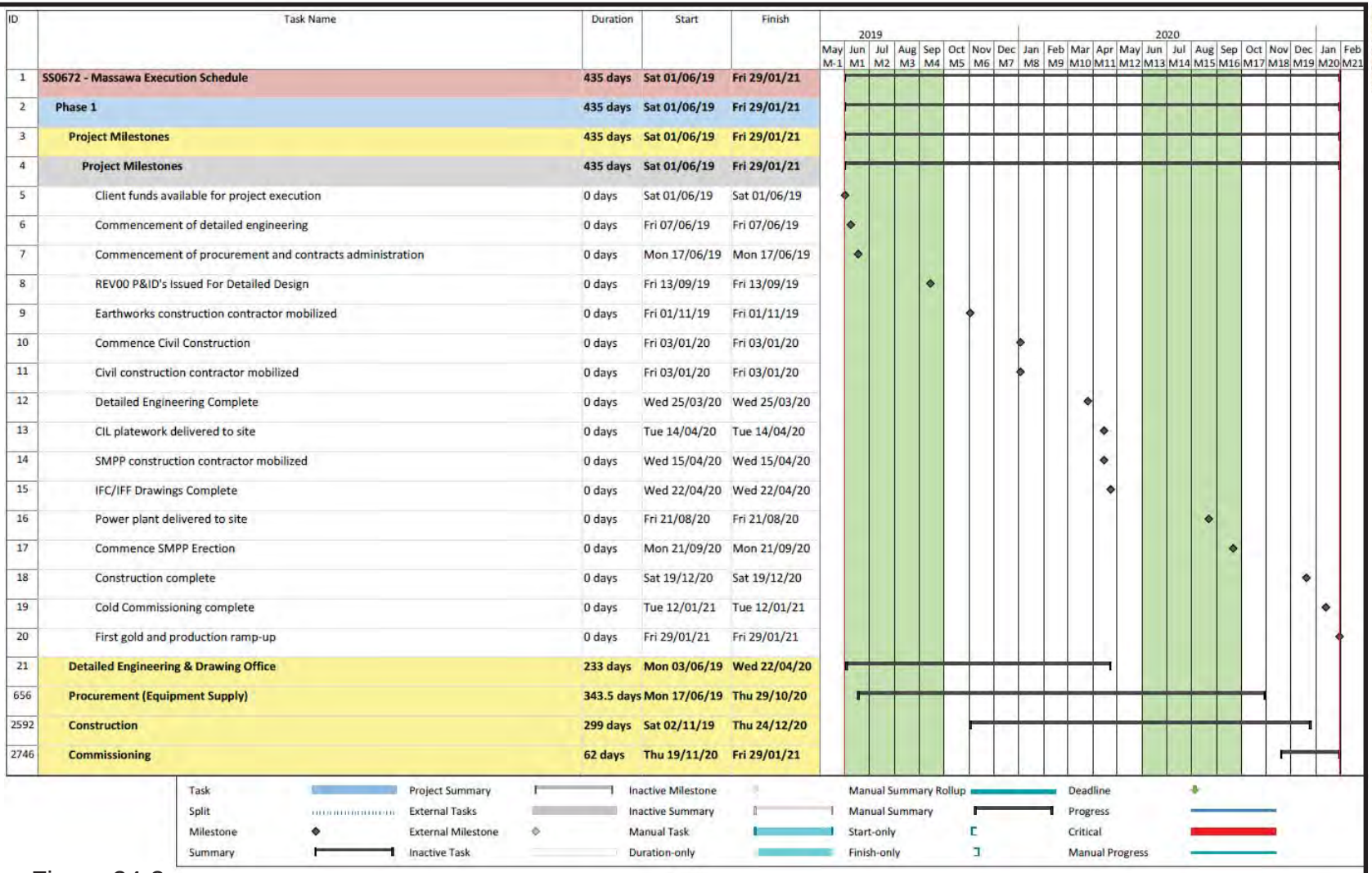
Phase 3:

- BIOX circuit and associated infrastructure.

The work packages are detailed in the Project Work Breakdown Structure (WBS) with final handover to Barrick to start with operation and production.

The project schedules assume that there will be a seamless advancement of the Project between the various phases of the Project evolution. It is also recognised that the schedules are very aggressive and that it will require diligent progress and coordination of all the parties involved.

The summarised project schedules for the various phases are shown in Figure 24-1, Figure 24-2, and Figure 24-3 respectively.



ID	Task Name	Duration	Start	Finish
1	**SS0672 - Massawa Execution Schedule**	**435 days**	**Sat 01/06/19**	**Fri 29/01/21**
2	**Phase 1**	**435 days**	**Sat 01/06/19**	**Fri 29/01/21**
3	**Project Milestones**	**435 days**	**Sat 01/06/19**	**Fri 29/01/21**
4	**Project Milestones**	**435 days**	**Sat 01/06/19**	**Fri 29/01/21**
5	Client funds available for project execution	0 days	Sat 01/06/19	Sat 01/06/19
6	Commencement of detailed engineering	0 days	Fri 07/06/19	Fri 07/06/19
7	Commencement of procurement and contracts administration	0 days	Mon 17/06/19	Mon 17/06/19
8	REV00 P&ID's Issued For Detailed Design	0 days	Fri 13/09/19	Fri 13/09/19
9	Earthworks construction contractor mobilized	0 days	Fri 01/11/19	Fri 01/11/19
10	Commence Civil Construction	0 days	Fri 03/01/20	Fri 03/01/20
11	Civil construction contractor mobilized	0 days	Fri 03/01/20	Fri 03/01/20
12	Detailed Engineering Complete	0 days	Wed 25/03/20	Wed 25/03/20
13	CIL platework delivered to site	0 days	Tue 14/04/20	Tue 14/04/20
14	SMPP construction contractor mobilized	0 days	Wed 15/04/20	Wed 15/04/20
15	IFC/IFF Drawings Complete	0 days	Wed 22/04/20	Wed 22/04/20
16	Power plant delivered to site	0 days	Fri 21/08/20	Fri 21/08/20
17	Commence SMPP Erection	0 days	Mon 21/09/20	Mon 21/09/20
18	Construction complete	0 days	Sat 19/12/20	Sat 19/12/20
19	Cold Commissioning complete	0 days	Tue 12/01/21	Tue 12/01/21
20	First gold and production ramp-up	0 days	Fri 29/01/21	Fri 29/01/21
21	**Detailed Engineering & Drawing Office**	**233 days**	**Mon 03/06/19**	**Wed 22/04/20**
656	**Procurement (Equipment Supply)**	**343.5 days**	**Mon 17/06/19**	**Thu 29/10/20**
2592	**Construction**	**299 days**	**Sat 02/11/19**	**Thu 24/12/20**
2746	**Commissioning**	**62 days**	**Thu 19/11/20**	**Fri 29/01/21**

Figure 24-3

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Phase 1 Project Schedule Summary

July 2019

Source: SENET, 2018.

ID	Task Name	Duration	Start	Finish
1	**SS0672 - Massawa Execution Schedule**	**269 days**	**Mon 02/08/21**	**Thu 11/08/22**
2	**Phase 2**	**269 days**	**Mon 02/08/21**	**Thu 11/08/22**
3	**Project Milestones**	**269 days**	**Mon 02/08/21**	**Thu 11/08/22**
4	**Project Milestones**	**269 days**	**Mon 02/08/21**	**Thu 11/08/22**
5	Client funds available for project execution	0 days	Mon 02/08/21	Mon 02/08/21
6	Commencement of detailed engineering	0 days	Tue 03/08/21	Tue 03/08/21
7	Commencement of procurement and contracts administration	0 days	Mon 30/08/21	Mon 30/08/21
8	REV00 P&ID's Issued For Detailed Design	0 days	Mon 13/09/21	Mon 13/09/21
9	Earthworks construction contractor mobilized	0 days	Mon 01/11/21	Mon 01/11/21
10	Detailed Engineering Complete	0 days	Mon 15/11/21	Mon 15/11/21
11	IFC/IFF Drawings Complete	0 days	Wed 22/12/21	Wed 22/12/21
12	Civil construction contractor mobilized	0 days	Tue 18/01/22	Tue 18/01/22
13	SMPP construction contractor mobilized	0 days	Thu 17/02/22	Thu 17/02/22
14	Commence SMPP Erection	0 days	Thu 17/02/22	Thu 17/02/22
15	Commence Civil Construction	0 days	Mon 21/02/22	Mon 21/02/22
16	Thickeners platework delivered to site	0 days	Thu 03/03/22	Thu 03/03/22
17	Construction Complete	0 days	Mon 25/07/22	Mon 25/07/22
18	Cold Commissioning complete	0 days	Fri 05/08/22	Fri 05/08/22
19	Phase 2 Ready For Product	0 days	Thu 11/08/22	Thu 11/08/22
20	**Detailed Engineering & Drawing Office**	**103 days**	**Mon 02/08/21**	**Wed 22/12/21**
202	**Procurement (Equipment Supply)**	**230 days**	**Mon 02/08/21**	**Fri 17/06/22**
490	**Construction**	**190 days**	**Tue 02/11/21**	**Mon 25/07/22**
515	**Commissioning**	**31 days**	**Thu 07/07/22**	**Thu 11/08/22**

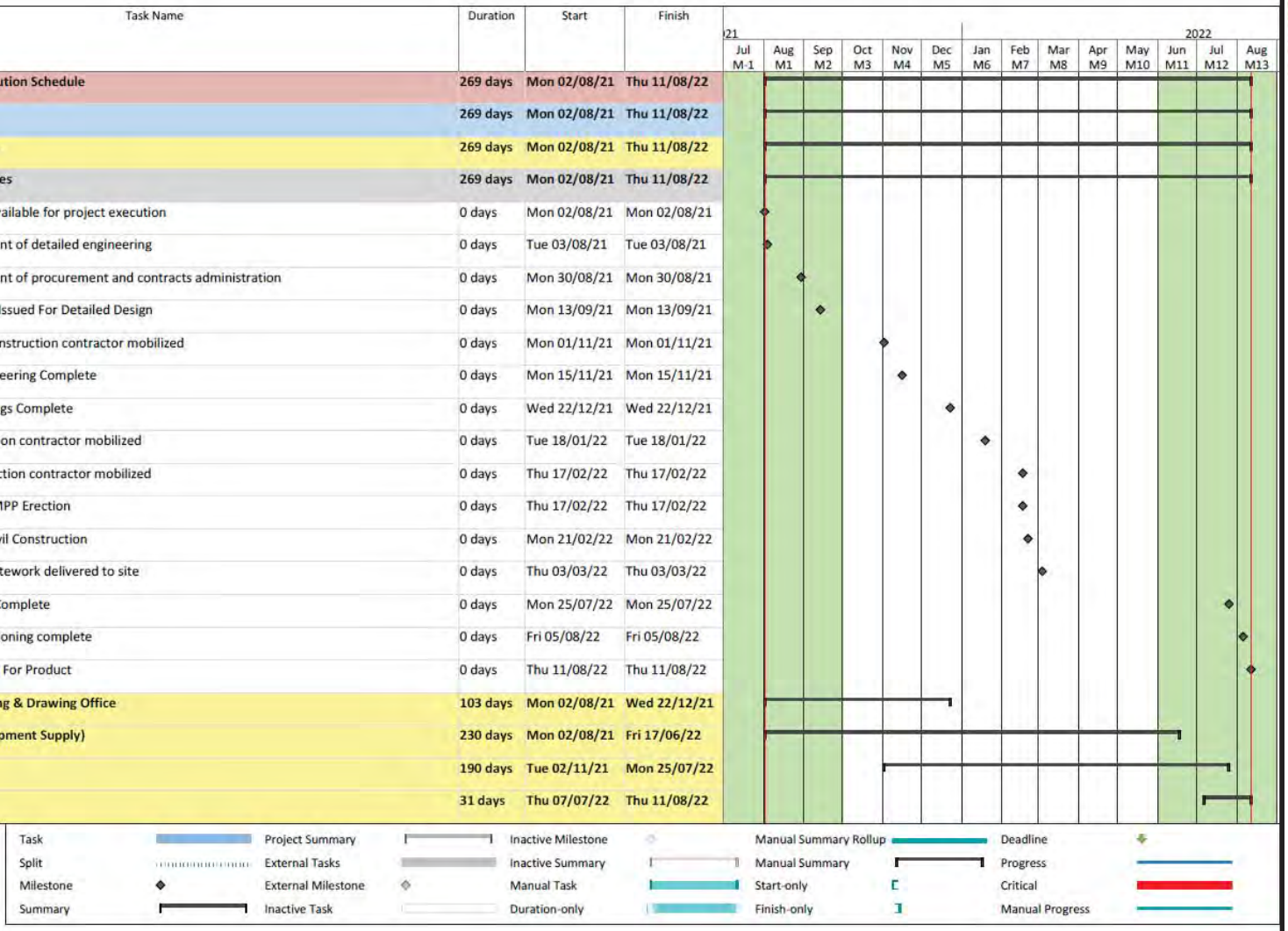


Figure 24-4

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Phase 2 Project Schedule Summary

July 2019

Source: SENET, 2018.

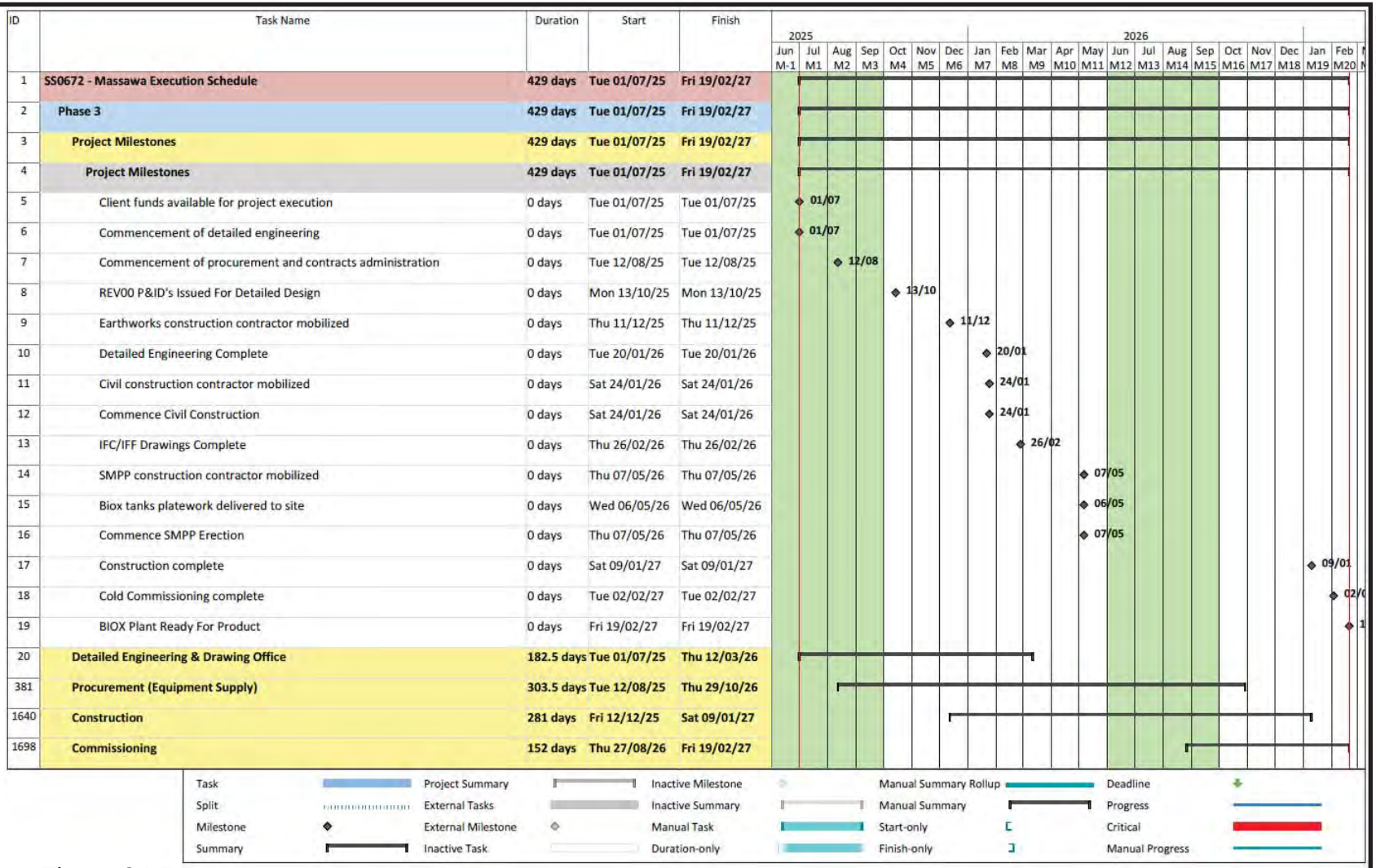


Figure 24-5

Barrick Gold Corporation

Massawa Gold Project

Sénégal

Phase 3 Project Schedule Summary

July 2019

Source: SENET, 2018.

24.2. Alternate Case - Ore Reserves and Economics at $1,200/oz Gold Price

24.2.1. Ore Reserve Summary at $1,200/oz Gold Price

Historically Barrick estimates its Ore Reserves using a gold price assumption of $1,200/oz. In order to investigate the effect on the Massawa Gold Project of using a $1,200/oz gold price, an alternate case has been prepared which results in the following upsides to the Project:

- A reduction in the cut-off grade results in an increase in the total Ore Reserves at a slightly lower grade, thus extending the mine life by 1.5 years.
- The size of the Massawa, Sofia, and Delya pits is increased by 6%.
- The increase in the total Ore Reserves results in a more efficient use of capital as the capital construction cost is expended over more tonnes.

For the alternate case the Ore Reserve was rerun at a gold price of $1,200/oz. No changes have been made to the mining dilution, ore loss, or gold recovery between the base case and the alternate case. For the alternate case, the cut-off grade for each of the deposits was re-estimated based on the higher gold price. A life of mine plan for the alternate case was determined based on the $1,200/oz Ore Reserve.

The 2018 Ore Reserve estimate for the alternate case at a $1,200/oz gold price includes an open pit (OP) Probable Ore Reserve of 7.8 Mt at 4.6 g/t Au for 1.2 Moz Au from the Massawa CZ; 5.2 Mt at 4.7 g/t Au for 0.8 Moz Au from the Massawa NZ; 7.1 Mt at 2.7 g/t Au for 0.6 Moz Au for Sofia; and 0.81 Mt at 4.21 g/t Au for 0.11 Moz for Delya. The OP Ore Reserves are those reserves occurring within a $1,200/oz pit design.

Total Massawa, Sofia, and Delya Ore Reserve estimates, as of 31st December 2018 within a $1,200/oz OP, are presented in Table 24-1.

Table 24-1 Massawa, Sofia, and Delya Ore Reserves as at 31st December 2018 at $1,200/oz Gold Price

Ore Reserve	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Attributable Gold (Moz)*
CZ Probable	7.8	4.59	1.15	0.96
NZ Probable	5.2	4.67	0.79	0.65
Sofia Probable	7.1	2.66	0.61	0.51
Delya Probable	0.81	4.21	0.11	0.091
Total OP Probable	**20.9**	**3.94**	**2.6**	**2.2**

*Attributable gold (Moz) refers to the quantity attributable to Barrick based on Barrick's 83.25% interest in the Massawa Project. Open pit Ore Reserves are reported at a gold price of $1,200/oz and include dilution and ore loss factors. Open pit Ore Reserves were generated by Shaun Gillespie, an employee of the company Table 24-1, under the supervision of Rodney Quick, MSc, Pr Sci Nat, an officer of the company and Qualified Person.

A financial model was run on the alternate case using a $1,200/oz gold price feeding the alternative case reserve mining schedule, together with a 3% royalty on revenue, seven-year tax holiday (two year construction, five years for operation), followed by corporate tax at a 25% rate, which produced a total net cash flow after tax of $696 million, and IRR of 28%. Payback is 2.5

years from start of production. Table 24-2 compares the base case financial model run at a $1,000/oz gold price versus the alternate case financial model run at a $1,200/oz gold price.

Table 24-2 $1,000/oz Gold Price Base Case Results versus Alternate Case $1,200/oz Gold Price

Item	Units	Base Case	Alternate Case
Gold Price	US$/oz	1,000	1,200
Total Ore Mined	Mt	18	21
Grade	g/t Au	4.19	3.92
Contained Gold	Moz	2.4	2.6
Total tonnes Mined	Mt	156	166
Strip Ratio	w:o	7.6	6.9
Mining Cost	$/t mined	3.55	3.55
Haulage Cost Average	$/t milled	1.20	1.25
Plant Costs	$/t milled	21.33	18.02
G&A Costs	$/t milled	8.76	8.60
Pre-production Capital	$ million	16.9	0
Capital Construction Cost	$ million	413	413
On-Going Capital	$ million	12	16
Rehabilitation Cost	$ million	23	23
Net After-Tax Cash Flow	$ million	258	696
$NPV_{5\%}$	$ million	114	421
$NPV_{10\%}$	$ million	24	251
IRR	%	12	28
Payback	Years	5	2.5

24.2.2. Dilution

The mining dilution was assumed to be 10% of additional waste for each tonne of ore in Massawa NZ, Sofia, and Delya Main and 36% in Massawa CZ. The CZ dilution and ore loss applied are based on a dilution study carried out by Maptek taking into account the very narrow nature of the CZ ore body. The 10% figure is based on experience with similar steep tabular ore bodies at Barrick's Loulo operation.

Ore loss was set at 3% for NZ, Sofia, and Delya and 8% for CZ, which is also based on historical information from nearby operations with similar geology.

24.2.3. Cut-Off Grade

To enable an improved cash flow, a full grade ore classification was implemented, with preferential treatment for high-grade ore. Marginal ore is treated only at the end of the mine life after being stockpiled, when mining has stopped.

The cut-off grade, g, is where revenue from the gold produced equals the cost of treating ore that includes the cost of refinery and shipping. Cut-off grades are calculated below as:

$$g = c \div r \times p$$

Where; **g** = cut-off grade

c = total operating costs $ /t treated = mining cost + metallurgical cost + G&A costs

r = metallurgical recovery

p = selling price $ /g = (gold price – (gold price x royalty)) ÷ 31.10348.

Massawa North Zone

The cut-off grade calculation for Massawa NZ Ore Reserve and all associated costs are broken down in Table 24-3.

Table 24-3 Massawa North Zone Ore Reserve Cut-Off Grade Calculation at $1,200/oz Gold Price

Parameter	Unit	Symbol	2018
Gold Price	$/oz	Gp	1,200
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	1,164
Met Recovery	%	Rec	83%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	11.37
G&A	$ /t milled	G_A	8.60
Mining	$ /t milled	Cr	40.20
Process Plant	$ /t milled	Cp	22.57
Maintenance/Engineering	$ /t milled	Mp	
Total Operating Costs	**$/t**		**70.97**
Full Grade Cut-off	**g/t Au**	**FGO**	**2.36**
Marginal Cut-off Grade	**g/t Au**	**MO**	**1.02**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**2.14**
		MO	**0.93**

The processing costs for Massawa NZ Mineral Resources are set materially higher than in other Mineral Resources to reflect the BIOX process treatment costs.

Massawa Central Zone

The cut-off grade calculation for Massawa CZ Ore Reserve and all associated costs are broken down in Table 24-4.

Table 24-4 Massawa Central Zone Ore Reserve Cut-off Grade Calculation for $1,200/oz Gold Price

Parameter	Unit	Symbol	2018
Gold Price	$/oz	Gp	1,200
Royalty		R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	1,164

Parameter	Unit	Symbol	2018
Met Recovery	%	Rec	76%
Dilution	%	Dil	36%
Ore Loss	%	Loss	8%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	5.84
G&A	$ /t milled	G_A	8.60
Mining	$ /t milled	Cr	22.37
Process Plant	$ /t milled	Cp	15.75
Maintenance/Engineering	$ /t milled	Mp	
Total Operating Costs	**$/t**		**46.32**
Full Grade Cut-off	**g/t Au**	**FGO**	**1.77**
Marginal Cut-off Grade	**g/t Au**	**MO**	**0.92**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**1.45**
		MO	**0.75**

Sofia

The cut-off grade calculation for Sofia Ore Reserve and all associated costs are broken down in Table 24-5.

Table 24-5 Sofia Ore Reserve Cut-off Grade Calculation for $1,200/oz Gold Price

Parameter	Unit	Symbol	2018
Gold Price	$/oz	Gp	1,200
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	1,164
Met Recovery	%	Rec	89%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	4.75
G&A	$ /t milled	G_A	8.60
Mining	$ /t milled	Cr	18.86
Process Plant	$ /t milled	Cp	16.65
Maintenance/Engineering	$ /t milled	Mp	
Total Operating Costs	**$/t**		**46.96**
Full Grade Cut-off	**g/t Au**	**FGO**	**1.45**
Marginal Cut-off Grade	**g/t Au**	**MO**	**0.87**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**1.32**
		MO	**0.79**

Delya

The cut-off grade calculation for Delya Ore Reserve and all associated costs are broken down in Table 24-6.

Table 24-6 Delya Reserve Cut-off Grade Calculation for $1,200/oz Gold Price

Parameter	Unit	Symbol	2018
Gold Price	$/oz	Gp	1,200
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	1,164
Met Recovery	%	Rec	91%
Dilution	%	Dil	10%
Ore Loss	%	Loss	3%
Mining Cost - Contractor	$ /t mined	MCC	3.35
Mining Cost - Owner's	$ /t mined	MCO	0.06
Mining Cost - Grade Control	$ /t mined	MCGC	0.14
Total Mining Cost	$ /t mined	TMC	3.55
Strip Ratio	Waste/Ore	SR	8.55
G&A	$ /t milled	G_A	8.60
Ore Crushing & Hauling	$ /t milled	Rd	4.20
Mining	$ /t milled	Cr	31.61
Process Plant	$ /t milled	Cp	21.82
Maintenance/Engineering	$ /t milled	Mp	
Total Operating Costs	**$/t**		**65.83**
Full Grade Cut-off	**g/t Au**	**FGO**	**2.04**
Marginal Cut-off Grade	**g/t Au**	**MO**	**1.06**
Diluted Cut-off Grades	**g/t Au**	**FGO**	**1.85**
		MO	**0.96**

24.2.4. Life of Mine Capital Expenditure at $1,200/oz Gold Price

The estimated LOM capital expenditure at a gold price of $1,200/oz is detailed in Table 24-7.

Table 24-7 LOM Estimated Capital Expenditure at $1,200/oz Gold Price

Item	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Construction & Project Capital	89,017,300	166,523,217	77,505,917	-	-	-	-	53,009,285	26,504,643	-	-	-	-	412,560,361
On-going Capital	-	-	1,112,500	1,112,500	1,112,500	1,112,500	1,112,500	1,712,500	1,712,500	1,712,500	1,712,500	1,662,500	1,662,500	15,737,500
Pre-Production Capitalised		-	-	-	-	-	-	-	-	-	-	-	(10,000,000)	(10,000,000)
Rehabilitation Asset	-	-	-	-	-	-	-	-	-	-	-	-	23,000,000	23,000,000
Total	**89,017,300**	**166,523,217**	**78,618,417**	**1,112,500**	**1,112,500**	**1,112,500**	**1,112,500**	**54,721,785**	**28,217,143**	**1,712,500**	**1,712,500**	**1,662,500**	**14,662,500)**	**441,297,861**

24.2.5. Operating Costs at $1,200/oz Gold Price

Mining and Haul Operating Costs

Mining operating costs at a $1,200/oz gold price are based on contractor mining as shown in Table 24-8, Table 24-9, and Table 24-10.

Table 24-8 LOM Material Movement at $1,200/oz Gold Price

Item	Units	Total
Ore	kt	20,927
Waste	kt	144,817
Total	kt	165,744

Table 24-9 Breakdown of Contractor Operating Costs at $1,200/oz Gold Price

Item Description	Total LOM Cost ($ '000)	Unit Cost ($ /t mined)
Establishment of Contractor's Facilities	5,856	0.04
Mobilisation of Contractor's Equipment	3,407	0.02
Demobilisation of Contractor's Equipment	3,220	0.02
Monthly Management Fee	115,872	0.64
Preparatory Works	1,101	0.01
Load and Haul	289.098	1.72
Drilling and Blasting	137,826	0.82
RC Grade Control Drilling	5,376	0.03
Ore Re-handle	21,222	0.11
Dewatering	3,674	0.02
Owners Costs	20,282	0.12
Total	**606,934**	**3.55**

Table 24-10 Breakdown of Fixed and Variable Operating Costs at $1,200/oz Gold Price

Mining Activity	Unit	Cost
Variable	$ '000	453,522
Fixed	$ '000	153,412
Total	**$ '000**	**606,934**
Variable	$/t	2.65
Fixed	$/t	0.90
Total	**$/t**	**3.55**

A total of 7.1 Mt of ore will be hauled from the Sofia pit to the ROM pad at Massawa. The hauling cost has been calculated at $3.25/t hauled. This amounts to a total of $23 million over the LOM based upon existing contracts from other Barrick's operating mines in West Africa.

Process Operating Costs

The process operating cost estimate has been split by the various ore types as shown in Table 24-11. The process operating costs were generated based on the various process routes possibly required.

Table 24-11 Process Operating Cost Estimate Summary at $1,200/oz Gold Price

Rock Type	Central Zone			North Zone		Sofia Main	Sofia North	Delya		
	WOL	BIOX – 100% CZ	BIOX – 25% CZ/ 75% NZ	WOL 100% NZ	BIOX 100% NZ	WOL	WOL	WOL 100% Delya	BIOX 100% Delya	BIOX 20% Delya 80% NZ
Oxide ($/t milled)	11.29	N/A	N/A	11.60	N/A	11.17	12.13	12.01	N/A	N/A
Oxide Trans ($/t milled)	14.35	N/A	N/A	13.41	N/A	12.95	12.84	13.76	N/A	N/A
Reduced Trans ($/t milled)	15.73	43.09	26.88	14.57	25.79	14.31	13.56	13.93	48.69	29.70
Fresh $/t milled)	17.40	43.09	26.88	15.74	25.79	15.68	14.30	15.62	48.69	29.70

G&A Costs

Table 24-12 is a summary of the G&A costs estimated for the first six years, followed by the final five years of production at Massawa. The model equates to $8.60/t of ore processed over the LOM at a gold price of $1,200/oz.

Table 24-12 G&A Costs Over the Life of Mine at $1,200/oz Gold Price

Item	Year 1 -6 ($)	Year 7 – 11 ($)
General Management	1,871,597	1,122,958
Site Administration	7,833,609	5,483,526
Finance	1,050,525	735,367
Supply Chain Management	940,452	705,339
Human Resources	402,940	322,352
Safety, Health and Environment	1,619,129	1,295,303
Social	446,812	446,812
Stores - Electrical & Mechanical	398,488	239,093
Labour	1,413,483	848,090
General Costs	2,212,068	1,548,448
Equipment Hire	206,196	144,337
Consultants	5,400	3,780
Power	371,798	223,079
Stores - Other	664,600	398,760
Total	**19,437,096**	**13,517,244**

24.2.6. Economic Analysis at $1,200/oz Gold Price

An economic assessment to confirm the reserve status of the Massawa, Sofia, and Delya pits at a $1,200/oz gold price was carried out based on the key parameters summarised below:

- Total ore mined from Massawa, Sofia Main, and Delya pits of 21 Mt of ore at an average grade of 3.92 g/t Au containing 2.6 Moz of gold.
- Strip ratio of 6.9:1 to give total tonnes mined of 166 Mt.
- Mining costs average of $3.55/t mined over the LOM.
- Haulage cost average of $1.25/t of ore milled over the LOM ($0.18/t km hauled).
- Plant cost average of $18.02/t ore but includes a range of costs dependant on ore feed and process route.
- G&A costs of $8.60/t ore milled over the LOM, including outside engineering costs.
- Capital construction cost of $413 million.

- On-going capital of $16 million over the LOM.
- Rehabilitation cost of $23 million at the end of the LOM.

The financial model is based on annual cash flow projections, with technical and economic parameters stated above using constant money terms. No escalation or de-escalation has been applied. In generating the financial model for the operations and project, the following principles were adopted:

- Financial implication on the methods of funding was not evaluated, since it has been assumed the Project will be financed by Barrick.
- Annual figures are based on financial years 1st January to 31st December.
- Real term annual cash flows were used to calculate the internal rate of return (IRR), net present values (NPV), and simple and discounted payback periods in real after-tax terms.
- Costs up to start of construction are considered as sunk costs.
- No salvage value for plant and equipment on cessation of operations was included.
- Calculations are based in US dollars ($).

24.2.7. Production and Cash Flow Forecast at $1,200/oz Gold Price

The estimated production and cash flow forecast at a $1,200/oz gold price are summarised in Table 24-13.

Table 24-13 Production and Cash Flow Forecast at $1,200/oz Gold Price

Year	-2	-1	0	1	2	3	4	5	6	7	8	9	10	Total
Production (koz)	-	-	221	227	201	245	237	206	245	115	162	175	25	2,060
Cashflow ($ Million)	(89)	(192)	65	115	83	134	127	37	132	59	105	113	7	696

24.2.8. Financial Analysis at $1,200/oz Gold Price

A financial model was run using a $1,200/oz gold price feeding the reserve mining schedule, together with a 3% royalty on revenue, seven-year tax holiday (two year construction, five years for operation), followed by corporate tax at a 25% rate, which produced a total net cash flow after tax of $696 million, and IRR of 28%. Payback is 2.5 years from start of production. A sensitivity table on NPV, IRR, and payback against gold price is supplied in Table 24-14.

Table 24-14 Project Financial Analysis at $1,200/oz Gold Price

Discount Rate	Gold Price ($/oz)		
	1,000	1,200	1,400
0%	339	696	1,051
5%	165	421	677
10%	60	251	441
IRR	15%	28%	41%
Payback	5 years	2.5 years	1 year

24.2.9. Government Revenue at $1,200/oz Gold Price

The government revenue earnings from the Project are sourced from the following assumptions incorporated into the financial model:

- 3% royalty on revenue;
- Tax rate of 25%. Payments are made throughout the year after the seven-year tax holiday from issuance of the mining permit.
- Dividends from 10% free carry share of the Project, which are payable after capital has been redeemed.
- Other taxes which include withholding taxes on dividends and salaries.

Under the current Kounemba Convention, Barrick will be exempt from all taxes, levies, and duties for a period of seven years from issuance of the mining title.

A sensitivity of expected government revenue based on the cash flow is detailed in Table 24-15.

Table 24-15 Government Revenue Sensitivity at $1,200/oz Gold Price

Revenue	Unit	Gold Price ($/oz)	
		1,200	1,250
Royalty	$ million	74	77
Company Tax	$ million	91	102
Dividends	$ million	62	69
Other Tax ($M)*	$ million	108	117
Total	**$ million**	**334**	**365**

*Other taxes include withholding taxes on dividends and salary taxes.

24.2.10. Sensitivity Analysis at $1,200/oz Gold Price

Discount Rate and Gold Price

The proposed Project is profitable at current gold prices, but it is sensitive to gold price fluctuations and therefore becomes a marginal project at gold prices lower than $1,000/oz, but very attractive at current or higher gold prices (Table 24-16).

Table 24-16 NPV Sensitivity at Different Gold Prices and Discount Rates ($1,200/oz Gold Price)

$ millions	Gold Price ($/oz)						
Discount	**900**	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
0%	163	339	518	696	874	1,051	1,229
5%	37	165	293	421	550	677	805
10%	(34)	60	156	251	346	441	536
15%	(76)	(4)	69	142	215	287	360
20%	(100)	(43)	14	71	127	184	241
25%	(113)	(68)	(22)	23	68	113	159

Grade and Gold Price

The Project is fairly resilient to grade changes and at current gold prices would be able to absorb a 10% reduction in grade (Table 24-17). The Project is sensitive to gold price fluctuations with a $1,000/oz and a 20% reduction in grade making the Project very marginal. Conversely, at higher gold prices than current, the Project is very resilient to a grade reduction.

Table 24-17 NPV at 20% Grade Variation at Different Gold Prices ($1,200/oz Gold Price)

Grade	$ millions	Gold Price ($/oz)						
		900	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
3.14	-20%	(177)	(18)	126	269	410	554	696
3.53	-10%	2	163	322	482	643	803	963
3.92	**0%**	163	339	518	696	874	1,051	1,229
4.32	10%	322	518	714	099	1,104	1,300	1,495
4.71	20%	482	696	909	1,122	1,335	1,548	1,760

Operating Cost and Gold Price

At current gold prices, the Project can absorb a 20% increase in operating costs and still remains largely profitable (Table 24-18). The Project will require a gold price of less than $1,000/oz and an increase of 20% in operating costs to become a marginal project.

Table 24-18 NPV at 20% Change in Operating Costs at $1,200/oz Gold Price

Operating Cost		Gold Price ($/oz)						
Operating Cost ($/t)	**$ millions**	**900**	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
44.72	-20%	369	548	726	903	1,081	1,259	1,436
50.31	-10%	266	443	622	800	978	1,155	1,332
55.91	**0%**	**163**	**339**	**518**	**696**	**874**	**1,051**	**1,229**
61.50	10%	59	236	413	592	770	948	1,125
67.09	20%	(48)	133	310	488	666	844	1,021

Capital and Gold Price

The Project is sensitive to increases in capital cost (Table 24-19). At a $1,200/oz gold price, an increase of 10% on capital reduces the NPV by $100 million, or 18% of the NPV.

Table 24-19 LOM Capital Cost Sensitivity at $1,200/oz Gold Price

Capital $ millions	Gold Price ($/oz)						
	900	**1,000**	**1,100**	**1,200**	**1,300**	**1,400**	**1,500**
-5%	181	358	536	714	892	1,070	1,247
-10%	199	376	555	733	911	1,088	1,265
-15%	218	395	573	751	929	1,107	1,284
0%	**163**	**339**	**518**	**696**	**874**	**1,051**	**1,229**
10%	125	303	481	659	837	1,014	1,192
25%	70	248	425	603	781	959	1,136
50%	(31)	155	333	510	689	867	1,044

25. Interpretation and Conclusions

Barrick has documented standard procedures for the drilling, logging, and sampling processes, which meet industry standards. The mineralisation wireframe parameters at Massawa, Sofia, and Delya are based on visibly identifiable geological contacts, which ensure that a geologically robust interpretation can be developed. Randgold's procedures ensure a reliable database of exploration information, but the implementation of a digital system is an opportunity for improving the systems and checks in place.

Massawa Mineral Resources are estimated using industry accepted methods. Portions of the Massawa mineralisation have been recognised to have significantly higher grades than the remainder of the mineralisation, and the top cutting, domaining, and estimation approach taken by Randgold to limit the effects of the high grades is considered to be appropriate. The Qualified Persons consider the Mineral Resource estimates at the Project to be appropriately estimated and classified.

The Qualified Persons concur with the parameters used in the Mineral Resource to Ore Reserve conversion process.

The strategic focus for Massawa has been to prioritise the Sofia and CZ ore over the refractory ores of the Massawa NZ. Consequently, increasing the reserves of non-refractory material will further benefit the Project and active exploration work is continually underway targeting additional areas on the permit.

The open pit mining operations at Massawa, Sofia, and Delya will consist of multiple open pits, i.e. CZ, NZ, Sofia Main, Sofia North, and Delya. The open pits are being planned to be mined by a mining contractor and a down-the-hole blasting service will be provided by an appropriate blasting contractor. The proposed mining method of conventional 90 t truck and excavator open pit mining is appropriate for the ore body and suitable dilution and ore loss factors have been applied. Randgold has significant experience in other mining operations in the region on similar ore bodies to Sofia, Delya, and NZ and has compared production, modification factors, and costs against these operations to ensure they are suitable. The CZ ore body has execution risk in that the bulk of the gold is hosted within thin lodes containing a large coarse gold component. Higher dilution and ore loss factors have been applied to this ore body to compensate. Any misallocation or misinterpretation of the CZ ore body will result in a loss of value. As such, a detailed GC programme will be an important requirement to successfully mine the CZ ore body.

Significant testwork has already been undertaken on the various ore types from oxide and free leaching ores of Sofia, to the high gravity and partially refractory ores of the CZ and the highly refractory ores of the Massawa NZ and Delya. Based on testwork completed, the overall recoveries of 78% for the Project are realistic.

The processing feed plan extends over a nine-year period where the WOL plant has a nameplate capacity of 2.4 Mtpa of fresh ore which can increase to 3.0 Mtpa when treating the softer oxides. The plant is divided into two streams, i.e. two parallel grinding ball mills form the hub of the

processing plant. Each subsequent process route is implemented sequentially from a single stream. This means that the flotation circuit is erected well into the mine life in conjunction with the BIOX circuit, which matches the phase capital schedule.

The completed ESIA analysis includes all required specialist studies. The Project is within the Niokolo-Koba headwaters and within a relatively environmentally pristine area. Significant effort has been made to generate an Environmental Management Plan that is practical and effective in minimising the impact of the Project on the environment. An inclusive transparent approach that utilises the concept of environmental offsets to benefit the regions biodiversity preservation will be implemented.

26. Recommendations

It is recommended that additional exploration be conducted to expand the non-refractory ore to extend the life of mine and improve the economics and pay back of the Project. It is recommended that the entire Massawa, Sofia, and Delya Mineral Resources be drilled to an AdvGC spacing suitable to the variography for each of the pit's ore types prior to commencement of mining.

Process operating costs have been estimated based on specialist studies on the variable ore types and it is recommended that optimisation studies be undertaken on the CZ and BIOX amenable ore types to further optimise the process recoveries and costs. Mining costs have been developed from the first round of mining tender and it is recommended that the final mine plan be submitted to the short list of tender contractors to obtain the most efficient cost, and a trade-off be done against an owner mining option.

A full updated ESIA has been completed for the Project, and Environmental and Social Management workshops will be required including all affected parties to find practical and effective management measures to leverage the benefits of the Project to the region and the Senegalese economy but minimise the negative impact on communities and environment.

It is recommended that Randgold, which owns the permit, submit an application to the Senegalese government to convert the Kanoumba Permit into a mining licence under the Senegalese 2003 Mining Code.

It is further recommended that, upon approval of the mining permit by the Senegalese government, a new entity company be registered in Senegal into which the mining permit will be transferred, and at this stage the State will receive its participation interest in that company. It is the intention that the newly registered company will have a new name and that ownership of the Kanoumba Permit will be transferred into the newly formed company.

As previously indicated, Randgold is now a wholly owned subsidiary of Barrick following the merger transaction which was completed on 1st January 2019.

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28. Date and Signature Page

This report titles 'Technical Report on the Feasibility Study of the Massawa Gold Project, Senegal' was prepared and signed by the following authors:

Signed *Rodney B Quick*

Dated at Johannesburg, South Africa
23rd July 2019

Rodney B Quick, MSc, Pr. Sci.Nat
Mineral Resource Management and Evaluation Executive
Barrick Gold Corporation

Signed *Simon P Bottoms*

Dated at Southampton, UK
23rd July 2019

Simon P Bottoms, MGeol, CGeol, FAusIMM
Senior Vice President, Africa & Middle East Mineral Resource Manager
Barrick Gold Corporation

Signed *Richard Quarmby*

Dated at Johannesburg, South Africa
23rd July 2019

Richard Quarmby, Pr Eng, CEng, SAIChE
Africa & Middle East Capital Projects Metallurgist
Barrick Gold Corporation

Signed *Graham E Trusler*

Dated at Johannesburg, South Africa
23rd July 2019

Graham E Trusler, MSc, Pr Eng, MIChE, MSAIChE
CEO
Digby Wells Environmental (Jersey) Limited

29. Certificate of Qualified Person

29.1. Rodney B Quick

I, Rodney B Quick MSc, Pr. Sci.Nat, do hereby certify that:

1. I am an employee of Barrick Gold Corporation (the 'Company') at 161 Bay Street, Suite 3700, Toronto, Ontario, Canada M5J 2S1.
2. I am an author of the technical report titled 'Technical Report on the Feasibility Study of the Massawa Gold Project, Senegal' (the 'Technical Report') of the Company.
3. I graduated with a Bachelor of Science Honours degree in Geology from the University of Natal Durban, South Africa in 1993 and with a Master of Science degree in Geology from the Leicester University, United Kingdom in 2000.
4. I am a Professional Natural Scientist registered (400014/05) with the South African Council for Natural Scientific Professions (SACNASP). I am a current Member of SACNASP.
5. I have worked as a geologist continuously since my graduation from University.
6. I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. I most recently visited the Massawa Gold Project from 1st November to 2nd November 2018.
8. I am responsible for the preparation of Sections 5, 15, 16, 18 to 22, and 24 and relevant disclosure in Sections 1, 2, 3, and 25 to 27 of this Technical Report.
9. I am not independent of the Company in accordance with Section 1.5 of NI 43-101.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is that I have previously undertaken a peer review on resource to reserve conversions of Sofia Main and have prepared a previous technical report on the property dated 12th May 2017.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that Instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, Sections 5, 15, 16, 18 to 22, and 24 and relevant disclosure in Sections 1, 2, 3, and 25 to 27of this Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd day of July, 2019.

Signed *Rodney B Quick*

Rodney B. Quick

MSc, Pr. Sci.Nat - Mineral Resource Management and Evaluation Executive - Barrick Gold Corporation

29.2. Simon P Bottoms

I, Simon P Bottoms, MGeol, FGS CGeol, FAusIMM do hereby certify that:

1. I am an employee of Barrick Gold Corporation (the 'Company') at 161 Bay Street, Suite 3700, Toronto, Ontario, Canada M5J 2S1.
2. I am an author of the technical report titled 'Technical Report on the Feasibility Study of the Massawa Gold Project, Senegal' (the 'Technical Report') of the Company.
3. I graduated with a Masters of Geology degree from the University of Southampton, United Kingdom in 2009.
4. I am a Chartered Geologist registered (1023769) with the Geological Society of London. I am a current Member of AusIMM.
5. I have worked as a geologist continuously for 8 years since my graduation from University.
6. I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. I most recently visited the Massawa Gold Project from 8th June to 10th June 2018.
8. I am responsible for the preparation of Sections 4, 6 to 12, 14, and 23, and relevant disclosure in Sections 1, 2, 3, and 25 to 27 of this Technical Report.
9. I am not independent of the Company in accordance with section 1.5 of NI 43-101.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is that I have previously undertaken a peer review on resource to reserve conversions of Sofia Main and prepared a previous technical report on the property dated 12th May 2017.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, Sections 4, 6 to 12, 14, and 23 and relevant disclosure in Sections 1, 2, 3, and 25 to 27 of this Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd day of July, 2019.

Signed *Simon P Bottoms*

Simon P Bottoms

MGeol, FGS CGeol, FAusIMM – Senior Vice President, Africa & Middle East Mineral Resource Manager – Barrick Gold Corporation

29.3. Richard Quarmby

I, Richard Quarmby, Pr Eng, C Eng, SAIChE, do hereby certify that:

1. I am an employee of Barrick Gold Corporation (the 'Company') at 161 Bay Street, Suite 3700, Toronto, Ontario, Canada M5J 2S1.
2. I am an author of the technical report titled 'Technical Report on the Feasibility Study of the Massawa Gold Project, Senegal' (the 'Technical Report') of the Company.
3. I graduated with a BSc chemical engineering degree from the University of the Witwatersrand in 1985 and earned a Master of Business Administration degree in 2005.
4. I have been registered, no. 910237 as a Professional Engineer (Pr Eng) with the Engineering Council of SA since 1991 and in 2010 was accepted to the UK equivalent institution i.e. Chartered Engineer with the Engineering Council UK (C Eng), no. 580441. Further, I have been a Member, no. 1361, of the South African Institution of Chemical Engineers (SAIChE) since1989, and am also a registered Member, no. 454225, of the Institute of Materials, Minerals and Mining (IoMMM) UK.
5. I have worked as an engineer continuously since my graduation from University in 1985.
6. I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. I most recently visited the Massawa Gold Project from 1st November to 2nd November 2018.
8. I am responsible for the preparation of Sections 13, 17, and 24 and relevant disclosure in Sections 1, 2, 3, and 25 to 27 of this Technical Report.
9. I am not independent of the Company in accordance with section 1.5 of NI 43-101.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is that I have been involved in the Massawa property in my capacity of Group Metallurgist, Projects for Randgold Resources Limited since my initial employment from September 2015. I have prepared a previous technical report on the property dated 12th May 2017.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, Sections 13, 17, and 24 and relevant disclosure in Sections 1, 2, 3, and 25 to 27 of this Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd day of July, 2019.

Signed *Richard Quarmby*

Richard Quarmby

Pr Eng, C Eng, SAIChE - Africa & Middle East Capital Projects Metallurgist – Barrick Gold Corporation

29.4. Graham E. Trusler

I, Graham E. Trusler, MSc, Pr Eng, MIChE, MSAIChE, do hereby certify that:

1. I am the CEO of Digby Wells and Associates (Pty) Ltd, of 48 Grosvenor Road, Bryanston, Gauteng, South Africa.
2. I am an author of the technical report titled 'Technical Report on the Feasibility Study of the Massawa Gold Project, Senegal' (the 'Technical Report') of the Company.
3. I graduated with a Masters of Chemical Engineering degree from the University of KwaZulu-Natal, South Africa, in 1988.
4. I have been registered, no. 920088 as a Professional Engineer (Pr Eng) with the Engineering Council of South Africa since 1992. Further, I have been a Member, of the South African Institution of Chemical Engineers (SAIChE) since1994. I am also registered as a Chartered Chemical Engineer with the Institution of Chemical Engineers, am a member of the Water Institute of South Africa and a lifetime member of the American Society of Mining and Reclamation. I have worked as an engineer continuously from 1990.
5. I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101. I have over 30 years of experience within the mining industry in metallurgical production, research, and environmental issues.
6. I visited the Massawa Gold Project from 19th November to 22nd September 2018.
7. I am responsible for the preparation of Section 20 and relevant disclosure in Sections 1, 2, 3, and 25 to 27 of this Technical Report.
8. I am independent of the Company applying the test set out in Section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
10. I have had prior involvement with the property that is the subject of the Feasibility Report. The nature of my prior involvement is that I have been involved in the Massawa property in my capacity of Social and Environmental project evaluation.
11. As of the effective date of this report, to the best of my knowledge, information and belief, Section 20, and relevant disclosure in Sections 1, 2, 3, and 25 to 27 of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated this 23rd day of July, 2019.

Signed *Graham E Trusler*

Graham E Trusler

MSc, Pr Eng, MIChE, MSAIChE - CEO - Digby Wells Environmental

30. Appendix

30.1. Field Duplicates

Figure 30-1 to Figure 30-23 provide graphical representations of original vs field duplicate samples analysed by all laboratories for the period.

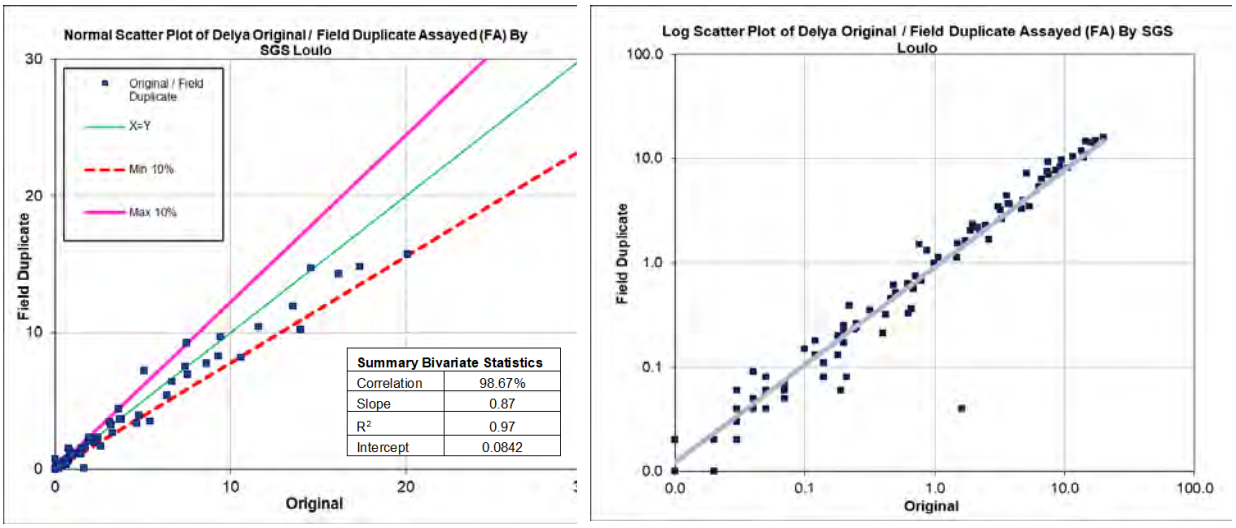

Figure 30-1 Normal and Log Scatter Plots of Delya Field Duplicates Assayed (Fire Assay) by SGS Loulo

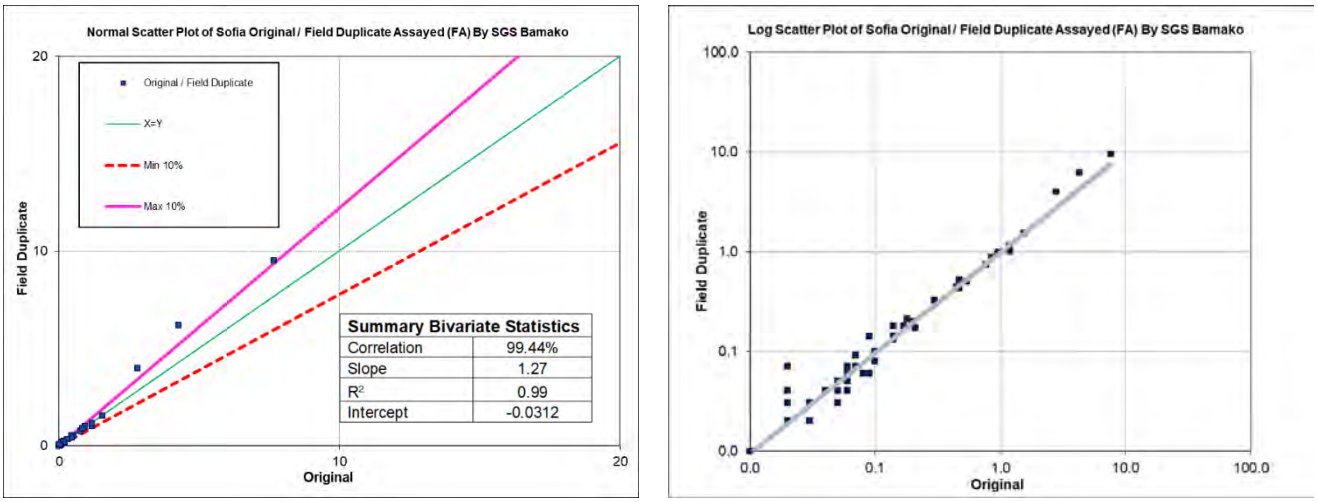

Figure 30-2 Normal Scatter Plot of Sofia Field Duplicates Assayed (Fire Assay) by SGS Bamako

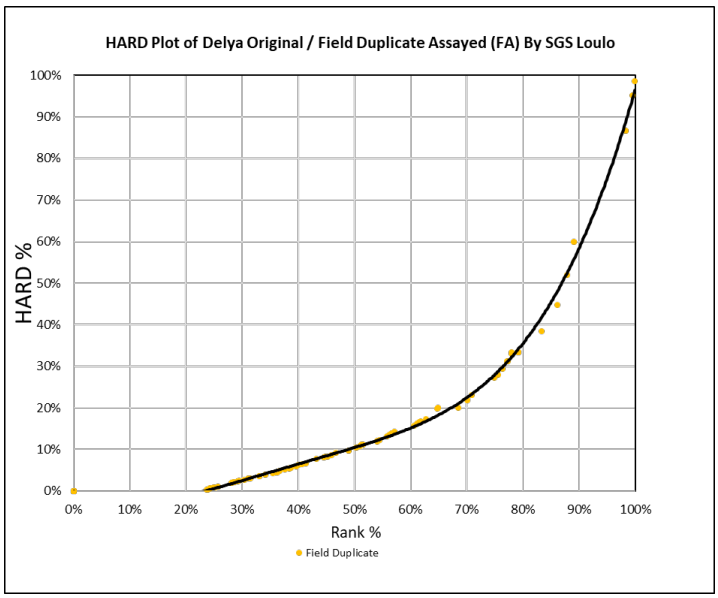

Figure 30-3 HARD Plot of Delya Field Duplicates Assayed (Fire Assay) by SGS Loulo

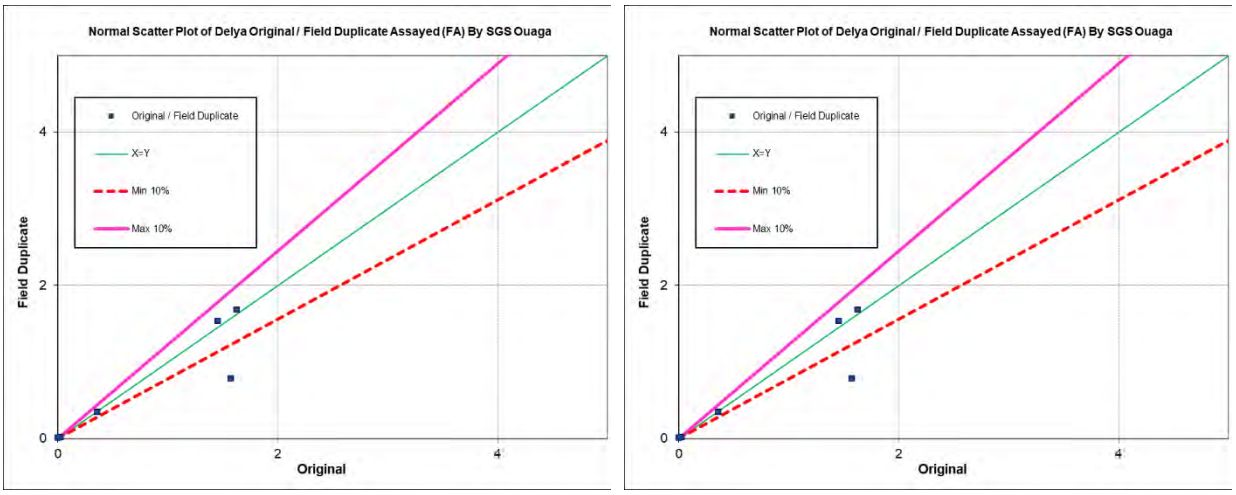

Figure 30-4 Normal and Log Scatter Plots of Delya Field Duplicates Assayed (Fire Assay) by SGS Ouagadougou

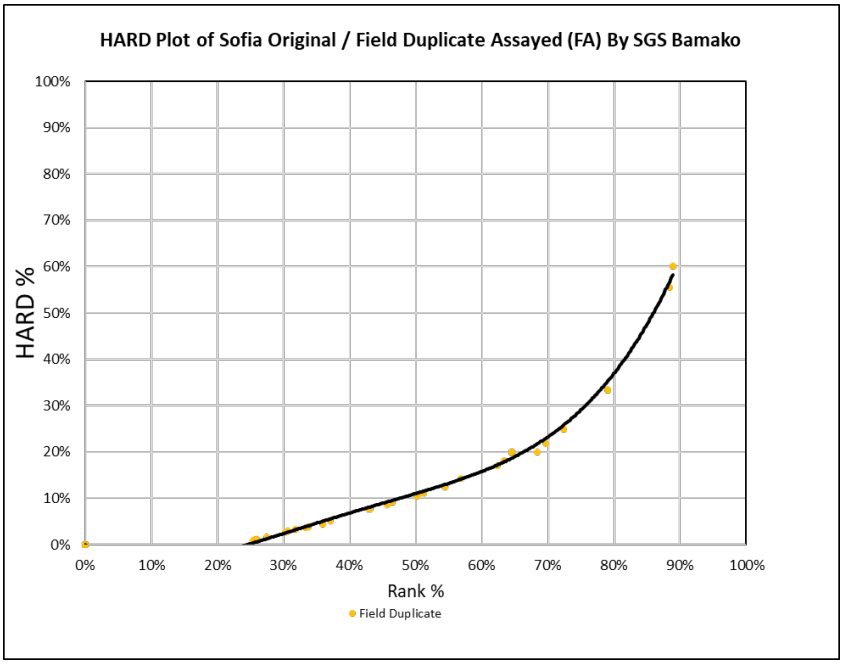

Figure 30-5 HARD Plot of Sofia Field Duplicates Assayed (Fire Assay) by SGS Bamako

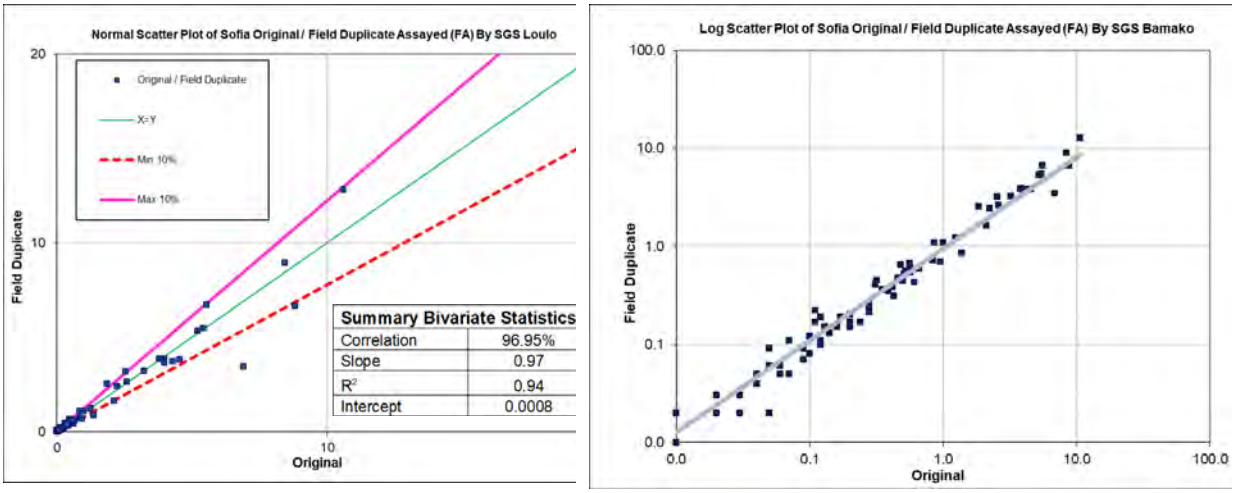

Figure 30-6 Normal and Log Scatter Plots of Sofia Field Duplicates Assayed (Fire Assay) by SGS Loulo

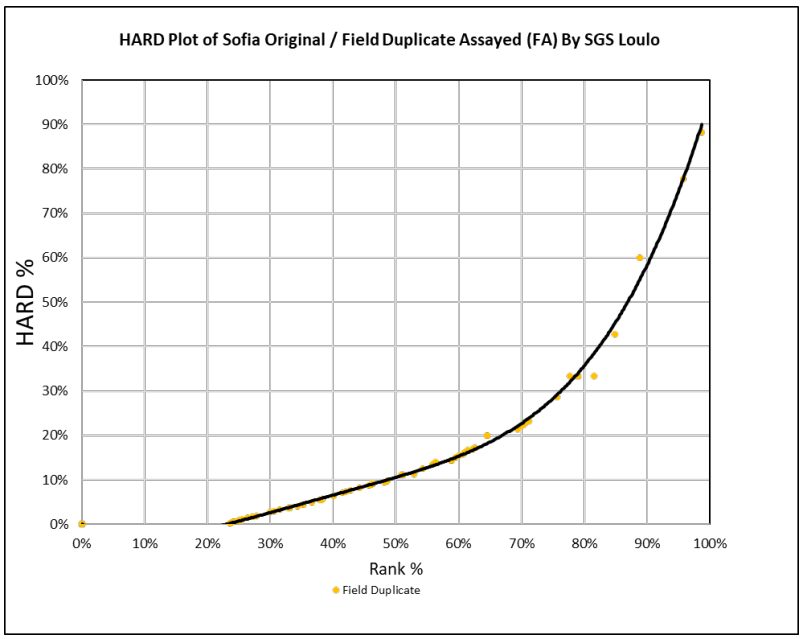

Figure 30-7 HARD Plot of Sofia Field Duplicates Assayed (Fire Assay) by SGS Loulo

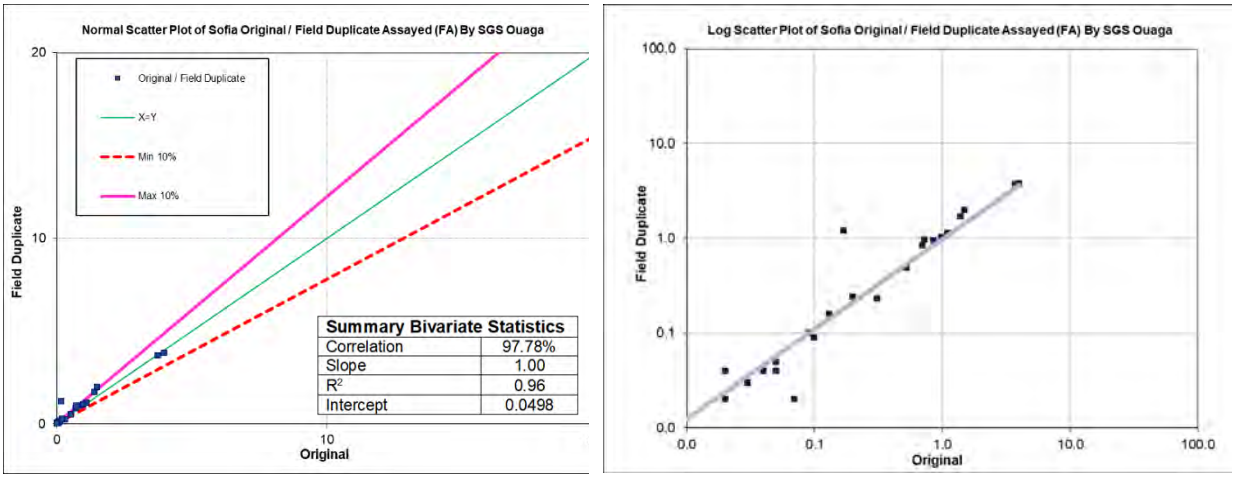

Figure 30-8 Normal and Log Scatter Plots of Sofia Field Duplicates Assayed (Fire Assay) by SGS Ouagadougou

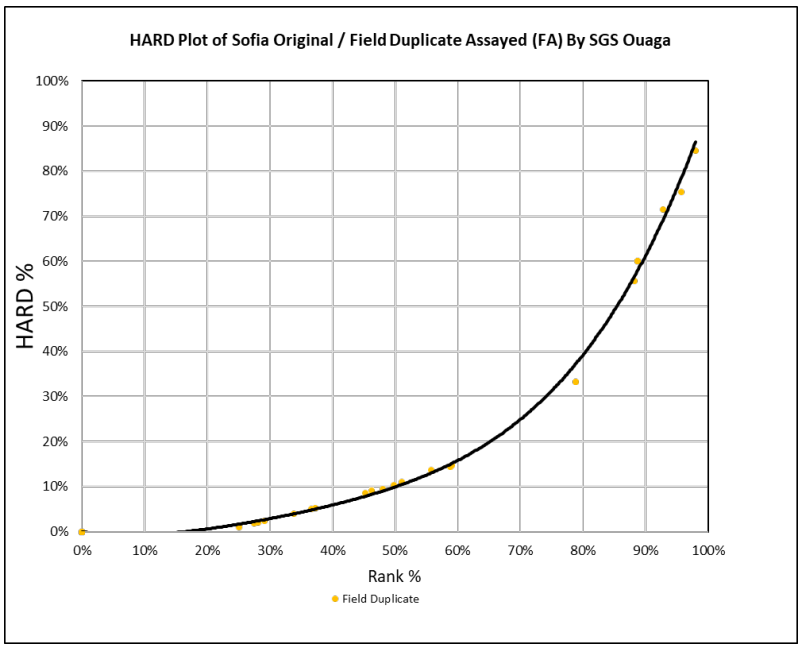

Figure 30-9 HARD Plot of Sofia Field Duplicate Assayed (Fire Assay) by SGS Ouagadougou

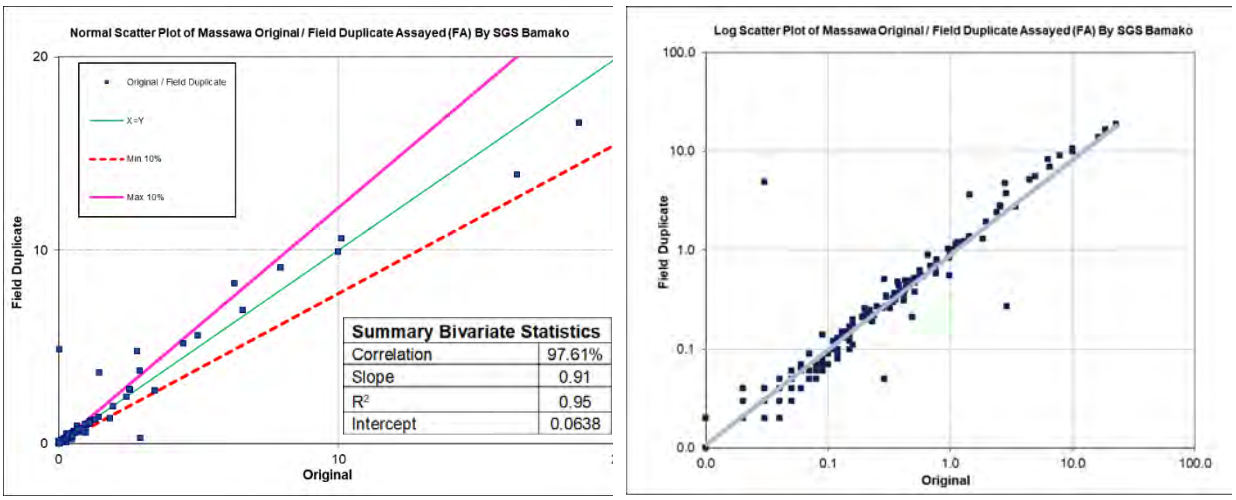

Figure 30-10 Normal and Log Scatter Plots of Massawa Field Duplicate Assayed (Fire Assay) by SGS Bamako

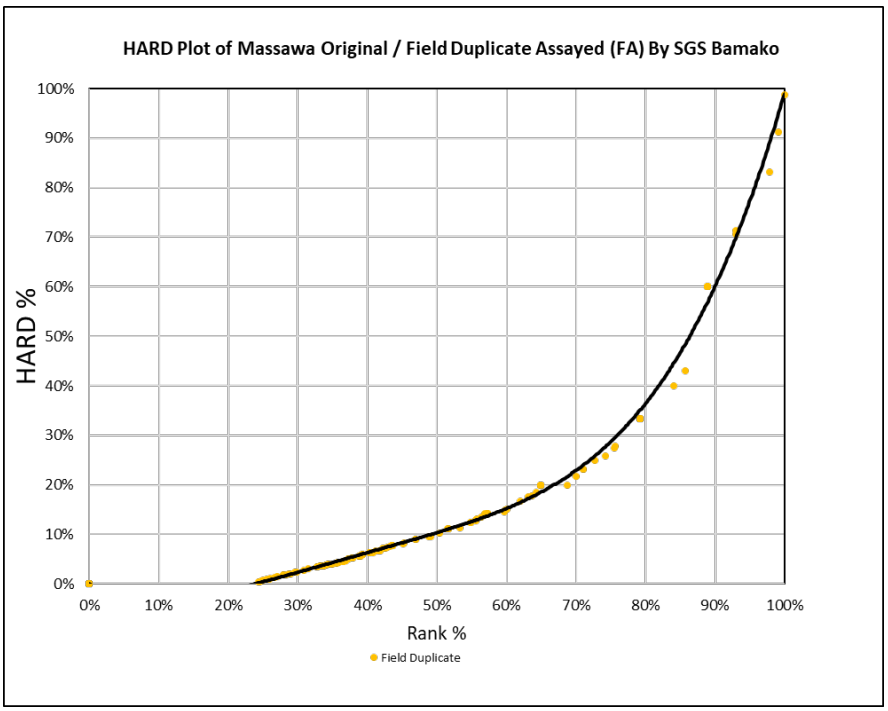

Figure 30-11 HARD Plot of Massawa Field Duplicate Assayed (Fire Assay) by SGS Bamako

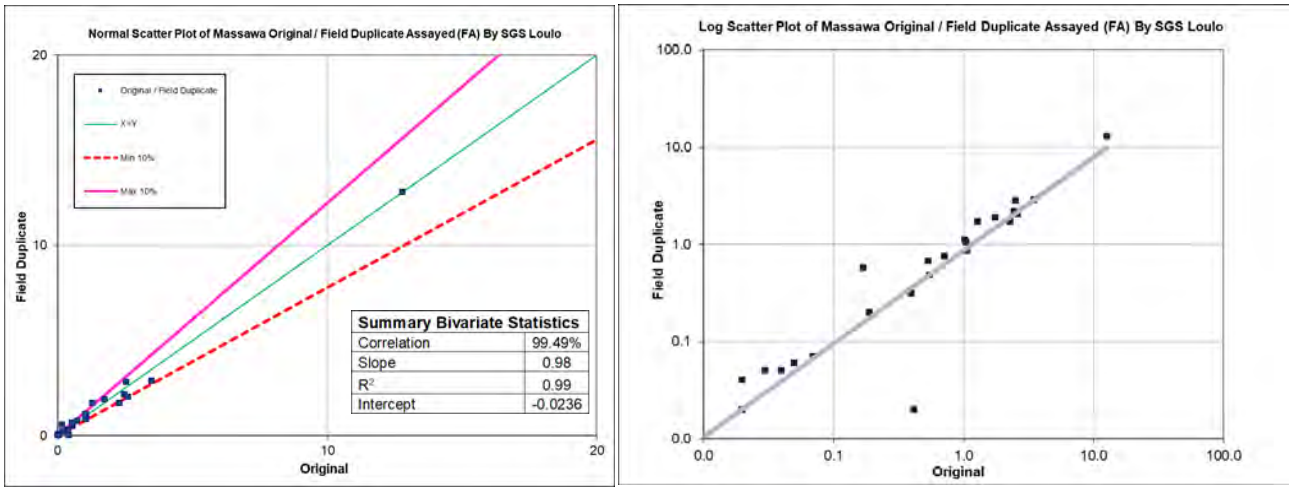

Figure 30-12 Normal and Log Scatter Plots of Massawa Field Duplicates Assayed (Fire Assay) by SGS Loulo

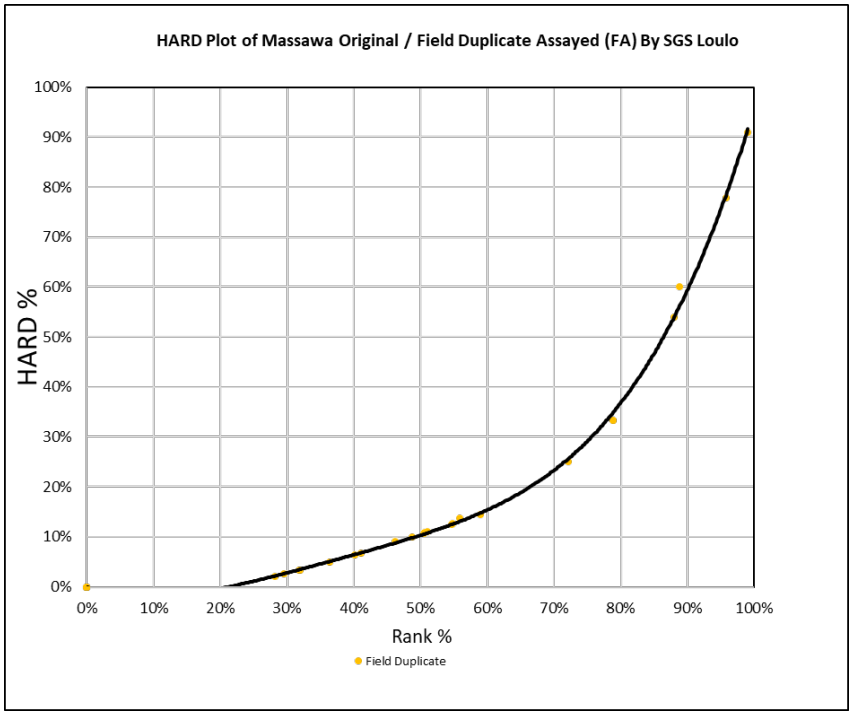

Figure 30-13 HARD Plot of Massawa Field Duplicates Assayed (Fire Assay) by SGS Loulo

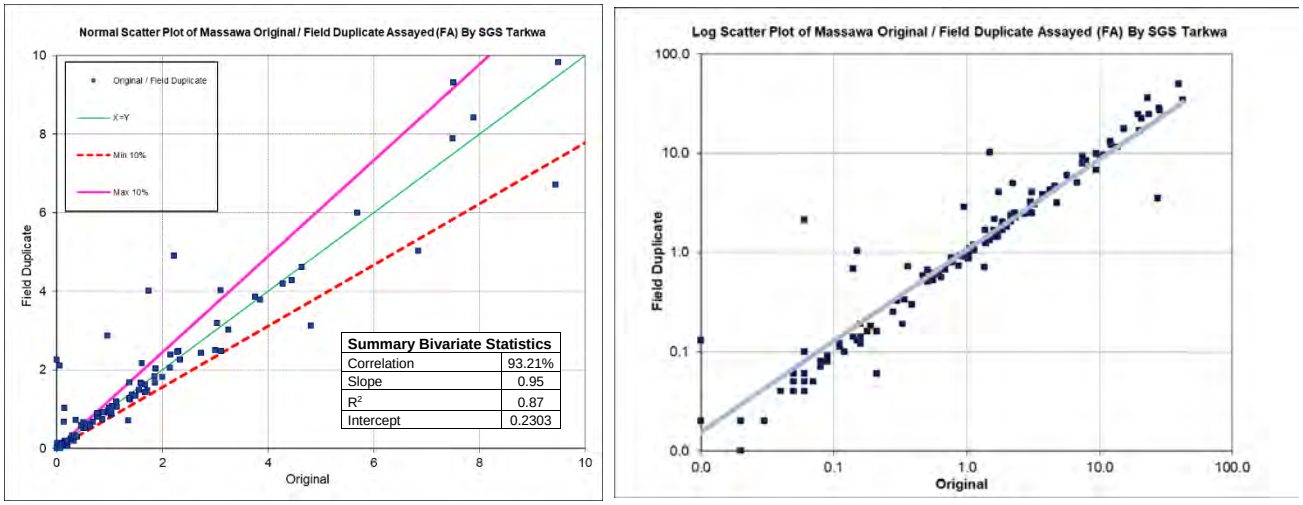

Figure 30-14 Normal and Log Scatter Plots of Massawa Field Duplicates Assayed (Fire Assay) by SGS Tarkwa

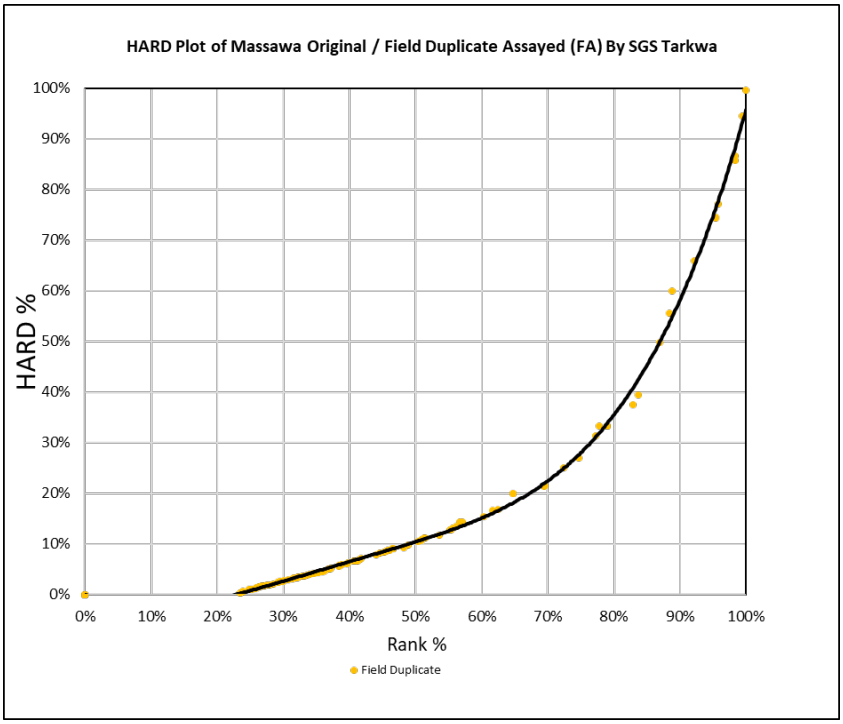

Figure 30-15 HARD Plot of Massawa Field Duplicates Assayed (Fire Assay) by SGS Tarkwa

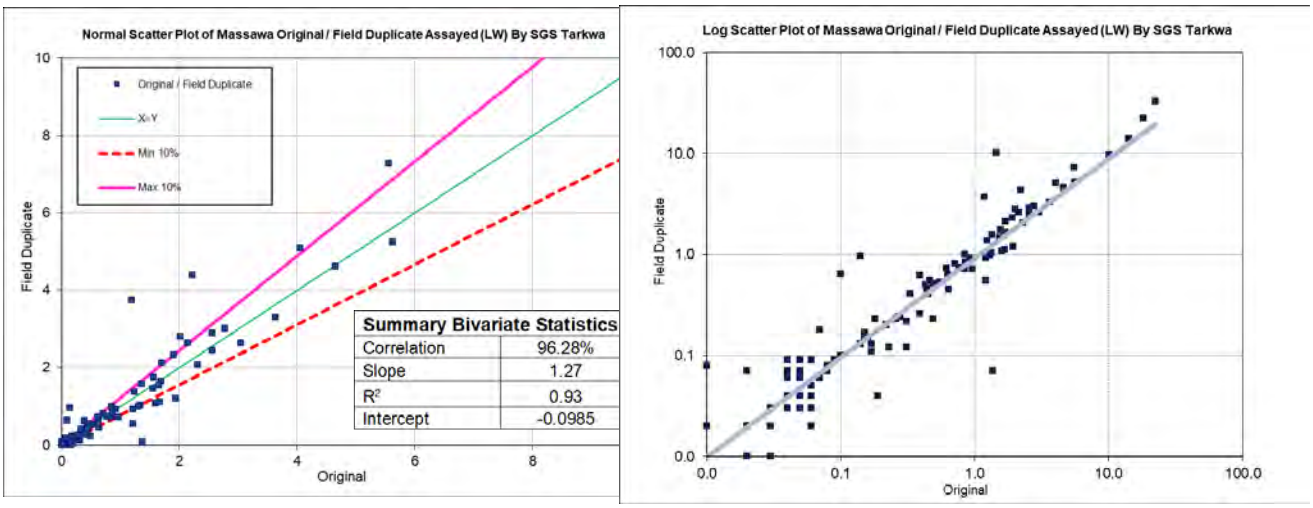

Figure 30-16 Normal and Log Scatter Plots of Massawa Field Duplicates Assayed (LeachWELL) by SGS Tarkwa

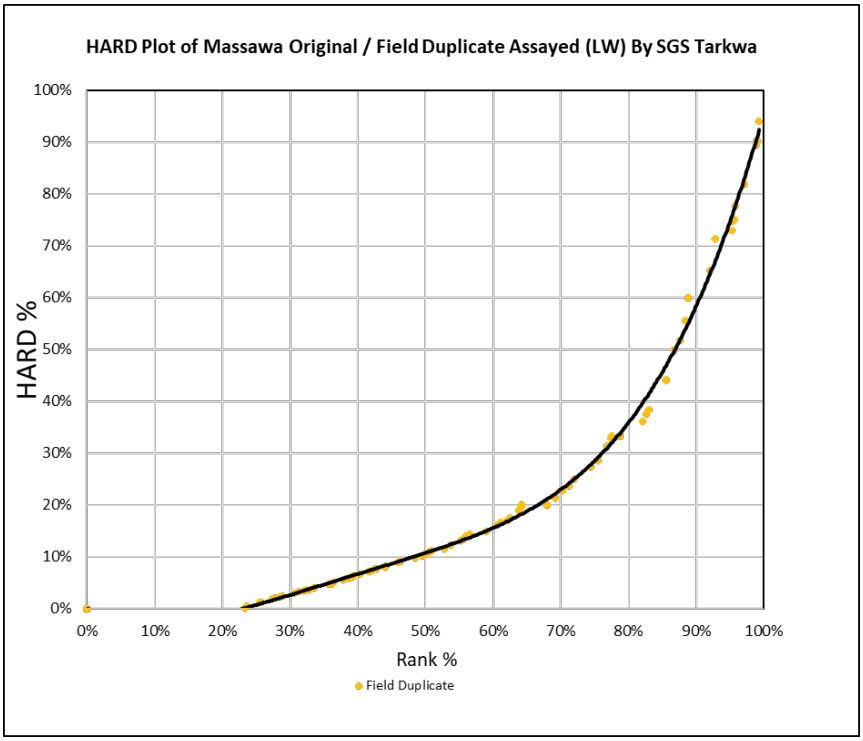

Figure 30-17 HARD Plot of Massawa Field Duplicates Assayed (LeachWELL) by SGS Tarkwa

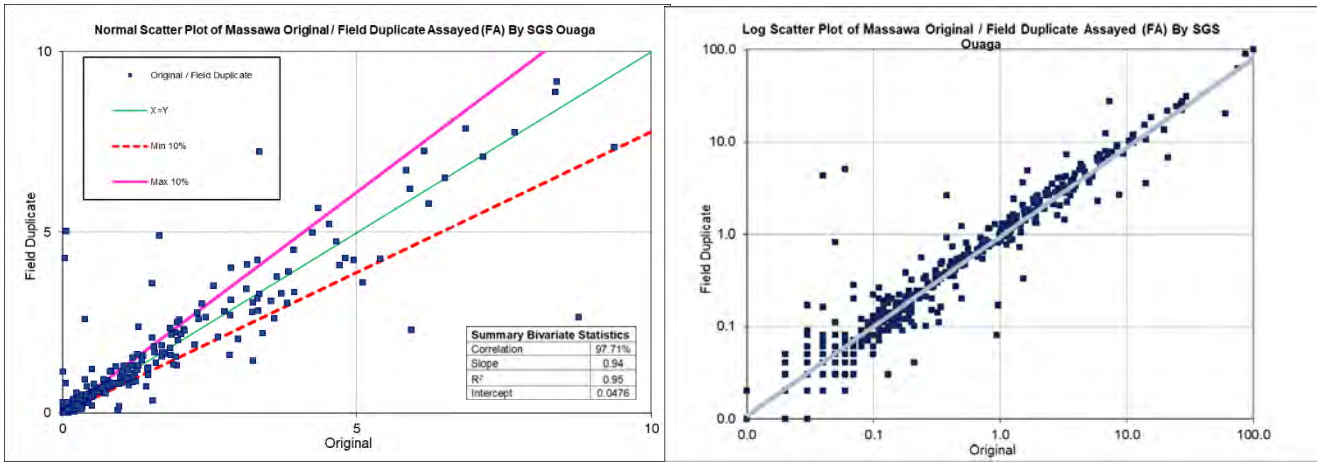

Figure 30-18 Normal and Log Scatter Plots of Massawa Field Duplicates Assayed (Fire Assay) by SGS Ouagadougou

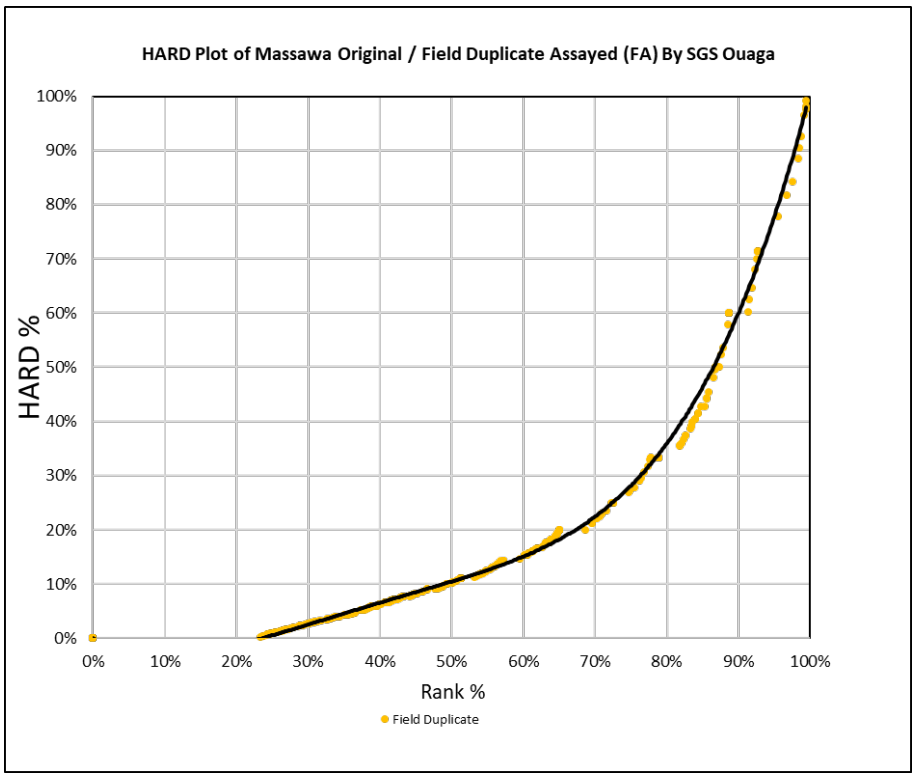

Figure 30-19 HARD Plot of Massawa Field Duplicate Assayed (Fire Assay) by SGS Ouagadougou

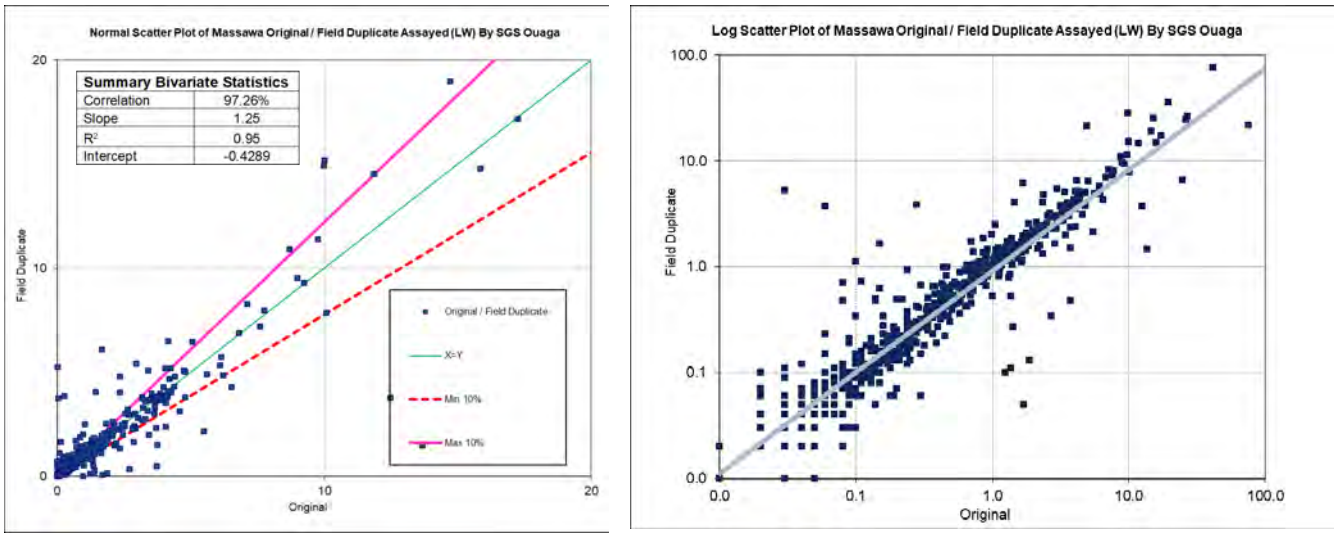

Figure 30-20 Normal and Log Scatter Plots of Massawa Field Duplicates Assayed (LeachWELL) by SGS Ouagadougou

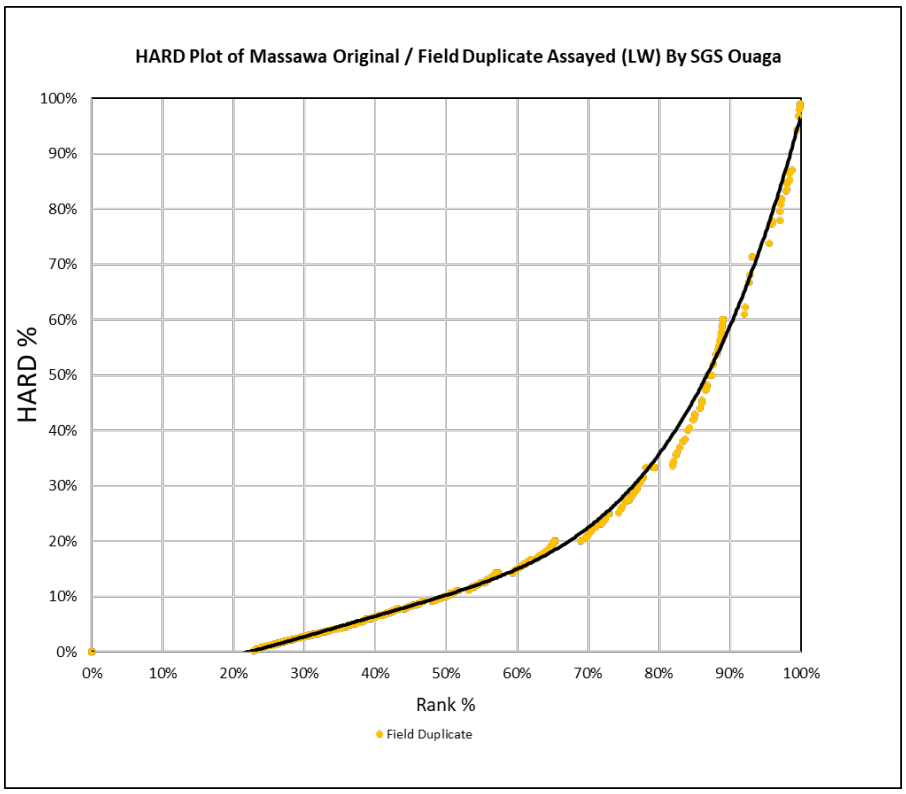

Figure 30-21 HARD Plot of Massawa Field Duplicates Assayed (LeachWELL) by SGS Ouagadougou

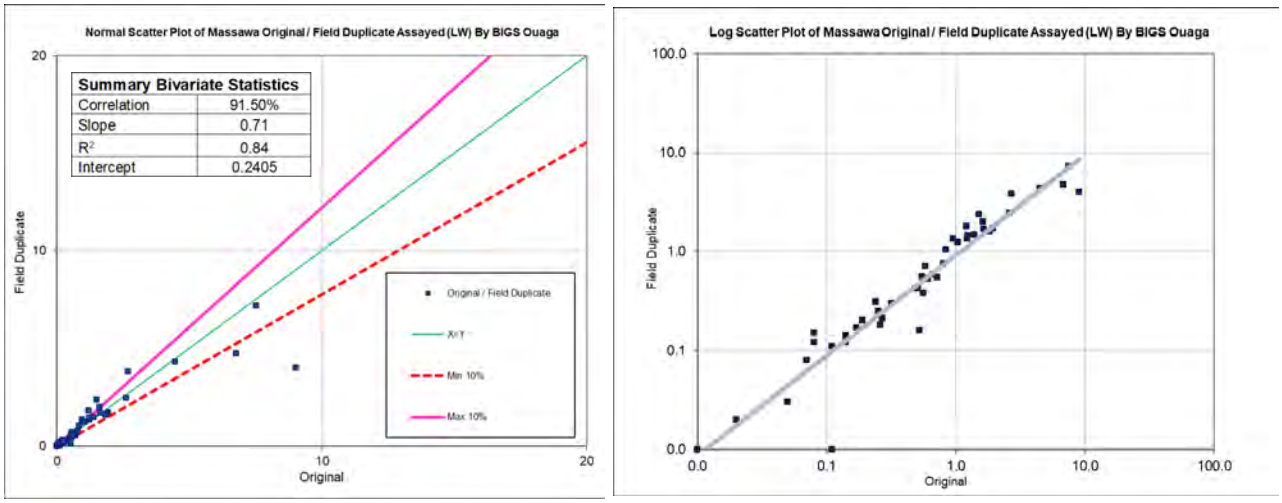

Figure 30-22 Normal and Log Scatter Plots of Massawa Field Duplicates Assayed (LeachWELL) by BIGS Ouagadougou

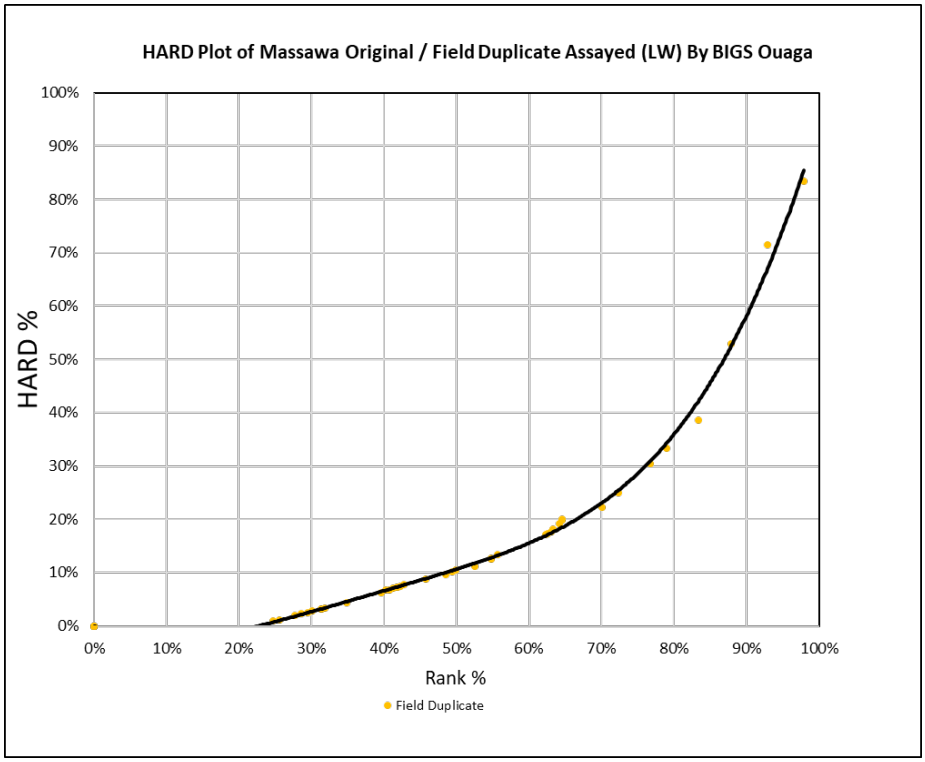

Figure 30-23 HARD Plot of Massawa Field Duplicate Assayed (LeachWELL) by BIGS Ouagadougou

30.2. Blanks

Figure 30-24 to Figure 30-35 provide a graphical representation of the performance blank sample results analysed by all laboratories during the period.

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.24	797	<0.01	3 (0.38%)	1 (0.13%)	793 (99.50%)	794 (99.62%)	3 (0.38%)

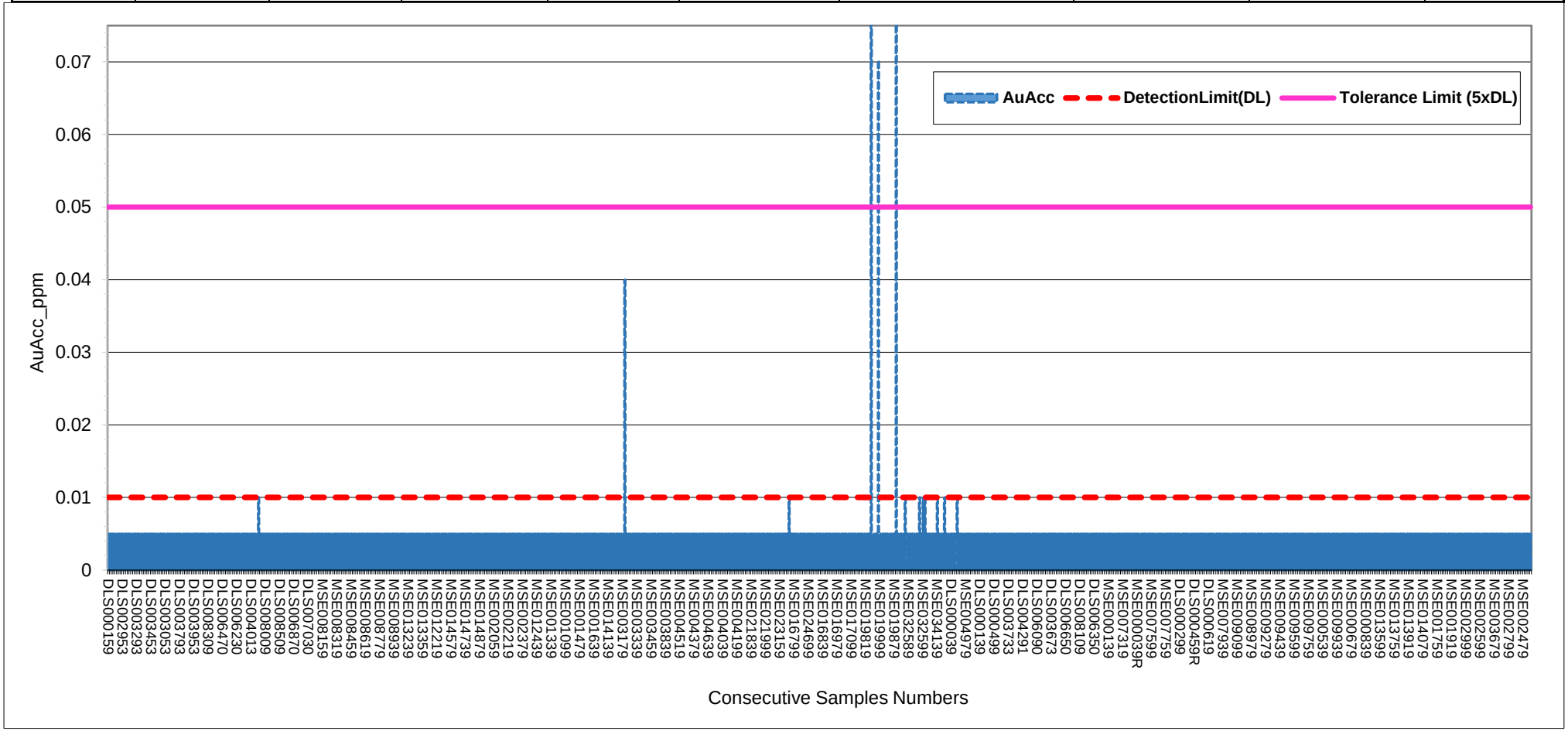

Figure 30-24 Delya Blank Samples Assayed (Fire Assay) by SGS Loulo

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.005	36	<0.01	0	0	36 (100%)	36 (100%)	0

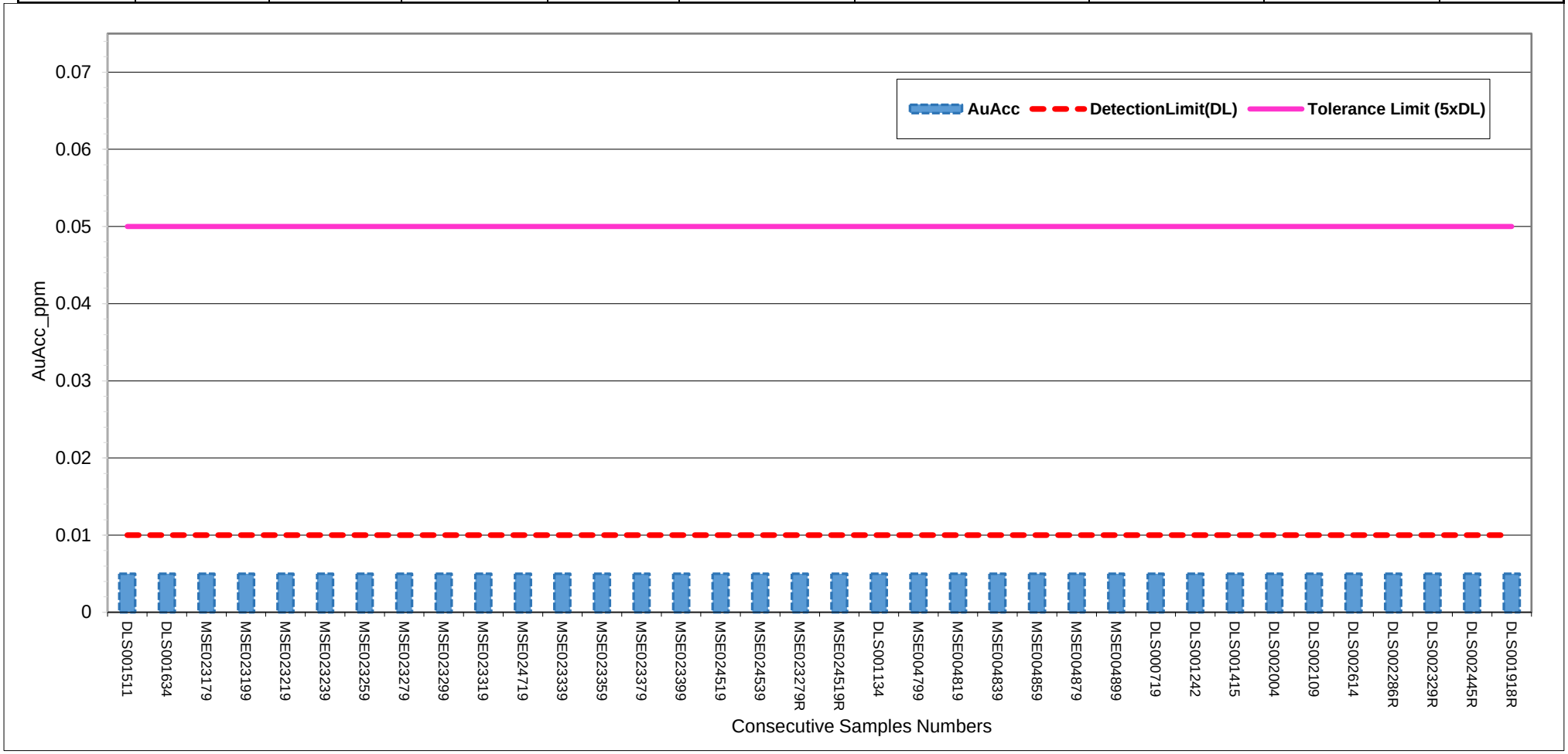

Figure 30-25 Delya Blank Samples Assayed (Fire Assay) by SGS Ouagadougou

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.01	269	<0.01	0	0	269 (100%)	269 (100%)	0

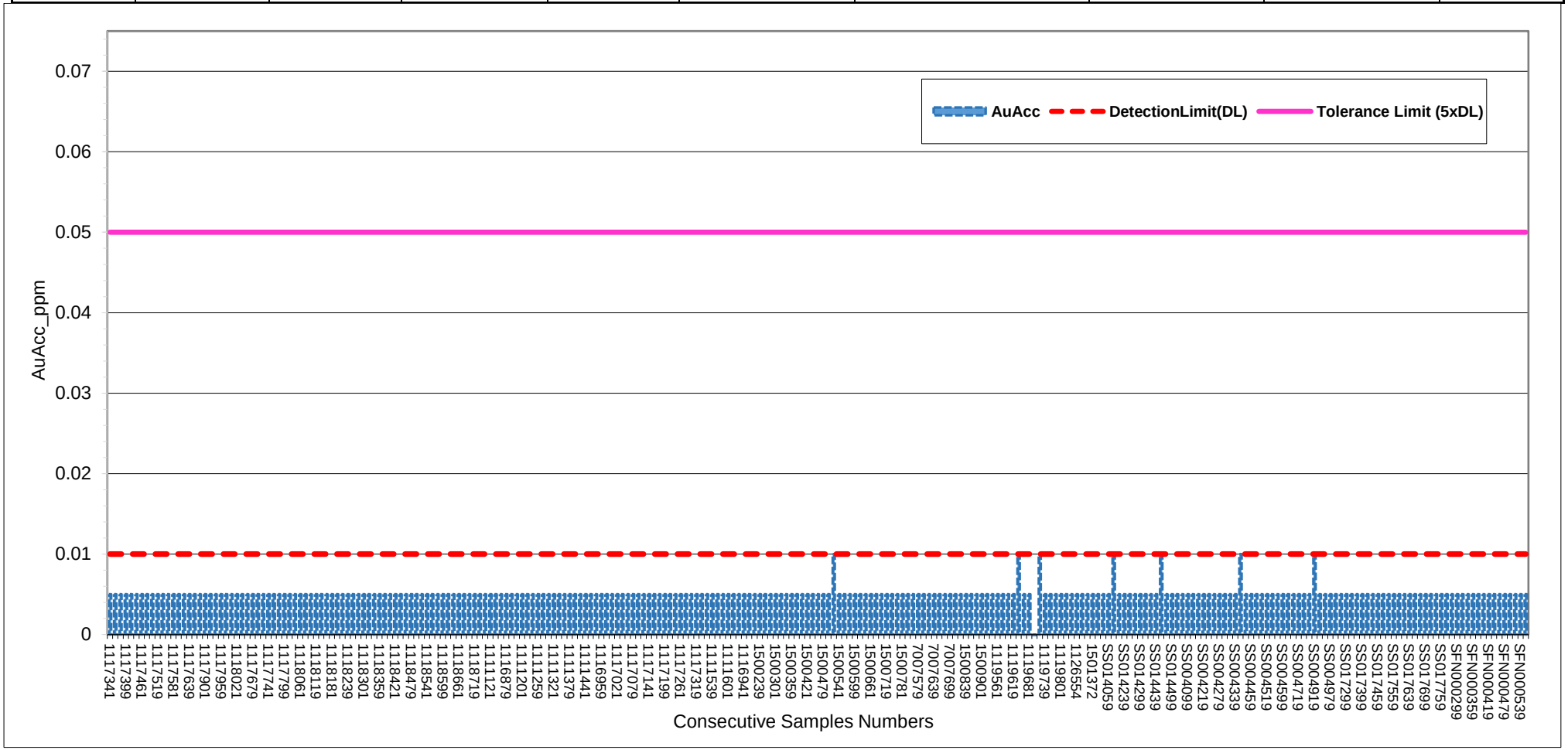

Figure 30-26 Sofia Blank Samples Assayed (Fire Assay) by SGS Bamako

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.02	844	<0.01	0	2 (0.24%)	842 (99.76%)	844 (100%)	0

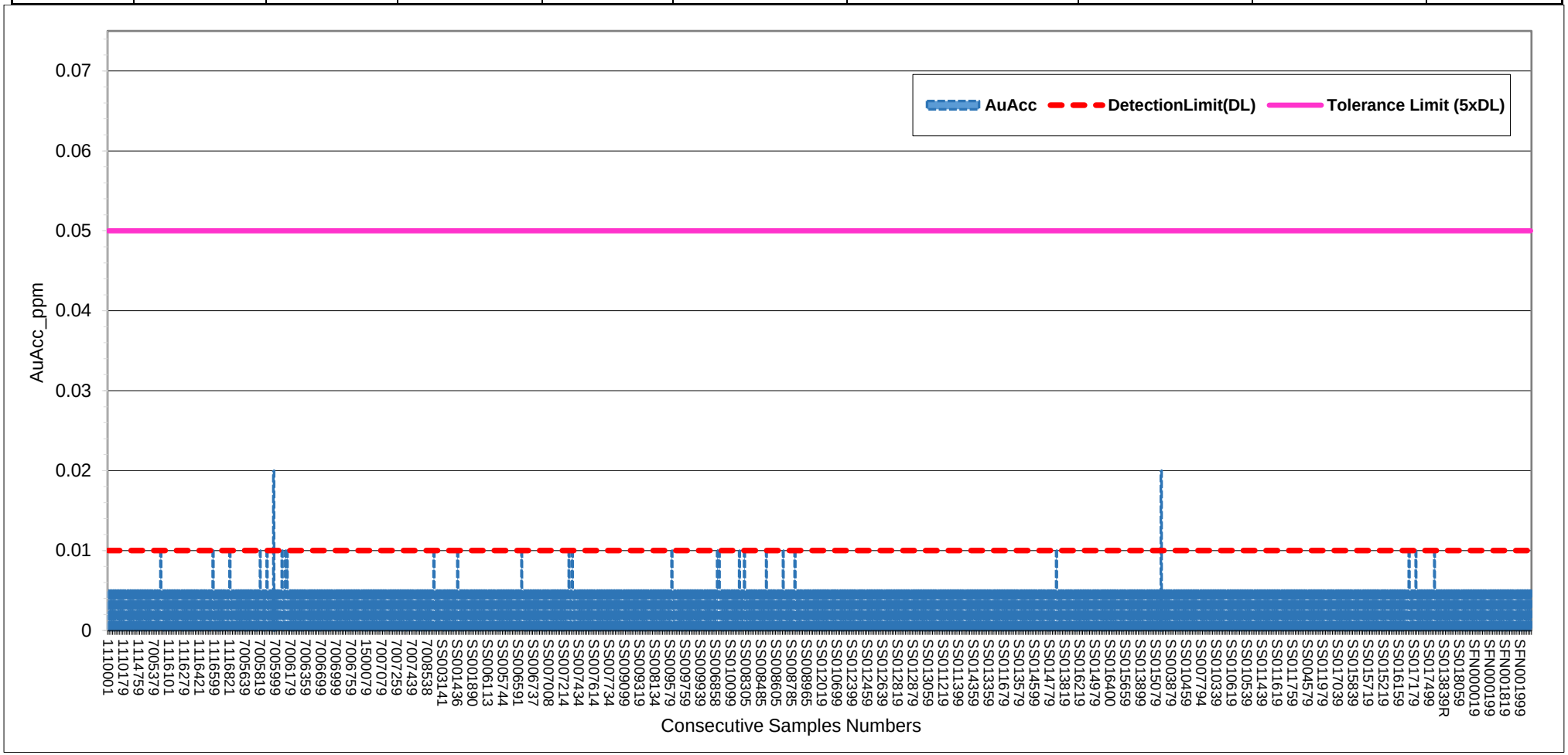

Figure 30-27 Sofia Blank Samples Assayed (Fire Assay) by SGS Loulo

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.005	141	<0.01	0	0	141 (100%)	141 (100%)	0

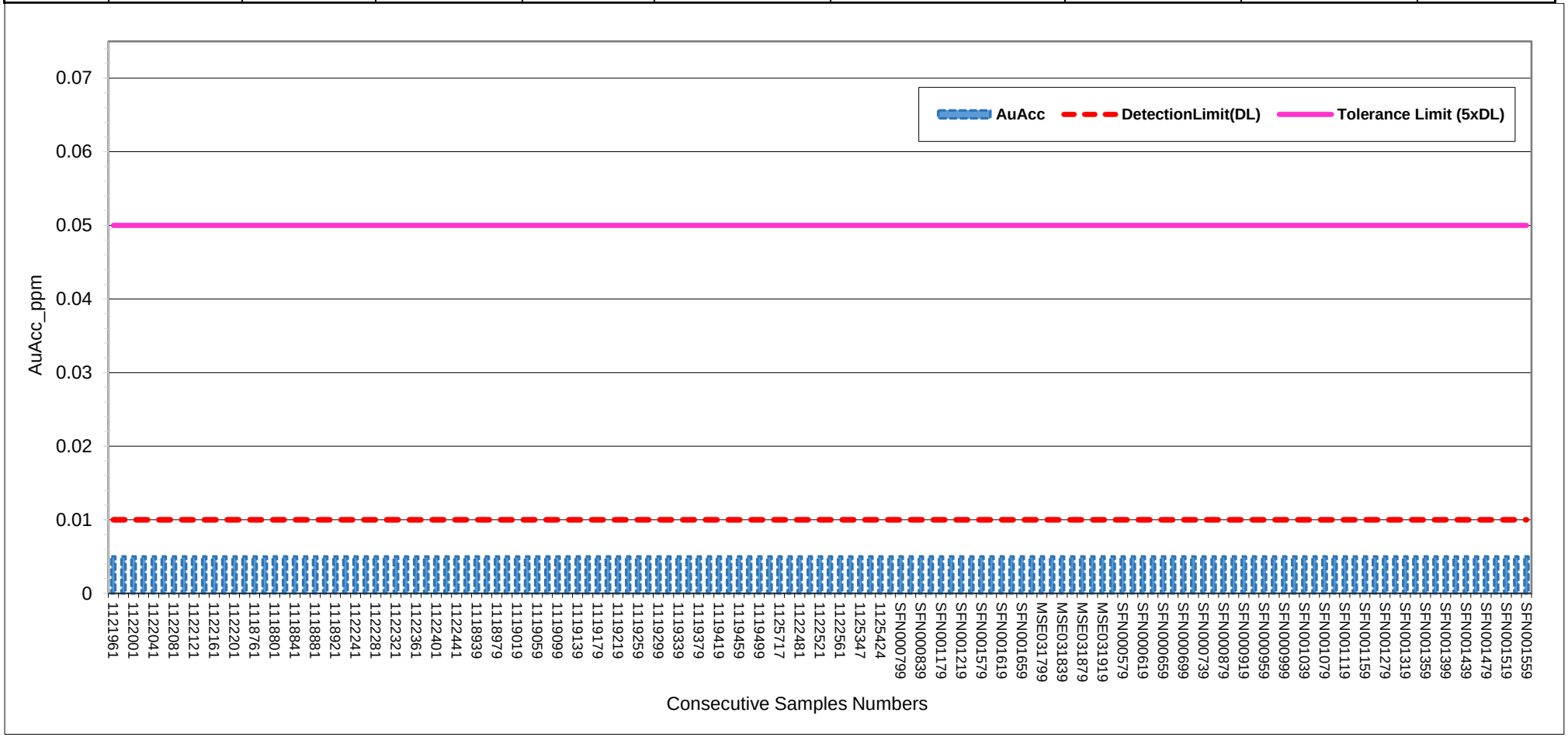

Figure 30-28 Sofia Blank Samples Assayed (Fire Assay) by SGS Ouagadougou

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.02	635	<0.01	0	13 (2.05%)	622 (97.95%)	635 (100%)	0

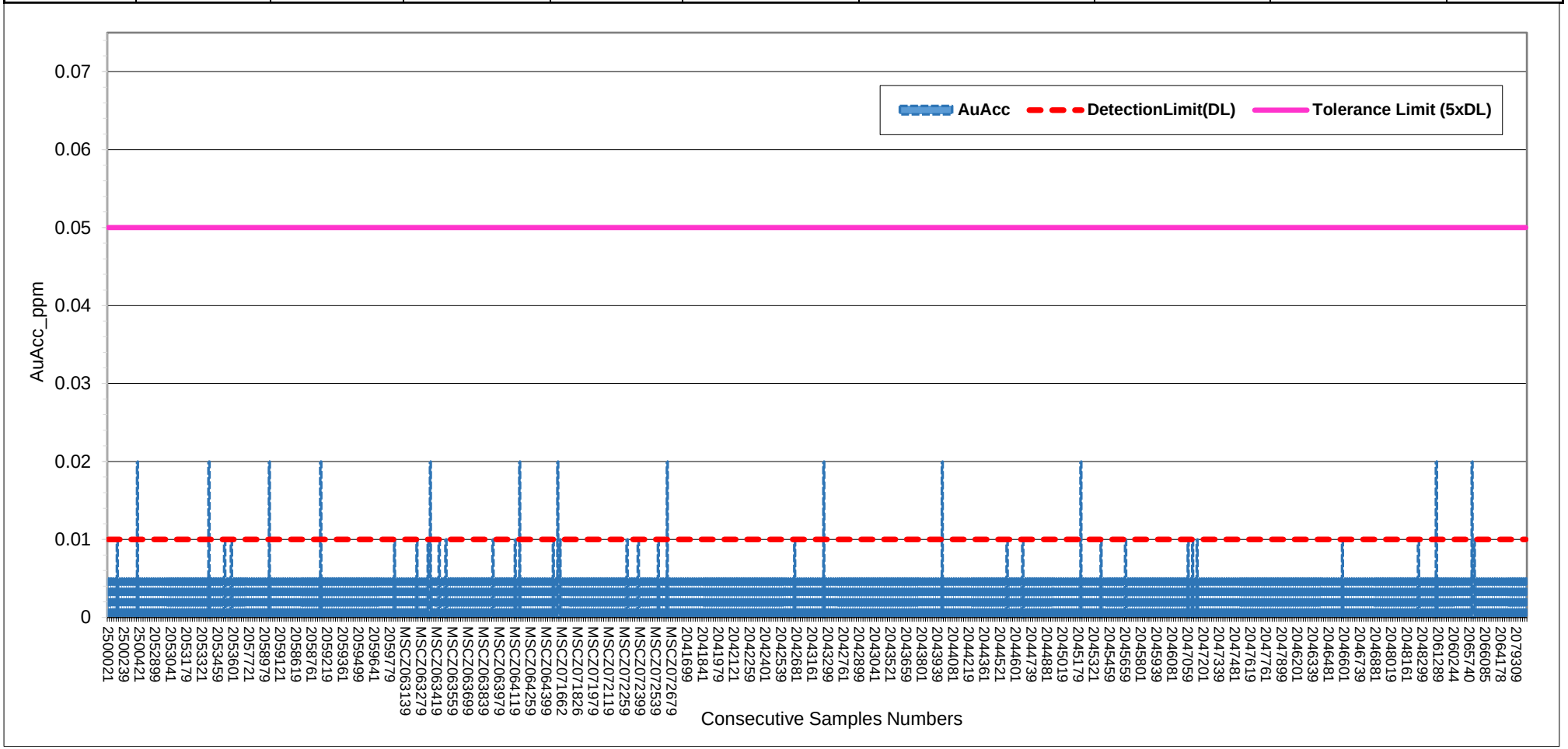

Figure 30-29 Massawa Blank Samples Assayed (Fire Assay) by SGS Bamako

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.01	229	<0.01	0	0	229 (100%)	229 (100%)	0

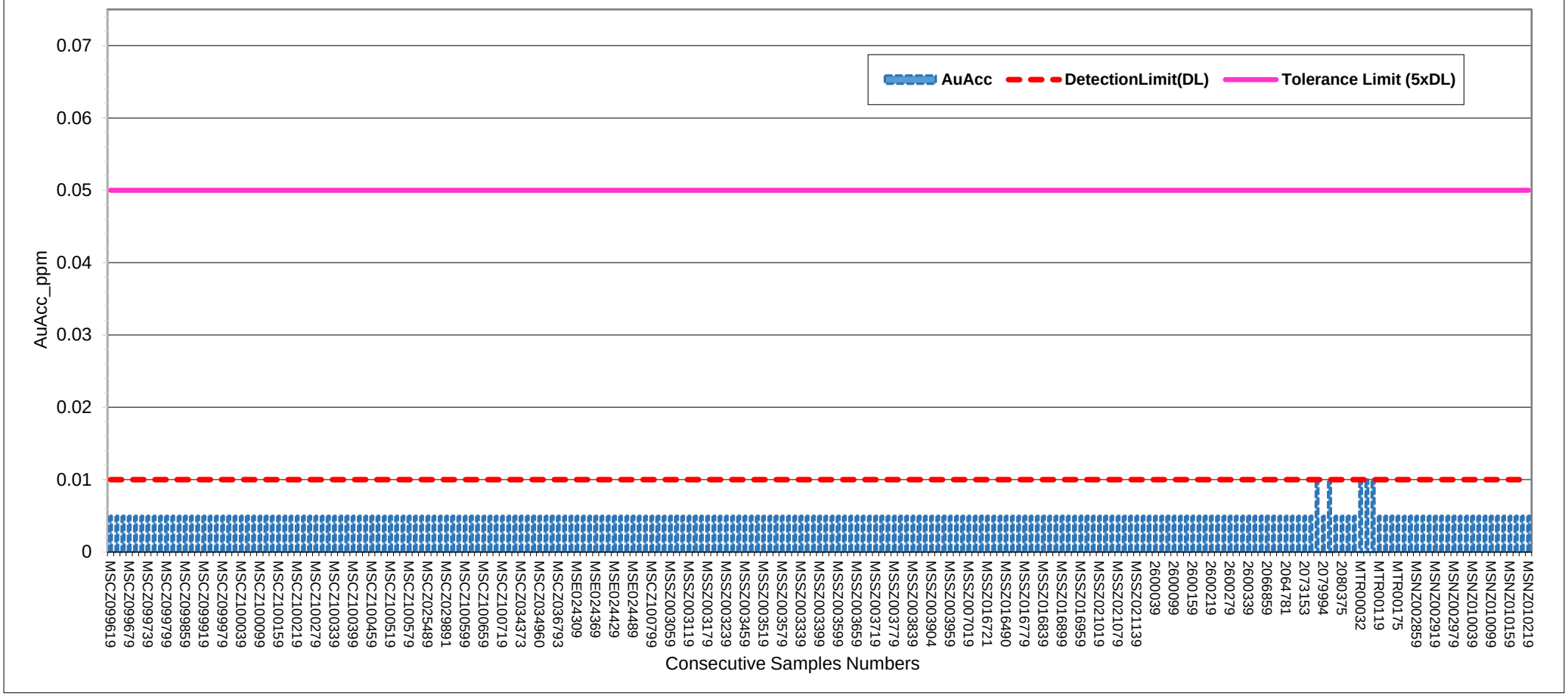

Figure 30-30 Massawa Blank Samples Assayed (Fire Assay) by SGS Loulo

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.02	220	<0.01	0	11 (5.00%)	209 (95.00%)	220 (100%)	0

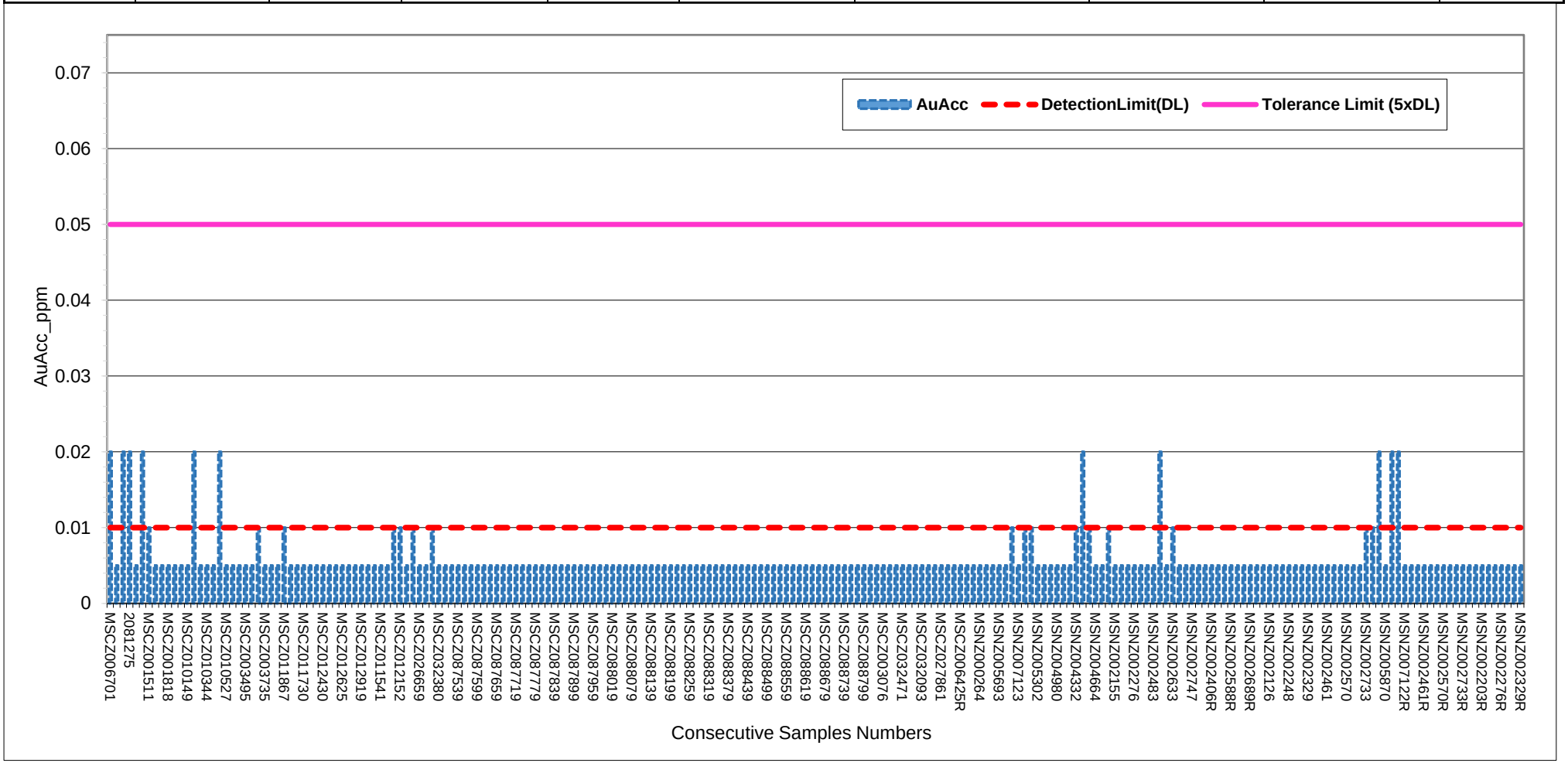

Figure 30-31 Massawa Blank Samples Assayed (Fire Assay) by SGS Tarkwa

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	0.02	345	<0.01	0	1 (0.29%)	344 (99.71%)	345 (100%)	0

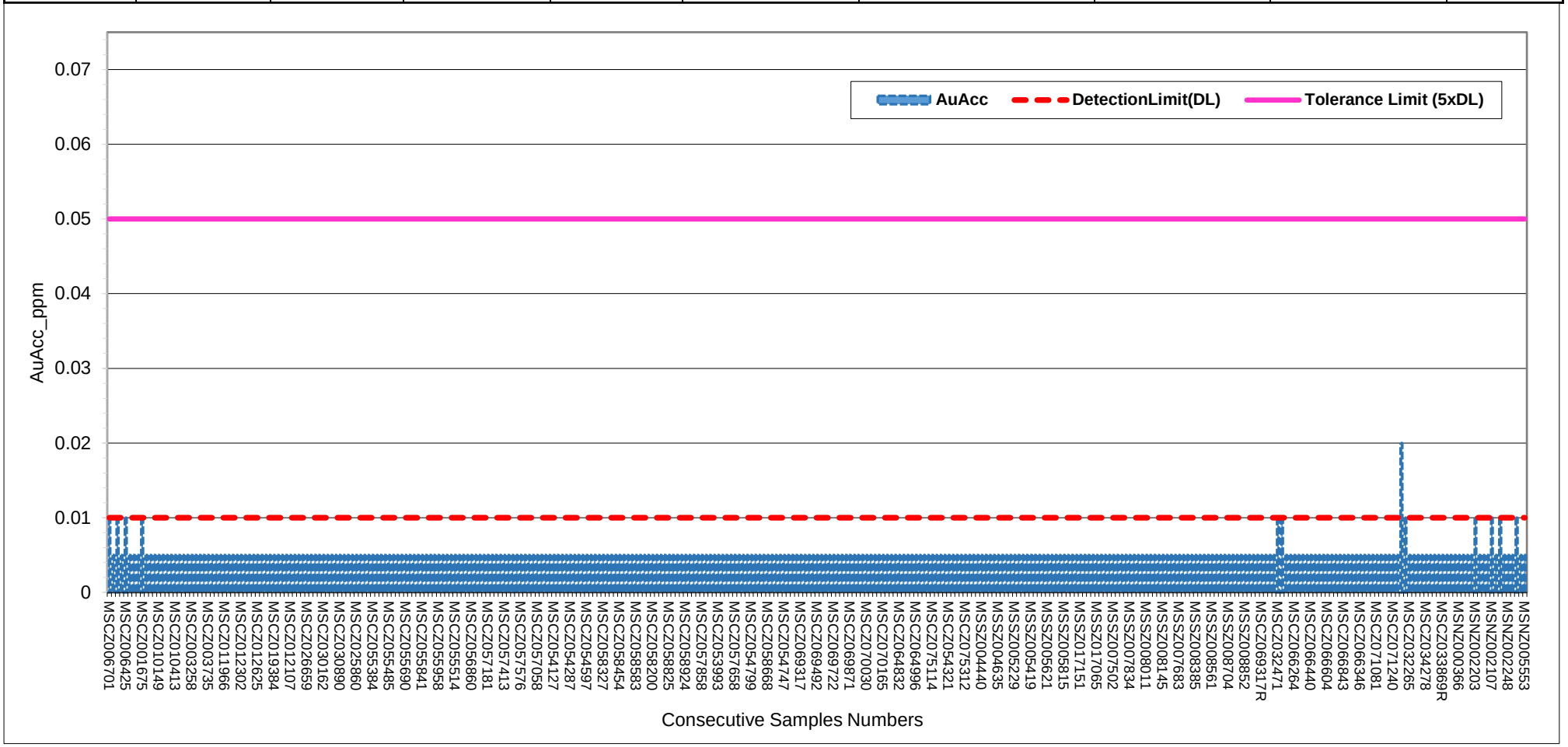

Figure 30-32 Massawa Blank Samples Assayed (LeachWELL) by SGS Tarkwa

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	2.09	943	<0.01	1 (0.11%)	11 (1.17%)	932 (98.83%)	942 (99.89%)	1 (0.11%)

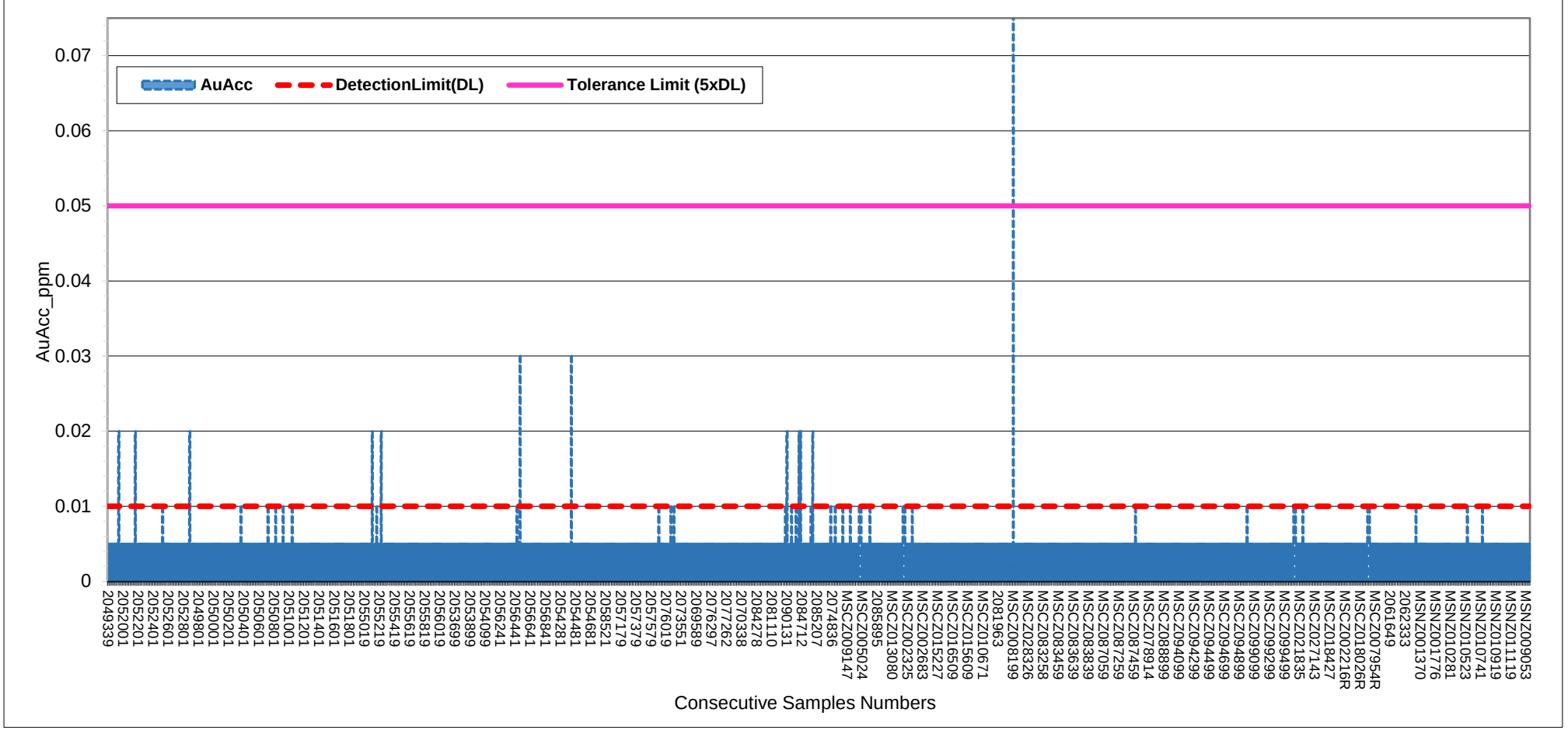

Figure 30-33 Massawa Blank Samples Assayed (Fire Assay) by SGS Ouagadougou

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.005	2.07	2471	<0.01	3 (0.12%)	14 (0.57%)	2457 (99.43%)	2468 (99.88%)	3 (0.12%)

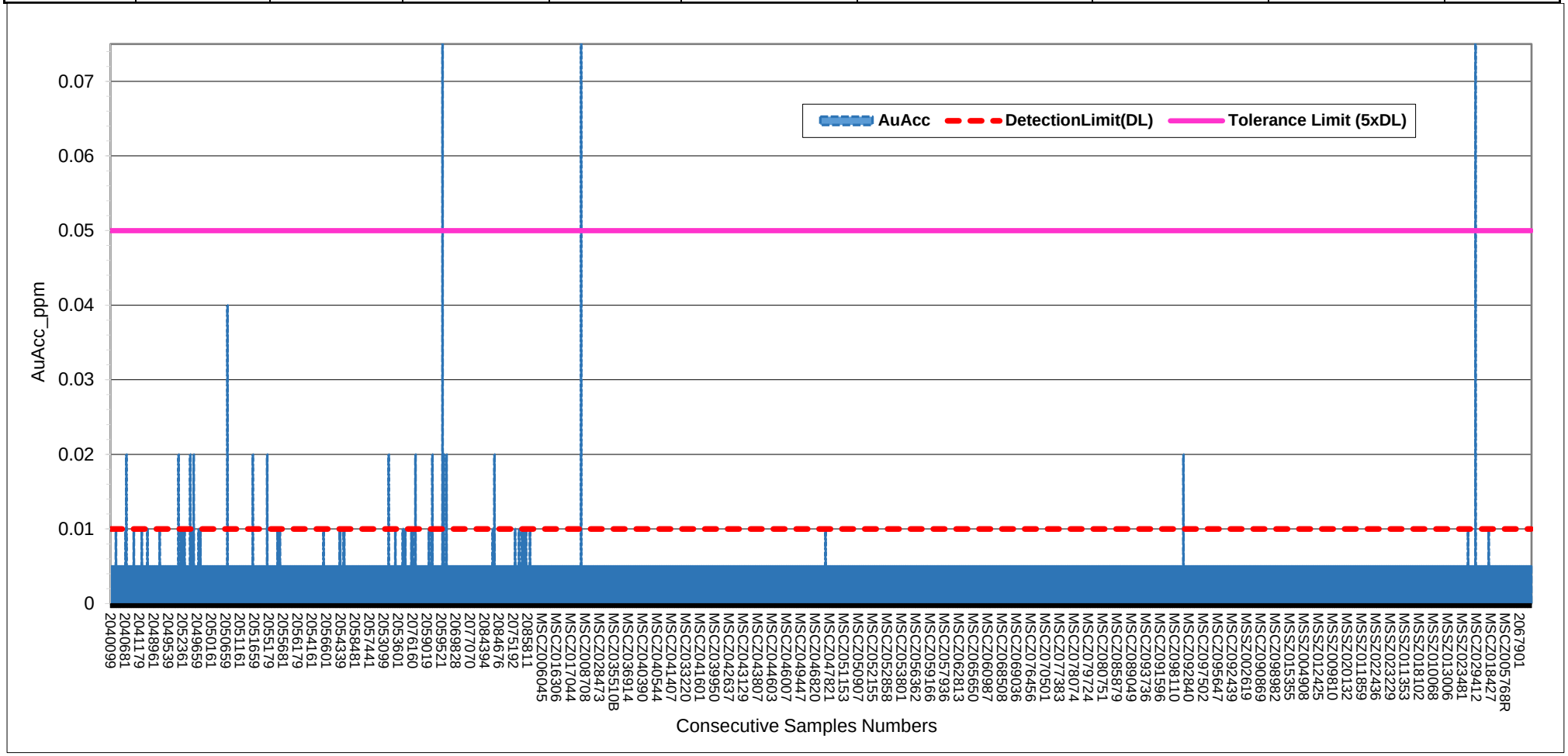

Figure 30-34 Massawa Blank Samples Assayed (LeachWELL) by SGS Ouagadougou

Type	Min Assay	Max Assay	N°Samples	Expected	Above TL	Between DL and TL	Below DL	%Pass	% Fail
Blank	0.0005	0.0005	251	<0.001	0	0	251 (100%)	251 (100%)	0

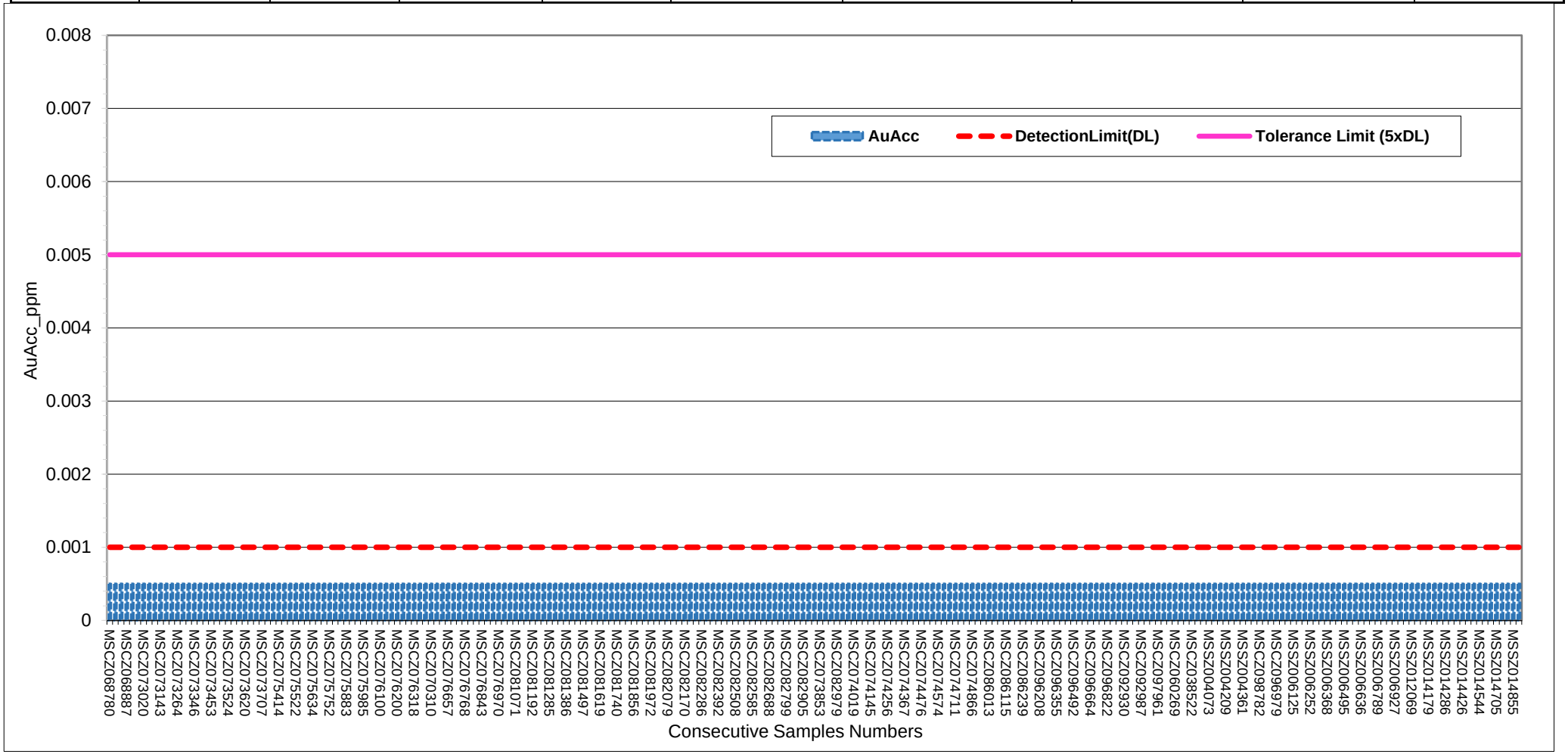

Figure 30-35 Massawa Blank Samples Assayed (LeachWELL) by BIGS Ouagadougou

30.3. CRMs

Figure 30-36 to Figure 30-47 show details of CRMs used and graphical representation of the performance of each CRM by all laboratories during the period.

SUMMARY OF DELYA CRMs ASSAYED (FA) BY SGS LOULO DURING THE PERIOD

CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 202	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.752	0.83	0.674	0.69	0.77	7
OREAS 209	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.58	1.712	1.448	1.51	1.58	2
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.49	5.946	5.034	5.46	5.6	11
OREAS 214	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.03	3.276	2.784	2.98	3.12	4
OREAS 217	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.338	0.368	0.308	0.32	0.37	77
OREAS 218	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.531	0.582	0.48	0.51	0.55	42
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.81	0.86	96
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.06	1.168	0.952	0.96	1.1	15
OREAS 222	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.319	1.121	1.21	1.27	24
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.15	2.309	1.991	2.1	2.2	14
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.29	0.33	113
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.48	0.54	155
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.63	0.71	122
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.352	1.088	1.19	1.28	47
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.55	2.778	2.322	2.49	2.63	31
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.08	4.341	3.819	4	4.23	13
OREAS 256	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	7.66	8.374	6.946	7.59	7.7	9
								782

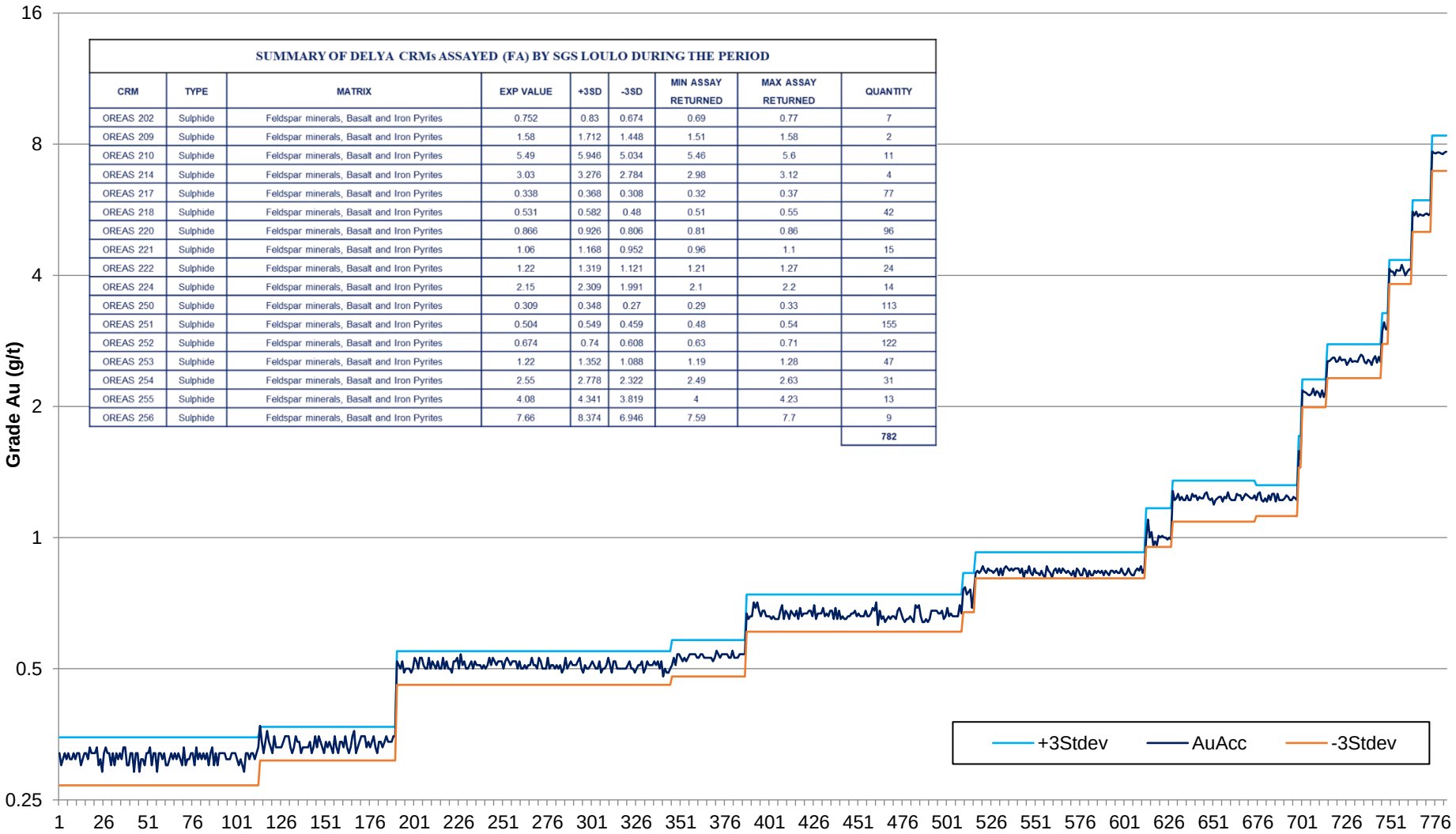

Figure 30-36 Tram Line of Delya CRMs Assayed (Fire Assay) by SGS Loulo Laboratory

SUMMARY OF DELYA CRMs ASSAYED (FA) BY SGS OUAGA DURING THE PERIOD

CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.49	5.946	5.034	5.04	5.26	5
OREAS 228	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	8.73	9.567	7.893	8.5	9.16	3
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.31	0.34	10
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.5	0.52	4
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.67	0.7	6
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.55	2.778	2.322	2.47	2.62	5
								33

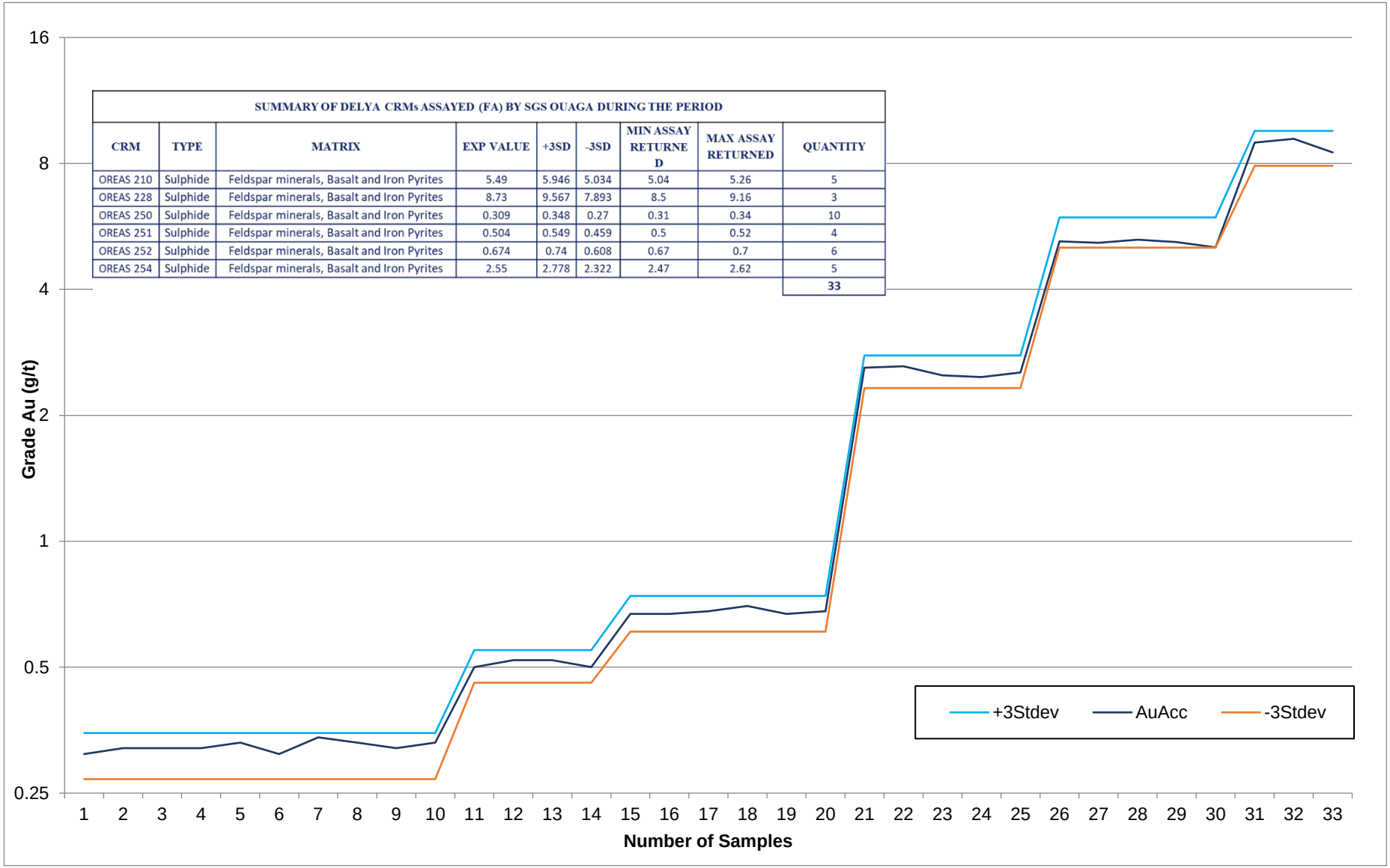

Figure 30-37 Tram Line of Delya CRMs Assayed (Fire Assay) by SGS Ouagadougou Laboratory

SUMMARY OF SOFIA CRMs ASSAYED (FA) BY SGS BAMAKO DURING THE PERIOD								
CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 202	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.752	0.83	0.674	0.69	0.8	20
OREAS 206	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.197	2.44	1.954	2.2	2.28	5
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.84	0.89	43
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.06	1.168	0.952	1.05	1.11	6
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.3	0.39	47
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.5	0.53	8
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.6	0.71	37
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.55	2.778	2.322	2.43	2.63	8
								174

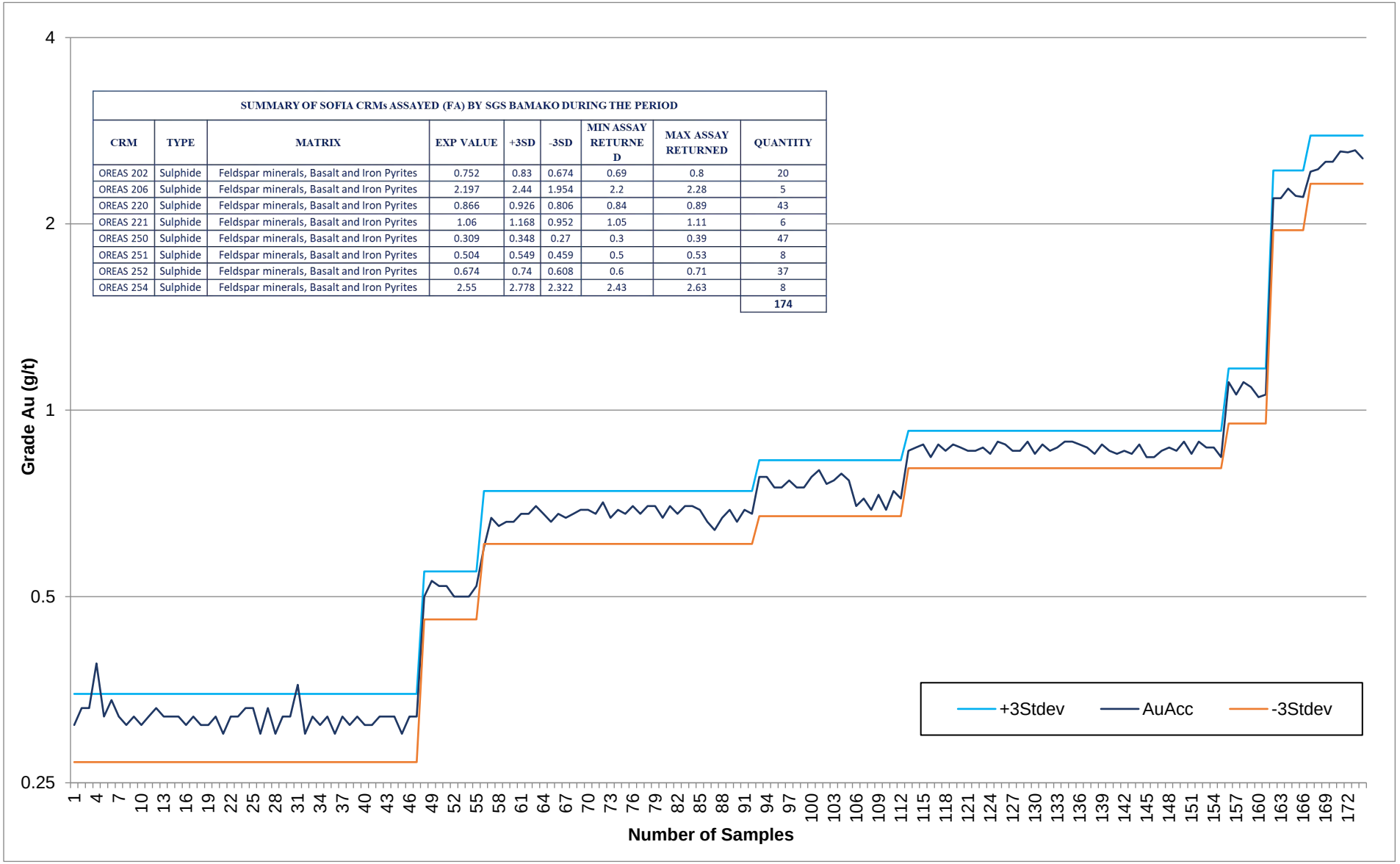

Figure 30-38 Tram Line of Sofia CRMs Assayed (Fire Assay) by SGS Bamako Laboratory

SUMMARY OF SOFIA CRMs ASSAYED (FA) BY SGS LOULO DURING THE PERIOD

CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 202	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.752	0.83	0.674	0.73	0.82	8
OREAS 203	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.871	0.961	0.781	0.84	0.9	34
OREAS 209	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.580	1.712	1.448	1.58	1.6	3
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.490	5.946	5.034	5.51	5.58	5
OREAS 214	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.030	3.276	2.784	2.92	3.05	16
OREAS 215	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.540	3.831	3.249	3.46	3.62	9
OREAS 217	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.338	0.368	0.308	0.33	0.35	6
OREAS 218	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.531	0.582	0.48	0.53	0.55	10
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.81	0.88	295
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.060	1.168	0.952	0.97	1.1	141
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.150	2.309	1.991	2	2.2	9
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.28	0.33	139
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.49	0.53	36
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.65	0.7	27
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.220	1.352	1.088	1.21	1.26	11
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.550	2.778	2.322	2.51	2.62	12
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.080	4.341	3.819	4.13	4.14	2
								763

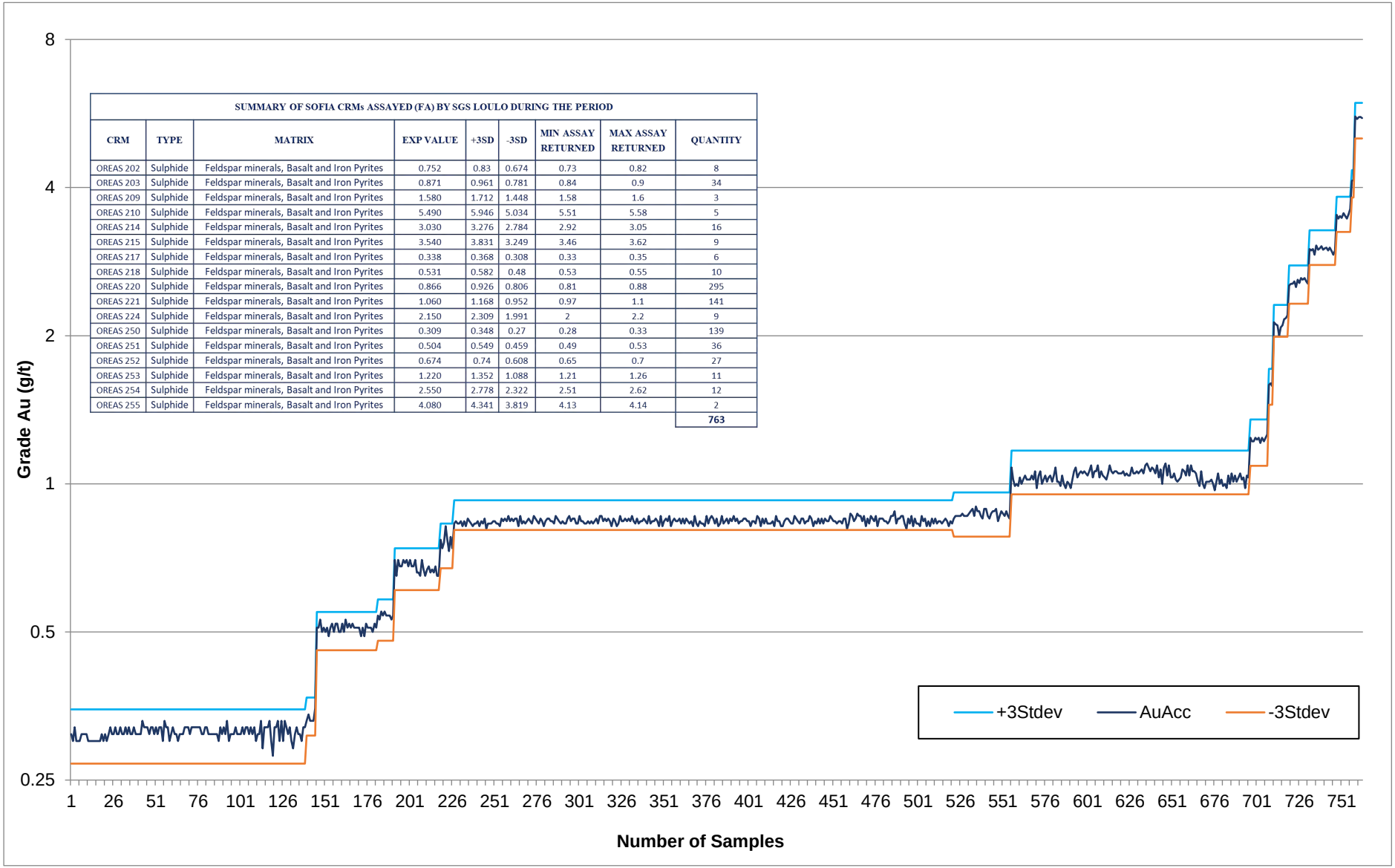

Figure 30-39 Tram Line of Sofia CRMs Assayed (Fire Assay) by SGS Loulo Laboratory

SUMMARY OF SOFIA CRMs ASSAYED (FA) BY SGS OUAGA DURING THE PERIOD								
CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 202	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.752	0.83	0.674	0.71	0.79	55
OREAS 206	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.197	2.44	1.954	2.17	2.26	10
OREAS 209	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.580	1.712	1.448	1.55	1.62	6
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.490	5.946	5.034	5.5	5.65	6
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.3	0.34	24
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.65	0.7	27
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.220	1.352	1.088	1.17	1.28	8
								136

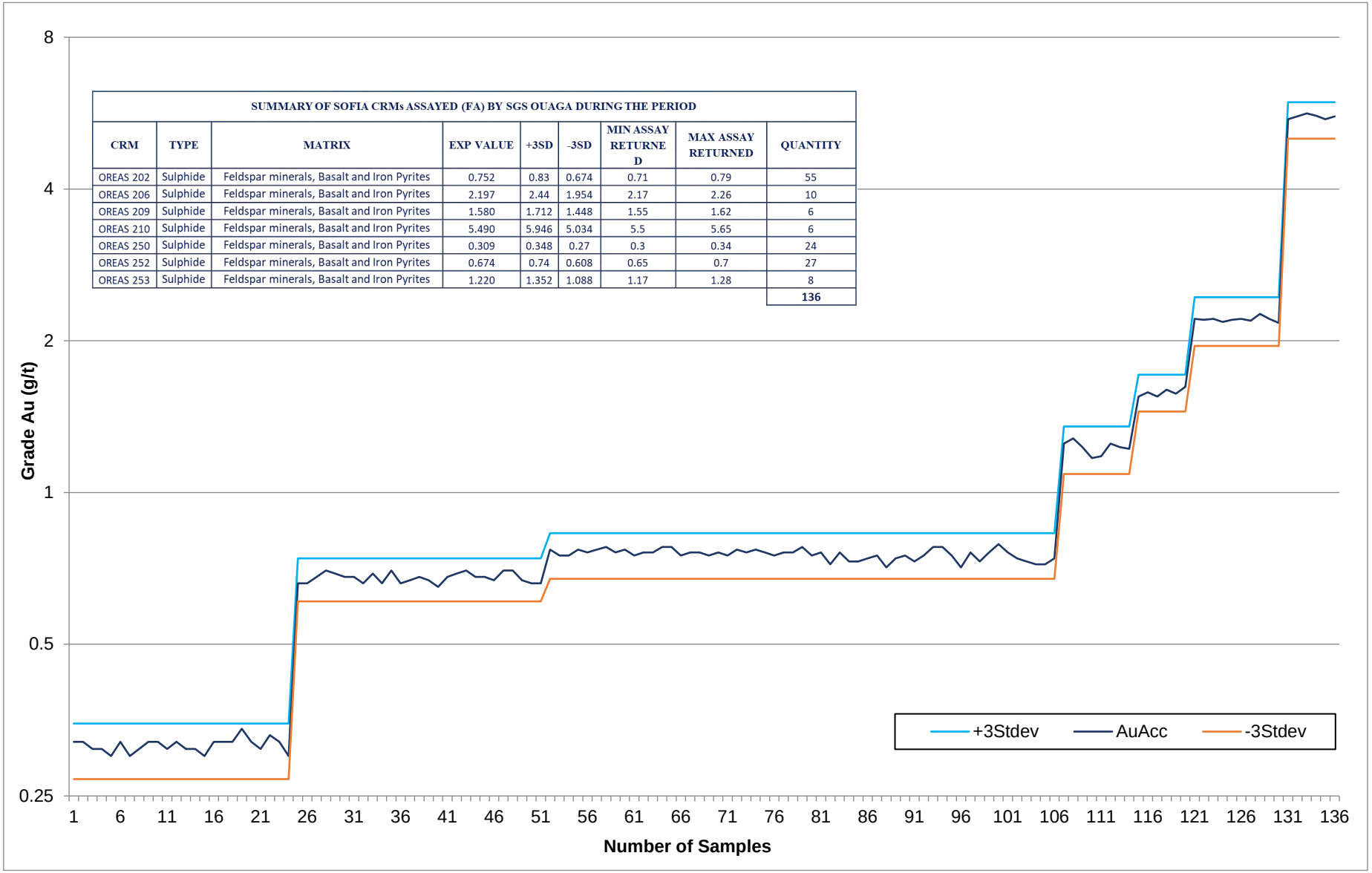

Figure 30-40 Tram Line of Sofia CRMs Assayed (Fire Assay) by SGS Ouagadougou Laboratory

SUMMARY OF MASSAWA CRMs ASSAYED (FA) BY SGS BAMAKO DURING THE PERIOD

CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 202	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.752	0.83	0.674	0.69	0.83	82
OREAS 203	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.871	0.961	0.781	0.84	0.87	2
OREAS 206	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.197	2.44	1.954	2.1	2.3	25
OREAS 208	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	9.248	10.562	7.934	9.4	9.6	2
OREAS 209	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.580	1.712	1.448	1.5	1.61	4
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.490	5.946	5.034	4.99	5.78	12
OREAS 214	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.030	3.276	2.784	3.04	3.19	5
OREAS 215	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.540	3.831	3.249	3.28	3.58	24
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.060	1.168	0.952	1.02	1.16	64
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.29	0.33	85
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.49	0.53	11
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.65	0.72	133
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.220	1.352	1.088	1.16	1.28	46
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.550	2.778	2.322	2.43	2.73	92
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.080	4.341	3.819	4.26	4.52	12
OREAS 216	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	6.66	7.125	6.195	6.58	6.65	2
								601

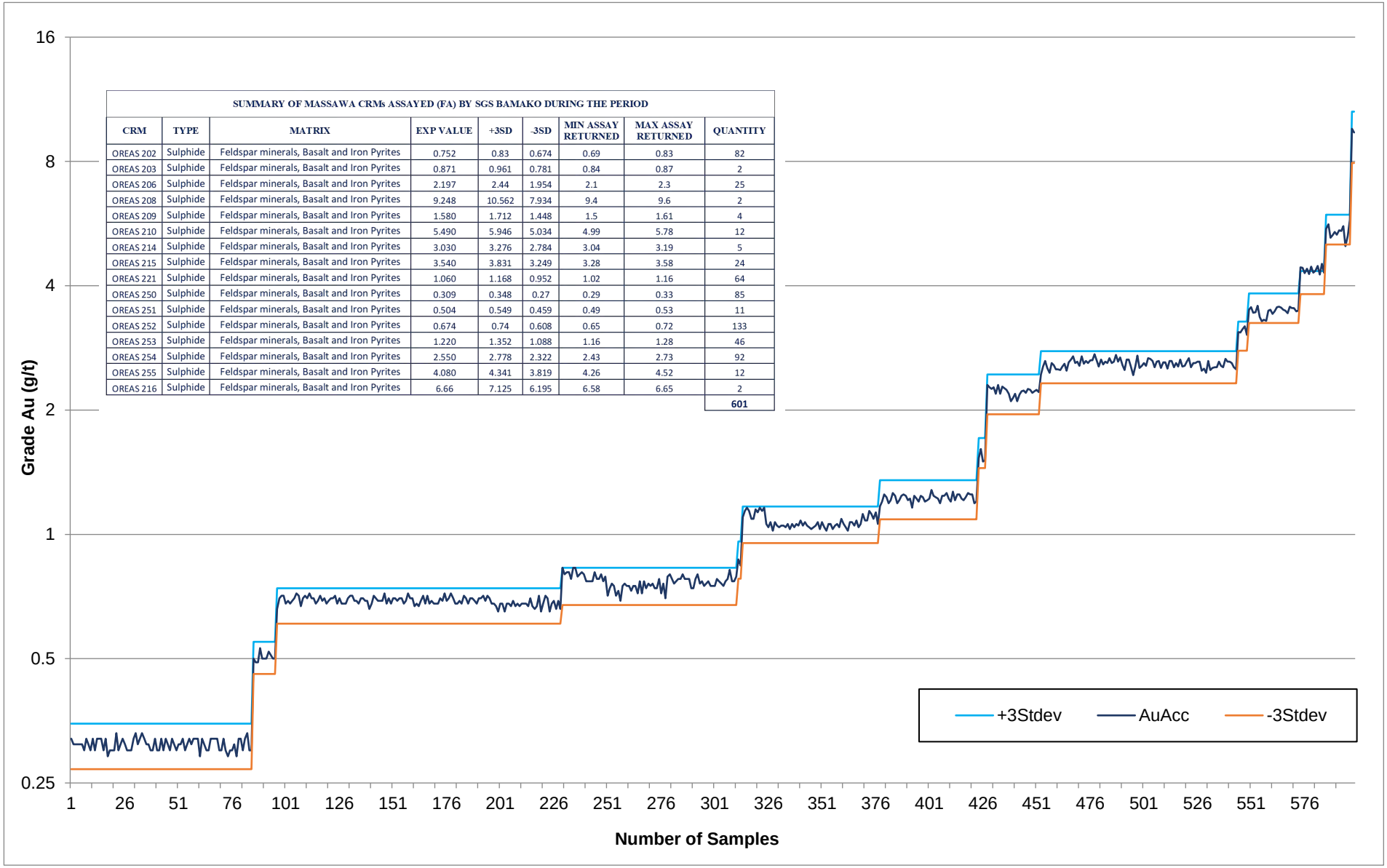

Figure 30-41 Tram Line of Massawa CRMs Assayed (Fire Assay) by SGS Bamako Laboratory

SUMMARY OF MASSAWA CRMs ASSAYED (FA) BY SGS LOULO DURING THE PERIOD

CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 203	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.871	0.961	0.781	0.85	0.88	5
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.490	5.946	5.034	5.44	5.65	19
OREAS 214	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.030	3.276	2.784	2.93	3.11	21
OREAS 215	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.54	3.831	3.249	3.46	3.56	4
OREAS 217	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.338	0.368	0.308	0.33	0.35	7
OREAS 218	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.531	0.582	0.48	0.52	0.53	3
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.83	0.86	8
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.06	1.168	0.952	0.99	1.05	7
OREAS 222	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.319	1.121	1.22	1.24	3
OREAS 223	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.780	1.915	1.645	1.76	1.84	15
OREAS 228	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	8.730	9.567	7.893	8.56	8.66	4
OREAS 229	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	12.110	12.728	11.492	12	12.2	2
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.3	0.33	26
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.48	0.53	50
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.66	0.71	10
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.352	1.088	1.22	1.26	4
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.550	2.778	2.322	2.5	2.67	28
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.08	4.341	3.819	4.11	4.14	4
OREAS 256	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	7.660	8.374	6.946	7.55	7.88	5
								225

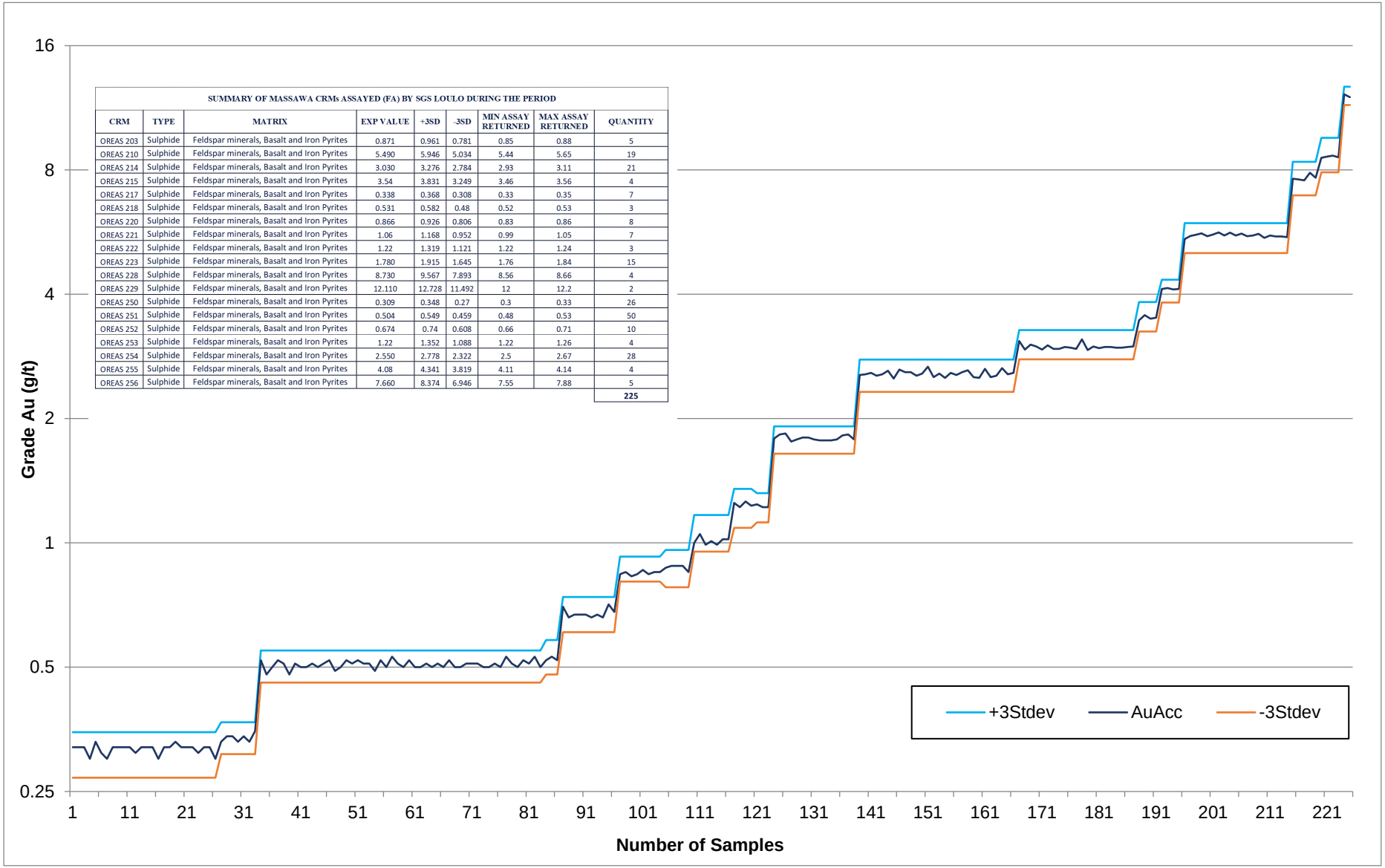

Figure 30-42 Tram Line of Massawa CRMs Assayed (Fire Assay) by SGS Loulo Laboratory

SUMMARY OF MASSAWA CRMs ASSAYED (FA) BY SGS TARKWA DURING THE PERIOD								
CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.490	5.946	5.034	5.26	5.67	10
OREAS 214	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.030	3.276	2.784	3	3.05	9
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.82	0.87	20
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.060	1.168	0.952	0.96	1.16	30
OREAS 222	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.220	1.319	1.121	1.21	1.23	3
OREAS 223	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.780	1.915	1.645	1.76	1.8	9
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.150	2.309	1.991	2.01	2.3	27
OREAS 228	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	8.730	9.567	7.893	7.9	8.77	6
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.3	0.31	16
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.5	0.51	16
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.6	0.74	14
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.220	1.352	1.088	1.1	1.3	8
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.550	2.778	2.322	2.33	2.61	32
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.080	4.341	3.819	3.98	4.08	5
								205

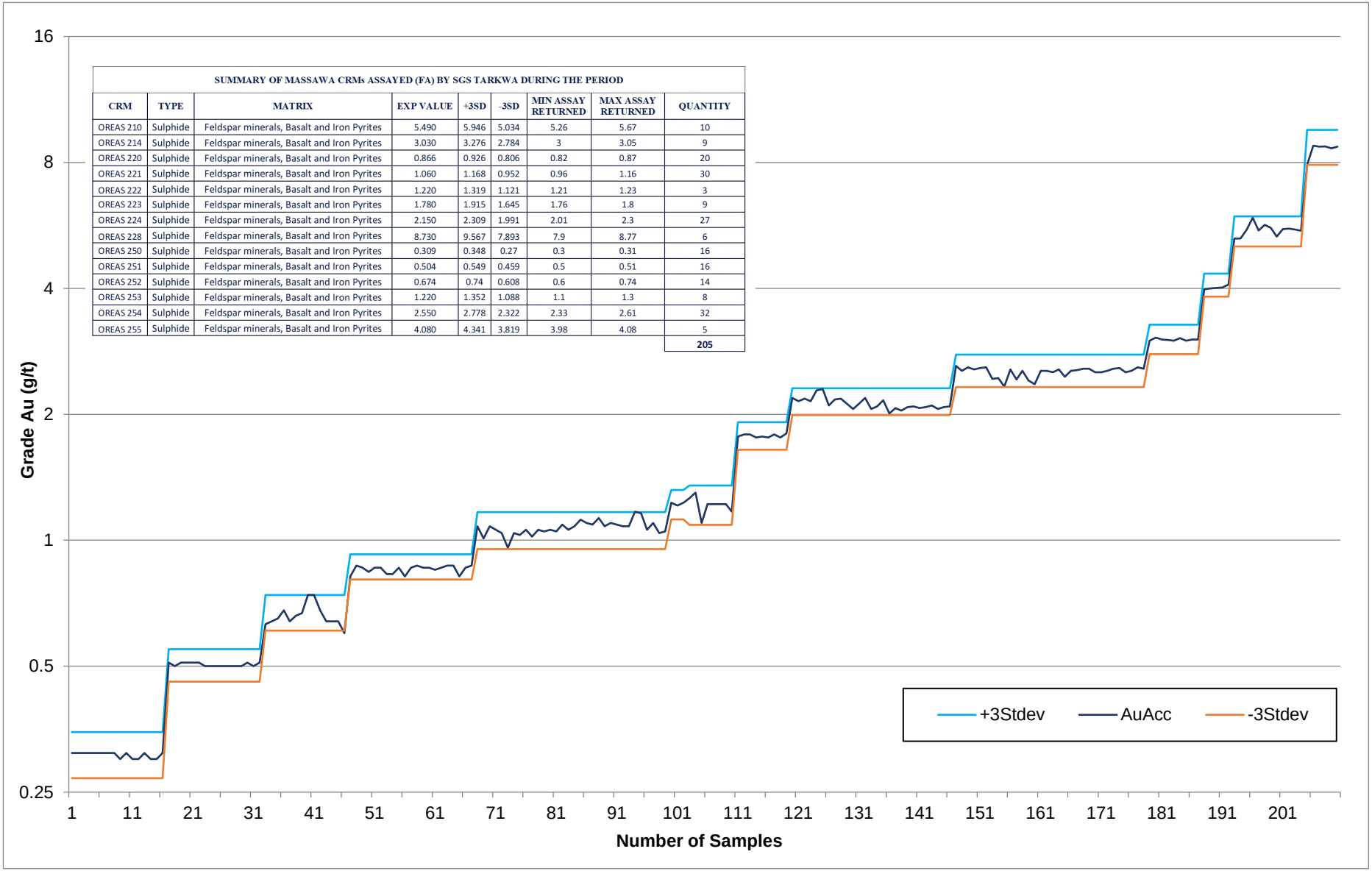

Figure 30-43 Tram Line of Massawa CRMs Assayed (Fire Assay) by SGS Tarkwa Laboratory

SUMMARY OF MASSAWA CRMs ASSAYED (LW) BY SGS TARKWA DURING THE PERIOD								
CRM	TPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.49	5.946	5.034	4.94	5.51	31
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.82	1.03	43
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.06	1.168	0.952	1	1.09	41
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.15	2.309	1.991	2.02	2.3	39
OREAS 228	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	8.73	9.567	7.893	8.73	8.8	4
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.62	0.72	32
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.352	1.088	1.09	1.29	55
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.55	2.778	2.322	2.47	2.58	34
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.08	4.341	3.819	4.08	4.08	2
OREAS 257	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	14.18	14.972	13.388	14	14.2	38
								319

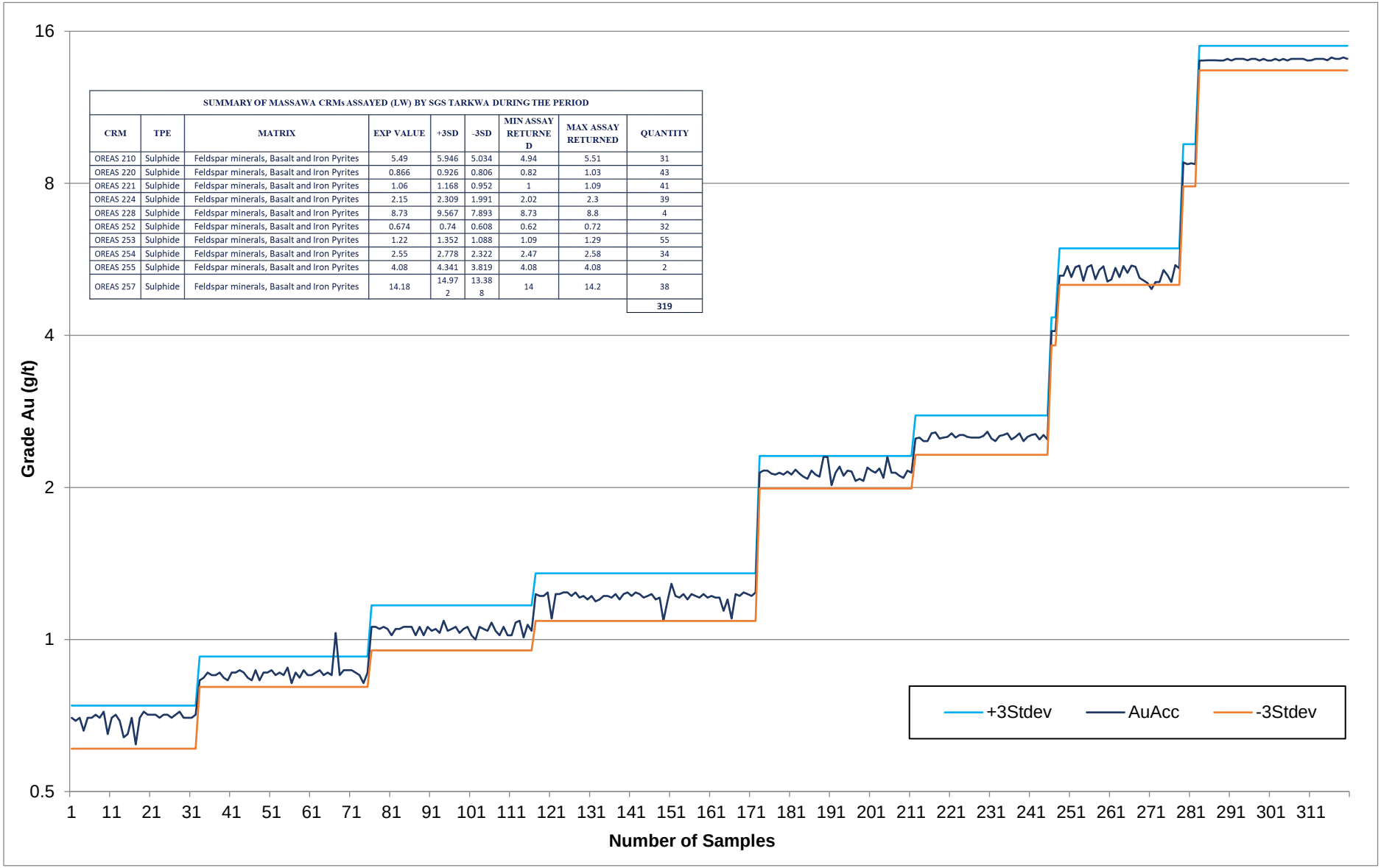

Figure 30-44 Tram Line of Massawa CRMs Assayed (LeachWELL) by SGS Tarkwa Laboratory

SUMMARY OF MASSAWA CRMs ASSAYED (FA) BY SGS OUAGA DURING THE PERIOD

CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 202	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.752	0.83	0.674	0.69	0.79	86
OREAS 206	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.197	2.44	1.954	2.1	2.25	57
OREAS 208	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	9.248	10.562	7.934	9.33	10	2
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.490	5.946	5.034	5.07	5.68	46
OREAS 214	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.030	3.276	2.784	2.86	3.16	18
OREAS 215	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	3.540	3.831	3.249	3.49	3.65	61
OREAS 216	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	6.660	7.125	6.195	6.55	6.69	6
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.81	0.9	63
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.060	1.168	0.952	0.98	1.14	94
OREAS 223	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.780	1.915	1.645	1.74	1.85	63
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.150	2.309	1.991	2	2.18	47
OREAS 228	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	8.730	9.567	7.893	8.48	8.93	4
OREAS 229	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	12.110	12.728	11.492	11.5	12.1	3
OREAS 250	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.309	0.348	0.27	0.3	0.34	97
OREAS 251	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.504	0.549	0.459	0.49	0.54	45
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.66	0.74	57
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.220	1.352	1.088	1.08	1.27	32
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.550	2.778	2.322	2.45	2.69	111
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.080	4.341	3.819	3.98	4.18	27
OREAS 256	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	7.660	8.374	6.946	7.5	7.66	5
								924

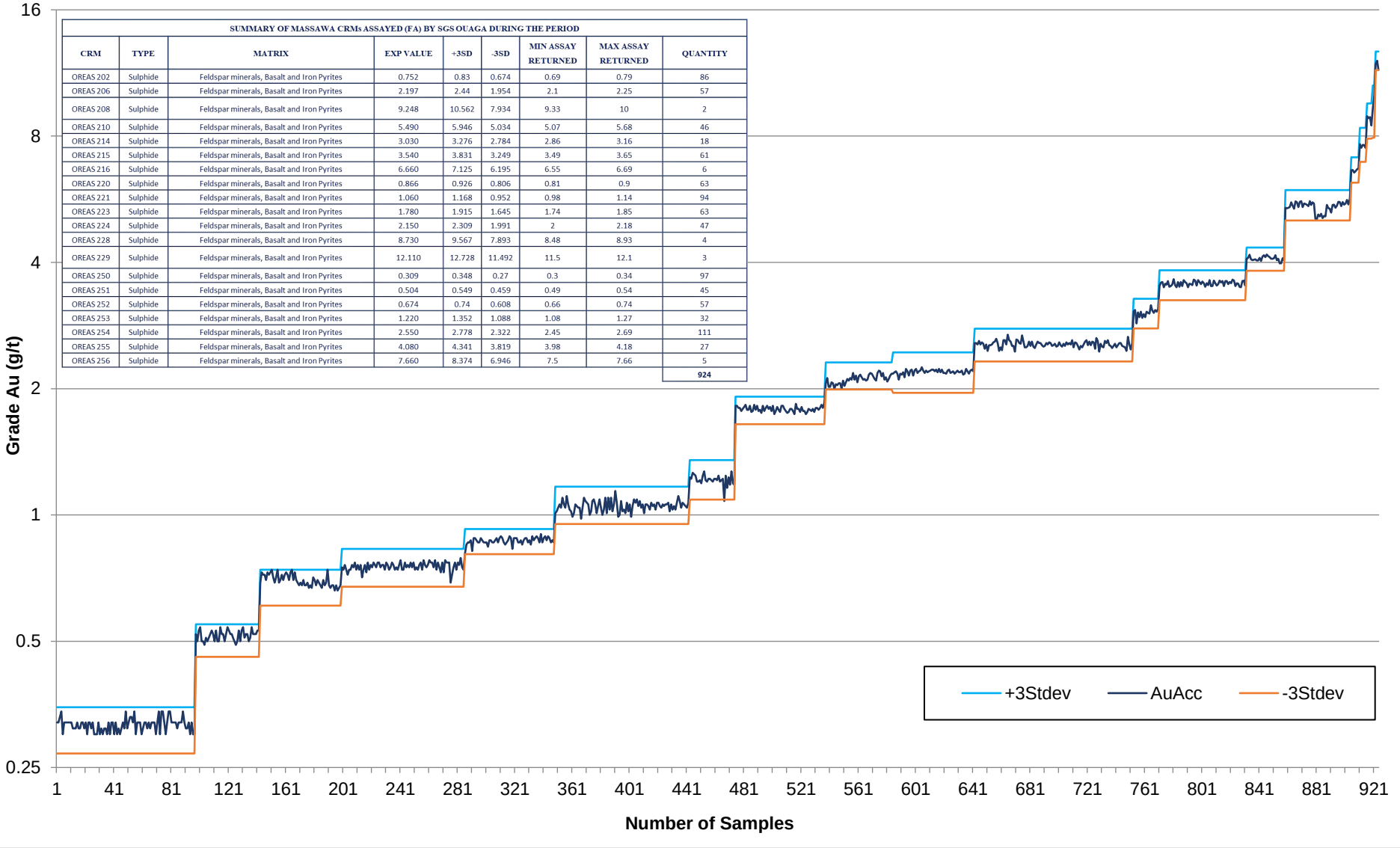

Figure 30-45 Tram Line of Massawa CRMs Assayed (Fire Assay) by SGS Ouagadougou Laboratory

SUMMARY OF MASSAWA CRMs ASSAYED (LW) BY SGS OUAGA DURING THE PERIOD

CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.49	5.946	5.034	5.11	5.69	152
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.84	0.925	243
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.06	1.168	0.952	1.02	1.155	281
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.15	2.309	1.991	2.03	2.26	146
OREAS 228	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	8.73	9.567	7.893	8.645	9.16	63
OREAS 229	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	12.11	12.728	11.492	12.075	12.305	4
OREAS 252	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.674	0.74	0.608	0.645	0.715	195
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.352	1.088	1.185	1.31	167
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.55	2.778	2.322	2.485	2.665	173
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.08	4.341	3.819	3.965	4.245	172
OREAS 257	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	14.18	14.972	13.388	13.88	14.55	115
								1711

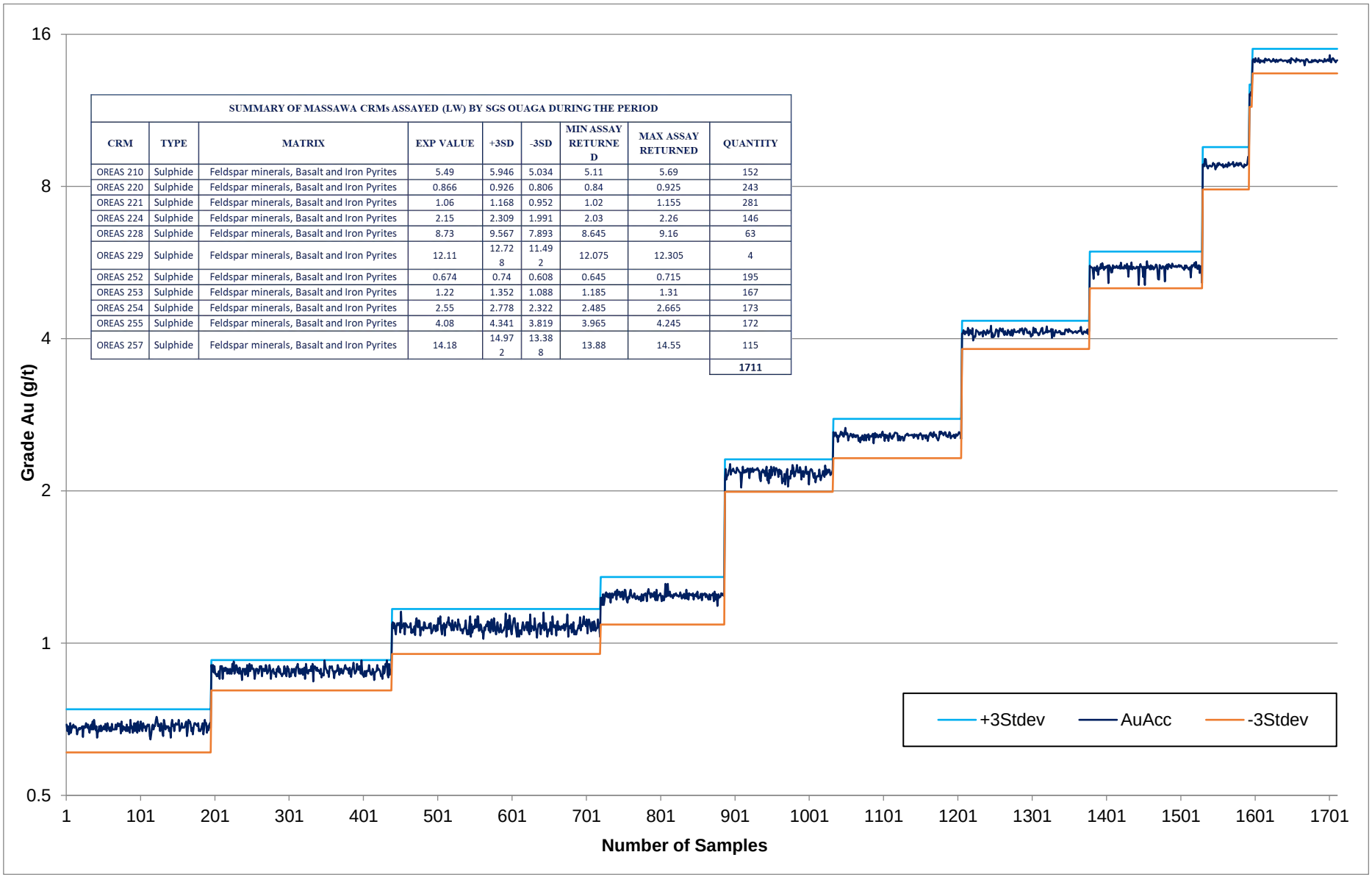

Figure 30-46 Tram Line of Massawa CRMs Assayed (LeachWELL) by SGS Ouagadougou Laboratory

SUMMARY OF MASSAWA CRMs ASSAYED (LW) BY BIGS OUAGA DURING THE PERIOD								
CRM	TYPE	MATRIX	EXP VALUE	+3SD	-3SD	MIN ASSAY RETURNED	MAX ASSAY RETURNED	QUANTITY
OREAS 210	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	5.49	5.946	5.034	5.536	5.83	16
OREAS 220	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	0.866	0.926	0.806	0.8275	0.8545	65
OREAS 221	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.06	1.168	0.952	1.065	1.16	24
OREAS 224	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.15	2.309	1.991	2.133	2.236	7
OREAS 253	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	1.22	1.352	1.088	1.213	1.257	17
OREAS 254	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	2.55	2.778	2.322	2.519	2.583	15
OREAS 255	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	4.08	4.341	3.819	4.116	4.236	59
OREAS 257	Sulphide	Feldspar minerals, Basalt and Iron Pyrites	14.18	14.972	13.388	13.985	14.288	47
								250

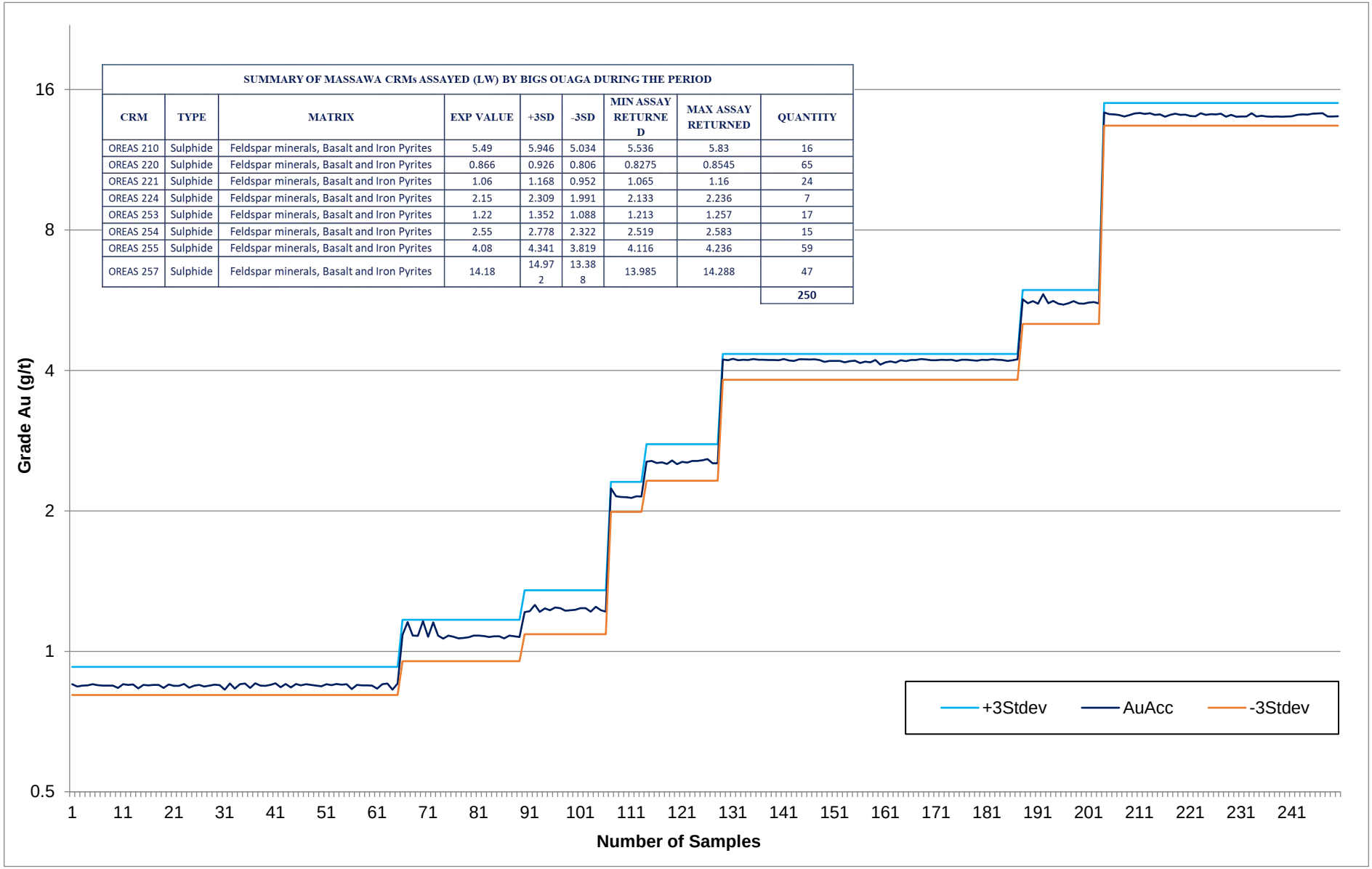

Figure 30-47 Tram Line of Massawa CRMs Assayed (LeachWELL) by BIGS Ouagadougou Laboratory

30.1. Appendix 1 – JORC 2012 Edition – Table 1 – Massawa Gold Project

The following table provides a summary of important assessment and reporting criteria used at the Massawa and Sofia deposits for the reporting of Mineral Resources and Ore Reserves in accordance with the Table 1 checklist in The Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code, 2012 Edition). Criteria in each section apply to all preceding and succeeding sections.

30.1.1. Section 1. Sampling Techniques and Data

JORC 2012 Checklist of Assessment and Reporting Criteria

Criteria	Commentary
Sampling techniques	• Diamond drilling (DD) core samples are taken within geological units, and are normally between 0.8 m and 1.5 m. The core is halved along the apex of the structure in the downhole direction. This is to have equal Au value for both halved core. o Massawa Central Zone (CZ) core is cut along the apex of the structure in the downhole direction within the two main mineralised structures, being hanging wall (VC, QFP, GAB) and footwall (GRW and CARSCH/CS). o Massawa Northern Zone (NZ) core is cut along the main ore zone (MOZ) structure host greywacke, greywacke/subordinate carbonaceous schist in the downhole direction. This is to have equal Au value for both halved core. o Sofia core is cut within the two main mineralised structures, being hanging wall (Quartz Diorite) and footwall (Gabbro). • Reverse circulation (RC) samples have been riffle split and composited to 1 m down hole for sampling. RC chips are stored in chip trays. • Grab, channel, and soil samples are used for early stage exploration only and not estimation. • Trenches have been cut into the oxide portion of the CZ and NZ pits of which 15 are included in the CZ resource database and eight are included in the NZ resource database.
Drilling techniques	• DD is utilised at Massawa for exploration and resource development. PQ rods (85.0 mm) and HQ rods (63.3 mm) are used in the weathered saprolite (oxides) with NQ (47.6 mm) being used in the un-weathered consolidated rock. DD core is stored in core trays that are marked up and logged before transfer to the core storage. • RC drilling is used at Massawa for grade control (GC) as well as exploration/resource development. RC chips are sieved and analysed by the geologist before being logged and placed in chip boxes for storage. • Rotary air blast (RAB) and Air Core drilling has previously been used at the Massawa Gold Project for exploration and sterilisation purposes, however, neither is used for resource estimation purposes. • Currently core orientation is measured using the 'Reflex ACT II Core Orientation Tool' (ACT). • All drill holes currently surveyed with Reflex EZ-GYRO multi shot system with measurements every 10 m down hole. Some historic holes measured with Reflex EZ-TRAC or Reflex conventional GYRO.

Criteria	Commentary
Drill sample recovery	• Core recovery is measured in the field and during detailed logging. Core loss is marked out clearly. • RC sample recovery is measured by weighing the total weight of sample collected over one-metre intervals and comparing this to the theoretical expected weight for the lithological unit and weathering type. • External Gilson riffle splitters are used on site for splitting RC samples and field duplicates. • Bias tests between equally represented areas of DD and RC drilling in NZ and Sofia Mineral Resources indicate that there is no issue with drilling support due to sample volume differences. • Bias tests between equally represented areas of DD and RC drilling in CZ Mineral Resources have shown that smaller sample sizes such as NQ and HQ core do not provide a fully representative repeatable sample, due to the nuggety nature of the vein related free gold. Consequently, all CZ drilling since 2013 has been completed with RC or PQ diamond core – which through field duplicate analysis have shown a good level of repeatability.
Logging	• DD core is logged geologically including weathering, mineralisation, alteration, lithology, structure, and redox. Data is logged directly into paper log sheets and the data is transcribed into an Excel entry template. This has been replaced with direct digital entry on rugged tablets using Maxwell LogChief during Q1 2017 so that data will auto synchronise with the newly installed onsite Maxwell Datashed SQL database. All drill core is photographed. • Geotechnical logging is only performed on holes specifically drilled for geotechnical assessment, as required. • RC chip samples are logged on one-metre intervals due to the sampling method. RC chips are sieved and placed into chip boxes for each interval.
Sub-sampling techniques and sample preparation	• Half core is cut for DD samples and submitted. Drill core length for each assay is determined by the geology and alteration. The core is halved along the apex of the structure in the downhole direction. This is to have equal Au value for both halved cores. • RC samples are submitted on one-metre intersections. Sub-sampling of RC intervals for GC and field duplicates is carried out at the rig using a riffle splitter. Wet samples are dried before manually splitting. • To date, field duplicates of CZ samples have shown a good correlation of results when using riffle splitters despite the nuggety nature of the free gold associated with the vein mineralisation. • To improve the splitting, all riffle splitters were replaced with Gilson splitters in Q1 2018. This is industry best practice for nuggety ore bodies. • Historical studies at Massawa NZ and Sofia ore bodies have confirmed there to be no significant grade bias through sample recovery rates. • CZ sample analysis has shown that smaller sample sizes such as half NQ and HQ core do not provide a fully representative sample that is repeatable, due to the high nugget factor within the high-grade (proximal) zones. Larger whole core and RC sampling methods are required to collect a sample that is representative enough, such that it can be split to produce a repeatable result when analysed using a bulk sample gold content analysis method. • Consequently, all CZ drilling since 2013 has been completed with RC or PQ diamond core, and the gold content has been analysed using bulk 1 kg LeachWELL (LWL69M) plus tail concentrate (FAA505T) fire gold content analysis. • Testwork completed to date has indicated that historic half core DD core data analysed with 50 g fire assay will underestimate the gold content relative to larger samples with bulk analysis methods; as such, this data is currently used as part of the estimation dataset within the Inferred portions of the CZ Mineral Resource.

Criteria	Commentary
Quality of assay data and laboratory tests	• All 50 g fire assay samples for Massawa NZ and Sofia have been analysed at either SGS Bamako or SGS Loulo in Mali. These samples are analysed using Fire Assay High Level (FAA505) with results in parts per million (ppm). • Since 2013, all Massawa CZ samples have been analysed by 1 kg LeachWELL (LWL69M) at SGS Ouagadougou, Burkina Faso. • Blank samples are course blank material. Currently these are inserted at least 1 in 20 (5%) with the logging geologist inserting them specifically within mineralised zones to improve contamination testing. Blank sample results show that cross sample contamination is currently low. • RC field duplicates are inserted at 1 in 20 (5%). DD field duplicates procedure has been updated to include coarse reject duplicates which are submitted blind to the laboratory. • Insertion of Certified Reference Materials (CRM) is following industry standard procedures. CRMs are currently inserted at 1 in 20 or 25 (4% to 5%). A limitation on the Massawa quality assurance and quality control (QA/QC) programme has been that no CRMs have been utilised within the 1 kg bulk sample LeachWELL analysis (LWL69M). However, Massawa CRMs inserted blind to the laboratory have been utilised to verify the 50 g fire assay of the remaining tail concentrate (FAA505T) from the 1 kg bulk sample LeachWELL analysis (LWL69M). This limitation has been addressed as part of the commencement of the 2017 feasibility programme where Massawa commissioned OREAS to create custom 1 kg matrix matched shear hosted orogenic gold CRMs that will be used as blind CRMs within the 1 kg bulk sample LeachWELL analysis (LWL69M). • From commencement of the feasibility programme in 2017 only pre-packaged CRMs were used, which eliminates any potential sample contamination at the internal prep stage. • A further limitation of the Massawa QA/QC programme is that umpire laboratory tests were last completed with the significant drill programme in 2010 and have not been routinely completed afterwards due to the non-continuous nature of the work programme. Various campaigns of round robin testwork has been carried out on the Massawa deposit during the pre-feasibility and feasibility programmes. These studies largely concentrated on the CZ and tested the reproducibility and reliability of the LeachWELL analysis and the SGS Ouagadougou laboratory. • An umpire assaying programme was previously completed on a significant drill programme in 2010. This umpire analysis tested the SGS Loulo and SGS Kayes (Mali) laboratories against the OMAC Laboratory, Ireland. These results indicated that the OMAC umpire results returned higher grades than those of SGS Loulo and SGS Kayes, although they were impacted by one significant outlier. Once the outlier was removed the results were significantly closer overall. A slight negative bias was outlined by Randgold against the SGS Loulo and SGS Kayes laboratory, however, both were considered as being acceptable, with the SGS Loulo laboratory noted for its higher precision. The more recent round-robin LeachWELL analysis has provided confidence in the general SGS Ouagadougou laboratory procedures and analysis in light of any umpire analysis.
Verification of sampling and assaying	• A number of RC and DD twins have been completed at Massawa CZ and NZ. The twins in NZ showed good repeatability of assays with an acceptable level of variability. • During 2013, six RC twins were drilled in CZ to utilise sample for the evaluation of different gold content analysis methods. This tested the repeatability of conventional 50 g fire assay, against 300 g screen fire assay, in addition to 1 kg and 2 kg LeachWELL sample analysis. The results indicated that 50 g fire assays have a low level of repeatability, 300 g screen fire assays had a moderate level of repeatability, relative to the bulk 1 kg and 2 kg LeachWELL analysis which showed a good level of repeatability. • The different size LeachWELL analysis tests (2 kg and 1 kg samples) showed no significant variance in the total gold content reported from the sample. However, significant differences were observed in the amount of gold recovered during leaching against the gold content in the leach tails, as such, the sample size selection was based upon optimisation of the leach recovered gold.

Criteria	Commentary
	• Consequently, all CZ drilling since 2013 has been completed with RC or PQ diamond core and the gold content has been analysed using bulk 1 kg LeachWELL (LWL69M) gold content analysis. • Testwork completed to date has indicated that historic drill sample data analysed with 50 g fire assay will underestimate the gold content relative to bulk analysis methods; as such, this data is currently used as part of the estimation dataset within the Inferred portions of the CZ Mineral Resource. • All project data is secured in an industry standard Maxwell Datashed SQL database for optimal validation through constraints, library tables, triggers, and stored procedures. All site software application databases will be set up to link back to the main database for information retrieval via ODBC. • During the migration to the SQL database in 2017, all assay data was migrated from the access database – but subsequently, assay data was re-imported directly from assay certificates from the laboratory and ranked such that they have a higher priority than the MS Access imported data. • Data loggers for collection of sample data in the field are used to minimise data transcription errors.
Location of data points	• The Massawa Project uses UTM ZONE 29N datum WGS84 grid coordinates. • All drill hole locations are surveyed using DGPS. • Since 2016, all downhole surveys are performed with a multi shot Reflex EZ-GYRO. Previously Reflex EZ-TRAC and reflex conventional GYROs were used. • Digital terrain model surfaces are created from LIDAR surveys.
Data spacing and distribution	• Resource classification is clearly defined for the deposit. The Mineral Resources are classified as Measured, Indicated, and Inferred based on drilling density, geological continuity and confidence, the variogram range continuity and the slope of regression. Drill densities in CZ and NZ Mineral Resources have been optimised from specific drill spacing studies. CZ Mineral Resources also take into account the representability of the sample size and gold content analysis • Sample compositing is applied, after top cutting, for samples that are used in the resource estimation. A 1 m composite is currently used with a 0.5 m tolerance for mass. Composite samples below 0.5 m are excluded for resource estimation.
Orientation of data in relation to geological structure	• Drilling is undertaken perpendicular to the strike of the main body of mineralisation. Commonly, a 50° to 60° dip angle for the drill holes is used to cut the steeply dipping mineralised lenses at as high an angle as possible.
Sample security	• Samples on the rigs are bagged and tied with custom Massawa and Sofia tags as well as being weighed and documented. The samples are stored in a secure facility and are delivered to the laboratory by company personnel.
Audits or reviews	• No external audits have been completed on Massawa and Sofia Mineral Resources in their current form as additional drill data for both CZ and NZ has been completed in a number of small local orientation blocks. • A full external audit of the sampling methods and procedures was undertaken by Roscoe Postle Associates Inc. (RPA) in 2017. RPA did not identify any areas material issues. Two high priority issues were identified, the collection of bulk densities which did not follow Randgold Standard Operating Procedures (SOP) from other sites, and laboratory certification and selection with regard to LeachWELL assaying. Bulk density sampling has been updated to meet the same standard as at other Barrick Africa and Middle East sites. LeachWELL certification does not currently exist, however, Randgold undertook a programme of laboratory round robin assaying using their current assaying procedures and 1 kg custom OREAS CRM standards which indicated there to be no issue with repeatability of these samples. All other minor 'housekeeping' issues outlined in the RPA audit have been reviewed and acted upon where required.

Criteria	Commentary
	• No external audit has been undertaken on the Massawa Mineral Resource models, however, the general procedures used to generate the Mineral Resources are common across Barrick Africa and Middle East operations which are externally audited on three-year cycles. The QP for the Mineral Resource (Mr Simon Bottoms) is the QP for all Barrick Africa and Middle East operations and has ensured that the Massawa Mineral Resources are of the same quality as all other Barrick Africa and Middle East operations

30.1.2. Section 2. Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	Commentary
Mineral tenement and land tenure status	• The Massawa gold deposit is located in Eastern Senegal and is located approximately 700 km SE of the capital city, Dakar, and 90 km due west of the Barrick Loulo Mine in Mali. The deposit is positioned in the centre of the Kanoumba Permit which was granted to Randgold by Presidential Decree in 2003 and renewed by arêtes issued by the Ministry of Energy and Mines in 2010, which merged the original permit with the Kanoumering Permit, into the single Kanoumba Permit. • The Senegalese government maintains a 10% free carried interest in all projects. The government's interest is activated at the Exploitation Stage, and a local mining subsidiary must be created. The subsidiary company is required to finance the 10% share of the State in the capital. The State does not contribute to the expenses for exploration, feasibility studies, development, or mining. In the case of an increase in the capital during the mine life, the State will receive a free 10% free carried interest in the new investment in order to keep its participation at 10%. • Conventions under the 1988 Mining Code (Law No 88-06 of 26th August 1988) covered the original Kanoumering (2002) and Kounemba Permits (2003). Subsequent to the 1988 Mining Code an updated mining code was passed in November 2003 (Law No 2003-36 of 24th November 2003). Randgold exercised the right to apply the 2003 Mining Code to the merged permit area on 14th April 2010, when two permits were combined into the single Kanoumba Permit of 621 km^2. The arête was approved on 21st May 2010.
Exploration done by other parties	• The southern portion of the Massawa target had been subject to a considerable amount of exploration work by AngloGold Ashanti Limited (AngloGold Ashanti or AGA) between December 1996 and January 2000. During this period AngloGold Ashanti undertook regional geochemistry, mapping, and an airborne survey. In addition, some RAB and 21 DD holes consisting of 3,406 m were drilled at priority targets, mostly around the Tinkoto pluton, however, none of which were drilled within any of the currently defined Mineral Resources. • The Massawa and Sofia Mineral Resources were all greenfield discoveries by Randgold in 2004 after completion of the Kounemba Permit regional soil survey. The regional soil sampling programme at 1,000 m by 100 m spacing identified a total of 11 targets, among which seven were ranked as priority for detailed work. Due to the low tenor of the Massawa anomaly it was only selected as a secondary target. A detailed soil grid was completed in mid-2005 which identified a 3.5 km long, 100 m to 400 m wide soil anomaly at above 50 ppb gold in soil. Subsequent soil sampling in 2008 extended the anomaly to the south and north by a further 3.4 km for a total of 6.2 km. The first trench was positioned over the anomaly in November 2006. The first trench, MWTR001 which was cited on the SW part of the soil anomaly returned an encouraging result and was followed up by exploratory RAB drilling. Positive results from the RAB drilling were followed up with a number of phases of DD between 2007 and 2010.
Geology	• The Project is located within the Kedougou-Kenieba inlier which is divided into the Mako volcanic series to the west, and an overlying Dialé–Daléma sedimentary basin to the east. The contact between the Mako and Dialé–Daléma series is structurally controlled and marked by the regional-scale, NE trending Main Transcurrent Shear Zone (MTZ)

Criteria	Commentary
	• Massawa is located along the eastern margin of the 150 km long Mako belt, where Birimian assemblages consist of greenstones and sedimentary rocks, dated between 2160 Ma to 2200 Ma, which were intruded by ultramafic to felsic plutons yielding ages of 2070 Ma to 2210 Ma. All rock types, excluding post-Birimian dykes, were metamorphosed to a lower green schist facies during the Eburnean orogeny. • The Massawa stratigraphy is dominated by a western package of volcaniclastic rocks and an eastern package of greywackes, with bedding striking at between 210° ± 10° and dipping steeply (75° to 80°) toward the west. This dominantly clastic sequence is intruded by a number of igneous rocks including sills of gabbro, felsic intrusions, and feldspar (and/or quartz-feldspar) porphyries. • The Massawa deposit occurs over a strike length of 4.5 km and is divided into two zones (CZ and NZ) that differ in terms of host rock geology, mineralogy, and structural controls. The CZ and NZ are separated by a 0.3 km gap zone, where less intense shearing is observed. Both Massawa CZ and NZ lie on a NE trending (035°) structure which is thought to be a second order splay off the neighbouring MTZ. • The NZ consists of one main NNE trending mineralised structure with discontinuous footwall (FW) and hanging wall (HW) lodes. Mineralisation is localised in a damage zone adjacent to highly strained bands of fine- to medium-grained felsic and lithic wacke, wacke with subordinate carbonaceous shales, and gabbros. The NZ is sub-divided into two further zones based on structure (NZ1 and NZ2). • The southern 1.1 km of the NZ (NZ1) hosts discontinuous, weak gold mineralisation (average grade of 1 g/t to 1.5 g/t Au). The weakly silicified, brittle-ductile, mineralised shear is less than 10 m in thickness and is sub-vertical to steeply dipping to the ESE. • NZ2 represents the northern and highest grade (>4 g/t Au) portion of the deposit. Mineralisation is confined to a single, continuous, narrow zone (10 m to 15 m average width), which is sub-vertical to steeply dipping (>70°) to the WNW. The mineralisation is bounded by two prominent carbonaceous shale horizons. Two mineralised domains are modelled in the NZ: • A higher-grade domain (average 5 g/t Au) consists of 7% to 10% disseminated sulphides (arsenopyrite>pyrite) associated with ductile shearing with extensional quartz-carbonate veins. • A lower-grade domain (average 1.5 g/t Au) consists of 1% to 3% disseminated sulphides (pyrite>arsenopyrite) associated with brecciation and extensional quartz-carbonate veins. • Mineralisation in the CZ is hosted by an anastomosing brittle-ductile shear network localised by pre-existing gabbro and felsic porphyry intrusive contacts. The CZ is divided into four blocks based on metallurgy (Block 1 to Block 4) generally becoming more refractory to the north associated with higher arsenopyrite content and a change in host lithology from volcaniclastics to sediments (greywacke). • There is a deposit scale correlation in the CZ between increasing brittle-ductile strain and increasing gold grade, with the grade of mineralisation variable along strike and down-dip related to the variable strain associated with the structural framework. The continuity of mineralisation is localised along gabbro and felsic porphyry intrusive contacts with high-grade mineralisation associated with high strain and arsenopyrite. • The grade and continuity of mineralisation in the CZ is characterised by alteration style, deformation intensity, and intrusive contacts. • Low-grade (+1 g/t Au) mineralisation in the CZ is associated with weak to moderate shearing with silica-carbonate alteration and disseminated sulphides with weak strain. Arsenopyrite is rare. High-grade (+3 g/t Au) mineralisation is associated with high strain including brecciation, extensional and shear veins, with moderate to strong silica-carbonate alteration and sulphides. Arsenopyrite is the dominant sulphide associated with gold, with arsenopyrite and pyrite also observed as vein selvedges +/- visible gold. • In the CZ, veins identified by trenching and diamond drilling vary in style and include extensional, sheared, and boudinage veins. Veining associated with +1 g/t Au mineralisation is sub-parallel in orientation to primary strain (shearing) highlighting the genetic link between deformation and mineralisation.

Criteria	Commentary
	• To the south and north of Massawa, the NE trending shears are dextrally offset by discordant N-S structures, resulting in dilation and mineralisation. Sofia is situated to the west of Massawa along the NE trending Sabodala Shear Zone. • The Sofia Mineral Resource is sub-divided into the Sofia Main Zone (MZ) consisting of brittle-ductile structures that strike 040°. The Sofia MZ consists of compressional structures and splays, with zones of high strain developed over at least 1.4 km of strike. Several syngeneic weakly mineralized (sub 1 g/t Au) sub-parallel zones are hosted in the HW and FW stratigraphy. • At Sofia North the main mineralized structure that controls the strain and alteration is developed at the eastern contact of the western mafics that turns in strike from 040° to 010° and has been delineated along a strike of over 2 km. A second, high grade, narrow vein hosted mineralised zone, is present in the HW stratigraphy at the contact between gabbros and lavas, suggesting that the strain has transferred from the FW to the HW on this strike orientation. • The mineralisation at Sofia is thought to be emplaced in a compressive system (thrust model/transgression) where changes in HW geometry from sub-vertical to steep/moderate west dipping has a structural control on the mineralisation. The mineralisation in Sofia is defined by a strong occurrence of fine disseminated pyrite accompanied by strong silica, K-feldspar, and carbonate alteration.
Drill hole Information	• No exploration results are released in this report.
Data aggregation methods	• Exploration results are generally not reported at Massawa. Where applicable, top cutting exploration results are reported as per the capping levels identified during previous Mineral Resource estimation.
Relationship between mineralization widths and intercept lengths	• In Massawa all Mineral Resources drilling is perpendicular to the mineralised zones with holes dipping between 50° and 60°. • The southern 1.1 km of the NZ (NZ1) mineralised shear is below 10 m in thickness and is sub-vertical to steeply dipping to the ESE. • The northern portion of the NZ (NZ2) has an average mean true thickness of 13 m, which is sub-vertical to steeply dipping (>70°) to the WNW. • Massawa CZ consists of eight broad discontinuous Phase 1 mineralised zones modelled over 1.6 km in length striking 023° to 025° and dipping 75° to 85° towards the NW, with variable width from 2 m to 12 m, with an average mean true thickness of 8 m. The Phase 2 structures are situated within the broad Phase 1 mineralised zones and consist of six main shears striking 023° to 025° over entire length of the CZ. • Phase 1 and Phase 2 intercepts are reported separately in all results as the Phase 2 data is hard bounded for the purposes of resource estimation due to the distinctly separate grade distributions between the two phases. • The Sofia Mineral Resource is sub-divided into the Sofia MZ consisting of a NE-sub-vertical mineralised structure with an average mean true thickness of 8 m to 9 m. In contrast, the NZ is hosted by an N-S-trending dextral shear with only narrow dykes of felsic intrusives.
Diagrams	• No new discoveries are reported in this document.
Balanced reporting	• No exploration results are released in this report.

Criteria	Commentary
Other substantive exploration data	• No exploration results are released in this report.
Further work	• The main exploration objective at Massawa is to increase Ore Reserves to greater than 3 Moz of gold. For this purpose, the potential exploration discovery and definition of an additional 600 koz of Measured and Indicated Mineral Resources are required. • For 2019, the priority exploration target areas are KB, Samina (Delya strike extensions), Tina, and Tiwana South (Bakan Corridor). Each of these target areas are associated with anomalous soils and litho-samples and, apart from Samina, are located on transfer and/or second-order ENE, NE, and NNW structures. RC and Air Core drilling in combination with trenching and additional fieldwork is planned for 2019 to further define potential at these priority target areas.

30.1.3. Section 3. Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	Commentary
Database integrity	• All project data is stored in an industry standard Maxwell Datashed SQL database for optimal validation through constraints, library tables, triggers, and stored procedures. All site software application databases are set up to link back to the main database for information retrieval via ODBC. A full-time database administrator is employed at site to manage the database and is overseen by the Loulo regional database administrator. • During the migration to the SQL database in 2017, all assay data was migrated from the access database – but subsequently, assay data was re-imported directly from assay certificates from the laboratory and ranked such that they have a higher priority than the MS Access imported data. • Data loggers for collection of sample data in the field are used to minimise data transcription errors.
Site visits	• Qualified persons (Simon Bottoms MGeol, FGS, CGeol and FAusIMM and Rodney Quick Pr. Sci. Nat, SACNASP, MSc) regularly visit site. During 2018, several visits took place to review both the on-going exploration and resource development programme.
Geological Interpretation	• Cross sections were generated for all deposits by drawing vertical sections approximately perpendicular to the mineralised zones' strike direction in Maptek Vulcan; Seequent Leapfrog, and Micromine. • The mineralisation wireframes for Sofia, Delya, and Massawa NZ were generated through drawing a string outlining the edges of mineralisation intercepts on vertical sections in Vulcan. The vertical sections are generally spaced 10 m apart, except in areas where the GC orientation drilling has been completed which are spaced at half of the drill spacing (e.g. 5 m sections for 10 m along strike drill spacing). This allows for two intermediate section strings between each line of drilling. The use of intermediate strings allows for the wireframes to be smoothed between sections, particularly when the mineralised body anastomoses both horizontally and vertically. The vertical section strings are snapped to the exact intersection within all forms of data including surface trenches, RC, and DD holes thereby ensuring that the points that define the wireframe intersect the drill hole or trench. Thereafter, the strings on each section are connected to form a 3D triangulated solid. Where a mineralised zone terminates, the wireframe is projected along strike by 50% of the drill spacing (mean distance of 25 m).

Criteria	Commentary
	• The wireframes are again checked against the hardcopy hand-drawn sections, and then cut to the 5 m resolution LIDAR topographic surface, for Massawa and Sofia, the sections are cut to the 10 m resolution DGPS topographic surface for Delya. • For the CZ, cross sections were hand drawn for each drill line and flitches were then generated every 10 m vertical depth. These interpretations were then scanned and georeferenced into Seequent Leapfrog in order to create the 3D wireframes. The wireframes were generated in Seequent Leapfrog through selecting the geological intervals on vertical and horizontal sections and then using the vein-modelling tool to model these intercepts in 3D space. The Massawa deposit is composed of multiple sub-parallel shear zones, and the wireframe modelling attempts to model the major structures. The mineralised zones are identified by a combination of sulphide identification, alteration, and shearing. Full geological interpretation is completed on the sections, using the drilling, lithological, and alteration logging in the trench, RC, and DDH data, then from this interpretation, strings representing the outlines of the ore body are digitised. All strings are snapped to the exact interval on the drill or trench trace and the strings are connected to form a 3D triangulated representation of the ore body. • The wireframes are again checked against the hardcopy hand-drawn sections, and then cut to the LIDAR topographic surface, and further divided into weathering profiles namely saprolite, saprock, and fresh portions using the updated weathering surfaces.
Dimensions	• The Massawa CZ and NZ deposits occur over a strike length of 4.5 km. The NZ has a strike length of 2.5 km. • The southern 1.1 km of the NZ (NZ1) mineralised shear is less than 10 m in thickness and is sub-vertical to steeply dipping to the ESE. The northern portion of the NZ (NZ2) has average mean true thickness of 13 m, which is sub-vertical to steeply dipping (>70°) to the WNW. • The Sofia Mineral Resource is subdivided into the Sofia MZ consisting of a 2.1 km of NE striking sub-vertical mineralised structure with discontinuous FW lenses and the Sofia NZ consisting of a 2.1 km N-S trending dextral shear zone.
Estimation and modelling techniques	• All Massawa, Sofia, and Delya Mineral Resources are estimated using an Ordinary Kriging (OK) estimation methodology, and apply parent cell estimation for the relevant GC or exploration parent cell size, thereby ensuring that sub-cells are effectively grouped to provide sufficient sample support per estimated block. • Qualitative Kriging Neighbourhood Analysis (QKNA) was utilised to determine the optimal block size, minimum number of samples, search radius, and block discretization for each pass, by evaluating the kriging efficiency (KE) and slope of regression (SR) across multiple block centroid locations to ensure that the selected parameters are robust and appropriate for different areas within the relevant domain. • All Massawa CZ search radii, number of samples used, and QKNA estimation parameter optimisations were restricted to RC drill data with bulk sample LeachWELL (plus tail assay) determination in order to ensure that the respective grade distributions and continuity were not skewed by the known negative bias of sample grades within the smaller volume DD. Block size optimisations were performed to take into account the mining SMUs. • The estimation search strategies for Sofia, Massawa NZ, and Delya were tested using QKNA based on the modelled variogram ranges taking into account that in nugget shear hosted gold systems the full variogram range is often artificially extended for the last 3% to 5% of variability. Additionally, the estimation search strategy takes into consideration the data distribution for both the geological and the data spacing sub-domains. The resultant search ellipsoid orientations are verified visually against the domain wireframe and, where appropriate, the search ellipse and associated variogram directions are locally re-orientated to reflect bifurcation and local changes in the strike and dip of the mineralisation wireframes. • In 2018, dynamic anisotropy (DA) was introduced to the Massawa CZ deposit to better reflect the anastomosing nature of the mineralisation and local changes in orientation across the deposit. • All Mineral Resources use multi-pass estimation searches, in order to estimate all blocks, whereby each pass uses a varying degree of restrictions before any given block can be estimated. Accordingly, the Mineral Resource classification is adjusted per the search pass during which the blocks from an area were estimated.

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>

Tina and Bambaraya have been estimated using inverse distance squared using one pass of the search ellipse. All searches have been orientated into the average strike of each of the deposits.
Top cutting is determined using a multifactor analysis, whereby an appropriate top cut is assessed utilising individual methods of analysis including raw composite percentile disintegration, analysis of the grade distribution, and log probability plots. Generally, the chosen top cutting value is expected to occur between the 95th and 99.9th percentiles. Subsequently, the final top cutting value is chosen taking into consideration the results of each of the methods of analysis.
NZ sample grade distributions were assessed using conventional multifactor analysis methods. The grade distributions for NZ1 domains were grouped for the purposes of top cut optimisation. The grade distributions for NZ2 were also grouped for the purposes of top cut optimisation but split by domains with similar grade distributions.

<table>
<tr><th rowspan="2">Domain</th><th colspan="3">NZ1</th><th colspan="3">NZ2</th></tr>
<tr><th>1103</th><th>1104</th><th>115-1108</th><th>2111 (HG)</th><th>2101 & 2103</th><th>2102</th></tr>
<tr><td>Top Cut (g/t Au)</td><td>13</td><td>27</td><td>21</td><td>65</td><td>39</td><td>7</td></tr>
<tr><td>Number of Samples Cut</td><td>3</td><td>11</td><td>20</td><td>8</td><td>6</td><td>5</td></tr>
</table>

All CZ GC samples top cutting optimisations were restricted to RC drill data with bulk sample LeachWELL plus tail assay determination. Due to the separate grade distributions of CZ mineralisation, the domains were grouped for top cutting based on similarities in grade distributions.
Grade distributions in the CZ low-grade domains (top cutting group 1) were assessed using conventional multifactor analysis methods. Grade distributions were assessed both independently for each of the mineralised zones and also as grouped domains to compare the results of the top cut evaluations.

An additional restricted search ellipse constraint has been placed on all CZ low-grade samples above 28 g/t Au within the GC domains, where the search ellipse is restricted to 33% of the total variogram range (21 m), which is almost equal to the range of the third structure.
Within the wider spaced exploration domains, the search ellipse for estimation of all samples above 11 g/t Au is restricted to 66% of the total variogram range (42 m), which is almost equal to the range of the second structure. The search ellipse restriction is reduced in the exploration domains relative to the GC domains, due to the wider spacing of the drill sample data.
This combination of a restricted search and extreme outlier top cutting is considered to be the most appropriate method for estimation within a nuggety gold system, as it retains the areas of local very high-grade material, but restricts their influence whilst also top cutting the actual outliers. The resultant impact on the contained metal ounces is the same as the impact from applying a straight top cut.

The grade distributions of the CZ high-grade domains were assessed both independently for each mineralised zone and then grouped to compare the results of the top cut evaluations. Due to the partial coverage of the RC data with bulk LeachWELL assay determination, some lodes were under sampled with respect to others and as such the grouped distribution was used to determine the relevant top cut.
Due to the nuggety nature of the CZ high-grade (proximal) domains, the range of potential top cuts established using conventional multifactor analysis methods varied significantly (by >200%). Accordingly, it was necessary to run an additional ‘metal at risk’ simulation which has been utilised at other nugget vein gold mine operations. The metal at risk simulation was run on all 15 m by 10 m AdvGC RC drilling analysed using bulk LeachWELL analysis. The simulation was restricted to a tight estimation data analysis envelope area that is restricted to half the drill spacing away from the last GC grid intersection.

The results of these simulations indicate that the top cut value was 55 g/t Au for CZ top-cutting group 1,150 g/t Au for CZ top-cutting group 2&3, and 465 g/t Au for CZ top-cutting group 4. In addition, a high yield restriction was used to limit the metal generated by high grades in cutting group 1 and 4. For top-cutting group 1 and top cutting 2 HW & HW Bridge this was 28 g/t Au, and for top-cutting group 4 this was 230 g/t Au.

</td></tr>
</table>

Criteria	Commentary

- Sofia sample grade distributions were assessed using conventional multifactor analysis methods. The grade distributions for Sofia Main domains were grouped for the purposes of top cut optimisation. The grade distributions for Sofia North were split by geological domains with similar grade distributions for the purposes of top cut optimisation.

Domain	Sofia Main	Sofia North	
	All	6500, 6700, 6900	6400, 6600
Top Cut (g/t Au)	25	16	7
Number of Samples Cut	237	9	17
High Grade Search Restriction (g/t Au)	17	13	-

- Delya sample grade distributions were assessed using conventional multifactor analysis methods. The grade distributions for Delya domains 6200 and 6300 were grouped for the purposes of top cut optimisation.

Domain	Delya	
	6000	6100, 6200, 6300
Top Cut (g/t Au)	33	8.5
Number of Samples Cut	11	5
High Grade Search Restriction (g/t Au)	16	-

- Massawa CZ RC samples used in the Mineral Resource estimation sample datasets were taken on one-metre intervals, therefore the samples were flagged within their respective estimation domain wireframes, giving one-metre composites. Massawa CZ trench and DD samples, as well as Massawa NZ, Sofia, and Delya Mineral Resource estimation sample datasets were composited to one-metre composites within their respective estimation domain wireframes with a tolerance of 0.5 m. This results in a maximum composite size of 1.5 m and a minimum composite size of 0.5 m, however, over 95% of the composites are equal to 1.0 m. Any residual composites with a length below 0.5 m were excluded from the final estimation dataset.

Residual Statistic	NZ	CZ	Sofia	Delya
Number Removed	111	22	24	-
Mean Grade	4.4	2.1	0.7	-

- Block models volume checks at all Massawa Mineral Resources have been validated to be within less than 1% variance from the volume of the corresponding wireframes. The CZ low grade domains have displayed a higher (2.5%) variance as a result of the relatively low and anastomosing nature of the block model.
- Due to the complex thin anastomosing shape of the high-grade wireframes, the CZ block model has been built entirely based upon the GC block size to achieve a close volume reconciliation between the block model and wireframes. Consequently, the exploration sub-domains use a parent cell size that is exactly double that of the GC block size, meaning that the estimation will assign the grades into groups of eight blocks when writing estimated grades to represent parent cell estimation.
- In the Sofia block model, the thin anastomosing wireframe shapes in Sofia Main dictate that the entire block model be built entirely based upon the GC block size to achieve a close volume reconciliation between the block model and wireframes. Consequently, the Sofia North sub-domains use a parent cell size that is exactly double that of the GC block size in strike length (Y axis) to account for the wider spacing along strike. This means that the estimation will estimate a single grade into the groups of two blocks when writing estimated grades, representative of the parent cell size.
- The block model parameters are listed below.

Block Details	Massawa NZ	Massawa CZ	Sofia	Delya
Parent Block Size (X, Y, Z (m))	1.5 by 8 by 5	2.25 by 4.5 by 2.5	3 by 15 by 12	2 by 9 by 7
Sub Cell Size (X, Y, Z (m))	0.5 by 1.6 by 1	0.25 by 0.5 by 0.5	0.5 by 1.5 by 1.2	0.5 by 1.8 by 1.75
GC Domain Block Size (X, Y, Z (m))	0.5 by 1.6 by 1			
Rotation	117°	117°	120°	35°

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>

- All Massawa Mineral Resource block models are interpolated using Maptek Vulcan 9.

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<tr><td>Moisture</td><td>

- Tonnage estimates are on a dry basis. Saprolite samples are dried prior to SG determination.

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<tr><td>Cut off Parameters</td><td>

- Massawa NZ processing costs are set materially higher than that of CZ, Sofia or Delya due to reflect the BIOX treatment costs.
- Massawa NZ underground Mineral Resources are reported at a cut-off grade of 2.5 g/t Au which reflects all costs above except the mining cost where the nearby Loulo UG actual costs are applied ($26.79 per tonne mined development and stoping plus $12.15 per tonne mined backfill).
- Tina and Bambaraya Inferred open pit Mineral Resources are reported at a 0.50 g/t Au flat cut-off grade, because the modifying factors affecting the marginal gold cut-off have not yet been defined.

Parameter	Unit	Symbol	NZ OP	CZ	Sofia	Delya
Gold Price	$/oz	Gp	1,500	1,500	1,500	1,500
Royalty	%	R	3%	3%	3%	3%
Selling cost	%	S	0%	0%	0%	0%
Net Gold Price	$/oz	Ng	1455	1455	1455	1455
Met Recovery	%	REC	83%	76%	89%	89%
Dilution	%	Dil	10%	36%	10%	10%
Ore Loss	%	Loss	3%	8%	3%	3%
Mining Cost - Contractor	$/t mined	MCC	3.08	3.03	3.08	3.05
Mining Cost - Owner's	$/t mined	MCO	0.06	0.06	0.06	0.06
Mining Cost - Grade Control	$/t mined	MCGC	0.20	0.20	0.20	0.20
Total Mining Cost	$/t mined	TMC	3.34	3.29	3.34	3.31
Strip Ratio	Waste/Ore	SR	14.32	6.34	4.94	8.78
G&A	$/t milled	G_A	8.00	8.00	8.00	8.00
Ore Crushing & Hauling	$/t milled	rd	-	-	3.25	4.25
Mining	$/t milled	Cr	30.30	24.15	19.84	30.30
Process Plant	$/t milled	CP	32.12	15.93	16.93	22.14
Maintenance/Engineering	$/t milled	Mp	-	-	-	-
Total Operating Costs	**$**		**70.42**	**48.08**	**48.02**	**64.69**
Full Grade Cut-off	**g/t**		**1.87**	**1.47**	**1.19**	**1.6**
Marginal Cut-off Grade	**g/t**		**1.07**	**0.73**	**0.7**	**0.85**

Parameter	Unit	Symbol	NZ UG
Gold Price	$/oz	Gp	1,500
Royalty	%	R	3%
Selling Cost	%	S	0%
Net Gold Price	$/oz	Ng	1455
Met Recovery	%	Rec	86%
Dilution	%	Dil	12%
Ore Loss	%	Loss	4%
OPEX Development	$/t mined	MCC	11.74
OPEX Stoping	$/t mined	MCO	15.05
Backfill	$/t mined	MCGC	12.15
Fixed Cost	$/t mined	TMC	15.25
Grade Control	$/t mined	SR	3.38
Total Operating Cost	**$/t**	**G_A**	**57.57**
Mine Sustaining Capital	**$/t**	**Rd**	**166.67**
All in Cost	**$/t**	**Cr**	**224.24**
Full Grade Cut Off*	**g/t Au**		**7.7**
Marginal Cut Off Grade	**g/t Au**		**2.5**

</td></tr>
<tr><td>Mining Factors or Assumptions</td><td>

- The mining dilution was assumed to be 10% of additional waste for each tonne of ore in Massawa NZ, Sofia, and Delya Main and 36% in Massawa CZ. The CZ dilution and ore loss applied are based on a dilution study carried out by Maptek taking into account the very narrow nature of the CZ ore body. The 10% figure is based on experience with similar steep tabular ore bodies at Randgold's Loulo operation.
- Ore loss was set at 3% for NZ, Sofia, and Delya and 8% for CZ, which is also based on historical information from nearby operations with similar geology, in the Randgold Group.

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<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td>Metallurgical Factors or Assumptions</td><td>

The Massawa deposits consist of free-milling ore in Sofia and most of CZ, and a refractory portion in the northern part of CZ as well as NZ and Delya fresh. The refractory ores have been proven to be highly recoverable through BIOX as an oxidative step.
The Massawa process plant design utilises well-known and established carbon-in-leach (CIL) and gravity technologies to recover gold from oxide and fresh ores that will be processed during the first and second phases of the operation, i.e. whole ore leach (WOL). These technologies will be utilised in combination with flotation and BIOX leach technologies to recover gold from refractory ores that will be processed during the third phase of the operation.
The proposed Massawa process plant design has been compiled for the treatment of different types of ores that will be processed in three phases:

Phase 1 – Oxide ore will be processed through primary crushing (mineral sizer), milling, and gravity recovery, CIL, gold recovery stages (acid wash, elution, electrowinning, regeneration), and detoxification of tails prior to disposal.
Phase 2 – Fresh (sulphide) ore will be processed through primary (jaw crusher) and secondary (cone crusher) crushing, mill feed bin, milling, and gravity recovery, CIL, gold recovery stages from carbon, and detoxification of solids prior to disposal.
Phase 3 – Refractory ore will be processed through primary and secondary crushing, mill feed bin, milling, and gravity recovery, flotation, regrind mill, BIOX, CIL, gold recovery stages from carbon, and detoxification of tails prior to disposal.

The recoveries used for the study are as shown in the table below:

<table>
<tr><th colspan="2" rowspan="2">WOL 2.7, 2.4 and BIOX 1.2 Mtpa</th><th colspan="3">Central Zone</th><th colspan="2">North Zone</th><th>Sofia Main</th><th>Sofia North</th><th colspan="3">Delya</th></tr>
<tr><th>WOL</th><th>BIOX - 100% CZ</th><th>BIOX - 25% CZ / 75% NZ</th><th>WOL 100% NZ</th><th>BIOX 100% NZ</th><th>WOL</th><th>WOL</th><th>WOL 100% Delya</th><th>BIOX 100% Delya</th><th>BIOX 20% Delya 80% NZ</th></tr>
<tr><td rowspan="4">Recoveries</td><td>Oxide</td><td>94%</td><td>N/A</td><td>N/A</td><td>90%</td><td>N/A</td><td>92%</td><td>91%</td><td>94%</td><td>N/A</td><td>N/A</td></tr>
<tr><td>Oxide Trans</td><td>94%</td><td>N/A</td><td>N/A</td><td>89%</td><td>N/A</td><td>92%</td><td>91%</td><td>90%</td><td>N/A</td><td>N/A</td></tr>
<tr><td>Reduced Trans</td><td>Geomet</td><td>57%</td><td>57%</td><td>51%</td><td>57%</td><td>90%</td><td>89%</td><td>52%</td><td>57%</td><td>57%</td></tr>
<tr><td>Fresh</td><td>Geomet</td><td>87%</td><td>88%</td><td>13%</td><td>88%</td><td>89%</td><td>85%</td><td>9%</td><td>92%</td><td>89%</td></tr>
</table>
</td></tr>
<tr><td>Environmental Factors or Assumptions</td><td>

Independent consultants Digby Wells Environmental (Jersey) Limited together with Tropica Environmental Consultants (Tropicana) were contracted to conduct the environmental and social impact assessment (ESIA) of the Massawa project.
As is common with mining projects in West Africa, several potential positive and potential negative impacts have been identified. An environmental and social management plan (ESMP) has been completed to ensure these impacts are systematically and correctly managed.
The ESMP includes a number of recommended mitigation measures that, if implemented effectively, will enhance the positive impacts of the Project, and minimise the negative effects.
No impacts which could present a fatal flaw to the successful execution of the Project have been identified to date. With this effective implementation of the ESMP, none of the negative impacts are believed to be sufficiently significant to prevent the development of the proposed Massawa project. The potential positive impacts associated with the local and national Senegalese economies are expected to be significant.

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<tr><td>Bulk Density</td><td>

Bulk density values were measured by applying the Archimedean principles (density = weight (in air) ÷ (weight (in air) – weight (in water)). All results are grouped by weathering profile and lithology. The density measurement procedure differed slightly for saprolite, transition, and fresh material:

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<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>

o Saprolitic density measurements were primarily obtained from trenches, although some drill hole core was used. For trench samples, cubes of approximately 25 cm by 25 cm by 25 cm were excavated and the insitu weight measured to estimate the moisture content. The sample is then dried out and wrapped in a waterproof membrane.
o Fresh and transition density measurements were primarily obtained from drill core. The procedure followed involved the selection of 10 cm to 15 cm pieces using the water immersion method.

<table>
<tr><th colspan="4">Massawa CZ and NZ Mean Densities</th></tr>
<tr><th>Lithology</th><th>Saprolite Density</th><th>Transition Density</th><th>Fresh Rock Density</th></tr>
<tr><td>Porphyry</td><td>1.80</td><td>2.41</td><td>2.74</td></tr>
<tr><td>Gabbro</td><td>1.66</td><td>2.37</td><td>2.85</td></tr>
<tr><td>Volcaniclastics</td><td>1.77</td><td>2.33</td><td>2.75</td></tr>
<tr><td>Greywacke</td><td>1.78</td><td>2.25</td><td>2.77</td></tr>
<tr><td>Carbonaceous Shale</td><td>1.75</td><td>2.27</td><td>2.75</td></tr>
<tr><td>NZ2 Mineralisation</td><td>1.87</td><td>2.25</td><td>2.8</td></tr>
<tr><td>NZ2 HG Mineralisation</td><td>2.21</td><td>2.2</td><td>2.83</td></tr>
<tr><td>NZ1 Mineralisation</td><td>1.85</td><td>2.26</td><td>2.81</td></tr>
<tr><td>CZ Low grade</td><td>1.66</td><td>2.35</td><td>2.8</td></tr>
<tr><td>CZ High grade</td><td>1.66</td><td>2.25</td><td>2.72</td></tr>
</table>

<table>
<tr><th colspan="4">Sofia Mean Densities</th></tr>
<tr><th>Lithology</th><th>Saprolite Density</th><th>Transition Density</th><th>Fresh Rock Density</th></tr>
<tr><td>Transported Cover</td><td>1.89</td><td></td><td></td></tr>
<tr><td>Quartz-Feldspar Porphyry</td><td>1.76</td><td>2.63</td><td>2.76</td></tr>
<tr><td>Quartz Diorite</td><td>1.88</td><td>2.62</td><td>2.8</td></tr>
<tr><td>Gabbro</td><td>1.85</td><td>2.57</td><td>2.84</td></tr>
<tr><td>Dolerite</td><td>1.81</td><td>2.55</td><td>2.83</td></tr>
<tr><td>Dacite</td><td>1.8</td><td>2.56</td><td>2.82</td></tr>
<tr><td>Tonalite</td><td>1.55</td><td>2.6</td><td>2.78</td></tr>
<tr><td>Tuff</td><td>1.79</td><td>2.53</td><td>2.79</td></tr>
<tr><td>Tuff SAB</td><td>1.7</td><td>2.57</td><td>2.78</td></tr>
<tr><td>Ultra Basic</td><td>1.86</td><td>2.6</td><td>2.82</td></tr>
<tr><td>Sofia Main Mineralisation</td><td>1.9</td><td>2.6</td><td>2.8</td></tr>
<tr><td>Sofia North Mineralisation</td><td>1.9</td><td>2.47</td><td>2.77</td></tr>
</table>

<table>
<tr><th colspan="4">Delya Mean Densities</th></tr>
<tr><th>Lithology</th><th>Saprolite Density</th><th>Transition Density</th><th>Fresh Rock Density</th></tr>
<tr><td>Dolerite</td><td></td><td></td><td>2.88</td></tr>
<tr><td>Mafic Dyke</td><td>1.85</td><td>2.38</td><td>2.81</td></tr>
<tr><td>Carbonaceous Shale</td><td>1.9</td><td>2.5</td><td>2.77</td></tr>
<tr><td>Chert</td><td>1.87</td><td>2.54</td><td>2.9</td></tr>
<tr><td>Andesite</td><td>1.9</td><td>2.77</td><td>2.86</td></tr>
<tr><td>Greywacke</td><td>1.86</td><td>2.56</td><td>2.71</td></tr>
<tr><td>Schist</td><td>1.81</td><td>2.54</td><td>2.83</td></tr>
<tr><td>Mineralised Zone</td><td>1.77</td><td>2.34</td><td>2.82</td></tr>
</table>

Density values are hard coded into all block models based on the lithology and weathering. Where density data does not exist in inferred satellite resources (Inferred Mineral Resources only), the density has been inferred from the nearest deposit on the same shear zone with the same weathering and lithology combination.

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<tr><td>Classification</td><td>

Massawa Gold Project Mineral Resources are classified as Measured, Indicated, and Inferred based on geological continuity, data density, variogram range, and resultant estimation search pass, as well as estimation quality in form of SR and KE.
The classification parameters are presented in the following table:

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<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>
<table>
<tr><th>Statistic</th><th>Deposit</th><th>Measured</th><th>Indicated</th><th>Inferred</th></tr>
<tr><td>Minimum Samples</td><td>All</td><td>6</td><td>4</td><td>3</td></tr>
<tr><td>Minimum Consecutive Sections</td><td>All</td><td>6</td><td>4</td><td>Good Geological Continuity</td></tr>
<tr><td rowspan="3">Maximum Drilling Density</td><td>CZ</td><td>5 m by 5 m RC with LeachWELL</td><td>15 m by 10 m RC with LeachWELL</td><td>50 m by 50 m</td></tr>
<tr><td>NZ</td><td>-</td><td>40 m by 40 m</td><td>80 m by 50 m</td></tr>
<tr><td>Sofia and Delya</td><td>-</td><td>40 m by 40 m</td><td>80 m by 60 m</td></tr>
<tr><td>Estimation Search Pass</td><td>All</td><td>GC Pass 1 or
GC Pass 2</td><td>GC Pass 2 or GC Pass 3 or
EXP Pass 1 or EXP Pass 2</td><td>EXP Pass 2 or
EXP Pass 3</td></tr>
<tr><td rowspan="4">Slope of Regression</td><td>CZ</td><td>-</td><td>-</td><td>-</td></tr>
<tr><td>NZ</td><td>-</td><td>>0.6</td><td><0.6</td></tr>
<tr><td>Sofia</td><td>-</td><td>>0.6</td><td><0.6</td></tr>
<tr><td>Delya</td><td>-</td><td>>0.5</td><td><0.5</td></tr>
<tr><td rowspan="4">Kriging Efficiency</td><td>CZ</td><td>-</td><td>-</td><td>-</td></tr>
<tr><td>NZ</td><td>-</td><td>>0.5</td><td><0.5</td></tr>
<tr><td>Sofia</td><td>-</td><td>>0.6</td><td><0.6</td></tr>
<tr><td>Delya</td><td>-</td><td>>0.5</td><td><0.5</td></tr>
<tr><td>Estimation Method</td><td>All</td><td>Kriging</td><td>Kriging</td><td>Kriging or IVD</td></tr>
</table>
</td></tr>
<tr><td>Audits or Reviews</td><td>

No external audits have been completed on Massawa and Sofia Mineral Resources in their current form as additional drill data for both CZ and NZ has been completed in a number of small local orientation blocks.
A full external audit of the sampling methods and procedures was undertaken by Roscoe Postle Associates Inc. in 2017. RPA did not identify any areas material issues. Two high priority issues were identified, the collection of bulk densities which did not follow Randgold SOP’s from other sites, and laboratory certification and selection with regard to LeachWELL assaying. Bulk density sampling has been updated to meet the same standard as at other Barrick Africa and Middle East sites. LeachWELL certification does not currently exist, however, Randgold undertook a programme of laboratory round robin assaying using their current assaying procedures and 1 kg custom OREAS CRM standards which indicated that there to be no issue with repeatability of these samples. All other minor ‘housekeeping’ issues outlined in the RPA audit have been reviewed and acted upon where required.
No external audit has been undertaken on the Massawa Mineral Resource models, however, the general procedures used to generate the Mineral Resources are common across Barrick Africa and Middle East operations which are externally audited on three-year cycles. The QP for the Mineral Resource (Mr Simon Bottoms) is the QP for all Barrick Africa and Middle East operations and has ensured the Massawa Mineral Resources are of the same standard as all other Barrick Africa and Middle East operations

</td></tr>
<tr><td>Discussion of Relative Accuracy/ Confidence</td><td>

The resource estimation completed at Massawa follows standard industry protocols and the estimation is a progression of previous models. The Mineral Resources have been generated using
For all block models, the following validation is carried out:

A volume reconciliation between the block model estimation domains and related wireframes is undertaken.
All models are visually checked against the composite data on both cross section and long section basis, to validate the estimation and ensure that estimated grades and metal show no clear bias or trends that would not be expected from the variography or estimation domaining.
A check of the number of the blocks estimated using the negative kriging weight is completed. Any blocks estimated using negative kriging weight have been reset to the anisotropic nearest block grade of the closest sample.

</td></tr>
</table>

Criteria	Commentary
	o A comparison between the data minimum, maximum, mean, and the estimated grade at 0 g/t Au cut off for each of the domains (within the open pit or underground reporting areas) is created. This is completed to check for possible over or under estimation. o Swath plots are created for each geological domain to validate the estimated grade variability compared to the composite along X, Y, and Z axes. This is to check that the model estimate follows the trends seen in the data and that there is no general bias with over or under estimation. Areas with less data support are also highlighted for further drilling and geological work. • The swath plots for Massawa, Sofia, and Delya Mineral Resources show that the correlation between the estimation and the source data is acceptable and that conditional bias is minimised. The swath plot analysis does, however, highlight a significant increase in estimation smoothing within the exploration sub-domains relative to the GC sub-domains. This is considered acceptable and is a function of the wider data density and the fact that the estimations in the exploration sub-domains use samples from a much wider search area relative to the more local GC sub-domains. The application of thorough exploratory data analysis and quantitative neighbourhood kriging analysis gives confidence in the model. • A pilot plant scale test was completed on the CZ Mineral Resource, which provides a source of reconciliation data to test against the GC blocks estimated within the Mineral Resource.

30.1.4. Section 4. Estimation and Reporting of Ore Reserves

As required by Section 4 of the 2012 JORC Code for reporting of Ore Reserves, a Table 1 is provided that contains a summary of the estimation and reporting criteria relevant to the Ore Reserve declared. Below is the Table 1 covering the 2018 Open Pit Ore Reserves for Massawa and Sofia.

Criteria	Commentary
Mineral Resource Estimate for Conversion to Ore Reserves	• The Ore Reserve statement is based upon the Mineral Resource declared as at 31st December 2018 by Randgold. Mineral Resources are reported inclusive of Ore Reserves. Mineral Resources are supported by a dedicated Competent Person Report.
Site Visits	• Qualified person Rodney Quick Pr. Sci. Nat, SACNASP, MSc regularly visit site. During 2018, several visits took place to review both the on-going exploration and resource development programme.
Study Status	• A technical and financial study has been conducted by Randgold to support the disclosure of updated Ore Reserves. The study is based on an open pit mining project whereby the ore is mined and fed through an onsite metallurgical plant to produce gold in doré.
Cut-Off Parameters	**Full grade cut-off grade** • The full grade cut-off is the ore material that is profitable at $1,000/oz, considering all operating costs of mining, haulage, processing, and general and administrative costs, as well as the appropriate recovery, dilution, and realised gold price post Royalty at $1,000/oz spot gold price. It is the principal material fed to the plant.

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td><u>Marginal cut-off grade</u>
• The marginal cut-off grade is the ore material mined within the $1,000/oz pit design that is profitable on a marginal basis (FGO OPEX less mining costs). Both marginal and full grade ore form the reserve since it forms part of the LOM plan feed schedule. The marginal ore is the reserve cut-off grade.</td></tr>
<tr><td>Mining Factors or Assumptions</td><td>• The mining dilution was assumed to be 10% of additional waste for each tonne of ore in Massawa NZ, Sofia, and Delya Main and 36% in Massawa CZ. The CZ dilution and ore loss applied are based on a dilution study carried out by Maptek taking into account the very narrow nature of the CZ ore body. The 10% figure is based on experience with similar steep tabular ore bodies at Randgold's Loulo operation.
• Dilution is applied to both tonnes and grade (increases the tonnes and drops the grade) and simulates the waste rock that is included into the ore during the mining operation. It is not included in the resource model and is applied in the optimisation and mining schedules.
• Ore loss was set at 3% for NZ, Sofia, and Delya and 8% for CZ, which is also based on historical information from nearby operations with similar geology, in the Randgold Group.</td></tr>
<tr><td>Metallurgical Factors or Assumptions</td><td>• Recoveries vary depending on the ore type and process route. All oxide, oxidised transition, and Sofia material will go through a gravity circuit and straight CIL circuit. The reduced transition and fresh material from Massawa CZ will pass through gravity, flotation, UFG of concentrate and leaching of both the concentrate and floatation tails. Reduced transition and fresh material from the NZ, north of Block E will pass through gravity, flotation, and Bio-oxidation. Flotation tails will be released. The recoveries used for the study are as shown in the table below:
<table>
<tr><th colspan="2" rowspan="2">WOL 2.7, 2.4 and BIOX 1.2 Mtpa</th><th colspan="3">Central Zone</th><th colspan="2">North Zone</th><th>Sofia Main</th><th>Sofia North</th><th colspan="3">Delya</th></tr>
<tr><th>WOL</th><th>BIOX - 100% CZ</th><th>BIOX - 25% CZ / 75% NZ</th><th>WOL 100% NZ</th><th>BIOX 100% NZ</th><th>WOL</th><th>WOL</th><th>WOL 100% Delya</th><th>BIOX 100% Delya</th><th>BIOX 20% Delya 80% NZ</th></tr>
<tr><td rowspan="4">Recoveries</td><td>Oxide</td><td>94%</td><td>N/A</td><td>N/A</td><td>90%</td><td>N/A</td><td>92%</td><td>91%</td><td>94%</td><td>N/A</td><td>N/A</td></tr>
<tr><td>Oxide Trans</td><td>94%</td><td>N/A</td><td>N/A</td><td>89%</td><td>N/A</td><td>92%</td><td>91%</td><td>90%</td><td>N/A</td><td>N/A</td></tr>
<tr><td>Reduced Trans</td><td>Geomet</td><td>57%</td><td>57%</td><td>51%</td><td>57%</td><td>90%</td><td>89%</td><td>52%</td><td>57%</td><td>57%</td></tr>
<tr><td>Fresh</td><td>Geomet</td><td>87%</td><td>88%</td><td>13%</td><td>88%</td><td>89%</td><td>85%</td><td>9%</td><td>92%</td><td>89%</td></tr>
</table></td></tr>
<tr><td>Environmental</td><td>• Independent consultants Digby Wells together with Tropica were contracted to conduct the ESIA of the Project.
• As is common with mining projects in West Africa, several potential positive and potential negative impacts have been identified. An ESMP has been completed to ensure these impacts are systematically and correctly managed.
• The ESMP includes a number of recommended mitigation measures that, if implemented effectively, will enhance the positive impacts of the Project, and minimise the negative effects.
• No impacts which could present a fatal flaw to the successful execution of the Project have been identified to date. With this effective implementation of the ESMP, none of the negative impacts are believed to be sufficiently significant to prevent the development of the proposed Massawa project. The potential positive impacts associated with the local and national Senegalese economies are expected to be significant.</td></tr>
<tr><td>Infrastructure</td><td>• No permanent infrastructure currently exists at Massawa.</td></tr>
<tr><td>Costs</td><td>• The mining costs have been based on contractor mining and include GC drilling, drill and blast, load and haul, and crusher feeding. Updated processing costs have been generated based on the different process routes. G&A costs of $8.0/t per tonne processed have been allocated to the OPEX costs</td></tr>
</table>

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>• Customs duties, taxes, charges, and logistical costs are included in all relevant reagent costs for the Project.</td></tr>
<tr><td>Revenue Factors</td><td>• A gold price of $1,000 per troy oz was employed for the Ore Reserve process.</td></tr>
<tr><td>Market Assessment</td><td>• The gold market is highly liquid and benefits from terminal markets (London, New York, Tokyo, and Hong Kong) on almost a continuous basis. Gold prices were in a general downward trend from 1980 to 2000, when it traded down to approximately $250/oz. From 2000 the price increased annually until 2011 and 2012 where the price peaked at just under $1,800/oz. There was a sharp correction to the gold price in 2013 with the end of Quantitative Easing monetary policy by the US Reserve Bank. Since then god has remained range bound between $1,350/oz and $1,050/oz.
• Gold produced at the mine site is shipped from site, under secure conditions, to a refining company. Under pre-established contractual conditions, the refiner purchases the gold from the mine with the proceeds automatically credited to the mines' bank account. Gold production will be sold on the spot market, with no plan currently to hedge any sales.</td></tr>
<tr><td>Economic</td><td>• Gold Price: $1,000/oz flat
• Royalty: 3%
• Income tax: 25% After a five year tax holiday
<table>
<tr><th>Item Description</th><th>Total LOM Cost ($ '000)</th><th>Unit Cost ($ /t mined)</th></tr>
<tr><td>Establishment of Contractor's Facilities</td><td>5,775</td><td>0.04</td></tr>
<tr><td>Mobilisation of Contractor's Equipment</td><td>3,360</td><td>0.02</td></tr>
<tr><td>Demobilisation of Contractor's Equipment</td><td>3,175</td><td>0.02</td></tr>
<tr><td>Monthly Management Fee</td><td>103,874</td><td>0.64</td></tr>
<tr><td>Preparatory Works</td><td>1,086</td><td>0.01</td></tr>
<tr><td>Load and Haul</td><td>281,574</td><td>1.72</td></tr>
<tr><td>Drilling and Blasting</td><td>134,028</td><td>0.82</td></tr>
<tr><td>RC Grade Control Drilling</td><td>5,301</td><td>0.03</td></tr>
<tr><td>Ore Re-handle</td><td>18,377</td><td>0.11</td></tr>
<tr><td>Dewatering</td><td>3,623</td><td>0.02</td></tr>
<tr><td>Owners Costs</td><td>20,000</td><td>0.12</td></tr>
<tr><td>Total</td><td>580,173</td><td>3.55</td></tr>
</table>
• Processing cost as per the table below:
<table>
<tr><th rowspan="2">Rock Type</th><th rowspan="2">($/t milled)</th><th colspan="3">Central Zone</th><th colspan="2">North Zone</th><th>Sofia Main</th><th>Sofia North</th><th colspan="3">Delya</th></tr>
<tr><th>WOL</th><th>BIOX – 100% CZ</th><th>BIOX – 25% CZ/ 75% NZ</th><th>WOL 100% NZ</th><th>BIOX 100% NZ</th><th>WOL</th><th>WOL</th><th>WOL 100% Delya</th><th>BIOX 100% Delya</th><th>BIOX 20% Delya 80% NZ</th></tr>
<tr><td>Oxide ($/t milled)</td><td>Oxide</td><td>12.59</td><td>N/A</td><td>N/A</td><td>12.90</td><td>N/A</td><td>12.46</td><td>13.39</td><td>13.31</td><td>N/A</td><td>N/A</td></tr>
<tr><td>Oxide Trans ($/t milled)</td><td>Oxide Trans</td><td>16.44</td><td>N/A</td><td>N/A</td><td>15.50</td><td>N/A</td><td>14.97</td><td>14.86</td><td>15.88</td><td>N/A</td><td>N/A</td></tr>
<tr><td>Reduced Trans ($/t milled)</td><td>Reduced Trans</td><td>17.95</td><td>57.05</td><td>41.52</td><td>16.71</td><td>40.37</td><td>16.41</td><td>15.75</td><td>16.17</td><td>62.41</td><td>44.25</td></tr>
<tr><td>Fresh $/t milled)</td><td>Fresh</td><td>19.58</td><td>57.05</td><td>41.52</td><td>17.93</td><td>40.37</td><td>17.84</td><td>16.48</td><td>17.81</td><td>62.41</td><td>44.25</td></tr>
</table></td></tr>
</table>

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>

Site G&A: $8.60/t
Capital construction cost of $413 million
A financial model run on the above parameters produced the following NPV sensitivity at different gold prices and discount factors:

<table>
<tr><th rowspan="2">Discount Rate</th><th colspan="3">Gold Price ($/oz)</th></tr>
<tr><th>1,000</th><th>1,200</th><th>1,400</th></tr>
<tr><td>0%</td><td>$258 million</td><td>$591 million</td><td>$925 million</td></tr>
<tr><td>5%</td><td>$109 million</td><td>$344 million</td><td>$579 million</td></tr>
<tr><td>10%</td><td>$22 million</td><td>$193 million</td><td>$363 million</td></tr>
<tr><td>IRR</td><td>12%</td><td>25%</td><td>37%</td></tr>
<tr><td>Payback</td><td>5 years</td><td>2.8 years</td><td>2.4 years</td></tr>
</table>
</td></tr>
<tr><td>Social</td><td>

A full updated ESIA is required for the Project and Environmental and Social Management workshops will be required including all affected parties to find practical and effective management measures to leverage the benefits of the mine to the region and the Senegalese economy, but minimise the negative impact on communities and environment.

<u>Potential Positive Impacts</u>

Employment opportunities;
Local procurement of goods and services;
Skills development and training;
Increase in local trade.
Increased contribution to the National economy through taxes and royalties; and
Infrastructure development and improvements for local community (such as roads and potable water).

<u>Potential Negative Impacts</u>

Loss of predominantly pristine wetlands’
Loss of habitat and reduction in biodiversity;
Potential downstream impacts to the Niokolo-Koba National Park due to surface water reduction and reduced water quality;
Reductions in groundwater levels due to pit dewatering;
Dust and noise generation;
Visual impact due to the development of infrastructure and waste facilities
Loss of land use (including agriculture and artisanal mining); and
Population influx and associated pressure on natural resources and increased social ills.

</td></tr>
</table>

Criteria	Commentary
Other	• There is a requirement to complete a full feasibility showing the viability of the project to obtain the mining permit. Environmental certification is also required on the completion of the full ESIA.
Classification	• The Ore Reserve is classified as Probable based upon the confidence levels determined in the Mineral Resource and accurately reflects the Competent Person's views of the deposit.
Audits or Reviews	• No external audits or reviews have been carried out yet, but the process of converting Mineral Resources to Ore Reserves follow similar procedures to those at other Barrick Africa and Middle East operations which are regularly audited.
Discussion of Relative Accuracy/ Confidence	• The Ore Reserve estimate is rated as probable in-line with the Mineral Resource classification. • Due to the complex nature of the deposit it is recognised there will be local deviations to the Ore Reserve estimate hence the level of confidence is placed as Probable. The highly nuggety nature of the gold mineralisation in the CZ brings complexity of sampling and assay. This has been mitigated by additional RC drilling to obtain a larger sample and bulk 1 kg LeachWELL plus tail fire gold content analysis to more accurately analyse the samples. • The modifying factors used are considered realistic for this Project based on the geotechnical environment and mining methods selected and assuming good practise is applied. The values applied are in-line with published data from similar scale operations.