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**UNITED STATES  
SECURITIES AND EXCHANGE COMMISSION  
Washington, D.C. 20549**

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**FORM 6-K**

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**Report of Foreign Private Issuer  
Pursuant to Rule 13a-16 or 15d-16  
of the Securities Exchange Act of 1934**

**Date: March 28, 2016  
Commission File Number 001-31528**

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**IAMGOLD Corporation**  
(Translation of registrant's name into English)

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**401 Bay Street Suite 3200, PO Box 153  
Toronto, Ontario, Canada M5H 2Y4  
Tel: (416) 360-4710  
(Address of principal executive offices)**

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Indicate by check mark whether the registrant files or will file annual reports under cover Form 20-F or Form 40-F.

Form 20-F                       Form 40-F

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(1):

**Note:** Regulation S-T Rule 101(b)(1) only permits the submission in paper of a Form 6-K if submitted solely to provide an attached annual report to security holders.

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Indicate by check mark whether by furnishing the information contained in this Form, the registrant is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934.

Yes                       No

If "Yes" is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b): 82- \_\_\_\_\_

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**Description of Exhibit**

<u>Exhibit</u>	<u>Description of Exhibit</u>
99.1	Sadiola Sulphide Project (SSP) 2015, NI 43-101 Report, Mali dated and effective March 15, 2016

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**Signatures**

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

**IAMGOLD CORPORATION**

Date: March 28, 2016

By: /s/ Tim Bradburn  
Vice President, Legal and Corporate Secretary



# **Sadiola Sulphide Project (SSP) 2015, NI 43-101 Report Mali**

**Dated and effective March 15, 2016**

Prepared by:

**IAMGOLD CORPORATION**  
**401 Bay St #3200,**  
**Toronto, ON**  
**M5H 2Y4**

and:

**G MINING SERVICES INC.**  
7900 W Taschereau Blvd.  
Building D, Suite 200  
Brossard, Québec  
Canada J4X 1C2

**SNOWDEN MINING INDUSTRY CONSULTANTS**  
Technology House, Greenacres Office Park, Cnr. Victory and  
Rustenburg Roads, Victory Park  
Johannesburg 2195  
SOUTH AFRICA

## Sadiola Sulphide Project (SSP) 2015, NI 43-101 Report Mali

### IAMGOLD CORPORATION

401 Bay St #3200,  
Toronto, ON  
M5H 2Y4  
Tel: (416) 360-4710 .  
E-mail :  
Web Address: [www.company.com](http://www.company.com)

### G MINING SERVICES INC.

7900 W. Taschereau Blvd.  
Suite D-200Brossard, Québec  
Canada J4X 1C2  
Tel: (450) 465-1950 • Fax: (450) 465-6344  
E-mail: [l.gignac@gmining.com](mailto:l.gignac@gmining.com)  
Web Address: [www.gmining.com](http://www.gmining.com)

### Dated and effective March 15, 2016

Compiled by:

Philippe Gaultier, Ing. MSc (OIQ # 130381)  
Director of Development Projects  
IAMGOLD Corporation

Daniel Vallières, Ing. (OIQ # 107203)  
Director, Mining engineering  
IAMGOLD Corporation

Jérôme Girard, Ing., P. Eng (OIQ # 116471) (PEO # 100160489)  
Manager, Metallurgy  
IAMGOLD Corporation

### SNOWDEN MINING INDUSTRY CONSULTANTS

Technology House, Greenacres Office Park, Cnr. Victory and Rustenburg  
Roads, Victory Park  
Johannesburg 2195  
SOUTH AFRICA  
PO Box 2613, Parklands 2121  
SOUTH AFRICA  
Tel: +27 11 782 2379  
Fax: +27 11 782 2396  
Reg. No. 1998/023556/07  
[johannesburg@snowdengroup.com](mailto:johannesburg@snowdengroup.com)

Luc-B Denoncourt, Ing. (OIQ # 129874)  
Project Director  
IAMGOLD Corporation

Louis-Pierre Gignac, Ing. (OIQ 132995)  
Co President  
G Mining Services Inc.

Mark Burnett, MSc (MRM), B.Sc.(Hons); PGDTE; CBM; CAG; GCG; Pr.  
Sci. Nat (400361/12); FGSSA: MSAIMM  
SNOWDEN

---

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## 1. **SUMMARY**

### 1.1 **Introduction**

IAMGOLD Corporation (IAMGOLD) has prepared a Technical Report for its jointly owned Sadiola Gold Mine, located in Mali with the support of G Mining Services Inc. (G Mining) and Snowden Mining Industry Consultants (Snowden). The purpose of this Technical Report is to support the disclosure of the December 31, 2015 Sadiola Gold Mine Mineral Resources and Mineral Reserves estimate. Therefore, all the report was made on a 100% ownership basis.

This technical report discloses certain scientific and technical information sourced from reports authored by each of AGA and SEMOS. Such reports are listed at Section 27 of this technical report (References). Snowden has verified the data disclosed and used for the Mineral Resource estimates reported, including sampling, analytical and test data underlying the information or opinions contained in the AGA and SEMOS reports. Data verification for the information in the AGA and SEMOS reports used in Mineral Resource estimation was performed by Snowden.

The Sadiola Gold mine is located in western Mali some 77 km south of the regional capital of Kayes. The Sadiola gold deposit is mined by the *Société d'Exploitation des Mines d'Or de Sadiola S.A.* (SEMOS), the operating company formed through a joint venture agreement between AngloGold Ashanti (41%), IAMGOLD (41%) and the Malian Government (18%). AngloGold Ashanti, through its wholly owned subsidiary AngloGold Mali S.A., is the mine operator. The Malian Government shareholding is a free-carried interest.

SEMOS is bound by the original prospecting and exploitation agreement (including its subsequent legal modifications) entered into on April 15, 1990 between AGEM and the Mali Government, and the mining license is valid for the original mineral commodities until April 15, 2020. The identity number of the current exploitation area is " *DECRET No 00-080/PM-RM DU 06 MARS 2000* " and is a modification of all previous exploitation areas. The surface area defined by " *DECRET No 00-063/PM-RM DU 25 FEV 2000* ".

### 1.2 **Accessibility, Climate, Local Resources, Infrastructure & Physiography**

#### 1.2.1 **Access**

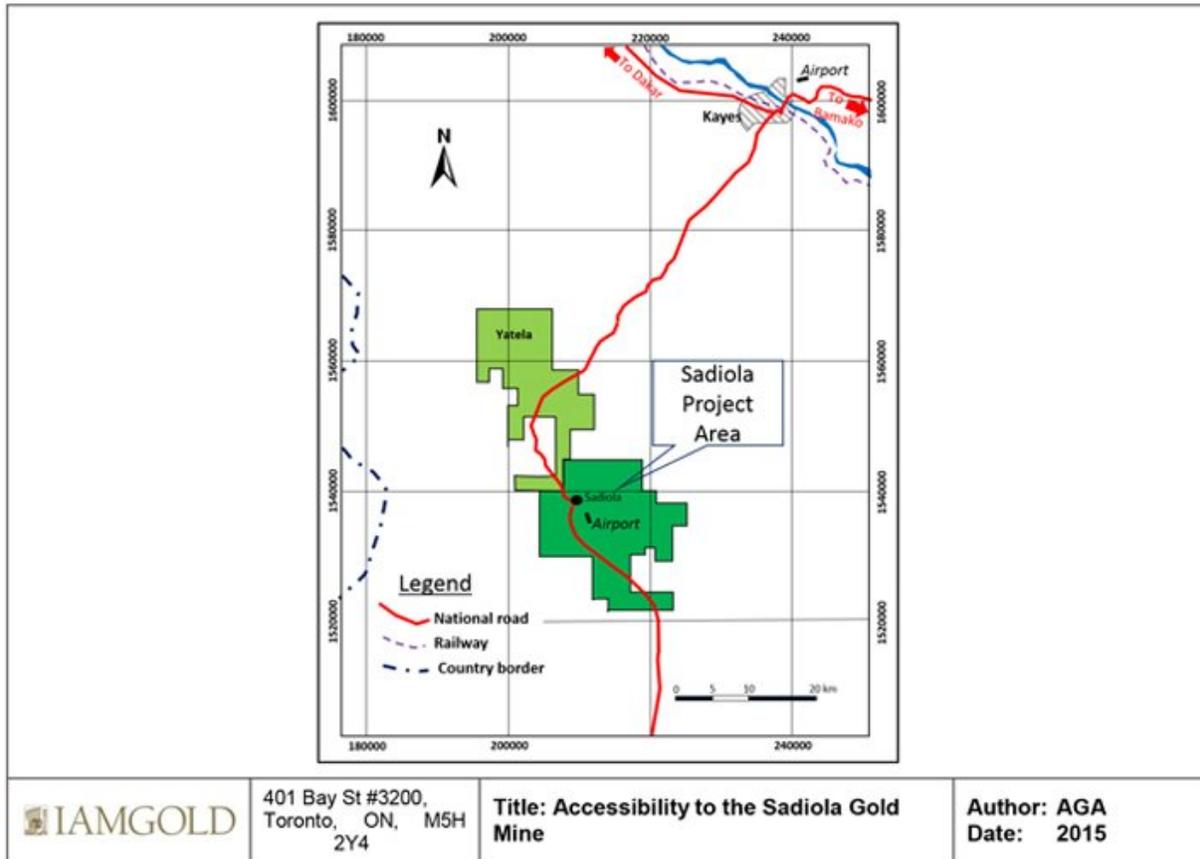
Mali, officially the Republic of Mali, is a landlocked nation in West Africa. Mali borders Algeria on the north, Niger on the east, Burkina Faso and the Côte d'Ivoire on the south, Guinea on the southwest, and Senegal and Mauritania on the west. As such, Mali is dependent on its neighbors for ocean-borne inbound materials and supplies.

The preferred entry point for freight to Sadiola is through the port of Dakar in Senegal and overland transport by either rail (to Kayes only) and/or road. Other ocean freight inbound routes exist via Lomé in Togo and Abidjan in Côte d'Ivoire.

The highway between Dakar and Kayes is paved along the whole of its 836 km extent, having been completed at the end of 2012. The section of road between Bamako and Kayes is 506 km on a paved highway.

Access to the Sadiola operation from Kayes is by an 80 km long, regional, compacted laterite surfaced, all-weather, single carriageway road. Kayes is serviced by rail, road and air from Bamako, the capital of Mali, and from Dakar, the capital of Senegal.

**Figure 1.1: Accessibility to the Sadiola Gold Mine**



Mali's main international airport is located in Bamako. There are daily flights to many other African and European destinations and the airport can receive large cargo carriers. The nearest national airport to the mine is located in Kayes. Kayes has a 2,700 m long sealed airstrip which has been widened and

expanded to allow for larger planes to land and take-off so that it can also cater for cargo carriers. The Sadiola Mine site is also equipped with a 1,800 m compacted, laterite airstrip.

Rail transport on the Dakar-Niger rail line is limited to 40-ton wagons, a width of 2.6 m, a height of 3 m and a length of 13 m. Road transport on national highways is limited to 7.5 Tonnes per axle and logistics providers can transport loads up to 120 Tonnes.

### **1.2.2**            **Climate**

The mine lies within the subtropical-to-arid climatic zone of Mali between the 700 mm and 900 mm isohyets. The climate is subtropical to arid with a dry and a rainy season. The dry season is from November to February with temperatures ranging from 15°C to 30°C. From March to June, temperatures range from 25°C to 45°C. The rainy season lasts from July to October.

Rainfall, in the form of high intensity convectonal thunderstorms, commonly accompanied by strong winds, occurs in summer. Annual rainfall exceeds 1,000 mm, and evaporation is twice the precipitation rate. Periods of water surplus and shortage occur in the wet and dry seasons respectively.

### **1.2.3**            **Local Resources**

The Sadiola Gold Mine is situated proximal to 46 officially recognized villages and several hamlets. The main villages are Farabakouta, Neteko, Sadiola and Borokone. A mine village has been established to the northeast and provides housing, a medical clinic, a park and recreation facilities for mine employees and dependents. Other facilities include guest accommodation, a post office, a supermarket, sewage treatment facilities and other amenities.

The Sadiola Mine employs more than 1,000 people, including those employed by outside contractors. The majority of Sadiola personnel are Malian nationals (~93%), with the remainder being expatriates from South Africa (~7%). The majority of the unskilled labor is sourced from the nearby town of Kayes, Sadiola and neighboring villages.

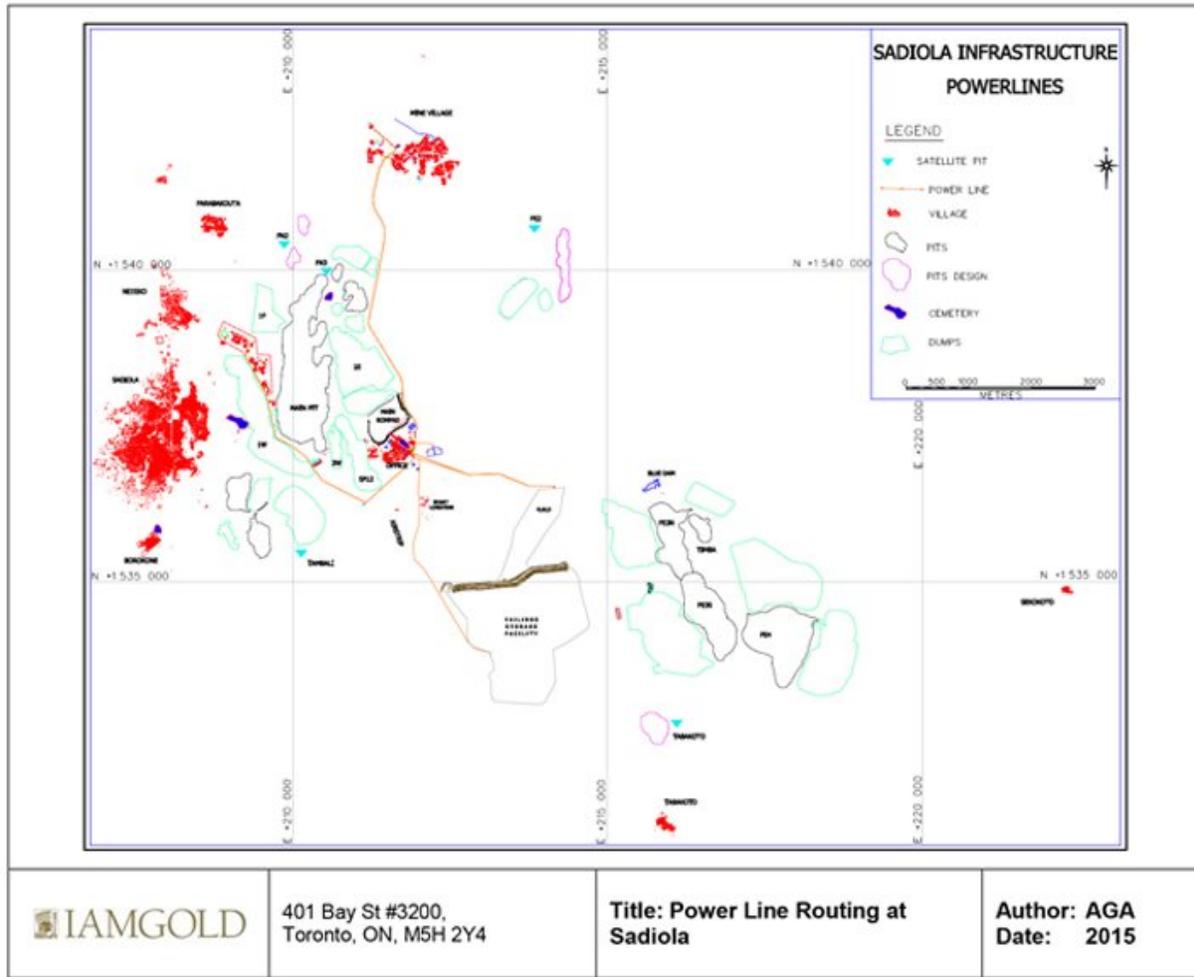
### **1.2.4**            **Infrastructures**

#### **1.2.4.1**           **Power**

Electrical power is currently provided through 18 x 1 megawatt (“MW”) medium-speed diesel generators which are capable of meeting an average demand of 16.7 MW. Approximately 4.73 M liters of diesel fuel per month are required for power generation and mining, under a contract with Total/ELF Petroleum Company. The 7 Mt national strategic fuel depot, situated in Kayes, is used as back-up storage in case of major road and/or rail disruptions.

The power infrastructure routing is outlined in yellow in Figure 1.2.

Figure 1.2: Power Line Routing at Sadiola



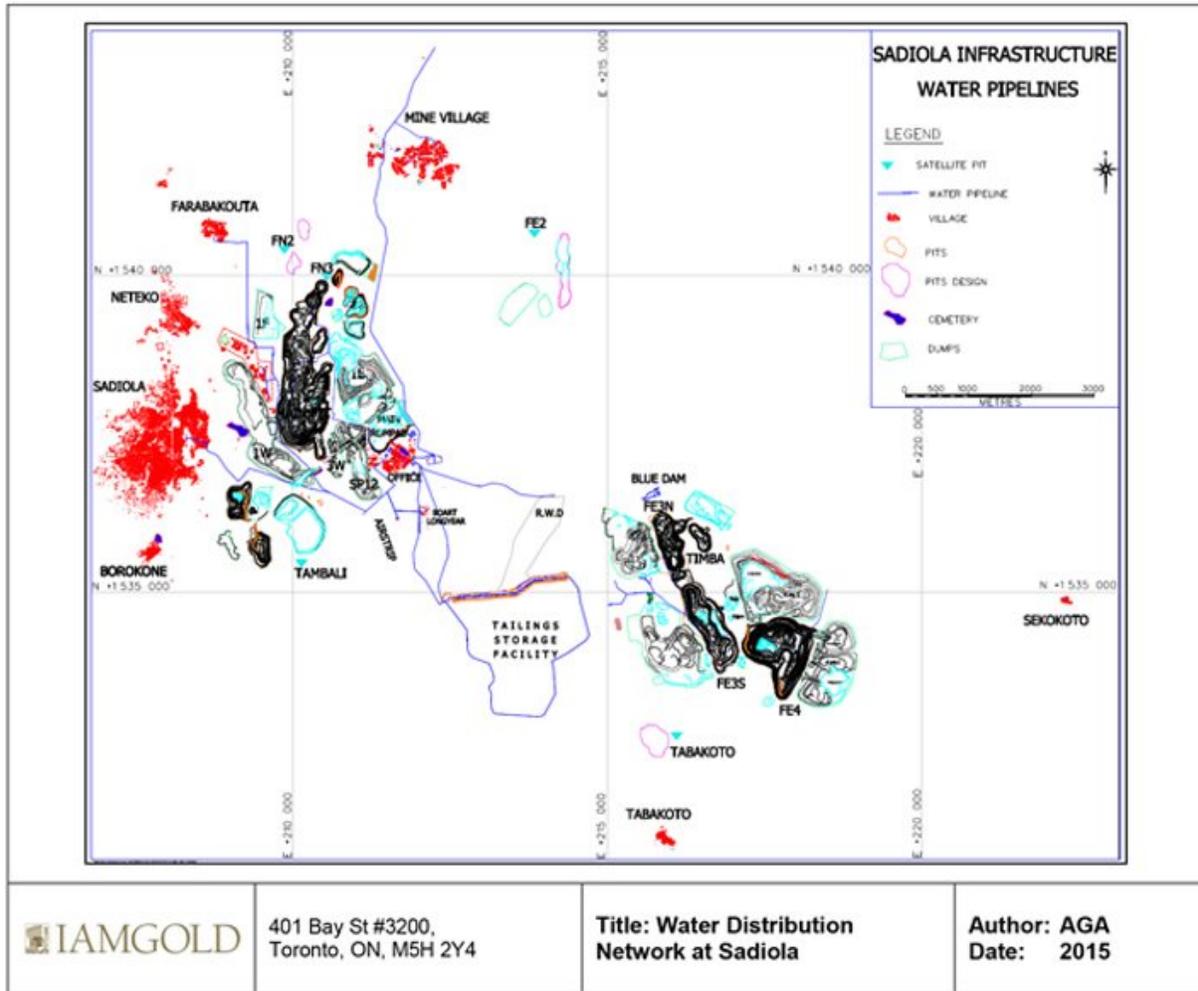
The current project includes an 89 km power line that will feed the processing plant and the infrastructure of the mine. The current generators will be used for emergency only.

1.2.4.2 Water

A 55 km pipeline from the Senegal River, the only reliable source of surface water in the region, was built to provide approximately 8 M m<sup>3</sup> per year of process water in order to ensure that the Sadiola Gold Mine does not impact on local water resources. Potable water for both the mine operation and the mine town site and local villages are supplied from the pipeline, as well as local boreholes, and treated prior to distribution.

The water distribution network within the Sadiola area is shown in Figure 1.3.

**Figure 1.3: Water Distribution Network at Sadiola**



**1.3 Project History**

**1.3.1 Existing Mine**

The Sadiola gold mine began commercial production in December 1996. The operation has processed over 86.3 M t of ore and has produced some 7.5 M ounces of gold as of the end of 2015. The gold grade and quantity of metal makes Sadiola a world class gold deposit.

The existing processing facility was designed for the processing of soft ore and can only introduce a small percentage of hard ore in the mill feed. Since the beginning of the operation, mining activities have been outsourced with mine engineering and geological services provided by SEMOS. All other activities on site such as processing are performed by SEMOS.

## 1.4 **Geological Setting and Mineralization**

The Sadiola deposit, previously known as the Sadiola Hill deposit, is located in west Mali within the Kedougou-Kenieba Inlier (KKI), a major Early Proterozoic window of volcano-sedimentary greenstone belts and calc-alkaline granite intrusions that comprise part of the Lower Birimian terranes of the West African Craton. The inlier is positioned at the northeast margin of the Kenema-Man Shield and is bound to the west by the Pan-African Mauritanide Hercynian Belt and concealed to the north, east and south by undeformed Neoproterozoic and Paleozoic sedimentary formations of the Taoudeni Basin.

The volcano-sedimentary sequences of the KKI are separated into two lithostratigraphic super groups which correspond reasonably well to the Mali West 1 classification. The Mako (or Saboussire) Super group is in the west and characterized by massive and pillowed tholeiitic basalt, calc-alkaline volcanic rocks and interbedded volcanoclastic sediments. To the east, the younger Dialé Daléma Super group comprises platform type sediments of carbonate, graywacke, sandstone and pelite, intruded by intermediate and felsic calc-alkaline rocks.

The super groups are separated by major, regional crustal scale structures. Regional metamorphism is to greenschist facies with amphibolites facies metamorphism only observed in the contact aureoles around major intrusions.

Gold mineralization in the Sadiola main pit has been mined for 2 km along strike. Mineralization occurs in all four rock types: graywacke, carbonate, diorite and quartz-feldspar porphyry (QFP), usually close to or within the contact of the Sadiola Fracture Zone (SFZ). The bulk of the mineralization is hosted in the footwall adjacent to the SFZ. The mineralization has a strong structural control and is spatially associated with a complex weathering and alteration pattern.

### 1.4.1 **Oxide**

The geometry of the extensive, soft, oxide deposit and its supergene enrichment of gold relates almost exclusively to the weathering history of the primary mineralization. Intense tropical weathering has produced deep troughs of white to grey, decarbonated, kaolin-rich saprolite, locally abundant nontronite and relative gold enrichment. Penetration of groundwater has caused oxidation of the primary sulphides and the formation of sulfuric acid, further promoting deeper argillization of the bedrock. The variable permeability of the deposit, controlled by faulting, shearing and porosity, has led to the irregular 'karst-like' weathering geometry from 30 m deep in the north to 220 m in the south. Weathering is deepest along the SFZ.

The deeply weathered saprolite was protected from erosion by a capping of hardpan laterite (ferricrete).

### 1.4.2 **Sulphide Mineralization**

Drilling of the (unweathered) primary mineralization has allowed detailed investigation of major and minor hydrothermal alteration processes that were active during the formation of the deposit.

Primary gold is extremely fine grained, dominantly less than 15 microns ( $\mu\text{m}$ ), with rare grains approaching 50  $\mu\text{m}$  and visible gold is rare. Gold mineralization is associated with arsenic and antimony dominated sulphide assemblages of arsenopyrite, pyrrhotite, pyrite, stibnite and gudmundite as well as potassic, calc-silicate, propylitic alteration and silicification. Much of the mineralization appears related to be related to deformation of the host rock.

### 1.5 **Exploration**

Eight key oxide targets, identified at a targeting workshop in Q4 2013, were the focus of exploration during 2014. During 2015 exploration drilling was undertaken on Sadiola North (FN) and Tabakoto. Scoping studies have been completed at Tambali and Sadiola North from new geological models. Estimates show potential for continued oxide and sulphide exploration, which is ongoing.

### 1.6 **Drilling**

Eight (8) key oxide targets, identified at a targeting workshop in Q4 2013, were the focus of exploration since 2014. The aim was to fast track the oxide Mineral Reserve development to extend the Life of Mine (LOM). No capital drilling was done to upgrade resource confidence during 2014. Drilling returned limited potential. Work included 20,536 m of Reverse Circulation (RC) drilling, outcrop and pit mapping and sampling, X-Ray fluorescence (XRF) analysis, and research by the Centre for Exploration Targeting (CET), University of Western Australia.

Some degree of sulphide exploration was conducted in the FE3 and FE4 pits. Encouraging results from both campaigns indicated an orebody that can potentially add flexibility to the mine plan and quickly access low cost sulphides for a mixed oxide/sulphide mining scenario.

Scoping studies have been completed at Tambali and Sadiola North and FN3 from new geological models. Estimates show potential for continued oxide and sulphide exploration.

### 1.7 **Metallurgical Test Work**

#### 1.7.1 **Metallurgical Testing**

The Sulphide ore was characterized by degree of weathering, lithology and by localization within the deposit. Calcite marble is the dominant rock type that will be processed during the hard sulphide production.

Metallurgical testwork performed in 2009-2010 included the following:

- The mineralogical and geochemical characterization of 58 ore samples taken from various locations within the deposit and of the various major rock types (calcite marble, greywacke and diorite).
- A general gold deportment and sulphide liberation study was undertaken to predict gold behaviour during processing for eight composite samples.
- Organic carbon was found to be less than 0.1% and preg-robbing is not expected to be a problem.
- Heavy liquid separation reported a mass pull of below 1.7% reporting to the sinks with excellent gold upgrading of between 20% and 50% to the heavy fraction. These results were matched by the gravity tests conducted.
- A grading analysis reported a higher gold to mass ratio in the coarse material from the southern parts of the ore body.
- X-ray diffraction analyses showed that the samples were composed of quartz, feldspar, carbonates and mica and contained minor to traces of amphiboles, chlorite, scapolite and molybdenite.
- The QEMSCAN trace mineral search concluded an average gold grain size between 3 and 7 microns, or a gravity average between four (4) and twelve (12) microns.

Comminution testwork included the following tests:

- Bond Impact Work Index used to determine crushing design parameters. Results indicated 13.4 kWh/t for diorite, 10.7 kWh/t for greywacke, and 12.2 kWh/t for calcite marble;
- The JKTech drop weight test was used to determine SAG mill capability;
- The SAG mill comminution (SMC) test gave a lithological weighted average value of AxB of 33.4;
- Bond Rod Mill Work Index was determined to be 14.86 kWh/t;
- Bond Ball Mill Work Index was determined to be 13.33 kWh/t;
- Pennsylvania Abrasion test is used to determine the consumption of steel media. It was determined that the greywacke material will consume a lot more steel media than other rock types, the lithological weight average was a value 0.082.

Leaching and gravity testwork includes the following:

- Gravity testwork to evaluate the recoverable gold and the associated design requirements indicated that a Falcon recovery of 24.8% with an 82.1% intensive leach recovery of the gravity concentrate;
- Leaching tests were conducted on both ROM ore and the gravity tails, resulting in highly variable recovery of gold from different parts of the orebody. Cyanide consumption was estimated to be 0.632 kg/t and lime consumption 0.61 kg/t;

- For the Optimization Study metallurgical testing was done to assess the recovery from the hard sulphide stockpile. The testwork and plant trial confirmed the 76% recovery established for the Feasibility Study and yielded a higher head grade.

The gold recovery by lithology and by location in the deposit for the hard sulphide ore (Table 1.1) is estimated to range from 68% to 83% with a weighted average recovery for the hard sulphide estimated at 76%. The recoveries for other rock types are based on actual plant results. The average recovery for soft oxides was set at 94% and 80% for soft sulphides. The recovery assumed for processing existing hard sulphide stockpiles is also set at 76% based on plant trials.

**Table 1.1: Average Gold Recovery for Hard Sulphide Ore**

Rock Type	Pit Location	Gravity	Intensive Leach	Leach Gravity tails	Predicted incl. Solution & Carbon Losses
		%	%	%	%
Calcite Marble	North	26.2	87.5	74.5	77.8
	Middle	24.3	78.2	76.0	76.0
	South	25.5	78.7	71.8	72.7
Greywacke	North	18.5	92.1	81.7	83.2
	Middle	13.9	74.7	70.3	70.5
	South	37.6	86.3	76.7	79.6
Diorite	North	30.9	68.5	80.5	76.0
	Middle	4.3	77.1	76.9	76.5
	South	7.1	58.1	69.0	67.8

### 1.8 Mineral Resource Estimate

The SEMOS Mineral Resources are reported in accordance with the guidelines of the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2012 edition (“the JORC Code (2012)”), and the AGA reporting guidelines (2014), above the Mineral Resource cut-off grade.

The classification categories of Measured, Indicated and Inferred Mineral Resources classified in accordance with the JORC Code (2012) are considered to be equivalent to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) categories of Measured, Indicated and Inferred Mineral Resources (CIM, 2014).

The majority of the samples used in the estimation of the Mineral Resources were completed by the reverse circulation (RC) drill method for the oxide pits and diamond coring (DD) for the Sadiola Main Pit oxides and sulphides. All assaying was completed with standard fire assay techniques.

A gamma density downhole probe was used to estimate global densities for the saprolitic grade control drilling and modelling. The probe was also used where possible in saprolitic exploration drillholes to support the resource density estimate. The SEMOS drillhole data is captured in a Structured Query Language (SQL) database with a customized CAE software front end.

Three-dimensional (3D) models were created for the deposits using block models and ordinary kriging (OK) with Datamine Studio 3™ software. Post processing of the panel estimates was carried out by applying the uniform conditioning (UC) technique, using Isatis 2014 and 2015™ software, to all the deposits to generate recoverable resources.

After assessing all items specified within the JORC Code (2012) such as sampling techniques, data quality and estimation techniques, the Mineral Resource was classified according to the AGA standard “15% rule”, as described in the AGA Mineral Resource and Ore Reserve Guidelines. Drillhole spacing and, in the case of Sadiola Main Pit North, kriging variance was also incorporated in the classification (Bloy Resource Evaluation, 2015a; Bloy Resource Evaluation, 2015b).

The AGA classification methodology provides an average grade above cut-off estimate with less than 15% relative error at 90% confidence. For an Indicated Mineral Resource, annual production, and for a Measured Mineral Resource, quarterly production, should meet these criteria.

The Mineral Resource has been classified as Measured Resource, Indicated Resource and Inferred Resource and reported within optimized pit shells, based on a gold price of US\$1,400 per ounce (oz). A summary of the Mineral Resources stated as at December 31, 2015 is presented in Table 1.2 and Table 1.3. The tonnages were estimated and are stated on a dry basis and are shown on a 100% ownership basis.

During 2015, a total of 151,000 oz were processed. Reconciliation of the resource can be reviewed in Table 1.4.

The oxide resources are made up of eight discrete in-situ areas (current and future open pits) and stockpiles located in Zone A of the SEMOS mining lease (around the main run of mine (ROM) pad and the FE3/4 Satellite ROM pads). Of the eight areas, two were mined during the course of 2015 (FE2 and FN3 pits).

A stockpile re-evaluation was completed in 2014 which resulted in a decrease of 145,000 oz in 2014 compared to 2013. Most of this change, 68%, was due to restating the hard sulphide stockpile tonnages following a drilling program and review of the survey data. The stockpile classifications have remained the same since the 2014 re-evaluation exercise.

Although Sadiola is considered to be a mature operation, it remains prospective for oxide and sulphide gold mineralization. The geochemistry and geophysical data continues to be used to enhance the geological model, which has been developed and debated through a number of workshops. A research project commenced during 2012 in conjunction with the Centre for Exploration Targeting (CET) at the University of Western Australia. The objective of this three-year project is to provide an effective assessment of the controls on the geometry and genesis of the Sadiola-Yatela deposit mineralization to aid exploration opportunities for near-mine oxide and sulphide (hard rock) resources at Sadiola-Yatela. The results of this study have been documented in Masurel *et al* (2015).

**Table 1.2: Sadiola Inclusive Measured and Indicated Mineral Resources by Area (December 31, 2015)**

Area	Cut-off (Au g/t) weighted average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Sadiola SSP	0.60					100,000	1.9	190,224	6,116	100,000	1.9	190,224	6,116
Area 1	0.70					1,968	2.48	4,889	157	1,968	2.48	4,889	157
Area 2	0.63					2,758	1.57	4,333	139	2,758	1.57	4,333	139
Stockpiles	-	1,462	1.68	2,451	79	14,155	1.09	15,405	495	15,617	1.14	17,856	574
<b>Total</b>	<b>0.60</b>	<b>1,462</b>	<b>1.68</b>	<b>2,451</b>		<b>118,881</b>	<b>1.81</b>	<b>214,851</b>	<b>6,908</b>	<b>120,342</b>	<b>1.81</b>	<b>217,302</b>	<b>6,986</b>

Source: AGA, 2016b

Notes: Mineral Resources are quoted inclusive of Ore Reserves. The Measured and Indicated Resources are inclusive of those Mineral Resources modified to produce the Ore Reserve. The Mineral Resources are quoted using a gold price of US\$1,400. Rounding of figures may result in computational discrepancies. Gram per ton (g/t); gold (Au); kilogram (kg).

**Table 1.3: Sadiola Inferred Mineral Resources by Area (December 31, 2015)**

Area	Cut-off (Au g/t) weighted average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Sadiola SSP	0.60	14,652	1.82	26,725	859
Area 1	0.70	71	2.86	203	7
Area 2	0.72	802	1.76	1,408	45
Stockpiles	-	0	0	0	0
<b>Total</b>	<b>0.61</b>	<b>15,524</b>	<b>1.83</b>	<b>28,336</b>	<b>911</b>

Source: SEMOS, 2016b

Notes: Rounding of figures may result in computational discrepancies. The Mineral Resources are quoted using a gold price of US\$1,400.

Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration. Gram per ton (g/t); gold (Au); kilogram (kg).

**1.9 Reconciliation**

A summary reconciliation between the 2014 Mineral Resource and the 2015 Mineral Resource declaration is presented in Table 1.4.

The primary reason for the increase in the Mineral Resource is the inclusion of Mineral Resources classified at the Inferred level of confidence in the Main Pit, not previously included in the Mineral Resource. AGA included this material to align the Mineral Resource Reporting undertaken at Sadiola with other AGA operations.

**Table 1.4: Sadiola Mineral Resource Reconciliation**

	<b>Tonnes (M t)</b>	<b>Au (‘000 oz)</b>	<b>Comments</b>
Previous (2014)	120.64	6,887	Published as at December 31, 2014
Depletion	-4.26	-151	Depleted in 2015 from FN3 and FE2 pits and stockpiles
Gold price	-9.77	-525	Reduction in the resource gold price assumption from \$1,600/oz in 2014 to \$1,400/oz in 2015 resulted in reduction to the Mineral Resource
Cost	2.68	148	Lower fuel price and improved efficiencies resulted in lower costs for 2015 optimizations
Exploration	2.62	135	Infill drilling at FE2 and SSP (northern portion)
Methodology	23.48	1,392	Including the Inferred Mineral Resource in the optimization of the shell defining the Mineral Resource (only Measured and Indicated used in 2014)
Other	0.49	11	Change as a result of stockpile adjustments
Current (2015)	135.87	7,897	As at December 31, 2015

Source: SEMOS, (2015b)

Note: The Sadiola resource reconciliation includes Measured, Indicated and Inferred resource categories.

There is no guarantee that these Inferred Mineral Resources will be converted to Mineral Reserves, given the low confidence classification.

**1.10 Mineral Reserves Estimate**

The mineral reserve estimated at December 31, 2015 includes reserves in Satellite pits, Stockpiles and the SSP pit.

Table 1.5: Sadiola December 31, 2015 Mineral Reserve

SSP Reserves	Tonnage (kt)	Gold (oz)	Grade (g/t)
Satellites Pits	-	-	-
Stockpiles	-	-	-
SSP Pit	-	-	-
<b>Total Proven</b>	-	-	-
Satellites Pits 2	1,503	108,595	2.25
Stockpiles 2	5,262	331,279	1.96
SSP Pit 1	63,030	3,916,724	1.93
<b>Total Probable</b>	<b>69,795</b>	<b>4,356,598</b>	<b>1.94</b>
<b>Total Proven &amp; Probable</b>	<b>69,795</b>	<b>4,356,598</b>	<b>1.94</b>

<sup>1</sup> Mineral Reserve was estimated by Louis Pierre Gignac, Vice-President Engineering at G Mining Services inc. He has been involved in mining engineering and financial evaluation for 14 years. He fulfills the requirements as a qualified person for the Purpose of NI 43-101.

<sup>2</sup> Mineral Reserve was estimated by Andrew Bridges, has a minimum of 5 years relevant experience to the type and style of mineral deposit under consideration and to the activity which is being undertaken to qualify as a Competent Person (or Recognized Mining Professional) as defined in the 2012 Edition of the JORC Code and the 2009 edition of the SAMREC code.

<sup>3</sup> Satellites pits and stockpile are estimated at \$ 1,100/oz and the SSP pit is estimated at \$ 1,200/oz.

<sup>4</sup> Cut-off-grade for satellites pits is 0.85 g/t Saprolite Oxide and 1.10 g/t Hard Sulphides. For the SSP pit, the cut-off-grade is 0.70 g/t Hard Sulphide.

### 1.11 Mining Method

A large push-back to the existing Sadiola Main Pit is needed to mine sulphide ore. The oxide ore production of the main pit have ceased in 2010. The most appropriate mining method for the Sulphides is an open pit truck and shovel (or excavator) method. Given the large tonnage, increased pit size and greater depth of the pit, larger equipment consisting of RH170s (20 m<sup>3</sup> shovels) and 150-tonne trucks (CAT 785) is deemed more appropriate than the fleet of RH120s and CAT 777s currently operated by mine contractor.

A study was undertaken to compare bulk mining and selective mining for the Sadiola Sulphide project based on the Prefeasibility study work. The study concluded that, given the structure of the mineralization of the Sadiola Sulphide ore body, there is no advantage of mining using either the bulk or selective mining methods since the ore body is generally sub vertical. However due to the simplified operation of bulk mining and the additional costs associated with selective mining, it was concluded that bulk mining would deliver cost savings over selective mining methods.

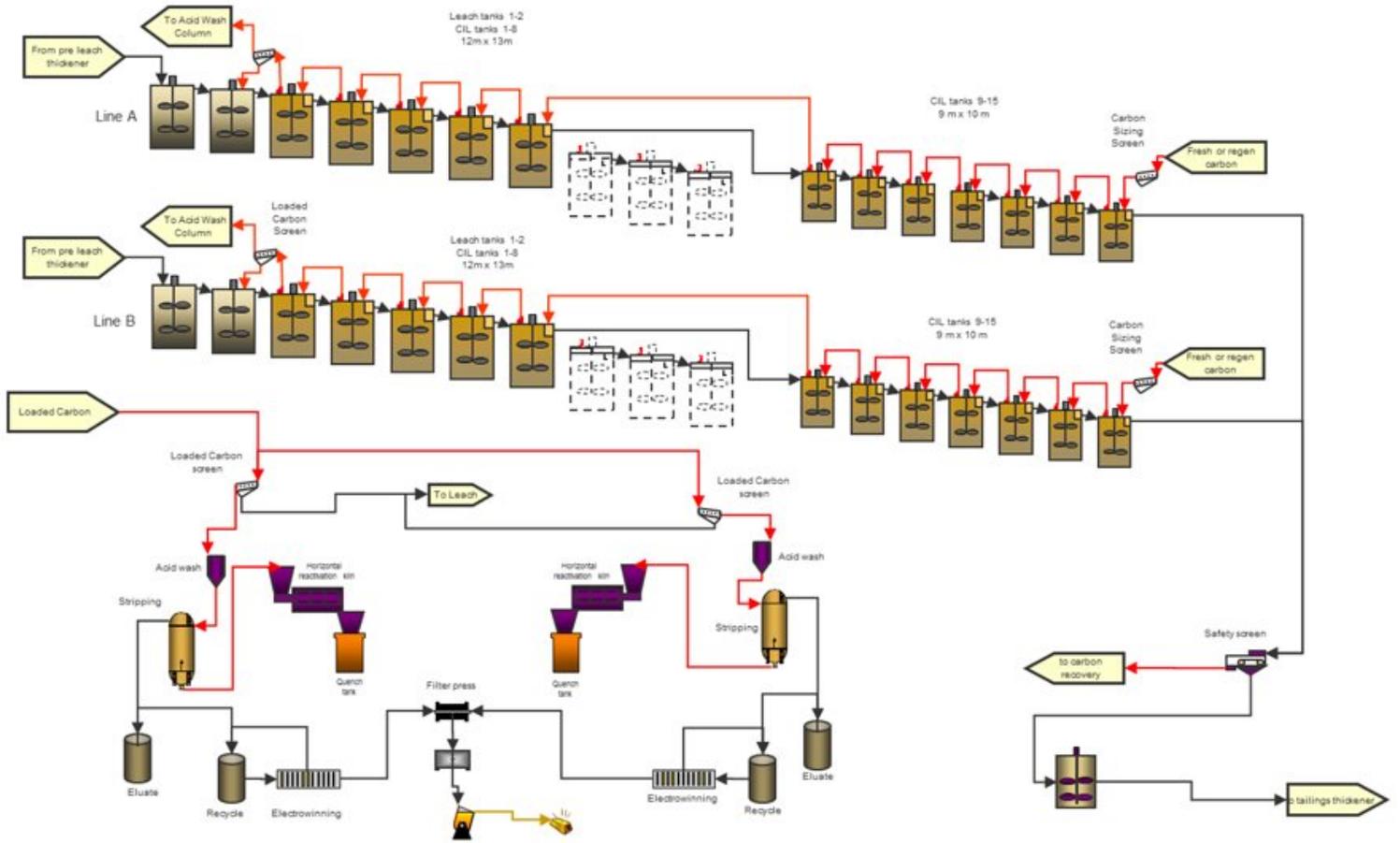
**1.12      Recovery Method****1.12.1      Process Plant Design**

The current facility is designed only for soft ore and will require additional capacity to process hard ore from SSP project. The upgraded processing plant which includes current and new equipment is designed to process 900 tph based on 92% operating time for an annual throughput of 7.2 Mtpa of hard ore. The design of the processing plant is based on the transition of processing soft ore to hard ore over the life of the mine. During the initial years of the hard sulphide operation, the existing plant will be kept in operation to process remaining oxide ore. When the mine production will reach its full capacity of hard sulphide, the two plant will be combined to process 7.2 Mtpa of hard rock.

A gyratory crusher (450 kW), SAG (2 x 7000 kW)/Ball mill (7000 kW) and CIL circuits will be added to the actual milling capacity as existing ball mills (3 x 2010 kW) to allow to reach a maximum capacity of 7.2 Mtpa (see Figure 1.4 and Figure 1.5).

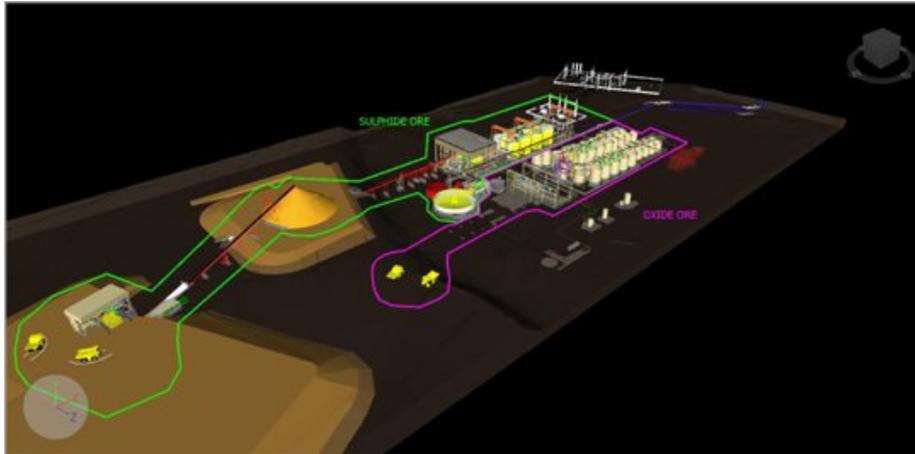


Figure 1.5: 900 tph Hard Rock Plant Simplified Flowsheet (Lixiviation)

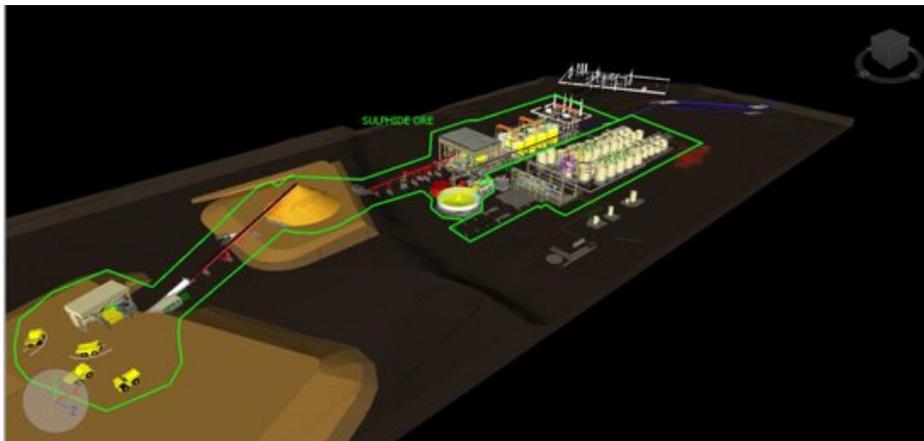


As mentioned, a new hard rock processing facility will be constructed adjacent to the old plant (Figure 1.6) and will operate in parallel until the mine has reached the full production of sulphide ore. At this time, the existing plant will be modified and both plants will be connected to process 7.2 Mtpa of sulphide ore.

**Figure 1.6: Plant General Arrangement (Both Plants in Parallel)**



**Figure 1.7: Plant General Arrangement (Combined Plants at 7.2 Mtpa of Sulphide Ore)**



## 1.13 Project Infrastructure

### 1.13.1 Infrastructure and Support Facilities during Construction

Most of existing infrastructure present at Sadiola will be used as part of the expansion project. Some, additional infrastructures are required for the expansion:

- Construction camp (area 1905): The construction camp includes a service building, kitchen, gatehouse /camp office and camp units within a compound. The camp includes a total of 140 rooms each with single twin bed, desk, TV, private bathroom;
- Construction offices (area 1905). The construction office (27 m x 29 m) consists of a collection of temporary containers. A VSAT system will be installed for the construction period to permit use of Internet services and management systems;
- Mine truck shop and warehouse (area 1530): The building is divided up into five main areas: offices, change rooms, workshop, warehouse and truck shop. The location of the mine truck shop was modified from the previous study. The truck shop design includes the following:
  - The truck shop has five bays (4+1) for heavy duty repairs and preventive maintenance. One bay is equipped with rails for maintaining large shovels;
  - The truck shop bays are serviced by three (3) lubrication stations. The entire shop is serviced by a 25 t overhead crane;
- The office section includes 16 closed offices, an open area with eight workstations and a map table, a larger room for dispatchers as well as one for IT, a conference room for eight people and a copy room;
- The workshop area links the truck shop area and the warehouse. The 234 m<sup>2</sup> area will accommodate floor presses, lathes, drills and other equipment as well as the tool crib;
- The warehouse with an area of 488 m<sup>2</sup> will house spare parts to support the mobile equipment;
- Wash bay;
- Fuel and lube storage (area 1540).

### 1.13.2 Power Line

Electrical power is currently provided through 18 x 1 megawatt (“MW”) medium-speed diesel generators which are capable of meeting an average demand of 16.7 MW. Approximately 4.73 M liters of diesel fuel per month are required for power generation and mining, under a contract with Total/ELF Petroleum Company. The 7 Mt national strategic fuel depot, situated in Kayes, is used as back-up storage in case of major road and/or rail disruptions.

The project includes building a 225 kV transmission line between the existing Kayes substation and the Sadiola Mine as well as three (3) substations connected to this line. The proposed line route is parallel to the existing Manantali-Kayes’ line for the first 38 km up to Diamou. The second portion of the line route, 51 km long, is in the same axis than the Diamou-Sadiola road. The total length of the proposed line is approximately 89 km.

A new substation will be built in Diamou, allowing EDM (Énergie du Mali) to feed at medium voltage the local communities. On the Sadiola site, two new substations are planned: one for Sadiola, to be operated by EDM, and one main substation for SEMOS mining facilities.

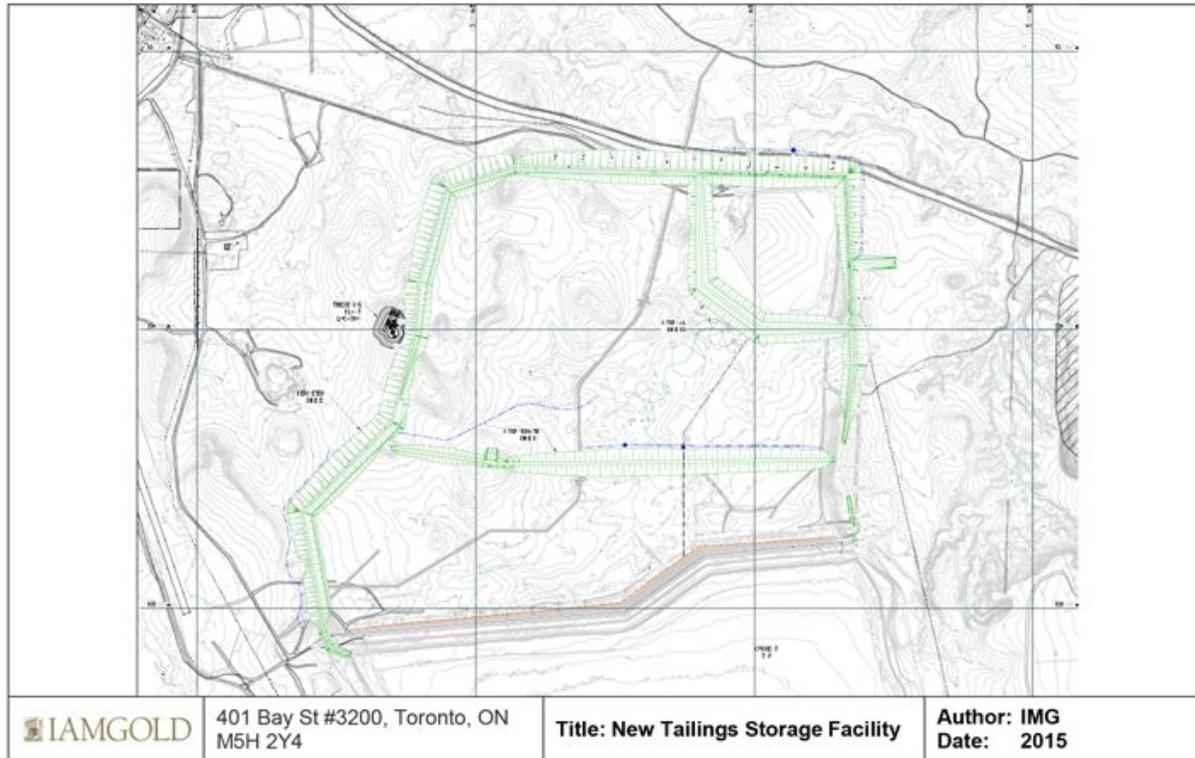
Beside reducing and securing the operating costs with a fixed kWh price, this will also reduce the carbon footprint of the mine by shutting down the existing generators. In addition to improving the project economics, this will also help the communities around Diamou and Sadiola access medium voltage electricity at lower cost and with a better reliability.

### **1.13.3 Tailings Management**

The current TSF will continue to receive “Oxides/soft Sulphides” tailings until the commissioning of the upgraded processing plant. An expansion of the Tailings Storage Facility (TSF) has been designed to accommodate the additional 75 M t of tailings planned in the Sadiola Sulphides project. The tailings from the “Hard Sulphides” zone milling process will be deposited over a period of approximately 10 years at an average production rate of 7.2 Mtpa.

The tailings management strategy put forward considers deposition of the “Hard Sulphides” stream in a New TSF area fully lined with a geo-synthetic membrane. Tailings are to be deposited from the dykes as slurry at 50% solids at the start of the New TSF operation. Towards the end of the operation, tailings will be thickened to a targeted 68% solids content and deposition will take place from service roads built directly on the tailings surface inside the New TSF area. Changing from slurry to thickened tailings will increase the storage capacity of the New TSF and shape its deposition surface to promote surface runoff and prevent water accumulation in the TSF. A phasing approach was integrated in the design allowing the project to defer some work in time.

Figure 1.8: New Tailings Storage Facility



#### 1.13.4 Water Management

A numerical water balance model was developed using the GoldSim software (GoldSim Technology Group, 2015) in order to:

- estimate the water storage capacity of the New TSF that will be required to limit the risk of spill to the environment over 10.4 years of operation to 1%, and
- estimate the volumes of water available for mine processes.

Provided local rainfall data were utilized to generate a 1,000-year long stochastic rainfall time series, which is representative for the site. This time series formed the basis for the probabilistic approach of the analysis. It is important to note that the short length of the historical records for daily rainfall leads to uncertainty in the estimation of extreme events.

The water balance model incorporated the relevant hydrologic components: bleed water released by consolidating tailings, the rainfall runoff production on the tailings surface, direct precipitation and evaporation at the water basin, and pumping from the TSF towards the process water reservoir.

Other than during the reclamation works for the current TSF (Year 3 and Year 4), no inflow from current TSF towards the reservoir of the New TSF is considered in the water balance model. It's assumed that the emergency spillway of Current TSF directs all overflow towards the environment.

#### 1.14 **Project Execution Plan**

The Sadiola Sulphide Project execution will be directly managed by IAMGOLD project management team which will include SEMOS people on site. The engineering, procurement and construction works will be contracted out to qualified firms. The construction work will be mainly contracted out to local (regional) contractors under the supervision of the Project team. Project control functions such as scheduling, cost control, project logistics and site supervision will be executed directly by the IAMGOLD project team.

An Owners' Steering Committee will be formed to oversee the project. Major project milestones are presented in Table 1.6.

**Table 1.6: Major Project Milestones**

Description	Start Date	Completion Date
Project approval	-	Q1 - Year 1
Detailed engineering	Q1 - Year 1	Q4 - Year 1
Permitting	-	Q2- Year 1
Truck shop	Q3 - Year 1	Q2 - Year 2
Plant construction 7.2 Mtpa	Q3 - Year 1	Q4 - Year 2
225 kV transmission line	Q3 - Year 1	Q4 - Year 2
Pit pre-stripping	Q2 -Year 2	Q4 - Year 2
TSF construction	Q3 - Year 1	Q4 - Year 2
Commissioning and Ramp Up		Q4 - Year 2
Plant commercial production	-	Q1 - Year 3

#### 1.15 **Environmental and Permitting**

An environmental and social impact assessment (ESIA) was completed for the Sulphide Project in 2010 and a separate ESIA for the powerline in 2012. The Government of Mali has issued several environmental and construction permits based on those ESIA studies. These permits have now expired.

Discussions are required with the Malian authorities to determine whether the new version of the SSP project is to be submitted to a new ESIA process or whether 2010 ESIA can simply be updated to account for the new version of the SSP project (as an addendum to the 2010 ESIA). Until such discussions are held, it is not possible to have a formal permitting plan.

Some important changes were made in the current version of the SSP project:

- Change in the location of the waste rock dumps from a North-East location to North-West and South location along with in-pit dumping;
- Decision to postpone the TTP and to install an impermeable liner underneath the new tailing storage facility (TSF).

The potential impacts of these changes on the community and the environment have not been accounted for in the 2010 ESIA.

Pending on discussions with the Malian authorities, the most likely permitting process will involve producing an addendum to the 2010 ESIA to support new environmental permit requests or permit renewals. This addendum would include the updated baseline studies, an updated impact assessment conducted on the changes made on the SSP project, a community consultation report as well as the new and updated mitigation measures. A formal environmental and social management plan would also have to be provided to the authorities along with the permit requests. For the project proponent, conducting these activities would lead to better risk assessment and risk management measures for the construction, the operation and closure phases and would help prevent potential issues or conflicts with the host community.

Unless stated otherwise by the Malian authorities, it is most likely that these activities need to be conducted prior to submitting new environmental permit requests or permit renewals to the authorities.

The timeline required to conduct these activities will be important for construction.

## **1.16 Capital and Operating Costs**

### **1.16.1 Basis of Estimate**

The accuracy level targeted by the 43-101 Technical Report for the capital and operating cost estimate is  $\pm 15\%$ . All costs have been stated in US dollars with foreign currency quotations and estimates converted using the following long term exchange rates.

**Table 1.7: Initial Capital Exchange Rate Assumptions**

Currency	Abbreviation	Exchange Rate Year 1	Exchange Rate Year 2
U.S. Dollar	USD	1	1
CAD Dollar	CAD	0.8	0.8
Great Britain Pound	GBP	1.55	1.55
CFA Franc	XOF	0.001681	0.001754
Euro	EUR	1.1	1.15
South African Rand	ZAR	0.076923	0.076923

Table 1.8 shows the gold prices assumptions for the project:

**Table 1.8: Gold Price Assumptions**

Years	Gold Price (USD)
Year 1	1,150
Year 2	1,225
Years 3-4	1,250
Years 5 +	1,275

The delivered fuel price assumption affecting mining costs is summarized in Table 1.9:

**Table 1.9 Fuel Price Assumption**

Years	Fuel Price (USD)
Year 1	0.78
Year 2	0.83
Years 3-4	0.86
Years 5+	0.89

### 1.16.2 **Capital Cost Summary**

The 7.2 Mtpa scenario capital expenditures (CAPEX) are estimated at \$379 M including a contingency of \$24.5 M. The construction capital cost summary is presented in Table 1.10 and excludes past expenditures \$141.8 M.

**Table 1.10: Initial Capital Construction Cost Summary**

Capital Expenditures	USD	
	Fixed Exchange	Variable Exchange
03 – Mining	78,136,000	78,136,000
04 - Transmission Line	37,169,204	37,169,244
05 - Other Infrastructure	12,313,640	12,336,871
06 – Plant	70,943,760	71,619,565
08 - Tailings Facilities	33,175,034	33,570,973
09 - Construction Management	89,189,971	89,203,151
Fuel	1,511,980	1,560,441
Existing Commitments on purchased long-lead items	14,161,000	14,161,000
10 - Owner Costs	2,360,233	2,363,235
998 - Contingency	24,515,184	24,515,184
999 - Management Fees	13,974,681	13,974,681
<b>Grand Total</b>	<b>377,450,687</b>	<b>378,610,345</b>

The investment program is scheduled over a 24 month period. The working capital required for the expansion with an owner mining strategy and considering a production increase at the end of Year 2 is \$32 M of additional inventory.

Sustaining capital for the incremental 7.2 Mtpa project is estimated at \$257 M and is mainly for initial waste development, equipment replacement, major repairs and rebuilds, TSF stages and plant stay-in business capital.

The incremental closure cost from the Sulphide project amounts to \$20.4 M.

### **1.16.3 Operating Cost Summary**

Operating costs presented in this Section use the fuel and exchange rate from corporate assumptions. Mining costs have been estimated at an average of \$2.99/t based on owner mining costs for the Sulphide pit.

Processing costs have been estimated by rock type based on specific reagent, grinding media and power consumption for each. In addition to this fixed costs for maintenance and labor are added. The power cost is based on a grid power cost of 70 XOF/kWh. The average processing cost estimated over the life of mine is \$15.28/milled.

General and administrative services are estimated at \$31.8 M average (excl. refining cost). Refining cost is estimated to \$4.88/oz produced. The G&A costs in the financial model include all administrative, support services at site and refining costs.

The operating costs are estimated to \$35.29/t milled or \$735/oz produced as presented in Table 1.11.

**Table 1.11: Operating Cost Summary**

<b>Category</b>	<b>Total Costs (M\$)</b>	<b>Avg. Cost (\$/t milled)</b>	<b>Avg. Cost (\$/oz)</b>
Mining	754	11.47	239
Processing	1,004	15.28	318
G&A	280	4.26	89
<b>Direct Cost</b>	<b>2,037</b>	<b>31.01</b>	<b>646</b>
Royalties	241	3.66	76
Management Fee	40	0.61	13
<b>Total Cost</b>	<b>2,318</b>	<b>35.29</b>	<b>735</b>

Total manpower is estimated at its peak to 1,431 employees with 532 in mining, 463 in processing and 436 in general administration.

## 1.17 Economic Analysis

### 1.17.1 Financial Summary

The financial model assumes a long term real price of gold from Year 1 of \$1,150/oz, in Year 2 \$1,225/oz, \$1,250/oz in Years 3-4 and staying at \$1,275/oz from Year 5 onwards to the end of the mine life.

The financial analysis of the Sulphide project is a differential result between the current mine plan and the combined current mine plan and expansion project. All calculation are after tax and calculated in a constant dollar. According to an agreement protocol with the Government of Mali, the Sulphide Expansion project is considered as a new project. It would be treated under the 1990 mining code as per the original Sadiola Project giving it a 5 year corporate tax holiday after production start, exempting the construction period plus 3 years of operation of the VAT and import duties and exempting withholding taxes for SEMOS and their sub-contractors for the construction period plus 5 years after production start. The corporate tax rate after the tax holiday period is 30%.

Table 1.12: Summary of Financial Highlights

<b>Sadiola</b>	<b>7.2 Mtpa Incremental</b>
Tonnes (M t)	312
Strip Ratio	3.9
Milled Tonnes (M t)	65.7
Max. Throughput (M t)	7.2
Recoverable Gold (Moz)	3.2
Average Recovery (%)	76.5%
Mine Life (yr)	10
<b>Gross revenue (\$M)</b>	<b>\$4,012</b>
Direct Cash Cost (\$/oz)	\$646
Cash Cost (\$/oz)	\$735
AISC (\$/oz)	\$816
<b>Initial Capital (\$M)</b>	<b>\$379</b>
Resale of Equipment (\$M)	%57*
Sustaining Capital (\$M)	\$257
<b>Total Capital</b>	<b>\$693*</b>
Closure Costs (\$M)	\$20
<b>SADIOLA (100%)</b>	
After tax NPV 0	\$740
After tax NPV 6%	\$325
After tax NPV 8%	\$234
After tax NPV 10%	\$158
<b>After tax IRR</b>	<b>16.0%</b>
Payback Period (yr)	5

\*Assuming equipment already acquired would be disposed in the base case.

### 1.17.2 **Project Sensitivities**

Sensitivities to certain key parameters were undertaken in the financial model to appreciate variations of the 7.2 Mtpa incremental results. The key parameters include gold price/grade, initial capital cost, XOF/USD exchange rate, operating costs and fuel price. Base case assumptions are varied from -10% to 10% with a 5% increment. Results are summarized in tables below.

**Table 1.13: Gold Price Sensitivity**

Gold Price (US\$/oz)/ Grade	-10%	-5%	BASE	5%	10%
Sadiola (100%):					
- After tax CF	393	564	740	913	1,083
- After tax 6% DCF	96	209	325	440	552
- After tax 8% DCF	32	132	234	334	433
- After tax 10% DCF	-20	68	158	247	335

**Table 1.14: Initial Capital Cost Sensitivity**

CAPEX (M\$)	-10%	-5%	BASE	5%	10%
Sadiola (100%):					
- After tax CF	812	777	740	703	666
- After tax 6% DCF	389	358	325	293	261
- After tax 8% DCF	295	265	234	203	172
- After tax 10% DCF	217	188	158	128	98

**Table 1.15: Operating Cost Sensitivity**

OPEX (M\$)	-10%	-5%	BASE	5%	10%
Sadiola (100%):					
- After tax CF	986	865	740	615	486
- After tax 6% DCF	499	414	325	238	147
- After tax 8% DCF	390	313	234	155	73
- After tax 10% DCF	299	230	158	87	13

**Table 1.16: Exchange Rate**

XOF/USD (\$M)	-10%	-5%	BASE	5%	10%
Sadiola (100%):					
- After tax CF	638	691	740	783	822
- After tax 6% DCF	254	292	325	356	383
- After tax 8% DCF	170	203	234	261	286
- After tax 10% DCF	101	131	158	183	205

**Table 1.17: Fuel Price**

Fuel Price (\$M)	-10%	-5%	BASE	5%	10%
Sadiola (100%):					
- After tax CF	767	753	740	726	713
- After tax 6% DCF	345	335	325	316	306
- After tax 8% DCF	251	242	234	225	217
- After tax 10% DCF	174	166	158	150	143

**1.18**      **Conclusions**

Sadiola gold mine is currently processing soft ore coming from satellite pits and existing stockpiles. The soft ore mineral reserve is now depleting and modifications to the current processing plant and related infrastructures are needed to pursue the actual operation. The Sadiola Sulphide Project - Feasibility study of the exploitation of an important hard sulphide ore deposit, presented in this technical report, has demonstrated attractive financial results. This recent study came with interesting CAPEX and OPEX reductions in comparison with previous work performed on this project.

The main risks associated with the project are related to permitting process and security situation in Mali. Environmental permits are currently expired and some changes in the project engineering will require work to update the current baseline.

The financial results of the 7.2 Mtpa scenario indicates that the Sadiola Sulphide Project is financially attractive to all the stakeholders and joint-venture partners and this project could assure the continuity of the Sadiola operation.

**1.19**      **Recommendations and Future Work Program**

Before final project approval, it is recommended to initiate the following work to mitigate projects risk and capture some opportunities:

- Expand sulphide exploration to satellite pits to maximise the processing rate of hard sulphide during Years 3 and 4 of the SSP project;
- Initiate discussion with GoM and begin environmental work to update the ESIA and to proceed with the renewal of permits;
- Initiate discussion with GoM regarding the opportunity to modify the routing of the power line and reduce the length;
- Initiate the engineering for the construction camp and power line; and
- Refine project execution plan and initiate equipment inspection for equipment that are already purchased and presently in storage.

## 2. **INTRODUCTION**

IAMGOLD Corporation (IAMGOLD) has prepared a Technical Report for its jointly owned Sadiola Gold Mine, located in Mali with the support of G Mining Services Inc. (G Mining). The purpose of this Technical Report is to support the disclosure of the December 31, 2015 Sadiola Gold Mine Mineral Resources and Mineral Reserves estimate. Therefore, all the report was made on a 100% ownership basis.

The Sadiola Gold mine is located in western Mali some 77 km south of the regional capital of Kayes. The Sadiola gold deposit is mined by the *Société d'Exploitation des Mines d'Or de Sadiola S.A.* (SEMOS), the operating company formed through a joint venture agreement between AngloGold Ashanti (41%), IAMGOLD (41%) and the Malian Government (18%). AngloGold Ashanti, through its wholly owned subsidiary AngloGold Mali S.A., is the mine operator. The Malian Government shareholding is a free-carried interest.

SEMOS is bound by the original prospecting and exploitation agreement (including its subsequent legal modifications) entered into on April 15, 1990 between AGEM and the Mali Government, and the mining license is valid for the original mineral commodities until April 15, 2020. The identity number of the current exploitation area is " *DECRET No 00-080/PM-RM DU 06 MARS 2000* " and is a modification of all previous exploitation areas. The surface area defined by " *DECRET No 00-063/PM-RM DU 25 FEV 2000* ".

### 2.1 **Source of Information**

This Technical Report was prepared by IAMGOLD, G Mining Services Inc. (GMSI) personnel and Snowden. The dates of personal inspections of the Sadiola Gold Mine by the Qualified Persons (QPs) are provided in Section 29 of this Technical Report.

The QPs and their responsibilities for this Technical Report are listed in Section 29 Certificate of Qualified Person.

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 27 References.

**2.2 Authors and Participants**

<b>Section</b>	<b>Sub Section</b>	<b>Qualified Person</b>	<b>Company</b>
<b>Section 1 (Executive Summary)</b>	Section 1.1 to 1.2.3, 1.3, 1.14, 1.16.1, 1.16.2, 1.16.3, 1.18, 1.19	Luc-Bernard Denoncourt	IAMGOLD
	Section 1.2.4, 1.12, 1.13.1, 1.13.2, 1.14, 1.15	Philippe Gaultier	IAMGOLD
	Section 1.4 to 1.6, 1.8, 1.9	Mark Burnett	SNOWDEN
	Section 1.10, 1.11, 1.15.3	Louis-Pierre Gignac	GMSI
	Section 1.7, 1.12, 1.13.3, 1.13.4,	Jerome Girard	IAMGOLD
	Section 1.17	Daniel Vallieres	IAMGOLD
<b>Section 2 (Introduction)</b>	Section 2	Luc-Bernard Denoncourt	IAMGOLD
<b>Section 3 (Reliance on Other Experts)</b>	Section 3	Luc-Bernard Denoncourt	IAMGOLD
<b>Section 4 (Property Description and Location)</b>	Section 4	Luc-Bernard Denoncourt	IAMGOLD
<b>Section 5 (Accessibility, climate, local resources, Infras and Physiography)</b>	Section 5.2 to 5.5	Luc-Bernard Denoncourt	IAMGOLD
	Section 5 (prior to 5.1), 5.1, 5.6.1, 5.6.2	Philippe Gaultier	IAMGOLD
	Section 5.6.3	Jérôme Girard	IAMGOLD
	Section 5.6.4, 5.6.5	Louis-Pierre Gignac	GMSI
<b>Section 6 (History)</b>	Section 6	Luc-Bernard Denoncourt	IAMGOLD
<b>Sections 7 to 12 (Geology)</b>	Sections 7 to 12	Mark Burnett	SNOWDEN
<b>Section 13 (Metallurgical Testing)</b>	Sections 13	Jérôme Girard	IAMGOLD
<b>Section 14 (Mineral Resource Estimate)</b>	Section 14	Mark Burnett	SNOWDEN
<b>Sections 15-16 (Mineral Reserve and mining methods)</b>	Sections 15-16	Louis-Pierre Gignac	GMSI
<b>Section 17 (Recovery Methods)</b>	Section 17	Jérôme Girard	IAMGOLD
<b>Section 18 (Infrastructure)</b>	Section 18 (except for 18.8)	Philippe Gaultier	IAMGOLD
	Section 18.8	Jérôme Girard	IAMGOLD
<b>Section 19 (Market Study)</b>	Section 19	Daniel Vallieres	IAMGOLD
<b>Section 20 (Environment)</b>	Section 20	Philippe Gaultier	IAMGOLD
<b>Section 21 (Capital Cost &amp; Operating Cost)</b>	Section 21.1 to 21.5	Luc-Bernard Denoncourt	IAMGOLD
	Section 21.6 (except for 21.6.4), 21.7	Louis-Pierre Gignac	GMSI
	Section 21.6.4	Jérôme Girard	IAMGOLD
<b>Section 22 (Economic Analysis)</b>	Section 22	Daniel Vallieres	IAMGOLD
<b>Section 23 (Adjacent Property)</b>	Section 23	Luc-Bernard Denoncourt	IAMGOLD
<b>Section 24 (Other relevant Data &amp; Info)</b>	Section 24	Luc-Bernard Denoncourt	IAMGOLD
<b>Section 25 Interpretation and Conclusion</b>	Section 25.1	Mark Burnett	SNOWDEN
	Section 25.2	Luc-Bernard Denoncourt	IAMGOLD
<b>Section 26 Conclusion</b>	Section 26	Luc-Bernard Denoncourt	IAMGOLD
<b>Sections 27 &amp; 28</b>	Sections 27 & 28	Luc-Bernard Denoncourt	IAMGOLD

**2.3 List of Abbreviation and Acronyms**

The units of measure presented in this report, unless noted otherwise, are in metric. The currency used for all costs is presented in US Dollars, unless specified otherwise.

A list of the main abbreviations and terms used throughout this Report is presented Table 2.1 .

**Table 2.1: List of Abbreviations**

<b>Abbreviations</b>	<b>Full Description</b>
As	Arsenic
Au	Gold
C	Carbon
CAD	Canadian Dollar
CIL	Carbon-in-leach
CoG	Cut-off Grade
Cu	Copper
DD	Diamond Drilling
DGPS	Differential Global Positioning System
ETP	Effluent Treatment Plant
F	Degrees Fahrenheit
FA	Fire Assay
Fe	Iron
FS	Feasibility Study
G	Giga – (000,000,000's)
g	Gram
G&A	General & Administration
g/L	Gram per litre
g Au/t	Grams of gold per tonne
gpm	Gallons per minute (US)
gpt or g/t	Grams per tonne
GPS	Global Positioning System
h	Hour
ha	Hectares
h/d	Hours per day
h/y	Hours per year
h/wk	Hours per week
HDPE	High-Density Polyethylene
hp	Horsepower
Hz	Hertz
IMG	IAMGOLD Corporation
IRR	Internal Rate of Return
ISO	International Organization for Standardization

Abbreviations	Full Description
k	Kilo – (000's)
kg	Kilograms
kg/t	Kilograms per tonne
km	Kilometer
km/h	Kilometer per hour
kPa	Kilopascal
kV	Kilovolts
kW	Kilowatts
kWh	Kilowatts per hour
l	Liter
lpm	Liter per minute
M	Mega or Millions (000,000's)
m	Metre
m <sup>2</sup>	Square meter
m <sup>3</sup>	Cubic meter
masl	Metres above sea level
mg	Milligram
mg/L	Milligram per liter
min	Minute
ml	Milliliter
mm	Millimeter
m/min	Metre per minute
Mo	Month
m/s	Metre per second
Mt	Million tonnes
Mtpd	Million Metric tonne per day
Mtpa	Million Metric tonnes per annum
MVA	Megavolt-ampere
MW	Megawatt
NPI	Net Profit Interest
NPV	Net Present Value
NQ	Drill Core Diameter (47.6 mm)
∅	Diameter
OCR	Off-Channel Reservoir
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
oz	Troy Ounce (31.10348 grams)
Pb	Lead
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study

Abbreviations	Full Description
PLC	Programmable Logic Controller
ppb	Parts per Billion
ppm	Parts per Million
psi	Pounds per square inch
PV	Present Value
RC	Reverse Circulation
RoM	Run-of-mine
rpm	Revolutions per minute
RWD	Return Water Dam
S	Sulphur
Sec	Second (time)
STP	Sewage Treatment Plant
t	Tonnes (1,000 kg) (metric ton)
t/d or tpd	Tonnes per day
t/h or tph	Tonnes per hour
t/m <sup>3</sup>	Tonnes per cubic metre
TRS	Tailings Reclaim Sump
TSF	Tailings Storage Facility
TTP	Thickened Tailings Plant
TWSP	Treated Water Storage Pond
t/y or tpa	Tonnes per year
USD	United States Dollar
V	Volt
VAT	Value Added Tax
wk	Week
XRF	X-ray Fluorescence
y	Year

Table 2.2: List of Acronyms

Acronyms	Definition
IMG	IAMGOLD
AFE	Active Front End
AGA	Anglo Gold Ashanti
ARU	Active Rectifier Units
CIL	Carbon-In-Leach
CIP	Carbon-In-Pulp
CRM	Certified Reference Material
DNACPN	Direction Nationale de L'Assainissement et du Contrôle des Pollutions et des Nuisances
EDM	Énergie du Mali
EDR	Enhance Data Rate
GMSI	G Mining Services Inc.
GoM	Government of Mali
GPM	Gallon Per Minute
f <sub>c</sub>	Compressive Strength
HDPE	High-Density Polyethylene
HMI	Human Machine Interface
HVAC	Heating, Ventilating, and Air Conditioning
Hz	Hertz
I/O	Input/Output connection
IEC	International Electrotechnical Commission
IFC	International finance corporation
ILR	Intensive Leach Reactor
mA	Milliampere
MCC	Motor Control Centers
MTO	Material Take-Off
Mtpa	Million Tonnes per year
NI 43-101	National Instruments 43-101- Canadian Standards of Disclosure for Mineral Projects
P&ID	Process and Instrumentation Diagram
PE	Protective Earthing
PFD	Process Flow Diagram
PLC	Programmable Logic Controller
PSI	Pounds Per Square Inch
RCD	Residual Current Devices
ROM	Run-of-Mine
RMS	Root Mean Square
RTD	Resistance Temperature Detector
SAG	Semi-Autogenous Grinding

<b>Acronyms</b>	<b>Definition</b>
SEMOS	Société d'Exploitation des Mines d'Or de Sadiola S.A.
SF6	Sulfur Hexafluoride
SRM	Standard Reference Material
SS	Stainless Steel
STP	Shielded Twisted Pairs
SSP	Sadiola Sulphide Project
THD	Total Harmonic Distortion
TSF	Tailings Storage Facility
TTOG	"Tuyau de tôle ondulée galvanisée"
TTP	Tailings Treatment Plant
USG	United States Gallon
UPS	Uninterruptible Power Supply
UPVC	Unplasticised Polyvinyl Chloride
VFD	Variable Frequency Drives
VSA	Vacuum Swing Adsorption
XLPE	Cross-Linked Polyethylene Insulation

### **3. RELIANCE ON OTHER EXPERTS**

#### **3.1 Report Responsibility and Qualified Persons**

This report has been prepared by IAMGOLD, G Mining and Snowden. The information is deemed to be valid and complete.

## 4. PROPERTY DESCRIPTION AND LOCATION

### 4.1 Location

The Sadiola Mine is an opencast gold mining operation located in the Kayes district in the western part of the Republic of Mali, in West Africa. The mine is situated 77 km south of the regional capital of Kayes and about 440 km northwest of the capital city of Bamako. The nearest international border is with Senegal to the west and Mauritania to the north, as shown in Figure 4.1.

The geographical coordinates of the mine are latitude 13°52' 40" N and longitude 011°38'35" W.

Figure 4.1: Location of the Sadiola Gold Mine



### 4.2 Operating Area

Current Mining activities take place in five (5) open pits and on-site processing infrastructure includes a 4.9 Mtpa CIL gold plant where the ore is eluted and smelted.

Short-term exploration has also been undertaken to determine the residual oxide and sulphide potential for the remaining duration of the Mining Permit (until July 2024). Assay results have demonstrated the

continuity and width of the mineralization which has potential for development of an underground resource.

The Sadiola Mine currently consists of integrated activities and supporting services:

- Mining from two open pits at Tambali and FN3 producing Run of Mine (“ROM”) from the soft saprolitic oxide ore body throughout an area of 51 ha;
- A 4.9 Mtpa Carbon in Leach (“CIL”) gold plant where the ore is eluted and smelted, covering an area of 22.0 ha;
- A TSF of 376.1 ha;
- A series of waste rock dumps (“WRDs”) on the periphery of the open pits with a total area of 804.8 ha;
- General infrastructural areas including offices, workshops, utility and waste water treatment areas and accommodation facilities dispersed over the Sadiola Mine Permit area, and covering approximately 145.1 ha;
- Access and haul roads extending over an area of 267.9 ha;
- An airstrip of 29.9 ha (1,800 m long).

These activities are supported by basic infrastructure and support services. On-site generators provide independent power to sustain operations and water is pumped a distance of 55 km to Sadiola.

Access to the Sadiola Mine is via a single carriageway, laterite road. Secondary roads around the mine are easily passable through most of the year, but become inaccessible during the wet season. The small airstrip on the property is functional for light aircraft.

#### **4.3 Location of Property Boundaries**

The location of the property boundaries are set out in Table 4.1. A plan of these coordinates is shown in Figure 4.2

Table 4.1: SEMOS Concession Coordinates in Latitude and Longitude

Position	West			North		
	Degree	Minute	Second	Degree	Minute	Second
A	11	44	0	13	54	55
B	11	42	18	13	54	55
C	11	42	18	13	57	33
D	11	36	0	13	57	33
E	11	36	0	13	55	0
F	11	35	0	13	55	0
G	11	35	0	13	54	0
H	11	32	40	13	54	0
I	11	32	40	13	52	9
J	11	33	40	13	52	9
K	11	33	40	13	49	12
L	11	35	4	13	49	12
M	11	35	4	13	50	11
N	11	35	44	13	50	11
O	11	35	44	13	49	43
P	11	36	51	13	49	43
Q	11	36	51	13	46	27
R	11	33	39	13	46	27
S	11	33	39	13	45	0
T	11	38	48	13	45	0
U	11	38	48	13	45	51
V	11	40	0	13	45	51
W	11	40	0	13	49	27
X	11	44	0	13	49	27

Source: IAMGOLD, 2010

#### 4.4 Type of Mineral Tenure

In Mali, mineral resources are the property of the State. Malian Mineral Rights are governed by the Mining Act dated February 27, 2012. The Mining Act is complemented by the Mining Decree dated June 21, 2012. The Ministry of Mines and the National Directorate of Geology and Mines ( *Direction Nationale de la Géologie et des Mines* (DNGM)) is the government authority responsible for regulation of the mining industry in Mali.

#### **4.5 Exploration and Exploitation Permit**

Permits for exploration and exploitation in Mali are issued for a designated area and a specific group of minerals, among the five (5) groups, as defined by the Mining Code. Gold is named under Group 2. Previously, the Mining Code distinguished between “precious stones” and “every other mineral”.

A “license to operate”, or the operating permit, entitles the permit holder, within the limits of its scope and depth, the exclusive and indefinite right to prospect, undertake exploration and exploitation of mineral substance(s) found within the perimeter which is the subject of the permit.

This operating permit can be attributed only to the holder of an existing permit, or a current prospecting license, for the mineral substances for which they are duly issued. The operating permit is granted by the Director of Mines if, the holder has fulfilled the obligations set out in the Mining Act, submitted a mining feasibility study, a community development plan (CDP), formulated in conjunction with the interested local communities as well as the local and regional authorities, and a mine closure plan. It is to be noted that the CDP must be updated every two years.

The permit also grants its holder the right to undertake processing operations under Article 21, within the borders of Mali, and to market the saleable products. The holders of an operating license are free to export mining products.

The holder of the operating permit is required to begin exploitation within three years of issuance of the permit. This permit is granted by decree for a period of 30 years, and can be extended in 10-year increments until depletion.

The permit holder must notify the Administration in charge of Mines of any significant changes in key parameters of the feasibility study on commencement of operations. If the changes affect the completion times and the viability of the proposed operation, the permit holder must submit a new feasibility study.

During the operational phase, the State requires the creation of a new company under Malian law, with the release of 10% of the shares accruing to the State. Based on its own discretionary capacity, the State can acquire an interest of up to an additional 10%, to be paid for in cash. In addition, the Mining Code has also introduced the option for domestic private investors to acquire, for cash, at least 5% of the shares of the mining company, under the same conditions as other private shareholders.

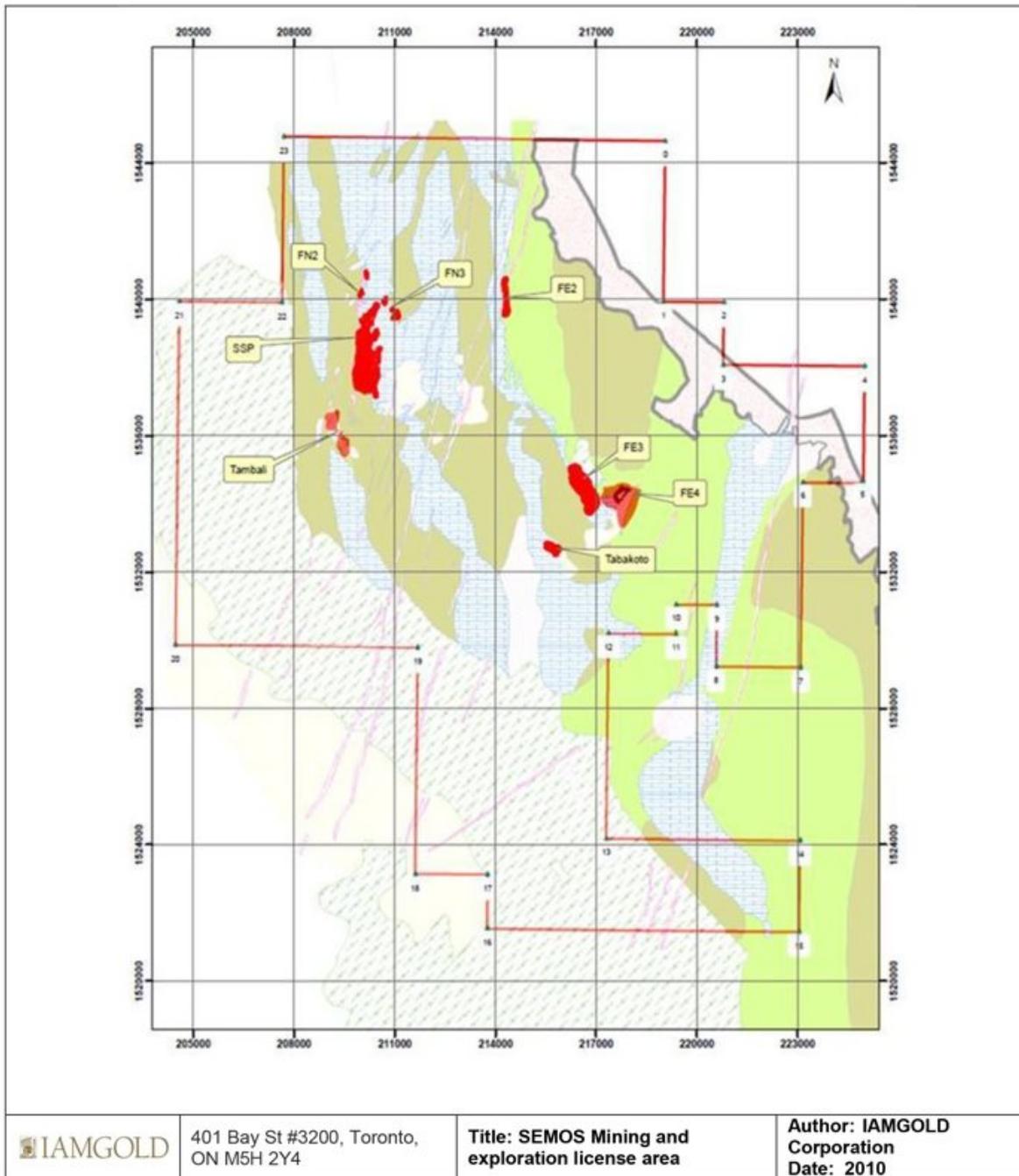
The operating permit expires when operations stop prematurely, and the permit holder renounces the permit, either totally or partially, by notifying the competent authorities. Cancellation or withdrawal of an operating permit is not final until it is accepted by the Minister of Mines. Once the renunciation is accepted, the permit becomes partly, or wholly, void.

The State has a right of first refusal regarding participation in the operating company, or the transfer of the operating permit.

#### **4.5.1**            **Mining Rights**

The Sadiola Mining Permit was granted on December 22, 1994, for an initial term of 30 years. The permit includes the sulphide ore to produce saleable gold, and extends over an area of 302 km<sup>2</sup>. Exploration activities are covered within this permit. The Mining Permit expires in July 2024. The permit is a modification of all previous exploitation areas. The permitting area attributed to SEMOS is illustrated in Figure 4.2. Full details are shown in Table 4.2.

Figure 4.2: SEMOS Mining and Exploration License Area



**Table 4.2: Legal Tenure and Permit Details**

Granted area (ha)	Title number	DOU (date of official granting by government)	Current status	Expiry date
30,200	<i>DÉCRET No 00-080/PM-RM DU 06 MARS 2000</i>	December 22, 1994	Active	July 31, 2024

Source: IAMGOLD, 2010

#### **4.6 Issuance of Exploration License**

The exploration license is issued by the Director of Mines, and granted on a “first come, first served” basis whether the application is made by a natural or a legal person, who has justified having the required technical capacity and financial resources.

Similarly to the operating permit, the exploration license is granted only for the area where the exploration is allowed and for the designated group of substances. The license is valid for three years, and renewable once, for two years (previously three years), based on the work performed. During the validity period of this license, no mining title can be granted on the perimeter thereof.

At the end of the period of validity of the license for exploration, and for a period not exceeding three years, the holder has a priority right to apply for an operating permit, or a prospecting license for the group of substances covered by this authorization.

Once mining rights in the form of an operating (or mining) permit is issued then the exploration permit becomes linked to the operating permit. Exploration is then valid for the period of the mining permit.

An exploration license is not assignable, transferable, or subject to lease. However, merging of contiguous exploration permits is allowable.

#### **4.7 Surface Rights and Servitudes**

The holder of an operating permit has the right, at its own expense, to use natural resources to achieve the objectives set out in the mining title, in accordance with laws and regulations.

Power lines, water distribution, communication channels and other facilities, associated infrastructure or works created by the holder of an operating permit, inside or outside the perimeter, or belonging to it, may be open to public use, or for use by neighbouring establishments.

#### **4.8 Water Rights**

The Sadiola operation is authorized to extract a maximum of 8.0 M m<sup>3</sup> annually from the Senegal River, subject to certain royalty payments, established in Resolution No. 250 of the permit/water use right from the Organisation de Mise en Valeur du Sénégal (OMVS), dated January 8, 1994.

Approximately 5.6 M m<sup>3</sup> of water per year is supplied to the mine via a 55 km pipeline from the Senegal River (IAMGOLD, 2004, as referenced in Digby Wells & Associates, 2011). This water services both the mine operations and the village settlement and is treated before distribution.

#### **4.9 Health and Safety**

Health and safety in mining are regulated by the Mining Act of 2012, in accordance with the Labor Code of Mali.

The mining operator must also provide housing for workers on site, comply with the terms and conditions of employment relating to the prevention and remedying of occupational accidents and occupational diseases, and comply with the general working conditions relating to professional associations and unions.

The mining operator must also contribute to the development or improvement of the health infrastructure and education, within a reasonable distance of each operation, for the needs of workers and their families.

#### **4.10 Environmental Approval and Permitting Process**

In accordance with the Mining Code of Mali, provisions relating to the environment, health and safety, occupational hygiene and employment, as well as customs, tax codes and foreign exchange regulations affect the mining industry.

The conduct of mining activities under the Mining Act must be accompanied by an EIA and a Social Instruction Manual. These documents are submitted by the mining companies and the operators of associated activities, together with the feasibility report which is required to obtain an operating permit.

The EIA includes the identification, description and assessment of the impact of the project on human beings, fauna and flora, soil, water, air, climate and landscape, including the interactions between these factors, cultural heritage and other property.

The holder of an operating permit must submit a first demand bank guarantee, for rehabilitation and security, from an internationally recognized bank, following the termination of works. The Mining Regulation states that the bank guarantee must be equivalent to 5% of the anticipated turnover, unless

such amount proves to be insufficient, in which case the mining company is required to cover any shortcomings.

#### **4.11 Environmental Obligations**

A number of environmental obligations are imposed on the holders of mining titles, including:

- Development and implementation of appropriate procedures to manage chemicals and to ensure transportation, warehousing, handling and secure means of disposal of such substances as well as fuels and lubricants;
- Building of on-site wastewater treatment facilities;
- Establishment of a program for waste reduction, sorting and recycling;
- Ensuring that containment walls surround areas used for storage of oil and lubricants;
- Implementation of a management plan for water and mud tanks on the site;
- Formulation and implementation of a site-specific program to monitor the quality of drained water, and water collected from the dumps, the tailings and the tailing sites well as surface water and groundwater that can be contaminated by mining activity;
- Establishment of regular procedures for inspection, monitoring, verification of the data recording and reporting pertaining to the tailings dams;
- Implementation of technical mechanisms to reduce greenhouse gas emissions.

#### **4.12 Mine Closure**

A mine closure plan must be submitted, along with the environmental and social impact assessment study, to the Regulatory Authority to obtain an operation permit. The closure plan must be updated every five years. The closure plan will also be resubmitted if changes to the mining plan justify changes to the closure plan, or if the mining administration believes it necessary for the mine permit holder to revise the closure and rehabilitation plan.

The mine closure and rehabilitation plan must indicate the methods that will be used to dismantle and/or recycle all mining installations including but not limited to mining installation, machinery and equipment specified in the application decree. It must also include plans for post-closure environmental monitoring and auditing.

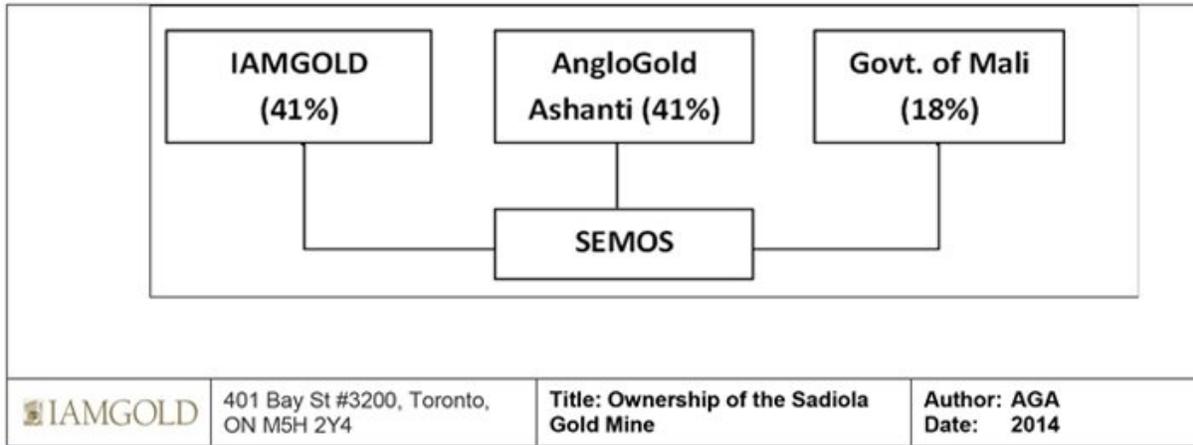
During the year the decision is made to close the mine, the mine operator, administration authorities and the local authorities must present a strategy for decommissioning of the mine and other practical uses/ recycling of mining equipment and installations for alternative socio-economic uses.

Civil liability is imposed on the holder of an operating permit in respect of damages or accidents caused by old equipment after mine closure and issue of an environmental discharge.

The Sadiola mine closure strategy will be similar to the one used for the Yatéla mine closure. The pits will not be backfilled. As the pit water is not expected to be safe to use, access to the pits will be prevented using safety berms. Where appropriate, the main access ramps will be dismantled. The waste rock dumps will be rehabilitated and revegetated using local plants. The tailings storage facility will be revegetated. Unless agreed otherwise with the Malian authorities, all the buildings and infrastructures will be dismantled and demolished. Before closure starts, a completed and detailed closure plan, including a community support aspect, will be submitted to the Malian authorities. The closure activities will begin once the plan is approved by the authorities.

#### 4.13 Issuer ' s Interest

Figure 4.3: Ownership of the Sadiola Gold Mine



#### 4.14 Location of Specific Items

Not applicable.

#### 4.15 Taxes, Royalties, Back-In Rights, Payments, Agreements, Encumbrances

Title holders must pay fixed fees for the grant, assignment, transfer and renewal of mining titles, as well as annual surface rights. The value of the fees is provided in the Mining Regulation, as adjusted by the Administration.

All general taxes and exemptions for holders of operating permits are applicable as identified in Mali legislation. Without limitation, they include gross revenue taxes, advalorem taxes, corporate taxes,

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customs and duties, labour and social security taxes, patent taxes, withholding taxes and value added taxes

#### **4.16**      **Permits**

A summary of details pertaining to the permits issued to the Sadiola operation are shown in Table 4.3 and Table 4.4.

Table 4.3: Geology and Mining Permits

Application date	Application area (km <sup>2</sup> )	Granted area (km <sup>2</sup> )	Title no.	Subject	Title	DOU (date of official granting by government)	Current status	Expiry date
-	-	-	-	Exploration and Mining Permit	Agreement of Establishment for the prospecting and exploration of gold, silver, related substances and platinoids between the Government of Mali and "AGEM" (Federal republic of Germany).	5 Apr 1990	Valid	5 Apr 2020
-	-	-	Arrêté N° 90-0079/MIHE-CAB	Exploration and Mining Permit	Arrêté N° 90-0079/MIHE-CAB du ministère chargé des mines octroyant un permis de recherche d'or et platinoides a la société AGEM.	13 Jan 1990	Valid	-
-	-	-	Décret N°90-191/P-RM	Exploration and Mining Permit	Décret N°90-191/P-RM portant approbation de la convention du 5 Avril 1990 par le Président de la Rép. du Mali.	May 12, 1990	Valid	-
-	-	-	-	Exploration and Mining Permit	Avenant N°1 à la Convention d'établissement du 5 Avril 1990.	24 Mar 1992	Valid	5 Apr 2020
-	-	-	Décret N°92-043/PM-RM	Exploration and Mining Permit	Décret N°92-043/PM-RM portant approbation de l'avenant N°1 à la convention d'établissement du 5 Avril 1990.	5 Feb 1992	Valid	-
11 Nov 1993	-	187	Décret N°94-257/PM-RM	Mining Permit	Décret N°94-257/PM-RM portant attribution à la société de recherche et d'exploitation aurifère "AGEM" d'un permis d'exploitation d'or, d'argent, de substances connexes et de platinoides.	1 Aug 1994	Valid	31 Jul 2024
7 Dec 1994	-	187	Décret N°94-440/PM-RM	Mining Permit	Décret N°94-440/PM-RM portant transfert au profit de la Société d'Exploitation des Mines d'Or de Sadiola "SEMOS SA". Du permis d'exploitation d'or, d'argent de substances connexes et platinoides attribué à la société de recherche et d'exploitation aurifère "AGEM".	22 Dec 1994	Valid	31 Jul 2024

Application date	Application area (km <sup>2</sup> )	Granted area (km <sup>2</sup> )	Title no.	Subject	Title	DOU (date of official granting by government)	Current status	Expiry date
14 Feb 1999	-	302	<i>DÉCRET No 00-080/PM-RM</i>	Mining Permit	<i>DÉCRET No 00-080/PM-RM DU 06 MARS 2000.</i>	6 Mar 2000	Valid	31 Jul 2024
-	-	-	-	Mining Permit	Shareholders Agreement BETWEEN AGEM/IAMGOLD, AMERCOSA, Government of Mali.	1994	Valid	-
19 Sep 2003	-	-	<i>Arrêté N° 04 0797/MMEE-MATCL</i>	-	<i>Arrêté N° 04 0797/MMEE-MATCL portant institution d'un périmètre de protection a SEMOS S.A.</i>	6 Apr 2004	Valid	-

Source: SEMOS, 2015

**Table 4.4: Environmental and Social Permitting and Authorizations**

Application date	Application area (km <sup>2</sup> )	Granted area (km <sup>2</sup> )	Title no.	Subject	Title	DOU (date of official granting by government)	Current status	Expiry date
			Resolution	Water Permit	Authorization granted to the Republic of Mali for pumping 8,000.000 m <sup>3</sup> per year from the Senegal River.	Jan 1994	Valid	
			Authorization	Determination of zones A and B	Authorization of the Ministry of Energy and Water and the Ministry of Territorial Administration about the Zone A and B.	Mar 1994		
			94-044	Law	Authorizing the Government of Mali to participate in SEMOS as shareholder.	8 Dec 1994	Valid	
			94-044	Law	Fixing the share of Malian Government to 20% from which 2% will go to IFC.	8 Dec 1994	Valid	
			<i>Arrêté Interministériel N° 95 MMEH-MATS</i>	Determination of zones A and B	Granting Protection Zones – Zone A and Zone B.	Sep 1995		
			<i>Arrêté ministériel N° 96-1280/MFAAC-SG</i>	Construction for Gendarmes	Authorization to construct a gendarmerie in Sadiola.	Aug 1996	Valid	

Application date	Application area (km <sup>2</sup> )	Granted area (km <sup>2</sup> )	Title no.	Subject	Title	DOU (date of official granting by government)	Current status	Expiry date
			V/L N°207	Letter of Approval From Ministry of Finance and Trade – Nation Direction of Customs	Approval of carrying out international flight from Sadiola.	Jul 1997	Valid	
			1181 MFC-DGD	Decision – Ministry of Finance and Trading	Organizing international flight.	Aug 1997	Valid	
			104	Decision – National Management of labor and safety	Setting up of a site clinic to treat employees and their family members.	1 Jul 1998	Valid	
			120	Resolution to appoint a new administrator	Proposal of Appointing P.J. Louw as an Administrator grade A to replace M.B.I. Tapson who has resigned.	1 Sep 2001		
				Authorization for the soft sulphide project approval	Approval of the Soft Sulphide Project by Ministry of Rural Development and Environment.	1 Jul 2002		
			03-0009	Authorization for North Pit extension	Approval of the North Pit Extension Project by the Ministry of Environment.	13 Oct 2003		
				Authorization	Approval of the proposed TORs of the West Waste Dump Extension Project National Sanitation and Pollution Control Department.	Apr 2004		
			N°04-0797/MMEE-MATCL	Determination of zones A and B	Authorization for the updated Zone A and B.	Apr 2004		
			N°04-006/MEA-SG	Authorization – dump extension	Approval of the West Waste Dump Extension Project by the Ministry of Environment.	19 Jul 2004		
			N°04-0007/MEA-SG	Authorization – extension of Mine Village	Approval of the Mine Village Expansion.	Jan 2005		

Application date	Application area (km <sup>2</sup> )	Granted area (km <sup>2</sup> )	Title no.	Subject	Title	DOU (date of official granting by government)	Current status	Expiry date
			N°05-0020/MEA-SG	Authorization – start FE3 and FE4	Approval to start with FE3 and FE4 Projects – Ministry of Environment.	19 Aug 2005		
			N°06-0035/MEA-SG	Authorization – dump extension	Approval to extend East Waste Dump.	21 Feb 2006		
			N°06-0055/MEA-SG	Authorization – power line	Approval to start the Manantali Grid Power Project.	19 Jun 2006		
				Authorization – construction of explosive magazine	Approval to start the Construction of an Explosive Magazine by FE3/4. Once the Environmental and Social Impact Assessment is complete, the final approval will be issue after the defense of the project in front the National Approval Committee.	Dec 2006, validation of report conducted in Dec 2007		
				Authorization – construction of Moolman accommodation	Approval to start the construction of 70 households to accommodate some Moolman residents not far from Neteko village. Once the Environmental and Social Impact Assessment is complete, the final approval will be issue after the defense of the project in front the National Approval Committee.	Validation of report conducted in Dec 2007		
			N°0014/GRK-CAB	Authorization - discharge pit water	Authorization to discharge pit water at main Pit and FE3&FE4 satellites pits.	20 Jan 2009		
			N°09/0002/MEA-SG	Authorization – new hazardous land field	Authorization to construct a new hazardous land field on waste dump.	24 Feb 2009		
			DRACPN-Kayes	Notice of Approval – medical incinerator	Authorization to operate medical waste incinerator.	14 Oct 2009		
			DRACPN-Kayes	Notice of Approval – road diversion	Authorization to divert Sekekoto – Sadiola road.	13 Jan 2010		
			DRACPN-Kayes	Notice of Approval – road re-profiling	Authorization to re-profiling National road Kayes- Keniéba on section from SEMOS offices Sadiola – Krouketo.	25 Feb 2011		

Application date	Application area (km <sup>2</sup> )	Granted area (km <sup>2</sup> )	Title no.	Subject	Title	DOU (date of official granting by government)	Current status	Expiry date
			N°011/0034/MEA-SG	Authorization – new hazardous land field	Authorization to construct a new hazardous land field site close to BLY workshop.	26 Apr 2011		
			N°011/0072/MEA-SG	Authorization - sulphide project	Authorization to execute Sadiola Sulphide Project.	22 Aug 2011		
			N°00176/DRACPN-KAYES	Notice of Approval – extension mine village	Authorization to execute Sadiola Mine village extension project 2011.	27 Sep 2011		
			N°012/0029/MEEE-SG	Authorization – power line	Authorization to connect Sadiola Mine to national Power Line.	May 28, 2012		
			N°2012/MEA-SG	Authorization – satellite pits and road diversion	Authorization to execute satellite pits mining Tambali, Tabakoto (formerly Sekekoto), Farabakouta Nord (FN2 FN3) Farabakouta Est (FE2) and National road (RN21) diversion.	3 Dec 2012		
			N°2013/1220/MM-MATDAT	<i>Arrêté Interministériel</i>	Modification of Zone A boundaries to include Tambali satellite pit in active mining area.	2 Apr 2013	Valid	
			N°2014/2126/MM-MIS-SG	<i>Arrêté Interministériel</i>	Modification of Zone A boundaries to include FE2, FN2, FN3 and Tabakoto (formerly Sekekoto) satellite pits in active mining area.	5 Aug 2014	Valid	

Source: SEMOS, 2015

## 5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURES AND PHYSIOGRAPHY

The Sadiola Gold Mine is located in the west of the Republic of Mali, West Africa near the border with Senegal, approximately 77 km south of Kayes, the regional capital.

### 5.1 Topography, Elevation and Vegetation

The Kayes region is flat and low-lying, 40 m above sea level, and is underlain by recent alluvial deposits of the Senegal River. The escarpment between Kayes and Sadiola, which is more rugged, is underlain by mid-late Proterozoic sedimentary rocks of the Kofi Formation (sandstones) and is generally 200 m to 300 m above sea level. Around Sadiola, the terrain is gently undulating with elevations ranging from 120 m to 200 m above sea level. Since 1994, the topography of the area within the SEMOS mining permits has been modified by the presence of the Main Pit (currently about 130 m deep), subsidiary pits, waste dumps and the TSF.

Four major vegetation communities have been described and mapped in the mining permit area:

- Sudanese gallery forests are located in natural depressions along streams favorable for the apparition of water dependent species such as *Mitragina inermis*, *Piliostigma* sp., *Acacia seyal*, *Ficus* sp., *Anogeissus leocarpus* and *Oxytenanthera abyssinica*. Grass cover is extremely abundant in the wet season. Perennial graminacés include *Antropogon gayanus*, *Oryza vulgaris*, and *Vetifera ziznioides*;
- Sparse woodland savannas are located on shallow soils and laterites. Shrub and grass cover is discontinuous, and wood volumes are less than 10 m<sup>3</sup> /ha, with the dominant species being *Combretum* sp, *Guiera senegalensis*, *Pterocarpus lucens*, and *Grevia* sp;
- Tree-bearing savanna, occupy the same areas as sparse woodland savanna, but is distinguished by the presence of large trees such as *Prosopis africana*, *Bombax costatum* and *Cordia pinnata*;
- Wooded savanna, which is the oldest savanna type, is characterized by wider diversity and a higher biomass potential. Trees in these areas include *Khaya senegalensis*, *Pterocarpus erinaceus*, *cordila pinnata* and *Bombax costatum* , while herbaceous vegetation comprises *Andropogon* sp . and *Penisetum pedicelatum* .

### 5.2 Access

Mali, officially the Republic of Mali, is a landlocked nation in West Africa. Mali borders Algeria on the north, Niger on the east, Burkina Faso and the Côte d'Ivoire on the south, Guinea on the southwest, and Senegal and Mauritania on the west. As such, Mali is dependent on its neighbors for ocean-borne inbound materials and supplies.

The preferred entry point for freight to Sadiola is through the port of Dakar in Senegal and overland transport by either rail (to Kayes only) and/or road. Other ocean freight inbound routes exist via Lomé in Togo and Abidjan in Côte d'Ivoire.

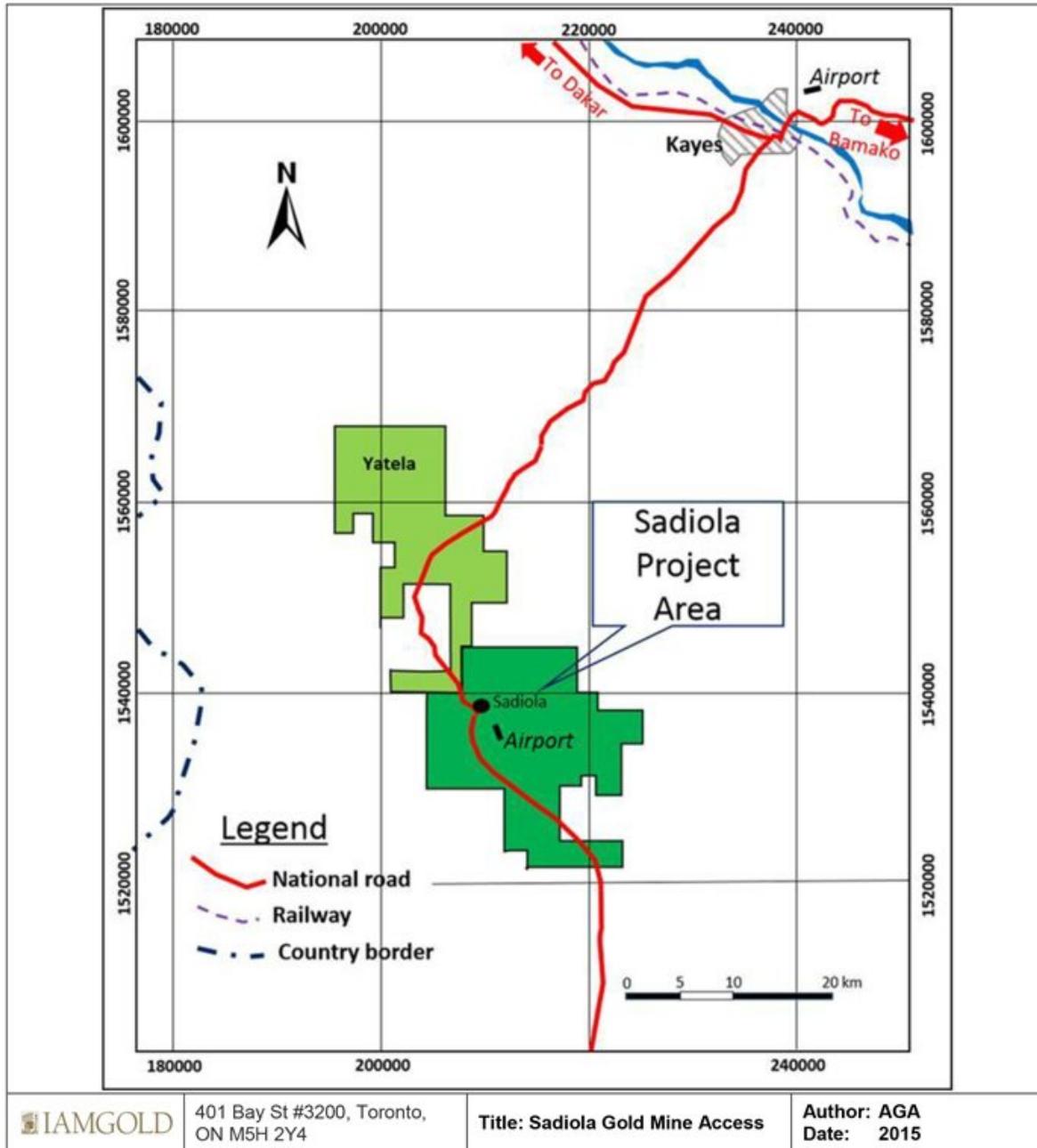
The highway between Dakar and Kayes is paved along the whole of its 836 km extent, having been completed at the end of 2012. The section of road between Bamako and Kayes is 506 km on a paved highway (Figure 5.1).

Access to the Sadiola operation from Kayes is by an 80 km long, regional, compacted laterite surfaced, all-weather, single carriageway road. Kayes is serviced by rail, road and air from Bamako, the capital of Mali, and from Dakar, the capital of Senegal.

Mali's main international airport is located in Bamako. There are daily flights to many other African and European destinations and the airport can receive large cargo carriers. The nearest national airport to the mine is located in Kayes. Kayes has a 2,700 m long sealed airstrip which has been widened and expanded to allow for larger planes to land and take-off so that it can also cater for cargo carriers. The Sadiola Mine site is equipped with a 1,800 m compacted, laterite airstrip.

Rail transport on the Dakar-Niger rail line is limited to 40-ton wagons, a width of 2.6 m, a height of 3 m and a length of 13 m. Road transport on national highways is limited to 7.5 tonnes per axle and logistics providers can transport loads up to 120 tonnes.

Figure 5.1: Sadiola Gold Mine access



**5.3 Proximity to Population Centre and Transport**

The Sadiola Gold Mine is situated proximal to 46 officially recognized villages and several hamlets. The main villages are Farabakouta, Neteko, Sadiola and Borokone. A mine village has been established to the northeast and provides housing, a medical clinic, a park and recreation facilities for mine employees and

dependents. Other facilities include guest accommodation, a post office, a supermarket, sewage treatment facilities and other amenities.

#### **5.4 Climate and Length of Operating Season**

The mine lies within the subtropical-to-arid climatic zone of Mali between the 700 mm and 900 mm isohyets. The climate is subtropical to arid with a dry and a rainy season. The dry season is from November to February with temperatures ranging from 15°C to 30°C. From March to June, temperatures range from 25°C to 45°C. The rainy season lasts from July to October.

Rainfall, in the form of high intensity convective thunderstorms, commonly accompanied by strong winds, occurs in summer. Annual rainfall exceeds 1,000 mm, and evaporation is twice the precipitation rate. Periods of water surplus and shortage occur in the wet and dry seasons respectively.

#### **5.5 Surface Rights**

The Sadiola Mining Permit was granted for an initial term of 30 years, expiring in July 2024, and may be extended by order of the President of Mali, if mining operations are ongoing.

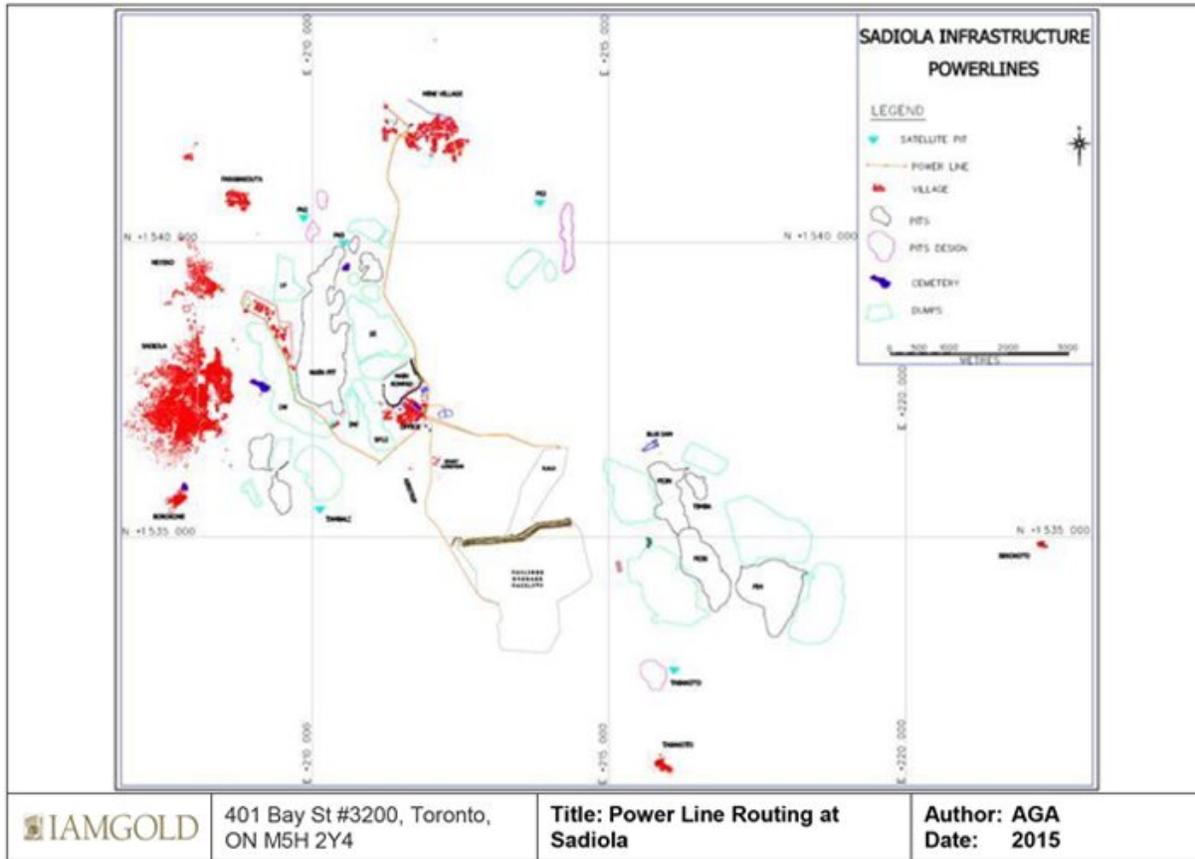
#### **5.6 Utilities, Mine Infrastructure and Personnel**

##### **5.6.1 Power**

Electrical power is currently provided through 18 x 1 megawatt (MW) medium-speed diesel generators which are capable of meeting an average demand of 16.7 MW. Approximately 4.73 M liters of diesel fuel per month are required for power generation and mining, under a contract with Total/ELF Petroleum Company. The 7 Mt national strategic fuel depot, situated in Kayes, is used as back-up storage in case of major road and/or rail disruptions.

The power infrastructure routing is outlined in yellow in Figure 5.2.

Figure 5.2: Power Line Routing at Sadiola



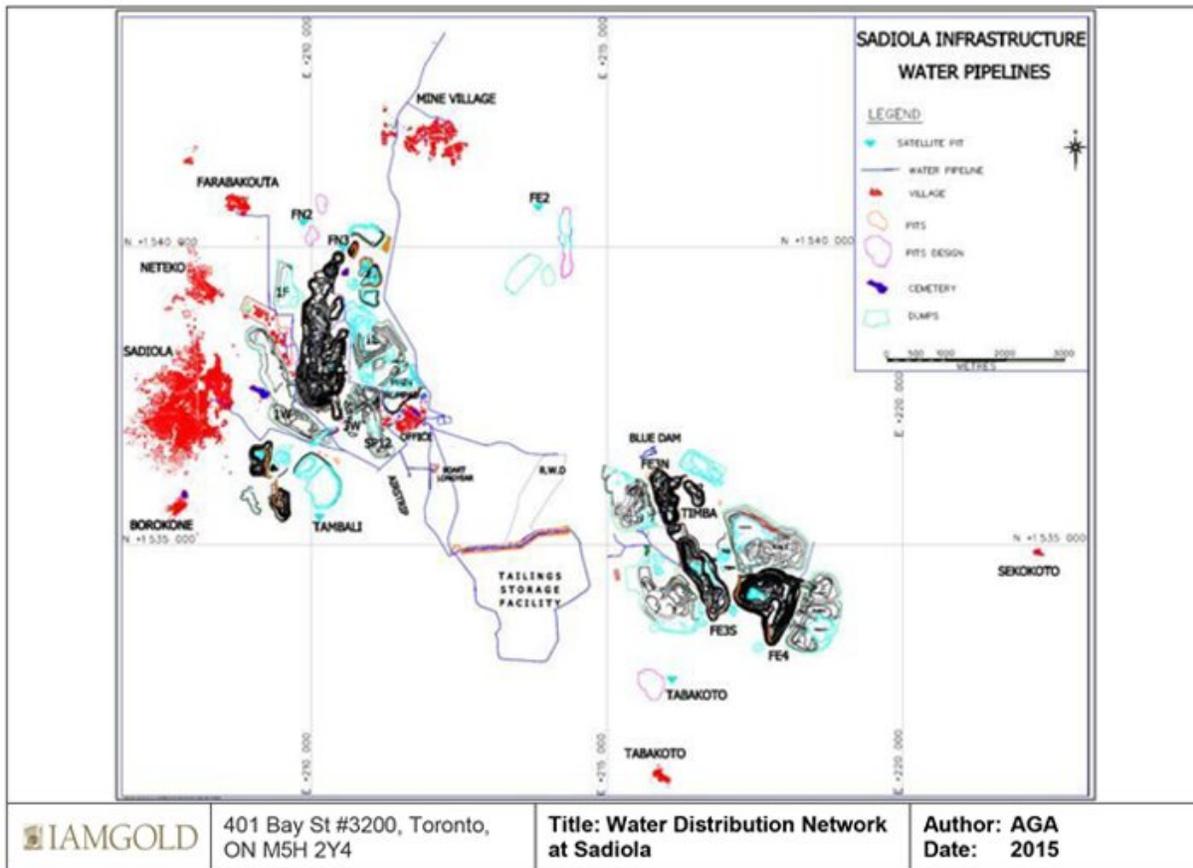
Current power infrastructure at site will not be sufficient of the expansion related to the SSP project. As such a high voltage (225 kV) power line will be added connecting the SOGEM power grid at Kayes to the Sadiola site, passing by Diamou. The power line will suffice existing power requirements of the site and the future expansion. This is further discussed in Section 18.7.

### 5.6.2 Water

A 55 km pipeline from the Senegal River, the only reliable source of surface water in the region, was built to provide approximately 8 M m<sup>3</sup> per year of process water in order to ensure that the Sadiola Gold Mine does not impact on local water resources. Potable water for both the mine operation and the mine town site and local villages are supplied from the pipeline, as well as local boreholes, and treated prior to distribution.

The water distribution network within the Sadiola area is shown in Figure 5.3.

Figure 5.3: Water Distribution Network at Sadiola



### 5.6.3 Tailings Storage Areas

The current tailings storage areas are shown in Figure 5.4. A proposed new TSF location will be situated in the valley downstream of the current TSF. This area consists of a large valley approximately 30 m deep and opening to the north. The valley is encased between two ridges on the east and the west sides. In the valley, the terrain is generally flat with some undulating surfaces and rocky outcrops. This area is bounded by the following:

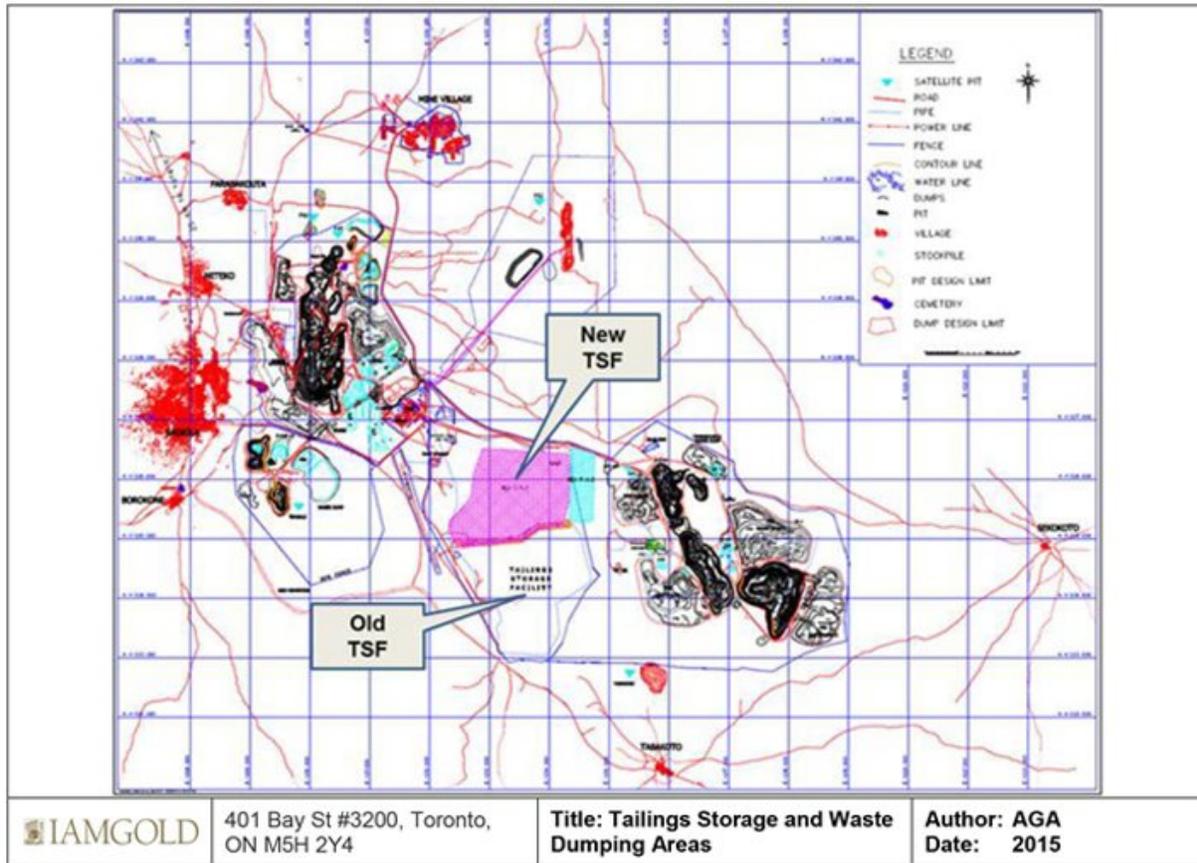
- The mine airstrip and a national road to the west;
- The mine workings and WRD to the east;
- The mining lease boundary to the north.

**5.6.4 Waste Disposal Areas**

The current waste disposal dumps are shown in Figure 5.4. Future mining based on the SSP will produce approximately 249 M t of waste during the 10-year life of the project. Two WRDs will be constructed to accommodate a volume of approximately 125 M m<sup>3</sup>. The smaller WRD will be positioned between the current western and north-western WRDs to serve as a noise and safety barrier between the mine and adjacent communities to the west. The larger WRD will be positioned to the northeast of the open pit with a footprint of approximately 226 ha, a maximum height of 80 m and overall side slopes of 1 V:3 H. The nearest village is located at a distance of 650 m from the WRD, beyond the required 500 m buffer zone.

The WRD will require a service corridor along its perimeter to include haul roads, protective berms and storm-water management and erosion prevention measures. A minimum clearance distance of 47 m will be maintained between the pit crest and WRD toe to accommodate the service corridor.

**Figure 5.4: Tailings Storage and Waste Dumping Areas**



**5.6.5 Mining Personnel**

The Sadiola Mine employs more than 1,000 people, including those employed by outside contractors. The majority of Sadiola personnel are Malian nationals (~93%), with the remainder being expatriates from South Africa (~7%). The majority of the unskilled labour is sourced from the nearby town of Kayes, Sadiola and neighbouring villages.

Additional manpower will be required during the construction of the new plant and other infrastructure required for implementation of the next phase of mining, and this has been included in future operational and project capital cost estimates.

The upgraded plant technology for the processing of hard sulphide rock will require specialised staff for its operation and maintenance. These skills are most likely to be obtained through expatriate staff.

## **6. HISTORY**

### **6.1 Prior Ownership and Ownership Changes**

The original prospecting and exploitation agreement with the Mali Government for the Sadiola area, dated April 15, 1990, was held by A.G.E.M. (Federal Republic of Germany). The following document (English translation available) outlines the terms and conditions of this agreement:

“CONVENTION D’ÉTABLISSEMENT POUR LA RECHERCHE ET L’EXPLORATION DE L’OR, DE L’ARGENT, DES SUBSTANCES CONNEXES AINSI QUE DES PLATINOIDS ENTRE LE GOUVERNEMENT DE LA RÉPUBLIQUE DU MALI ET LA SOCIÉTÉ DE RECHERCHE ET D’EXPLOITATION AURIFÈRE “A.G.E.M. (R.F.A.)” (1990).

A.G.E.M. was issued an exclusive prospecting license for gold, silver, related substances, and platinoids. The above exploration and exploitation agreement set out the terms, conditions, and circumstances under which A.G.E.M. were required to operate in Mali and outlined the commitments of both parties. A.G.E.M. undertook to conduct a feasibility study at its cost should any economically viable minerals be identified. On the basis of a feasibility study completed in 1993, an exploitation authorization (mining permit) was granted in August 1994 for a period of 30 years.

The State reserved its right to take up a share in the resultant mine and it was up to A.G.E.M. whether it wished to exploit the mineral deposit itself, obtain business partners, or sell the mineral rights. A.G.E.M. opted for the latter and the Sadiola deposit was procured by SEMOS S.A. a joint venture company formed between AAC (38%), IAMGOLD (38%), République du Mali (18%), and the IFC (6%). AAC subsequently spun off its gold mining assets to form a new company, AngloGold Limited, which subsequently merged in 2004 with Ashanti Goldfields Corporation to form AngloGold Ashanti. The IFC sold its shares in 2010, resulting in the current shareholding being split between AGA (41%), IAMGOLD (41%) and the Republic of Mali (18%). Annual gold production figures since 1997 are shown in Table 6.1.

SEMOS continues to be bound by the original exploration and exploitation agreement (including its subsequent legal modifications) entered into on April 15, 1990, between A.G.E.M. and the Mali Government. The mining permit is valid for the original mineral commodities (Group 2) until July 31, 2024.

### **6.2 Previous Exploration and Development Work**

Sadiola was identified as a favorable exploration area based upon the widespread evidence of artisanal gold workings and small scale mining by local inhabitants. Written records of mining at Sadiola reportedly date back 250 years, and the extent of the historical works suggests that mining may date back more than 1,000 years.

From October 1987 to August 1989, a large regional geochemical survey, known as Mali Ouest 1, was carried out for the Government of Mali by the German company, Klöckner INA (“Klöckner”), as part of an aid program financed by the European Development Fund. In addition to the 48,000 samples collected during this first-pass regional program, detailed geochemical sampling near the villages of Sadiola and Dinnguïlou confirmed high gold, arsenic and antimony anomalies.

In January 1990, exploration rights for the Sadiola area were granted by the Government of Mali. Klöckner was hired to conduct a large scale gold exploration program at Sadiola which identified the presence of a significant oxide gold deposit.

In 1991, WGM was retained by IAMGOLD to review the work of Klöckner, prepare a PEA of Sadiola and make recommendations for further work. The PEA yielded positive indications and WGM recommended a large exploration drilling program to delineate and confirm the Sadiola Mineral Resource. During 1991 and 1992, WGM assumed responsibility for the ongoing exploration effort. In December 1992, WGM estimated a probable reserve of 22.3 M t of oxide mineralization, with an average content equivalent to 3.3 g Au/t.

In October 1992, a joint venture agreement with AAC was signed for the construction and management of any mine developed at Sadiola. A feasibility study on the Sadiola gold deposit, dated December 1993 and prepared by AAC, was presented to the Government of Mali.

In August 1994, the Government of Mali issued an exploitation permit (the “Sadiola Mining Permit”). SEMOS was incorporated on December 14, 1994, as the joint venture company to hold the Sadiola Mining Permit, exploit the Sadiola gold deposit and carry out exploration activities within the Sadiola area.

The Sadiola Gold Mine poured its first gold bar on December 20, 1996 and since start-up, the mine has produced more than 7 M ounces.

Mining commenced in the FE3 pit in April 2001 and in the FE4 pit in November 2001. Mining at the latest pit (Tambali), south of the Main Pit, started in July 2013.

Several reports, maps and reviews are available at the SEMOS site office, with detailed descriptions and interpretations from many authors throughout the years of operation. This includes recent work carried out by Professor Kim Hein, from the University of the Witwatersrand in South Africa, spanning mid-2007 to early 2008, and Dr. Greg Cameron for 2009 and 2010.

### **6.3 Historical Mineral Resource and Mineral Reserve Estimates**

SEMOS has operated the Sadiola Mine since inception in 1996. As such, no historical Mineral Resource and Mineral Reserve estimates are relied on, but SEMOS has reported Mineral Resources on a 100%

ownership basis in accordance with the guidelines of the JORC Code (2012) and AGA reporting guidelines (2014).

#### 6.4 Production History

A summary of historical gold production for the Sadiola operation is shown in Table 6.1.

**Table 6.1: Summary of Annual Gold Production**

Year	Tonnage processed (t)	Head grade (g Au/t)	Gold production (oz)
1997	4,028,737	3.02	383,281
1998	4,954,755	3.32	506,113
1999	5,159,602	3.25	542,955
2000	5,344,785	3.76	611,442
2001	5,328,149	3.35	536,047
2002	5,038,017	3.50	479,910
2003	5,070,167	3.04	451,807
2004	5,146,827	3.76	458,671
2005	5,027,242	3.48	441,865
2006	4,821,402	3.89	498,990
2007	4,157,279	3.73	369,342
2008	4,116,776	3.93	453,245
2009	4,364,348	2.59	354,280
2010	4,370,613	2.22	287,177
2011	4,825,748	2.09	294,544
2012	4,638,155	2.03	244,768
2013	4,856,893	1.48	209,776
2014	5,026,930	1.47	206,332
2015	5,061,765	1.17	168,586

Source: SEMOS, 2015

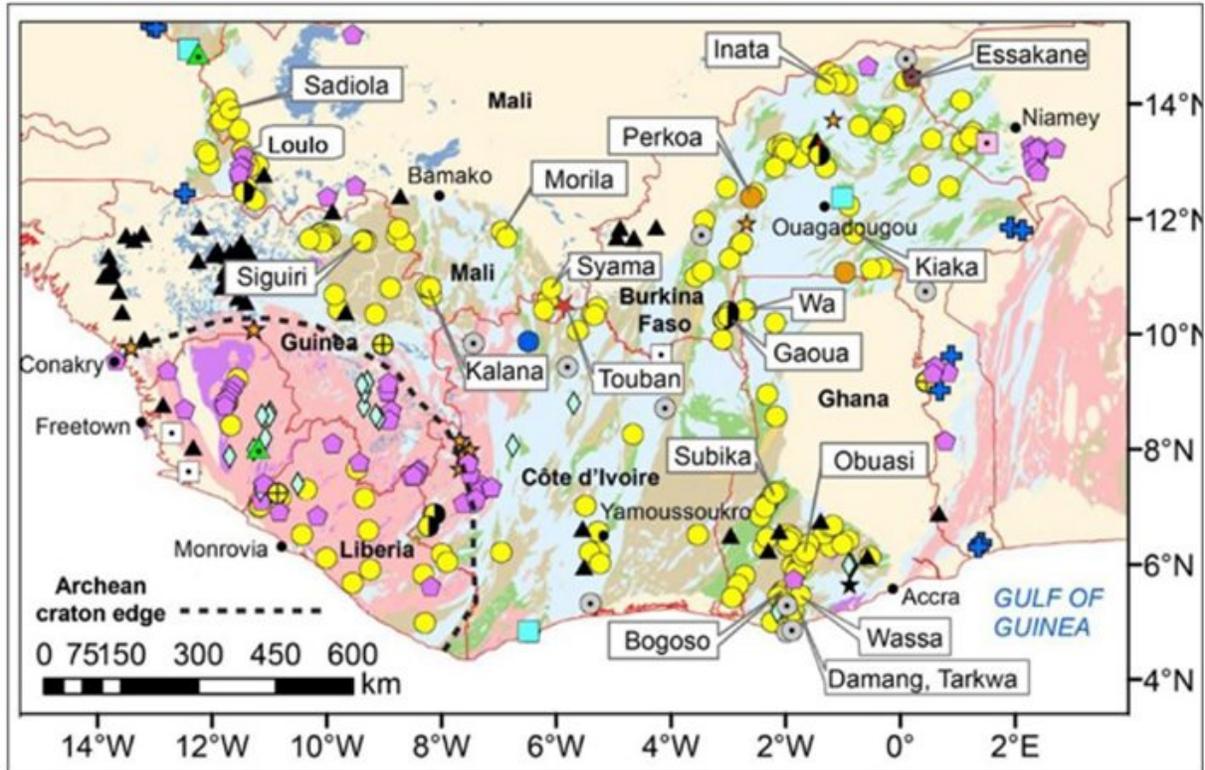
## 7. GEOLOGICAL SETTING AND MINERALIZATION

### 7.1 Regional Geology

Section 7.1 is reproduced from the SEMOS Competent Persons Report (CPR) (SEMOS, 2014), with minor modifications.

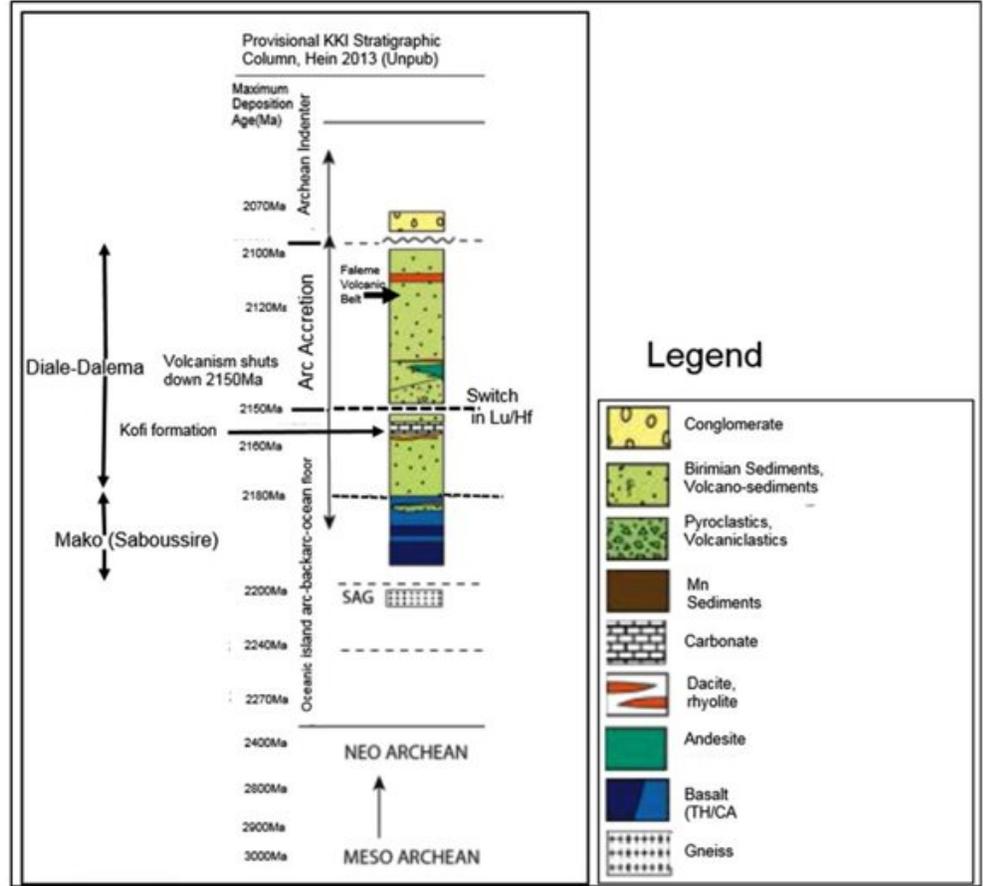
The Sadiola deposit, previously known as the Sadiola Hill deposit, is located in west Mali within the KKI, an Early Proterozoic window of volcano-sedimentary greenstone belts and calc-alkaline granite intrusions that comprise part of the Lower Birimian terranes of the West African Craton (Figure 7.1 and Figure 7.2 ). The inlier is positioned at the northeast margin of the Kenema-Man Shield and is bound to the west by the Pan-African Mauritanide Hercynian Belt. It is concealed to the north, east and south by undeformed Neoproterozoic and Paleozoic sedimentary formations of the Taoudeni Basin.

Figure 7.1: Regional Geological Setting



Source: Miller et al., 2013

Figure 7.2: Kedougou-Kenieba Inlier Stratigraphy



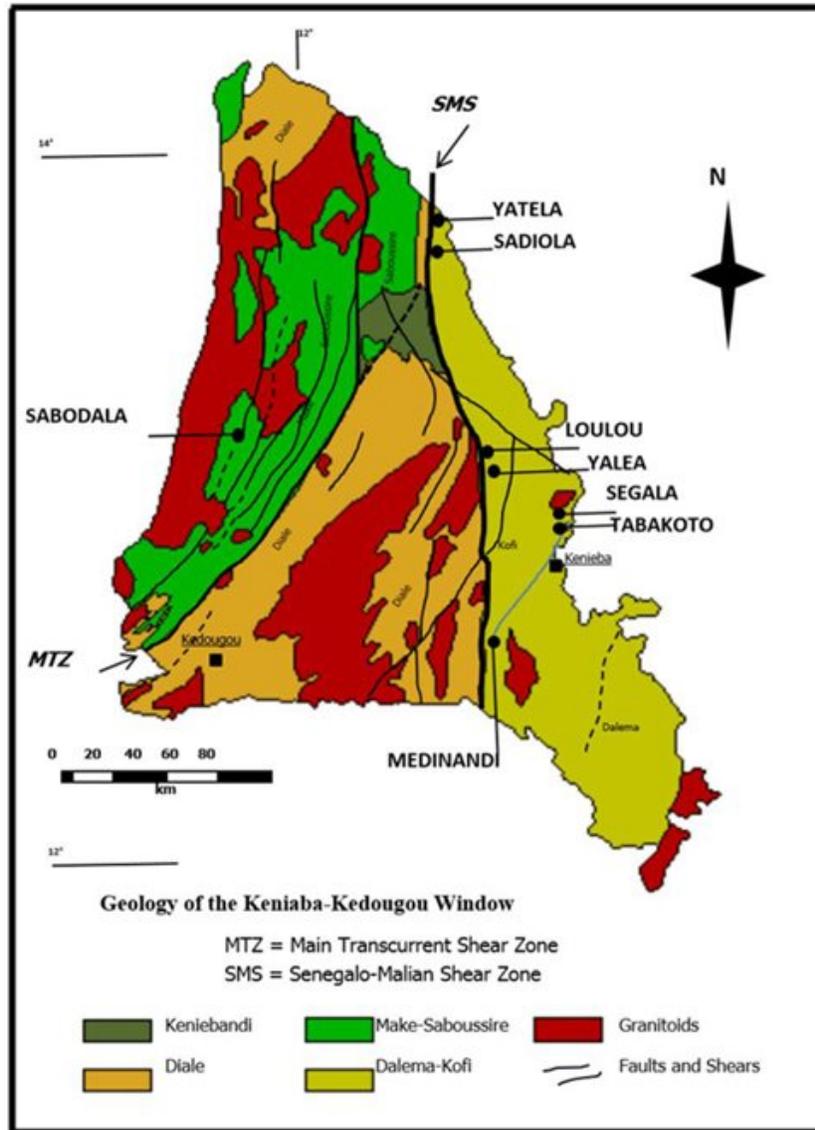
Source: AGA, 2015

The volcano-sedimentary sequences of the KKI are separated into two lithostratigraphic Supergroups (Gueye *et al.*, 2007). The Mako (or Saboussire) Supergroup is located in the west and is characterized by massive and pillowed tholeiitic basalt, calc-alkaline volcanic rocks and interbedded volcanoclastic sediments. To the east, the younger Dialé-Daléma Supergroup comprises platform type sediments of carbonate, graywacke, sandstone and pelite, intruded by intermediate and felsic calc-alkaline rocks.

The Supergroups are separated by a major, regional crustal scale structure. The northeast-trending Main Transcurrent Shear Zone and the north to north-northwest trending Senegal-Mali Shear (SMS) Zone separate the volcano-sedimentary Dialle group to the west from the dominantly sedimentary Dalema to the east (Figure 7.2). Regional metamorphism is greenschist facies with amphibolite facies metamorphism only observed in the contact aureoles around major intrusions (Gueye *et al.*, 2007).

Sadiola is positioned just east of the SMS terrane boundary within a corridor of gold deposits which are all spatially related to the SMS. This mineralized corridor extends from the Sadiola group of deposits in the north through Randgold’s Loulou and Yalea deposits in the south and on to the Guinea border (Figure 7.3). These deposits are all hosted in the Daléma Complex/Kofi Formation (Figure 7.3) and it is generally accepted that most of the mineralization is kinematically related to the SMS.

Figure 7.3: Geology of the Kedougou-Kenieba Inlier



Source: AGA, 2015

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## 7.2 Local Geology

The Sadiola Mining Permit is located in the Kofi Formation on the northern portion of the KKI. Within the Sadiola permit and beyond, the Kofi Formation occurs as alternating sequences of limestones (generally impure) and turbidites, comprising impure sandstone, black shales, siltstones, fine to medium grained graywackes and pelites, interpreted to be of distal origin. Minor felsic to intermediate volcanics also occur and the succession is intruded by numerous QFPs to intermediate diorite dykes and stocks.

To the west, the Kofi Formation is truncated by intermediate to mafic volcanics. To the east, it is overlain by sandstones of the Neoproterozoic Taoudeni Basin that forms a prominent escarpment along the strike of the inlier.

Gold mineralization within the Sadiola Mine Permit is hosted within two major trends. The Sadiola trend located to the west and the Farabakouta East (FE) trend to the east. On the Sadiola trend, gold mineralization is hosted within the north-south striking Sadiola Fracture Zone (SFZ) that exploits the contact between the graywacke to the west and the carbonate on the east along a 4 km strike. This zone is also associated with north-northeast to northeast trending mineralized splays. While the SFZ dips steeply towards the west, with localized flexures to the east, the north-northeast to northeast trending mineralization dips moderately towards the southeast.

The southern part of the Sadiola trend contains north-south and north-northeast trending mineralization. In this area, the mineralization straddles the contact between the graywacke and carbonates, but occurs dominantly within the carbonate. This contact is also exploited by deformed lenses of mineralized diorite. Towards the north, gold mineralization occurs as discrete lenses and/or shoots and is hosted within the carbonates in north-northeast trending structures.

On the FE trend, mineralization occurs along the contact between the western carbonates and eastern turbidites (metapelites) that have a prominent graphitic base.

The contact is brecciated and folded, suggesting a possible unconformable relationship between the strata. Gold mineralization is hosted within the weathered carbonates, with the lenses spread over a 10 km strike. Towards the southeast, the mineralization dips approximately 45°W, while in the western and northern parts, it dips at shallow angles towards the east.

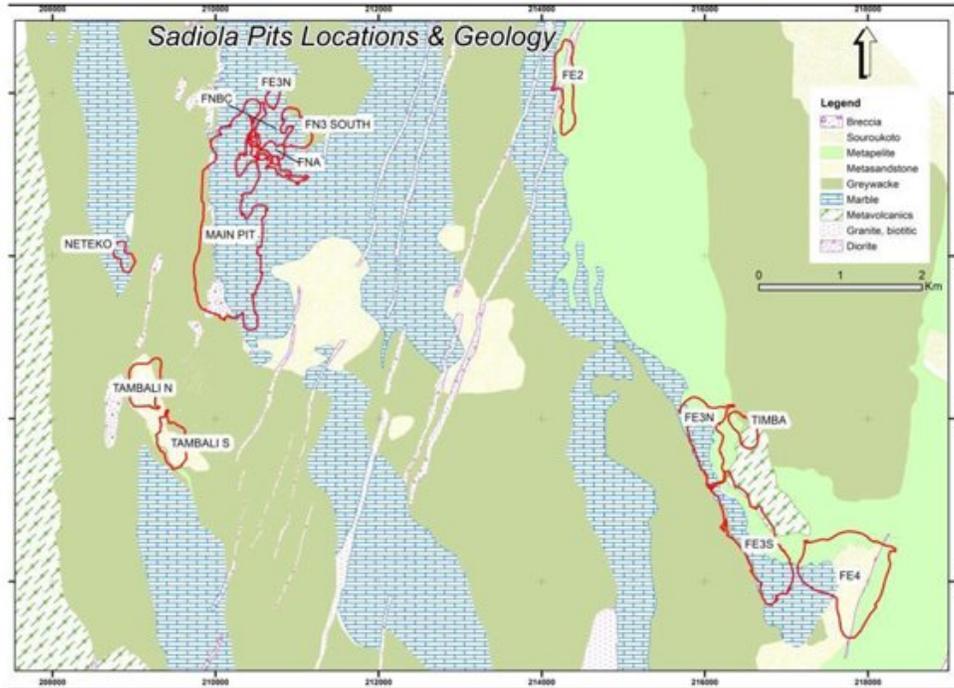
## 7.3 Property Geology

The Sadiola Gold Mine property comprises the following mining areas (illustrated in Figure 7.4):

- Sadiola trend (Sadiola Main Pit, SSP, FN2 and FN3);
- FE trend (FE2, FE3 and FE4 pits);

- Tabakoto (formerly Sekekoto);
- Tambali.

Figure 7.4: Property Geology and Pit Locations

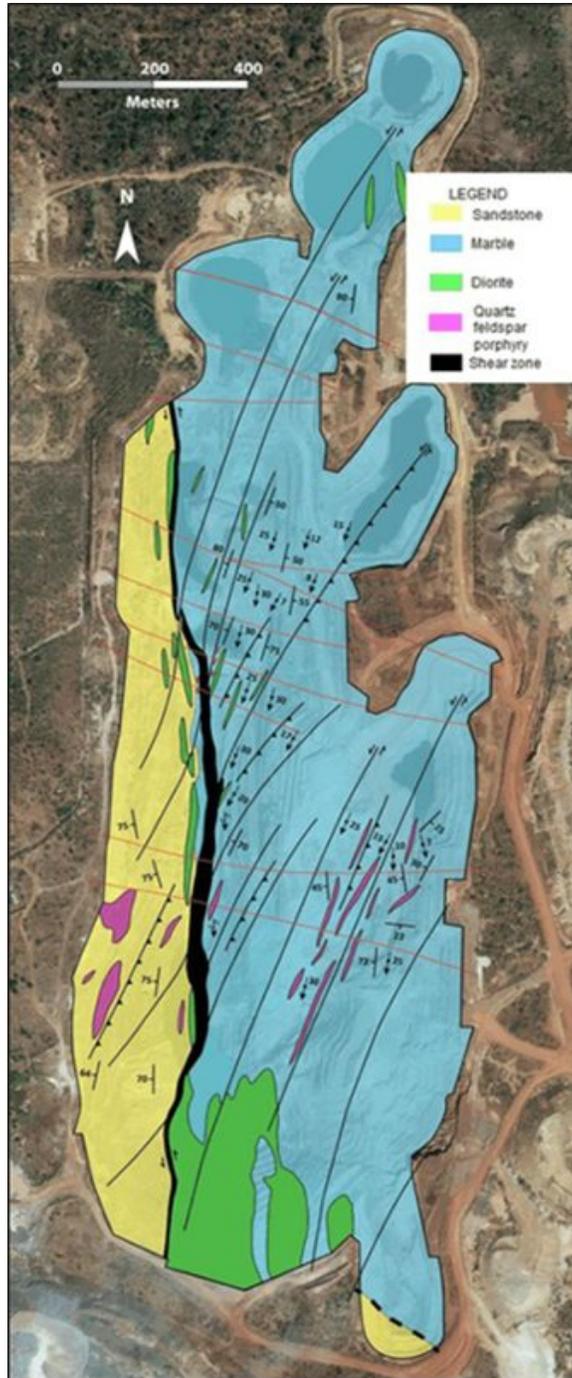


Source: SEMOS 2016

### 7.3.1 Sadiola Trend

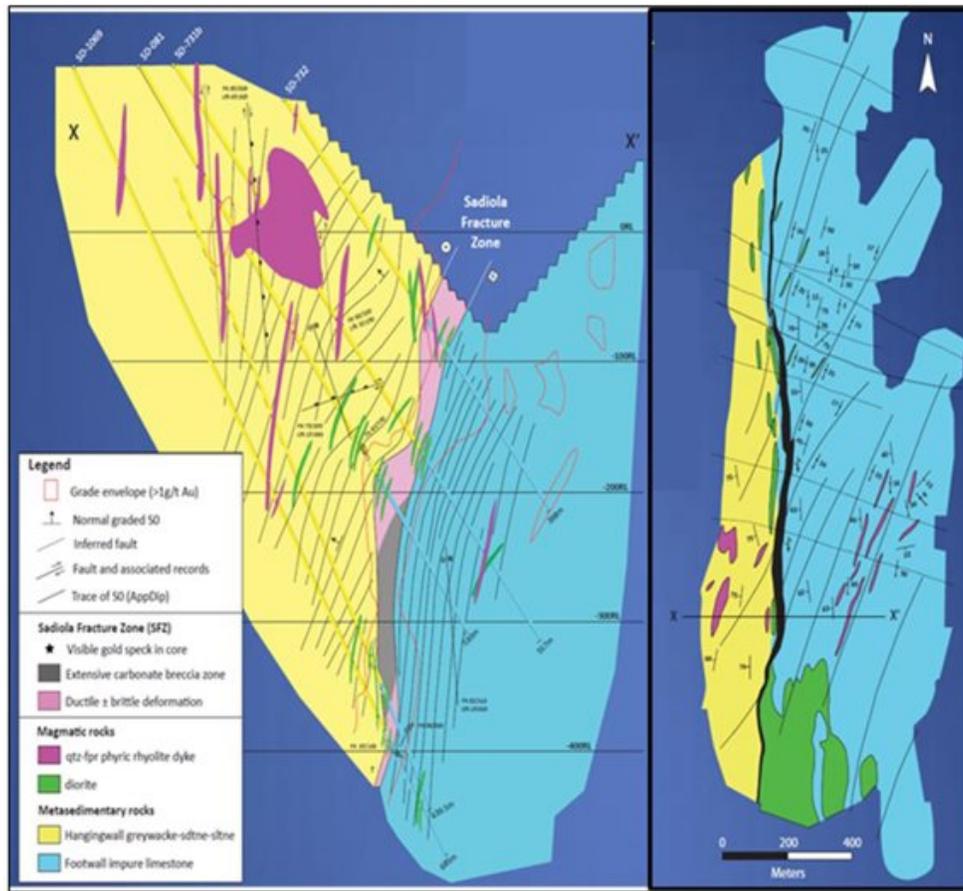
The Sadiola trend is a brittle-ductile shear zone-hosted deposit related to the interaction of a north-northeast striking fault array with a single major structural discontinuity, the north-south striking SFZ along the graywacke-carbonate contact shown in Figure 7.5 (Cameron, 2010).

Figure 7.5: Geology of the Sadiola Deposit



A west-east section illustrating the major lithologies in the area is presented in Figure 7.6 below.

**Figure 7.6: Sadiola WestEast Section**



Source: AGA, 2015

Gold mineralization occurs in all four rock types: graywacke, carbonate, diorite and QFP, usually close to or within the contact/SFZ. The bulk of the mineralization is hosted in the footwall adjacent to the SFZ. The mineralization has a strong structural control and is spatially associated with a complex weathering and alteration pattern.

Mineralization along the SFZ occurs over a drilled strike length of approximately 2,500 m and remains open to the north, south and at depth. Much of the mineralization appears related to incipient brecciation, fracturing and shearing of the host rocks. The location and geometry of mineralization may be spatially related to changes in strike and dip of the SFZ; specifically, left-hand flexures and steeper parts of the SFZ appear to be better mineralized. Leapfrog™ modelling shows a left-hand flexure (dilatational) in the footwall contact which corresponds to a large area of high grade mineralization from 5200 m N to

5700 m N. This suggests a sinistral sense of movement during mineralization, consistent with the generally accepted sinistral kinematic along the SMS. This zone of higher grade mineralization appears to be both open and improving with depth.

Post-mineralization deformation has complicated structural relationships in the deposit. Late stage faulting occurs over the length of the pit, the most important of which is the B6 fault.

Mining has been limited only to the southern 2,600 m due to consistent grades along the strike. Towards the north the mineralization weakens along the north-south trending SFZ, and stronger mineralization is hosted by shoots within the north-northeast to northeast trending shears. These shears appear to cross-cut and at times thrust off the upright folds. The change over to the north-northeast trend has led to the development of numerous satellite pits on the northern part of the Sadiola trend, mainly the FNE and FN3. On the last 1,000 m of the Sadiola trend is the FN2 resource that has not been mined due to low grades. At FN2 the mineralization occurs along the graywacke-carbonate contact.

### **7.3.2**            **FE Trend**

The FE trend is a carbonate-metapelite contact on the eastern part of the Sadiola Mine Permit. It is marked by prominent electromagnetic (EM) lineaments resulting from the strongly conductive eastern graphitic alteration in metapelite (Figure 7.7). The conductive zone can be traced over more than 12 km. This overlies the areas locally referred to as FE1 to FE4, where FE1 is on the extreme north and FE4 is on the southern end of the strike.

The carbonate is characterized by laminated impure marble grading into white pure carbonate and pink dolomite as the metamorphic facies changes.

The contact between the carbonate horizon and the eastern metapelite is brecciated. There is a major north plunging S-fold on the southern end of the FE trend between the FE3 and FE4 pits. From the FE3 pit to the FE2 resource, the lithological contact trends northwest-southeast and from FE2 northwards it trends approximately north-south to north-northeast.

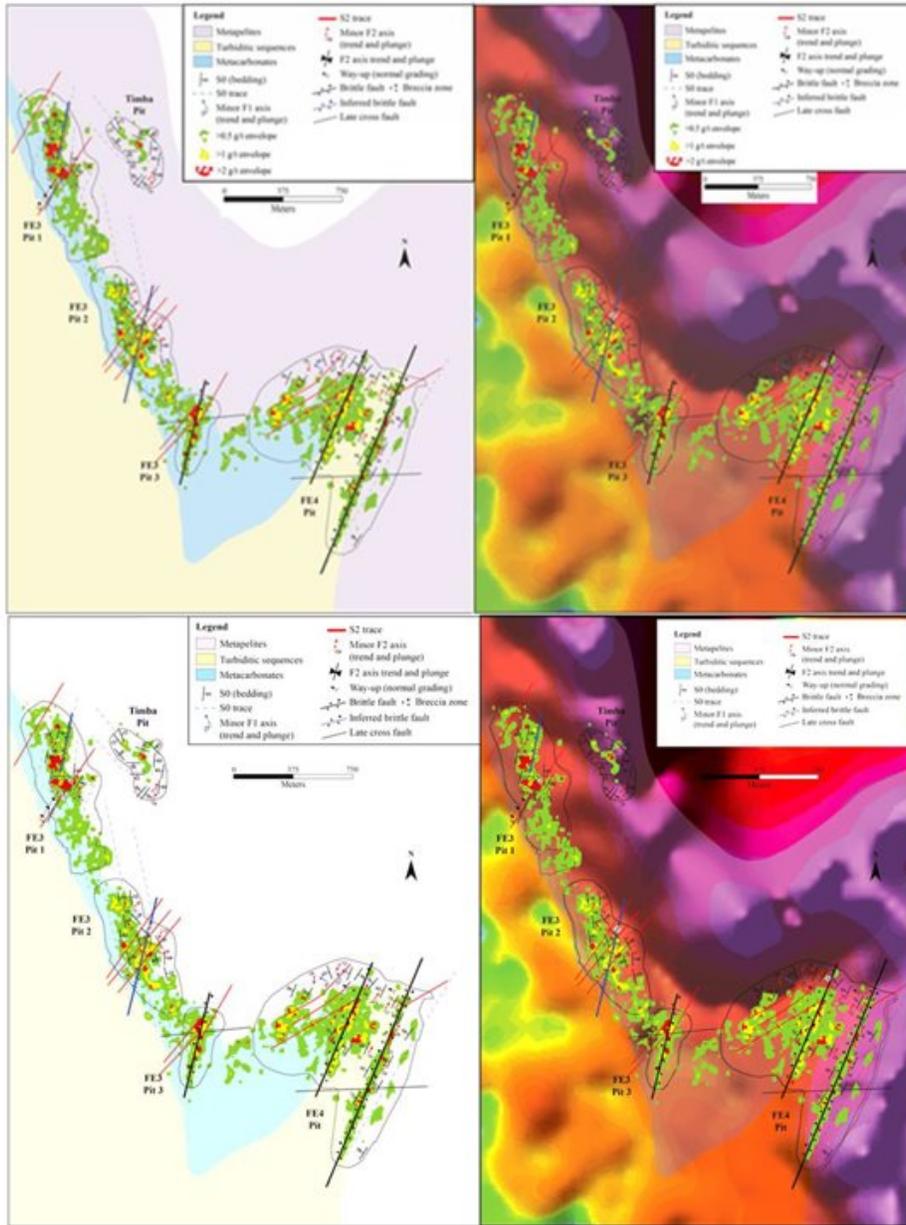
The FE4 pit is bounded by a short east-west trending fold limb to the north and a sheared longer limb to the east, the two limbs being separated by an antiform (Figure 7.8).

Primary mineralization along the FE trend is structurally controlled by a north-northeast trending sub-vertical shear set (Masurel *et al.*, 2015). Geometric and kinematic features of the ore-hosting structures indicate the latter were undergoing sinistral displacement at the time of mineralization.

Shallow high grade sulphide intersections have been encountered below the FE3 pit 3 and FE4 pits, but the strike lengths are very short. Along the FE trend the gold mineralization appears to be associated with copper mineralization which occurs in narrow malachite bands within the carbonate layers. Pyrite is the dominant sulphide species and is closely associated with minor chalcopyrite and traces of arsenopyrite and pyrrhotite together with tetrahedrite-tennantite. Traces of galena, sphalerite, and ullmannite do also occur but are consistently precipitated late in the paragenesis.

All the mining along the FE trend has been confined to the oxide zone where the lithological contact and the shears have played a major role in focusing the weathering.

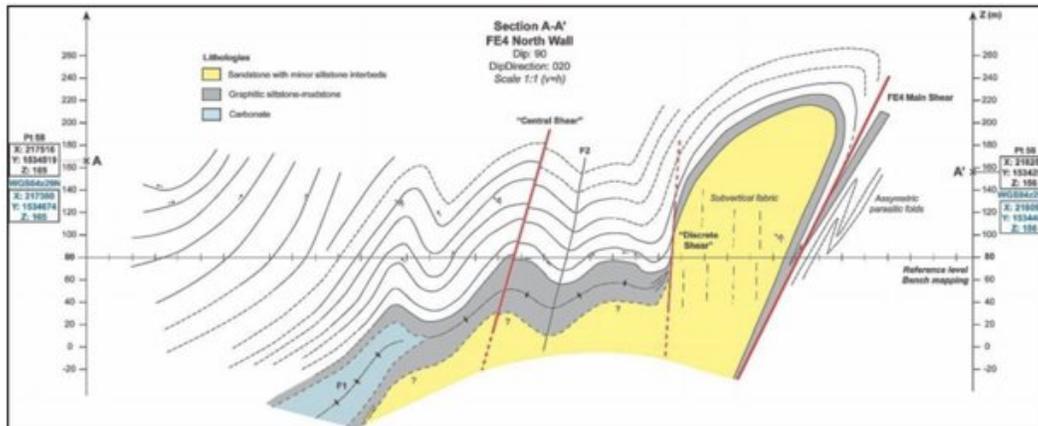
Figure 7.7: FE3 and FE4 Pits



Source: Masurel, et al., 2015

Note: Geological map with major structure and grade from Leapfrog™ modeling (left) overlain on EM Z7 (right)

Figure 7.8: Section through FE4 with Major Antiform to East



Source: Masurel et al., 2015

### 7.3.3 Tabakoto

The Tabakoto (formerly Sekekoto) target comprises a gold-in-soil anomaly spread over an area of 1.5 km by 1.5 km and is located about 5 km south-southeast of the Sadiola Main Pit. Gold mineralization occurs in the laterite and the underlying saprolite over a 300 m strike which is open on both ends and at depth.

The target is located along the northwest trending contact between the western carbonate and eastern turbidite strata. Weathering typically exceeds 70 m in depth, especially along the geological contact with mineralization occurring along the weathered northeast-dipping interface of east-dipping marbles and a sequence of metagraywackes/conglomerates.

The mineralization is interpreted as having been constrained by cross cutting east-northeast to west-southwest structures, consistent with observations in the FE trend. It also occupies the same stratigraphic horizon as the FE trend.

### 7.3.4 Tambali

The Tambali (TS1) prospect is located about 150 m southwest of the Sadiola Main Pit and has been investigated since 1998 when exploration commenced. High gold values in soil samples are closely associated with arsenic, copper and antimony. Two small pits (Tambali South and North) were mined during the period 2013 to 2014.

Host rocks consist of moderately-sorted wacke arenite and QFP with minor siltstone interbeds. A fine grained marly horizon marks the southwestern and western boundaries of the prospect. The bedding changes between the two pits from a moderate dip to the southwest on the south, to a moderate/steep dip to the west on the north.

At the Tambali South pit, the bedding has an average orientation of 150°/55°SW, whereas in the Tambali North pit, it has an average orientation of 170°/75°W (Masurel, 2015). North-northeast to northeast trending QFP felsic dykes cross cut the sediments. The QFP dykes are, in turn, cross cut by north-south to north-northeast trending, very narrow; diorite dykes in the Tambali South pit. This set of mafic dykes is tentatively associated with the northeast trending diorite dykes at Sadiola.

#### 7.4 **Mineralization**

Mineralization at Sadiola is referred to as mesothermal or shear hosted (Hanssen, 1998; Hein, 2008; Cameron 2010; Masurel, 2015). The gold preferentially precipitated within the highly faulted carbonates and QFPs, although the graywacke and diorites are also mineralized to a lesser extent. The mineralization is mostly associated with lens-shaped breccia zones with both arsenic and antimony dominated sulphide assemblages including arsenopyrite, pyrrhotite, pyrite, stibnite and gudmundite. Hydrothermal alteration assemblages identified to date include: calc-silicate, potassic, chlorite-calcite, carbonate and silicification.

Secondary mineralization as a result of extensive supergene processes resulted in enrichment of gold in the oxidized lateritic and saprolitic material. The saprolitic material type extends to a depth of about 200 m especially along the permeable structures in the rock formation, controlled mainly by geological contacts, faulting, shearing and porosity that allowed the deep penetration of ground water. This also resulted in the oxidation of the primary sulphides. Oxidation of pyrite (and other sulphide species) results in the formation of sulfuric acid further promoting the downward argillization of the carbonate bedrock to form the clay rich assemblages present in the saprolite. The irregular “karst like” (trough and crest) soft rock- hard rock contacts can in many cases be related to the extent of faulting and the original sulphide content of the underlying profile.

The Sadiola deposit shows good continuity of mineralization both along strike and down dip and has been mined for 2 km along strike. Structurally controlled high grade “pay shoots” typically occur within a more pervasive lower grade alteration halo. The location and geometry of high grade mineralization appears to be controlled by the confluence of the SFZ with the N20° splays resulting in steeply to vertically plunging zones within the plan of the SFZ, though the overall mineralization plunges at 15° to the south-southwest which is consistent with the fold axis in the carbonate.

The saprolitic oxide portion of the orebody, or the material above the soft-hard contact, is referred to as the Sadiola Main Pit Oxides. The Sadiola Sulphide Project (“SSP”) focused on a techno-economic assessment of the unweathered material below the main Sadiola orebody. The soft sulphides represent the transition zone from the weathered (overlying) oxides to the (underlying) unweathered hard sulphides and occur as a continuous unit underlying the saprolitic oxides. The contact between the oxides and soft sulphide mineralization could potentially be exploited by a deepening of the open pit, while deeper mineralization could be exploited by underground mining methods.

#### 7.4.1 **Oxide Mineralization**

The geometry of the extensive, soft, oxide deposit and its supergene enrichment of gold relates almost exclusively to the weathering history of the primary mineralization. Intense tropical weathering has produced deep troughs of white to grey, decarbonated, kaolin-rich saprolite, locally abundant nontronite and relative gold enrichment. Penetration of groundwater has caused oxidation of the primary sulphides and the formation of sulfuric acid, further promoting deeper argillisation of the bedrock. The variable permeability of the deposit, controlled by faulting, shearing and porosity, has led to the irregular “karst-like” weathering geometry from 30 m deep in the north to 220 m in the south. Weathering is deepest along the SFZ.

The deeply weathered saprolite was protected from erosion by a capping of hardpan laterite (ferricrete).

#### 7.4.2 **Sulphide Mineralization**

Drilling of the (unweathered) primary mineralization has allowed detailed investigation of major and minor hydrothermal alteration processes that were active during the formation of the deposit.

Primary gold is extremely fine grained, dominantly less than 15 µm, with rare grains approaching 50 µm. Visible gold is rare. Gold mineralization is associated with both arsenic and antimony dominated sulphide assemblages of arsenopyrite, pyrrhotite, pyrite, stibnite and gudmundite as well as potassic, calc-silicate, propylitic alteration and silicification. Much of the mineralization appears to be related to deformation of the host rocks.

## **8. DEPOSIT TYPES**

Sadiola has been classified as a “mesothermal – shear hosted” gold deposit which has implications regarding the distribution of mineralization and resource potential. Deposits of this type exhibit good continuity of mineralization both along strike and down dip. Structurally controlled, high grade “pay shoots” typically occur within a lower grade halo in these types of deposit.

Sadiola is a brittle-ductile shear zone-hosted deposit related to the interaction of a north-northeast striking fault array with a single major structural discontinuity, the north-south striking SFZ.

The FE trend hypogene mineralization exploits the intersection of the north-northeast trending shears and the northwest-northeast trending lithological contacts.

Most of the mineralization is hosted within the carbonate strata suggesting a litho-structural control.

Elements of skarn mineralization have been observed at both the FE and Sadiola trends; these could be linked to some deep seated intrusion as suggested by Theron (1997) and Hein (2008).

Supergene processes have upgraded the gold resources in the oxide zone. Along the FE trend the “high grade” oxide mineralization appears to be derived from very low grade sulphide mineralization. At Sadiola there was significant production from alluvial gold, indicating that the opportunities for secondary gold exist. There are areas on the streams draining the Sadiola trend which have been partially exploited by artisanal miners for alluvial gold. The extent of the remaining alluvial gold has not been qualified or quantified.

## **9. EXPLORATION**

Exploration work, other than drilling, completed since 2010 included field mapping, termite mound sampling and geophysics. The work was carried out by SEMOS exploration geologists where not otherwise mentioned in Sections 9.1 and 9.3.

### **9.1 Mapping**

This section has been reproduced, with minor changes, from SEMOS (2015c).

During 2015 the areas of mapping focus were:

#### **9.1.1 TB6**

TB6 is located to the south of Tambali and is covered by volcanic rocks that vary from crystalline, through porphyritic to pyroclastic textures. Low grade gold-copper veins were mapped and sampled and it was noted that all the copper-bearing samples are related to north northeast trending structures; however, their relationship with the regional northwest trending SMS fabric is not clear.

The potential for oxide resources is very small and can only be realized if the mineralization extends into the area of relatively deeper weathering on the southern boundary of the Sadiola Permit.

#### **9.1.2 Sadiola North**

Mapping targeted the area towards the north of the pit where the north northeast to northeast trending mineralization is dominant. The Sadiola north pit is generally underlain by carbonate rocks with numerous small scale folds (a few meters to a few tens of meters) with axes plunging at about 25° towards the southwest. Channel sampling along the 900 m north face confirmed that the mineralization splays into two, north-northeast trending zones that appear to dip towards the southeast. The mineralization appears to be concordant with bedding.

Very good gold grades were obtained from the channel sampling along the mineralized zones, confirming the extension of the mineralization towards the north. The western mineralization appears to be bounded by a steep east dipping breccia zone and it is possible that the breccia zone could be a splay from the SFZ, and a possible controlling structure for mineralization.

#### **9.1.3 FE2 Trench**

A dewatering trench located on the east side of the FE2 pit was mapped and sampled. The main lithology is metapelite overlain by pisolitic gravel. Thin sub-vertical quartz veins are present. Channel sampling returned assay results of less than 0.1 g Au/t indicating no oxide potential.

## 9.2 Sampling

Trench sampling was conducted in 2014 over the high grade quartz breccias at FE4 SE. Sampling results suggest that the north-northeast trending quartz veinlets/breccias do not carry gold mineralization. Trenches were dug 1.5 m deep, with one side wall sampled at 1 m intervals or where a change in lithology occurred.

Between 2010 and 2012, termite mound sampling was conducted over the Sadiola Mining Permit. Additional samples were collected on the neighboring S2 off-lease area in 2014. Termite mound sampling is believed to highlight a better geochemistry signature than conventional soil sampling. Samples were taken over a 50 m by 50 m grid and submitted to SGS for analysis, where low level gold analyses in parts per billion (ppb) were performed. In 2014 the exploration department commenced x-ray fluorescence (XRF) analysis of the reject pulps from the termite mound samples to compile a multi-element database. Elements analyzed for included arsenic, antimony and copper as these are good pathfinders for locating potential gold anomalies.

Analysis of historical termite mound sampling via multiple element analysis from XRF indicates a prominent arsenic anomaly along the FE trend (SEMOS, 2015c).

## 9.3 Geophysics

The following geophysical surveys have been undertaken on the Sadiola deposit since the late 1990s:

- Total magnetic intensity;
- EM by Spectrem (1999);
- Ground gravity (2003);
- Ground penetrating radar (GPR) by Terrateo Geoservices (2008);
- Airborne Electromagnetic (AEM) survey and radiometric survey (2009);
- Ground gravity survey by Gregory Symons Geophysics Contractors (2010 and 2011);
- Induced polarization (IP) gradient array, dipole-dipole and downhole IP by SAGAX Geophysique Inc. (SAGAX) (2011 and 2012).

The AEM and radiometric survey were very detailed and from these, several previously unidentified structural trends were identified.

The use of GPR technology showed very little potential, with a maximum depth of 15 m being attained.

The Sadiola style of mineralization is a good IP target due to the presence of sulphides, silicification and gold. The IP survey involved dipole spacing at 50 m and reading channels 1 to 10. The downhole IP was concentrated on two diamond drillholes at Tambali. Both holes had elevated gold grades in the deeper sulphide. Only 35% of the Sadiola Mining Permit is covered by IP.

No geophysical surveys have been undertaken since 2012. However, a new cover-weathering thickness grid was derived for Sadiola using the 1999 Spectrem AEM surveys. The comparison of the new grid to the drilling data shows a reasonable to good correlation, with areas of deeper weathering and/or thicker conductive cover (transported material) being readily identifiable. The thickness grid is used to estimate the effectiveness of soil and termite mound sampling in certain regions.

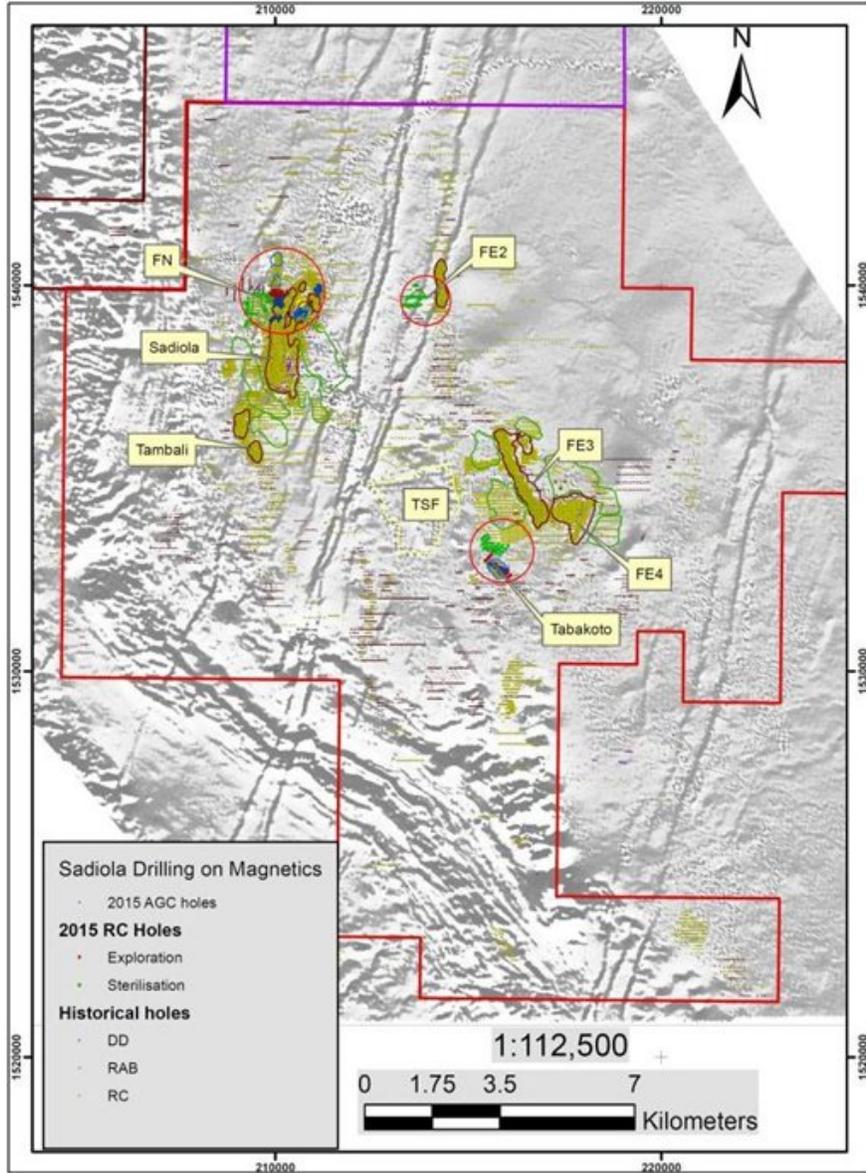
High resolution magnetic derivatives were used to map smaller veins, fractures and faults and to define similar favorable areas with cross cutting relationships which may be associated with economic gold concentrations. These were used to identify additional exploration targets in 2013.

**10. DRILLING**

**10.1 Extent of Drilling**

Exploration drilling has been conducted over the entire Sadiola permit area since the early 1990s, as shown in Figure 10.1.

**Figure 10.1: Location of Drillholes Collars**



Source: SEMOS, 2016

Rotary air blast (RAB) holes were drilled over various exploration targets before 2009. These holes were typically shallow (less than 50 m) to test the oxide mineralization just below the laterite interface.

Between 2010 and 2013, geological exploration drill sampling was undertaken using RC or DD as summarized in Table 10.1.

**Table 10.1: Exploration Drilling since 2010**

Year	RC	DD	Length (m)
2010	43,871	15,208	59,079
2011	68,184	3,628	71,812
2012	47,418	10,043	57,461
2013	28,038	-	28,038
2014	20,536	-	20,536
2015	13,110	-	13,110

Source: AGA, 2015

Note: The 2015 exploration drilling program included an additional 628 m of sterilisation drilling, which has not been included in the total drilled metres for 2015.

Most of the DD was done in and around the mining areas of Sadiola, FE and Tambali pits. No DD has been undertaken since 2013; all exploration drilling since 2013 has been undertaken using RC drilling.

In 2015 a total of 13,110 m of RC drilling was achieved focusing on the area to the north of the Sadiola Main Pit and the Tabakoto satellite deposit. In addition, 4,350 m of RC drilling was undertaken to upgrade the Mineral Resource on the northern extension of the Sadiola main pit. An additional 3,632 m of RC drilling was undertaken to define the north-northeast to northeast trending shears that occur in the Sadiola north area (SEMOS, 2015c).

At Tabakoto 2,874 m of RC exploration holes were drilled to infill the predominantly Inferred Mineral Resource. An additional 1,626 m of definition drilling was completed on the northern and southern extension of the mineralization trend (SEMOS, 2015c).

Table 10.2 provides a summary of all exploration drilling undertaken at Sadiola since 1993.

Table 10.2: Summary of Sadiola Exploration Drilling

Year		Hole Type DD			Hole Type RAB	Hole Type TR	Hole Type RC	CH	RC with HQ Finish	RC with HTW
		UNK	HQ	NQ						
1993	No. of holes drilled	6								
	Total meters	620.10								
	Average depth	103.35								
	Recovery % (average)									
1994	No. of holes drilled	7			1					
	Total meters	517.65			72.05					
	Average depth	73.95			72.05					
	Recovery % (average)									
1995	No. of holes drilled	12								
	Total meters	1,337.00								
	Average depth	111.42								
	Recovery % (average)									
1996	No. of holes drilled	2								
	Total meters	247.35								
	Average depth	123.68								
	Recovery % (average)									
1997	No. of holes drilled	2	7	5			40			
	Total meters	900.50	3,667.00	3,166.00			4,000.20			
	Average depth	450.25	523.86	633.20			100.01			
	Recovery % (average)									

Year		Hole Type DD			Hole Type RAB	Hole Type TR	Hole Type RC	CH	RC with HQ Finish	RC with HTW
		UNK	HQ	NQ						
1998	No. of holes drilled	548	1	8		2	286			
	Total meters	57,739.92	720.00	4,751.00		52.30	16,247.00			
	Average depth	105.36	720.00	593.88		26.15	56.81			
	Recovery % (average)									
1999	No. of holes drilled	5					363			
	Total meters	609.60					24,524.00			
	Average depth	121.92					67.56			
	Recovery % (average)									
2000	No. of holes drilled		1	10	685		1638		11	
	Total meters		198.00	5061.20	21,740.30		84,004.10		1,273.80	
	Average depth		198.00	506.12	31.74		51.28		115.80	
	Recovery % (average)									
2001	No. of holes drilled	18	1	25	499		1,566		21	1
	Total meters	4,479.85	501.00	8,032.90	16,212.50		97,181.70		1,923.00	267.20
	Average depth	248.88	501.00	321.32	32.49		62.06		91.57	267.20
	Recovery % (average)									
2002	No. of holes drilled	25		29	371		864		29	
	Total meters	5,581.35		12,345.50	13,366.00		82,688.50		2,008.00	
	Average depth	223.25		425.71	36.03		95.70		69.24	
	Recovery % (average)									
2003	No. of holes drilled	8		72	206		935		94	
	Total meters	2,213.30		21,375.30	8,561.00		83,896.30		10,663.05	
	Average depth	276.66		296.88	41.56		89.73		113.44	
	Recovery % (average)									

Year		Hole Type DD			Hole Type RAB	Hole Type TR	Hole Type RC	CH	RC with HQ Finish	RC with HTW
		UNK	HQ	NQ						
2004	No. of holes drilled	2	54			940		82		
	Total meters	153.27	23,143.20			74,786.70		11,453.00		
	Average depth	76.64	428.58			79.56		139.67		
	Recovery % (average)									
2005	No. of holes drilled	4	2	4	783	452		4		
	Total meters	552.50	214.50	1,059.40	30,998.00	35,860.00		265.00		
	Average depth	138.13	107.25	264.85	39.59	79.34		66.25		
	Recovery % (average)									
2006	No. of holes drilled	20		3	128	129		25		
	Total meters	1,470.30		585.10	4,458.00	12,492.00		3,405.00		
	Average depth	73.52		195.03	34.83	96.84		136.20		
	Recovery % (average)				100.00					
2007	No. of holes drilled	4	46	12		529		49		
	Total meters	430.00	8,132.00	3,899.60		38,982.00		7,864.00		
	Average depth	107.50	176.78	324.97		73.69		160.49		
	Recovery % (average)									
2008	No. of holes drilled	8	12	22	379	318				
	Total meters	1,873.00	1,272.50	9,201.00	16,591.00	22,057.00				
	Average depth	234.13	106.04	418.23	43.78	69.36				
	Recovery % (average)	100.00	100.00	100.00						
2009	No. of holes drilled		24			300				
	Total meters		8,028.00			31,496.00				
	Average depth		334.50			104.99				
	Recovery % (average)		100.00							

Year		Hole Type DD			Hole Type RAB	Hole Type TR	Hole Type RC	CH	RC with HQ Finish	RC with HTW
		UNK	HQ	NQ						
2010	No. of holes drilled	2	41			530		1		
	Total meters	535.00	9,962.40			65,950.00		246.00		
	Average depth	267.50	242.99			124.43		246.00		
	Recovery % (average)	75.33	97.84							
2011	No. of holes drilled	6	10	2		610				
	Total meters	1,209.00	1,952.00	349.00		72,174.00				
	Average depth	201.50	195.20	174.50		118.32				
	Recovery % (average)	81.10	78.47	78.20						
2012	No. of holes drilled	7	11			394				
	Total meters	2,556.00	6,016.00			44,460.00				
	Average depth	365.14	546.91			112.84				
	Recovery % (average)		96.92							
2013	No. of holes drilled		5			407				
	Total meters		874.00		9	39,506.00				
	Average depth		174.80		2,477.00	97.07				
	Recovery % (average)				275.22					
2014	No. of holes drilled				6	194	48			
	Total meters				66.50	20,768.00	4,038.00			
	Average depth				11.08	107.05	84.13			
	Recovery % (average)					N/A				
2015	No. of holes drilled					137				
	Total meters					13,110				
	Average depth					103.48				
	Recovery % (average)					N/A				

Source: AGA, 2015

**10.2 Diamond Drilling**

Diamond core drilling uses the conventional wireline method. Generally, the upper weathered zone of drillholes is cored using PQ or HQ core diameter with NQ core diameter being used once competent rock is encountered. Double and triple tube core barrels are used to capture the soft, friable, saprolite material. Core recoveries are measured and recorded in the drill logs. Table 10.3 provides an indication of average core recovery per year per weathering zone.

The current drilling contractor for exploration is Boart Longyear (since 2006). Prior to 2006, West African Drilling Services conducted this activity.

**Table 10.3: Annual ore Recoveries by Weathering Category**

Year	Hole Type	Redox	Average Recovery	Material Type
2008	DD	1	100.00%	Oxide
2009	DD	1	100.00%	Oxide
2010	DD	1	91.68%	Oxide
2011	DD	1	71.90%	Oxide
2012	DD	1	78.53%	Oxide
2013	DD	1	88.89%	Oxide
2008	DD	2	100.00%	Transition
2009	DD	2	100.00%	Transition
2010	DD	2	96.30%	Transition
2011	DD	2	30.91%	Transition
2013	DD	2	91.52%	Transition
2008	DD	3	100.00%	Fresh rock
2009	DD	3	100.00%	Fresh rock
2010	DD	3	97.95%	Fresh rock
2011	DD	3	87.70%	Fresh rock
2012	DD	3	96.94%	Fresh rock
2013	DD	3	96.76%	Fresh rock

Source: AGA, 2015

Since 2013 no further core has been drilled; all exploration, grade control and advanced grade control drilling is undertaken using RC methods.

All collar locations are surveyed to within 0.20 m by differential global positioning system (GPS) and the collar locations utilized for subsequent natural surface determinations by the SEMOS mine survey team. The mine grid has been independently verified as being accurate and all phases of drilling have been

surveyed. Holes are also surveyed downhole with a Reflex survey tool which provides azimuth, dip and magnetic readings for each sample point. Survey data is collected on average every 30 m downhole.

An ACE drilling tool is used to obtain oriented core for diamond drillholes.

### **10.3 Reverse Circulation Drilling**

RC drilling is undertaken using 115 mm dual tube drill rods fitted with a tungsten carbide drag bit. Compressed air is fed down the rod's annulus while the returning air carrying the freshly drilled sample is exhausted directly to surface up the center sample tube where it is captured via a cyclone. The RC face hammers used bit sizes ranging from 124 mm to 140 mm depending on ground conditions, using either a Schramm 685s or KWL 1600 drill rig with 500 psi compressors.

RC drill chips are logged on the drill site by geologists using coded field sheets. Chips for every 2 m drilled, are stored as screened (+2 mm for coarser material) and unscreened (mainly fine saprolitic material in the SEMOS context) portions in plastic chip trays. Wet samples and poor or no recovery samples are also recorded. RC drilling is stopped if insufficient air flow is recorded through the cyclone resulting in poor sample recovery.

RC sample recovery is routinely measured and reported on a monthly basis during grade control. Grade control holes are drilled with RC Drilltech D45KS rigs providing a 5-inch hole.

The sample splitter is cleaned with compressed air between each sample.

Density is measured from the grade control holes using a downhole gamma ray density probe.

### **10.4 2015 Drilling Results**

The following section has been reproduced, with minor modifications from SEMOS (2015c).

#### **10.4.1 Sadiola North (FN)**

RC drilling of 4,350 m was completed in 2015 to upgrade the Mineral Resource on the northern extension of the Sadiola Main Pit.

The Sadiola north area is characterized by mineralization contained within north-northeast to northeast trending shears and 3,632 m of RC drilling was completed to define shear extensions. These results together with information from pit mapping were used to update the geological model for the area.

Additional infill targets were generated from the new resource model for Sadiola north and an additional 628 m of follow-up drilling was undertaken. It appears that the identified target areas have low oxide potential; however, the drilling confirmed the presence of low grade sulphide mineralization along north-northeast structures.

#### **10.4.2            Tabakoto**

At Tabakoto 2,874 m of RC were drilled to upgrade the predominantly Inferred Resource model. An additional 1,626 m of definition drilling was completed on the northern and southern extension of the northwest trending mineralization.

The drilling campaign confirmed the deep weathering and mineralization associated with weathered carbonate. A review of the Tabakoto geology model suggests that the mineralization is controlled by steep northwest trending structures above which the laterite mineralization is located. There is a possibility of north-northeast to northeast cross-cutting structures consistent with the regional cleavage observed at the FE3/4 pits.

Results from the strike extension drilling indicate that there is oxide potential toward the northwest of the S12 target. The potential for significant mineralization towards the southeast is considered to be very low.

#### **10.4.3            Waste Dump Sterilization**

During 2015, 8,358 m of sterilization drilling was completed:

- At the FE2 pit the waste dump area was covered by 2,166 m of RC drilling;
- Mining at Sadiola north is scheduled to commence in early 2016 and 3,006 m of sterilization RC drilling was completed for the waste dump;
- The waste dump position at Tabakoto was also sterilized with 3,186 m of drilling.

In all cases, isolated very low grade oxide intersections were returned.

## **11. SAMPLE PREPARATION, ANALYSES AND SECURITY**

### **11.1 Sampling Method**

#### **11.1.1 Reverse Circulation**

The majority of the samples used in the Mineral Resource comprise exploration and grade control drill chips from RC drilling. Air core drilling was used in grade control and some exploration holes prior to 2010 but was not used for Mineral Resource estimation purposes.

Exploration RC rigs are fitted with cyclones providing routine samples at 2 m intervals. Subsamples are split at the rig using a three-tiered riffle splitter which yields a 2 kg to 2.5 kg field sample.

When RC samples are too wet to pass through the riffle splitters, they are dried in an oven overnight and are then split using the same three-tier riffle system. Drilling is normally stopped when the samples become too wet. Wet samples are flagged in the database.

Grade control holes are also sampled on a 2 m basis using the Sandvik Rotaport sampling system, which comprises a rotary cone splitter below the cyclone to produce an automatic subsample of 2 kg to 2.5 kg.

The following measures are taken in order to reduce the chance of sample contamination:

- The small sample bags are tied so as to prevent any contamination;
- The cyclone is scraped and cleaned every 30 m or when excessive water has passed through the cyclone. The grade control rigs have an automatic cleaning system for the cyclone using compressed air. The exploration rig cyclones are cleaned manually;
- The splitter is cleaned using compressed air after every 2 m sample has passed through the splitter. Scrapers and/or brushes are also used to remove any material that sticks in the splitter fins.

At the drill rig, individual samples are placed in large plastic bags with the list of samples contained in the bag. Samples are transported by the Exploration Section to the assay laboratory in a covered vehicle. At the laboratory the large bags are opened and the number of samples reconciled against the list.

The RC sampling recoveries are being measured for advanced grade control drilling. Exploration RC drilling recoveries are currently not measured due to the high proportion of wet samples being intersected. The Sadiola team is currently investigating how a recovery procedure may be implemented.

Recoveries measured for the 2015 AGC drilling program are:

FN area: 89.7%;

Tabakoto: 93%.

### 11.1.2 **Diamond Core**

Core from the diamond holes are also used in resource evaluation but no core drilling has taken place since December 2012. Core is logged and split in half by diamond saw along the orientated core axis. One half is bagged and dispatched for density determination and assaying, while the other half remains in the core tray as a record and for check sampling if required. The sampling interval for diamond core is 1 m, with geological boundaries being honored.

All core has been geologically and geotechnically logged with emphasis on structural measurements, alteration, lithology and mineralization (SEMOS, 2016a).

During 2013, a total of 90,000 m of core was imaged with a specially built hyperspectral scanner to produce shortwave infrared and color images of the core. A subset of 28,624 m (which included 232 randomly selected drillholes) was further processed to produce special features extractions and mineral presence maps to qualify alteration minerals for use in mining and geo-metallurgical studies. Images and quantitative data are stored as a subset of the geological database (SEMOS, 2016a).

### 11.1.3 **Assay Laboratories**

The laboratories used by Sadiola are listed below:

- SEMOS: On-site laboratory owned and operated by SEMOS. All exploration and grade control samples collected in 2014 and 2015 were submitted to this laboratory. The laboratory is not accredited. The laboratory assays include fire assay and aqua regia, along with moisture and density determinations. A 30 g aliquot is routinely used for fire assaying. The laboratory participates in the biannual Geostats Survey of International Laboratories;
- SGS Bamako: This is an independent laboratory located in Bamako, Mali. It was accredited by the South African National Accreditation System (SANAS) with International Organization for Standardization and International Electrotechnical Commission (ISO/IEC) 17025:2005, on September 7, 2015. The laboratory's unique accreditation number is T0652. During 2015, the laboratory acted as referee lab for the annual check assay as part of the quality control process;
- SGS Kayes: All samples from pre-2013 (exploration RC chips, diamond core and soil samples) were submitted to SGS Kayes which closed down at the end of 2013. SGS uses a 30 g aliquot for sulphide material and 50 g for oxide material in the fire assay. The laboratory participated in the biannual Geostats Survey of International laboratories, but was not accredited;

- SGS South Africa – Booyens: This laboratory was not used during 2014, but has been routinely used as an umpire laboratory for processing check assays. It is accredited by SANAS under ISO/IEC 17021:2011.

The assay lab was not visited during the site visit as no sample processing was being undertaken at the time.

## 11.2 **Sample Preparation and Assaying**

Since 2007, all samples are processed at one of the laboratories mentioned above. Prior to 2007, samples were prepared at the Sadiola exploration core yard. At the laboratory, drill core and RC cuttings (where necessary) are placed in an oven until dry (typically for eight hours), then passed through a jaw crusher which reduces the maximum particle size to 6 mm. The jaw crusher is cleaned using compressed air followed by a blank "flush", following each sample, or two samples, depending on the nature of the material being processed.

When undertaking fire assay, a riffle splitter is used to reduce the sample size to approximately 500 g which is then pulverized for a minimum of three minutes in a Labtech LM2 chrome steel pulverizer to achieve 90% passing 75 µm. The pulverizer is cleaned using a brush, followed by compressed air aspiration. This is then followed by a blank "flush", following each sample, or two samples, depending on the nature of the material being processed.

Depending on the laboratory and material type, 30 g of material is extracted for fire assay. Prior to 2015 sub sampling was undertaken using a manual method that comprised:

- Shaking;
- Spreading;
- Dividing;
- Spooning.

Since October 2015, sub sampling has been undertaken using a cascade rotary splitter.

The gold concentration is determined by Atomic Absorption Spectroscopy (AAS) with a minimum detection limit equivalent to 0.005 g Au/t. All samples that return grades in excess of 5 g Au/t are re-assayed using a gravimetric finish.

Rejects and pulps are kept in the SEMOS exploration core yard until all QAQC results (including third party lab checks) have been evaluated. The rejects from the exploration drilling are stored indefinitely, whereas the rejects from the grade control drilling are discarded after 30 days.

### 11.3 Quality Assurance and Quality Control

The QAQC measures utilized at Sadiola include the routine insertion of quality control (QC) samples into the sample stream as well as independent laboratory audits and job observations. The independent laboratory audits are undertaken by Assay Tech Inc. (South Africa).

QC material comprised Standard Reference Materials (SRMs), also known as Certified Reference Materials (CRMs), blanks, field and pulp duplicates and check assays.

These programs are run in addition to the normal QC insertions and monitoring undertaken in-house by the SEMOS and SGS Bamako laboratories.

### 11.4 Quality Control Samples

QC samples are assigned fixed positions within the sampling sequence by geologists. Prior to 2014, SEMOS grade control delivered the QC samples separately to the laboratory. The QC insertion rate has been streamlined and standardized according to AGA's recommended levels (Table 11.1).

**Table 11.1: Type and Insertion of QC Samples**

QAQC Material	Type	Amount	
		Grade Control (%)	Exploration (%)
Blanks	Coarse and pulp	1	5
Standards	CRM/SRM	2	5
Duplicates	Rig/Pulp	1	5
Check assays	Pulp	5% blank (minimum of 10) 5% CRM/SRM (minimum of 10)	5

Source: AGA, 2015

The assay laboratories insert their own QAQC materials and make the results available to SEMOS through their Laboratory Information Management System (LIMS). The results of the assay laboratory QAQC are not stored in the Geological Database Management System (GDMS).

QC results are monitored by SEMOS as part of the assay data validation process during the data loading. Sample submissions falling outside of acceptable rejection limits are investigated and resubmitted for re-assay if necessary. The assay results loading and feedback to the laboratory is typically completed within

24 hours after receipt of results. The internal laboratory standard results are also available to users through the LIMS query tool and are utilized when determining the pass/failure of a sample submission.

The pulp QC sample mass was reduced from 150 g to 30 g in 2014. The reason for the reduction was to supply the mass required for fire assaying (30 g); therefore, eliminating the possibility for test runs by the laboratory and reducing processing cost.

Mass loss measurements, where a sample mass is taken before and after crushing and milling, were introduced and implemented in October 2014 by the SEMOS laboratory. The test is undertaken at a rate of one in 20 samples. Mass measurement gives an indication of mass loss in the sample preparation steps and is part of the laboratory internal QAQC procedure to minimize sample loss, maintain representativeness and avoid introduction of bias. The maximum percentage loss during a processing step is 2%. Mass balance analysis is also being introduced at the sampling phase to determine the sample recovery.

A monthly QAQC report is produced by Sadiola according to QAQC guideline Rev 1.05 (AGA, 2014d) in which the QAQC activities for all labs are reported. An annual report is also published which includes the referee lab results and lab audits.

## **11.5 Certified Reference Material s**

The CRMs are commercially certified standards with a variety of gold grade ranges, purchased in bulk jars from Rocklabs Limited. Since October 2015, a rotary splitter was used to repackage the CRMs into 35 g sachets. Pre-2015, a total of 58 CRMs have been used at Sadiola, 34 of which were oxide CRMs and 22 were sulphide CRMs. In 2015, 26 CRMs were used, 18 of which were oxide and eight were sulphide.

### **11.5.1 Pre-2015**

A variety of CRMs have been used prior to 2015 as part of Sadiola's independent QAQC program. The CRM grades range from 0.0836 g/t to 18.34 g/t (Figure 11.2), providing a sufficient range of values to determine the accuracy of the assays undertaken.

Almost 95% of the CRM results are within the two standard deviation control limits and 99% are within the three standard deviation limits, indicating good analytical accuracy and precision has been achieved during assaying prior to 2015. The average of assays for each CRM is typically within 5% relative difference, indicating good analytical accuracy and low analytical bias. However, it is noted that the five highest grade CRMs, with expected grades greater than 8 g/t, display a consistent negative bias, although –the bias is still within 5% relative difference. This suggests that higher grade values may have been underestimated slightly in the assay process (Table 11.2). Snowden does not believe that this bias is material to the Mineral Resource estimate.

In Snowden's opinion, the overall analytical accuracy of the CRMs is satisfactory, especially post-2009; and it will not adversely affect the accuracy of the resource estimates.

#### **11.5.2            2015 Analysis**

SEMOS used an array of CRMs with grades ranging from 0.0826 to 18.34 g/t for the exploration and grade control QAQC programs. As part of the exploration QAQC, 13 oxide and four sulphide CRMs were submitted with the samples. In addition, a total of 10 oxide and five sulphide CRMs were submitted for the grade control QAQC program. CRMs with one assay were excluded from the statistical analyses presented below Table 11.3 and Table 11.4).

The statistical analysis of the CRMs indicates an acceptable accuracy for the exploration samples and good accuracy for the grade control samples (Table 11.3 and Table 11.4).

Table 11.2: Summary of Exploration CRM Analyses

CRM	Certified Mean Value	Mean of Samples	No. of Samples	Bias1 %	% of Samples within 2STDEV	% of Samples within 3STDEV	Date of Analysis	Comments
OXA89	0.08	0.10	591	15%	36	99	Aug 2013 to Nov 2014	Poor precision and positive bias
OXA71	0.08	0.31	4	0.96%	75	75	Jan 2013, May 2014	Positive bias due to one value plotting outside control limits
G306-1	0.41	0.41	5	0%	100	100	14-Nov	Good precision, good accuracy
OXD73	0.42	0.41	17	-2%	76	100	Dec 2009, Feb 2013 to Oct 2013	Good accuracy, precision needs to be improved
OXD87	0.42	0.41	61	-1%	97	100	April 2014 to Nov 2014	Good precision, good accuracy
G909-10	0.52	0.49	5	-4%	100	100	14-Nov	Good precision, good accuracy
SE29	0.59	0.60	62	1%	97	100	Mar 2013 to Sep 2013	Good precision, good accuracy
SE68	0.59	0.61	166	0%	99	99	Sep 2013 to May 2014; Dec 2014	Good precision, good accuracy
OXE106	0.61	0.61	118	0%	95	100	Jan 2014 to May 2014; Oct 2014	Good accuracy, good precision
SE44	0.61	0.62	5	2%	60	100	Jan 2011 to Sep 2013	Precision below acceptable levels, good accuracy
SE58	0.61	0.60	144	-1%	98	98	Dec 1999 to Sep 2013; mostly Nov 2011 to Sep 2013	Good precision, good accuracy
OXE86	0.61	0.61	110	-1%	100	100	Oct 2011 to Nov 2011; Mar 2014 to Apr 2014	Good precision, good accuracy
OXE74	0.62	0.60	11	-2%	91	91	Dec 2009, Jan 2011; Feb 2013; Dec 2013	Precision slightly lower than desired, good accuracy
OXF53	0.81	0.75	13	-8%	46	77	Feb 2013 to Mar 2013; Aug 2013; Nov 2014	Poor precision, minor negative bias
OXF41	0.82	0.87	31	6%	94	94	June 2012, Mar 2013 to Apr 2013	Acceptable precision, minor positive bias
OXG104	0.93	0.93	169	1%	99	100	Oct 2013 to Aug 2014	Good precision, good accuracy
OXG99	0.93	0.97	16	4%	63	100	Oct 2014; Dec 2014	Poor precision, acceptable accuracy

1 % relative difference

CRM	Certified Mean Value	Mean of Samples	No. of Samples	Bias1 %	% of Samples within 2STDEV	% of Samples within 3STDEV	Date of Analysis	Comments
OXG83	1.00	0.98	60	-1%	100	100	May 2013 to July 2013; Sep 2014 to Dec 2014	Good precision, good accuracy
OXG98	1.02	1.00	380	-1%	98	99	Most Dec 2012 to April 2013	Good precision, good accuracy
OXG46	1.04	1.02	34	-2%	100	100	Feb 2013 to Nov 2013	Good precision, good accuracy
OXH66	1.29	1.30	21	1%	100	100	13-Jul	Good precision, good accuracy
OXH37	1.29	1.25	15	-3%	80	80	Feb 2013 to Mar 2013	Poor precision, generally slight positive bias, but one failing value shows mean as negative bias
OXH52	1.29	1.30	13	1%	100	100	Jan 2013 to Nov 2014	Good precision
SH41	1.34	1.32	86	-1%	100	100	Oct 2012 to Dec 2012	Good precision, good accuracy.
SI54	1.78	1.81	4	2%	100	100	Jan 2013; May 2013	Good precision, good accuracy
Si64	1.78	1.79	65	1%	100	100	Mar 2014 to Dec 2014	Good precision, good accuracy
OXI96	1.80	1.81	611	0%	99	100	Sep 2013 to Dec 2014	Good precision, good accuracy
OXI81	1.81	1.81	77	0%	100	100	Apr 2012 to Jun 2012; Nov 2012; Apr 2013 to Jul 2013	Good precision, good accuracy
OXI67	1.82	1.77	30	-2%	97	97	Jun 2012; Mar 2013 to Apr 2013; Dec 2013	Good precision, good accuracy
OXI54	1.87	1.81	5	-3%	100	100	Feb 2013 to Dec 2013	Good precision, good accuracy
OXJ95	2.34	2.34	948	0%	99	99	2006 to 2014; most of the data analyzed between Dec 2012 to May 2014	Good precision, good accuracy
OXJ64	2.37	2.30	13	-3%	92	92	Feb 2013 to Mar 2013; Nov 2014	Precision slightly lower than desired, good accuracy
OXJ47	2.38	2.35		-1%	100	100	May 2013; Jul 2013	Good precision, good accuracy
OXJ36	2.39	2.45	10	2%	90	90	Feb 2013 to Mar 2013; Dec 2013	Good accuracy
SJ63	2.63	2.65	135	1%	100	100	Sep 2013 to Sep 2014	Good precision, good accuracy
SJ53	2.64	2.63	97	0%	100	100	Most analyzed between Oct 2012 and Dec 2012	Good precision, good accuracy

CRM	Certified Mean Value	Mean of Samples	No. of Samples	Bias1 %	% of Samples within 2STDEV	% of Samples within 3STDEV	Date of Analysis	Comments
SJ39	2.64	2.65	81	0%	98	98	Mar 2013; mostly Sep 2014 to Dec 2014	Acceptable precision. Most of the samples show slight negative bias, the failing samples balance off the mean
OXK35	3.49	3.53	2	1%	100	100	13-Feb	Good precision, good accuracy
OXK79	3.53	3.60	28	2%	96	96	Apr 2012 to Nov 2012	Precision acceptable, good accuracy
OXK94	3.56	3.63	536	2%	99	99	Dec 2009 to Nov 2014; mostly between late 2012 to 2013	Good precision, good accuracy
OXK69	3.58	3.43	17	-4%	88	94	Dec 2009 to Nov 2014	Precision below acceptable levels, good accuracy
OXK110	3.60	3.60	118	0%	100	100	Sep 2013 to Mar 2014	Good precision, good accuracy
SK62	4.08	4.08	576	0%	99	99	Most between Mar 2012 and Sep 2013	Good precision, good accuracy
SK43	4.09	8.62	7	0	57	57	Dec 2009 and Jan 2013	Poor precision, positive bias
SK52	4.11	4.02	55	-2%	98	98	Most between Oct 2012 and Nov 2012	Good precision, good accuracy
OXL34	5.76	6.00	27	2 %	81	100	Aug 2014 to Sep 2014	Precision slightly lower than desired, good accuracy
OXL93	5.84	5.95	41	2%	100	100	Aug 2013 to Sep 2013	Good precision, good accuracy
OXL51	5.85	5.12	8	-12%	88	88	Jan 2013 to Mar 2013, with one in Nov 2013	Poor precision, negative bias
SL46	5.87	5.84	2	0%	100	100	11-Jan	Good precision, good accuracy
OXL78	5.88	6.03	18	3%	94	94	Feb 2013 to Mar 2013; Dec 2013	Precision slightly lower than desired, good accuracy
SL51	5.91	5.91	12	0%	100	100	Apr 2012 to Jan 2013	Good precision, good accuracy
OXN33	7.38	7.16	36	-3%	94	94	May 2012 to Dec 2013, but mostly between Feb 2013 and Mar 2013	Precision slightly lower than desired, good accuracy.
OXN92	7.64	7.76	895	2%	99	99	Nov 2011 to Nov 2013, most between Dec 2012 and Nov 2013	Good precision, good accuracy

CRM	Certified Mean Value	Mean of Samples	No. of Samples	Bias1 %	% of Samples within 2STDEV	% of Samples within 3STDEV	Date of Analysis	Comments
OXN62	7.71	7.66	3	-1%	100	100	Feb 2013 to June 2013	Good precision, good accuracy
SN38	8.57	7.79	10	-9%	70	70	Jan 2000 and Dec 2009	Poor precision, negative bias
SN60	8.59	8.63	796	0%	99	99	Most between Dec 2012 and Aug 2013	Good precision, good accuracy
OXP61	14.92	14.22	43	-5%	91	91	Most of the data Nov 2012 and Apr 2013	Precision slightly lower than desired, negative bias is due to failing values. Data inside the control lines don't show bias
SP59	18.12	16.97	109	-6%	94	94	Most in Jan 2012 and May 2013	Precision slightly lower than desired, negative bias
SP49	18.34	17.43	80	-5%	95	95	Most in June 2013	Acceptable precision, negative bias

Source: Snowden, 2015

Note: The data as originally supplied to Snowden reported accuracy to four decimal places. The data was then rounded to two decimal places, as this is believed to be a better reflection of accuracy and precision.

Table 11.3: Summary of 2015 Exploration CRM Analyses

CRM	Certified mean value	Mean of samples	No. of samples	Bias2 %	% of samples within 2STDEV	% of samples within 3STDEV	Comments
OXA-89	0.08	0.08	6	-4%	100	100	Good accuracy, good precision
OXD-27	0.42	0.42	4	1%	100	100	Good accuracy, good precision
G909-10	0.52	0.48	3	-8%	67	100	Three samples, no conclusions can be made
SE-68	0.60	0.60	23	0%	100	100	Good accuracy, good precision
OXE-106	0.61	0.61	48	1%	98	98	Good accuracy, sample swapping with a high grade sample. Good precision
OXF-53	0.81	0.79	14	-3%	93	93	Good accuracy. Acceptable precision
OXG-104	0.93	0.92	49	0%	94	94	Good accuracy. Acceptable precision
OXH-37	1.29	1.31	18	2%	100	100	Good accuracy. Good precision.
OXH-52	1.29	1.30	11	1%	100	100	Good accuracy. Good precision
SH-13	1.32	1.32	17	0%	94	100	Good accuracy. Good precision
SI-54	1.78	1.78	31	0%	81	93	Good accuracy. Poor precision due to sample swops
SI-64	1.78	1.79	2	0%	100	100	Good accuracy, good precision
OXI-96	1.80	1.80	3	0%	100	100	Good accuracy, good precision
OXJ-95	2.34	2.16	13	-8%	62	69	A slight negative bias and sample swapping with low grade samples

Source: Snowden, 2015

Note: The data as originally supplied to Snowden reported accuracy to four decimal places. The data was then rounded to two decimal places, as this is believed to be a better reflection of accuracy and precision.

<sup>2</sup> % relative difference

Table 11.4: Summary of 2015 GC CRM Analyses

CRM	Certified mean value	Mean of samples	No. of samples	Bias %	% of samples within 2STDEV	% of samples within 3STDEV	Comments
OXA-89	0.08	0.08	195	-1%	99	99	Good accuracy. Good precision
SE-68	0.60	0.60	7	1%	86	100	Good accuracy. Acceptable precision
OXE-106	0.61	0.59	216	-2%	96	99	Good accuracy. One sample swap. Good precision
OXG-104	0.93	0.94	165	2%	98	99	Good accuracy. One sample swap. Good precision
OXH-52	1.29	1.29	22	0%	100	100	Good accuracy. Good precision
OXI-40	1.86	1.85	56	0%	100	100	Good accuracy. Good precision
OXJ-64	2.37	2.38	57	0%	100	100	Good accuracy, good precision
OXJ-47	2.38	2.37	58	0%	100	100	Good accuracy. Good precision
OXJ-36	2.39	2.40	69	0%	94	96	Good accuracy. Acceptable precision
SJ-63	2.63	2.65	52	0%	100	100	Good accuracy. Good precision
OXH-35	3.49	3.53	57	1%	100	100	Good accuracy. Good precision
OXK-110	3.60	3.63	131	1%	95	95	Good accuracy. Acceptable precision
SL-61	5.93	5.97	73	1%	99	100	Good accuracy. Good precision
SP-49	18.34	18.23	37	-1%	100	100	Good accuracy. Good precision

Source: Snowden, 2015

Note: The data as originally supplied to Snowden reported accuracy to four decimal places. The data was then rounded to two decimal places, as this is believed to be a better reflection of accuracy and precision.

**11.6 Blanks**

Blank material is sourced from the Souroukoto sandstone located near the SEMOS village. This material has been used historically and has been demonstrated to be barren. The material is crushed to <6 mm and samples of approximately 1 kg were inserted into the exploration and grade control sample streams in 2015. Additionally, 35 g of pulverized blank material was inserted into the exploration sample stream.

**11.6.1 Pre-2015**

Ten types of blanks, comprising 40,351 analyses were used by Sadiola from 2000 to 2014 to determine if contamination was occurring during the sample preparation and assaying processes (Table 11.5).

The blank assays are plotted over time, with the plot indicating an upper limit. The upper limit was set to a nominal value of 10 times the SEMOS laboratory detection limit of 0.01 g/t; to account for the high analytical variability often observed close to a detection limit.

A total of 218 samples, equating to 0.5% of the blanks, plot above the upper limit (Figure 11.1 and Figure 11.2). This indicates that negligible contamination occurred during the sample preparation and analytical processes. In Snowden's opinion, this would not have adversely affected the overall accuracy of the samples used for estimates.

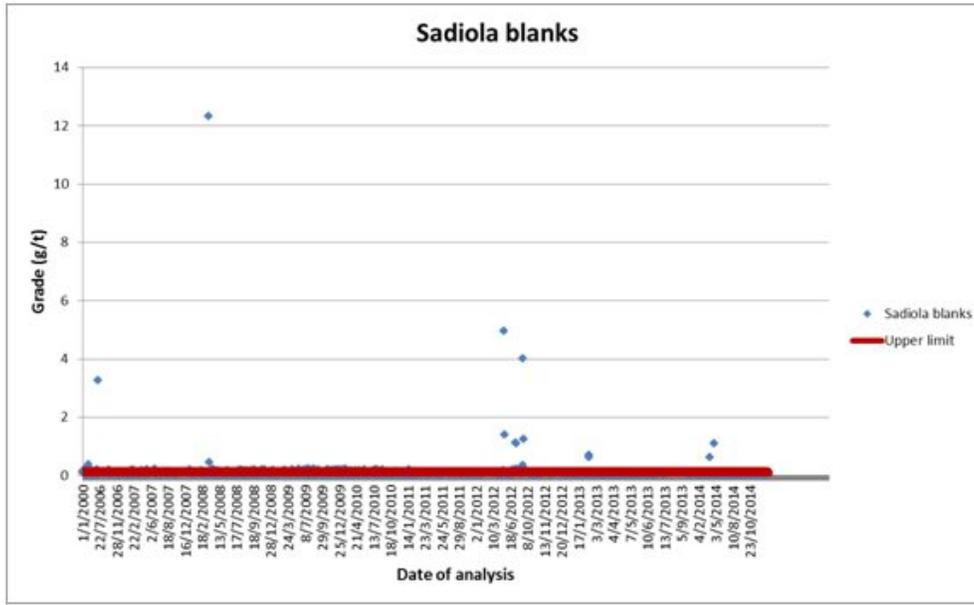
An improvement was noted in the lower grade ranges, from April 2011 to present (Table 11.5, Figure 11.1). This improvement coincides with the discontinuation of the CBLANKS ("coarse blanks"), which showed poorer results compared to other blank materials.

**Table 11.5: Summary of Blank Materials Used at Sadiola**

Blank material	Description
AuBlank33	Rocklabs pulp blank – certified material
BLANKA	Pulp blank accompanying samples destined for Analabs – prepared locally
BLANKS	Pulp blank accompanying samples destined for SEMOS – prepared locally
BLANKSn	Pulp blank accompanying samples destined for SEMOS (new set) – prepared locally At a point in time, a new set was prepared and coded differently with suffix "n"
CBLANKA	Coarse blank accompanying samples destined for Analabs – prepared locally
CBLANKAS	Coarse blank accompanying samples destined for Analabs-SGS – prepared locally
CBLANKASn	Coarse blank accompanying samples destined for Analabs-SGS (new set) – prepared locally At a point in time, a new set was prepared and coded differently with suffix "n"
CBLANKS	Coarse blank accompanying samples destined for SEMOS – prepared locally
CBLANKSn	Coarse blank accompanying samples destined for SEMOS (new set) – prepared locally At a point in time, a new set was prepared and coded differently with suffix "n"
Coarse Blank	Coarse blank

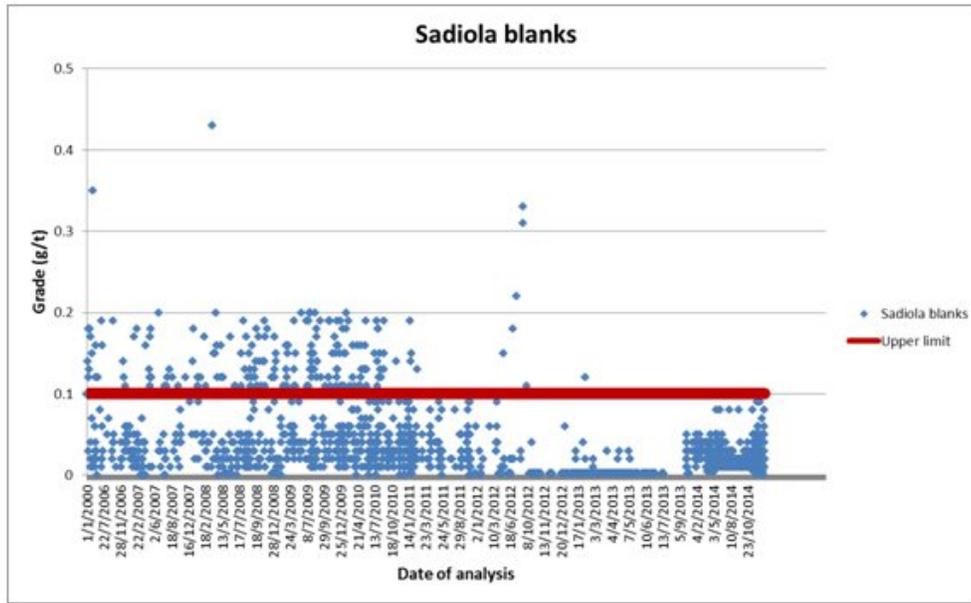
Source: AGA, 2015d

Figure 11.1: Sadiola Exploration Blank Assay Plots



Source: Snowden, 2015

Figure 11.2: Zoomed-in Sadiola Exploration Blank Assay Plots



Source: Snowden, 2015

Source: Snowden, 2015

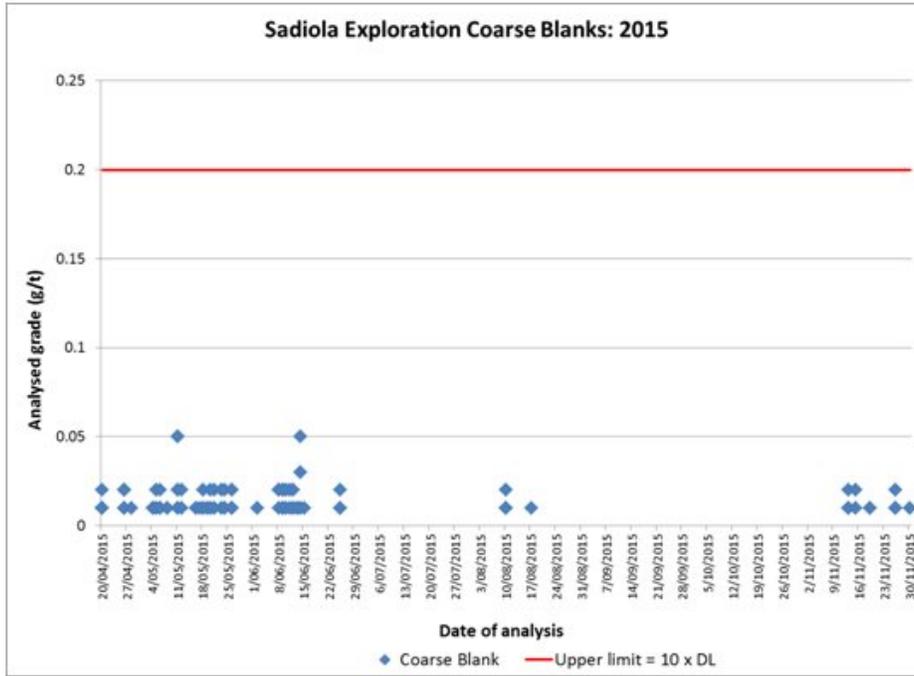
Note: The plot below is a zoom in of the plot above.

**11.6.2 2015 Analyses**

In 2015, only two types of blank samples were submitted with the samples. A total of 120 coarse blanks and 105 pulp blanks were submitted with the exploration samples. None of the blank samples plotted beyond the upper limit for blanks, which is set at 10 times the detection limit of 0.01 g/t.

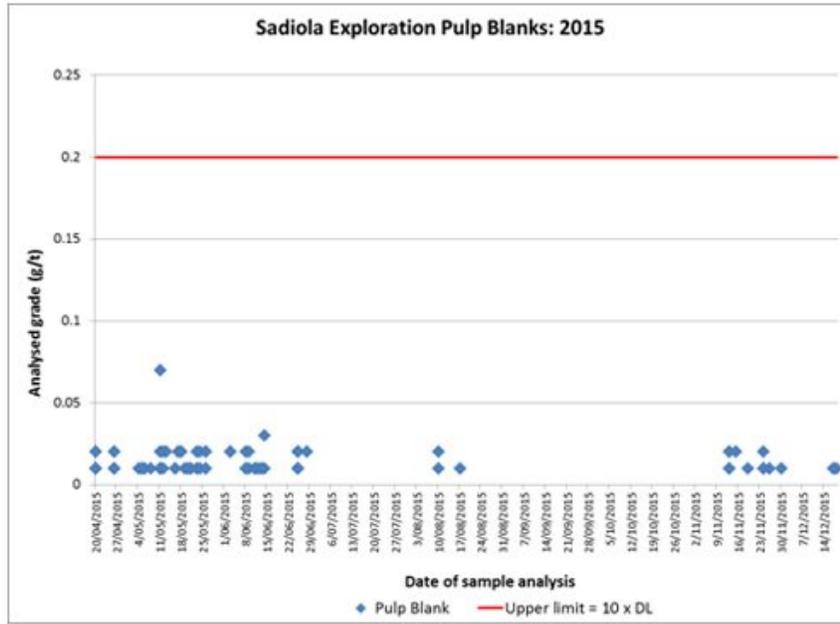
A total of 1,724 coarse blanks were submitted along with the grade control samples. Of these, only three samples plotted beyond the upper limit; this is equivalent to 0.17% failure rate. The QP is of the opinion that this is acceptable and indicates that cross-contamination is currently not an issue (Figure 11.3, Figure 11.4 and Figure 11.5).

**Figure 11.3: Sadiola 2015 Exploration Coarse Blank Assay Plots**



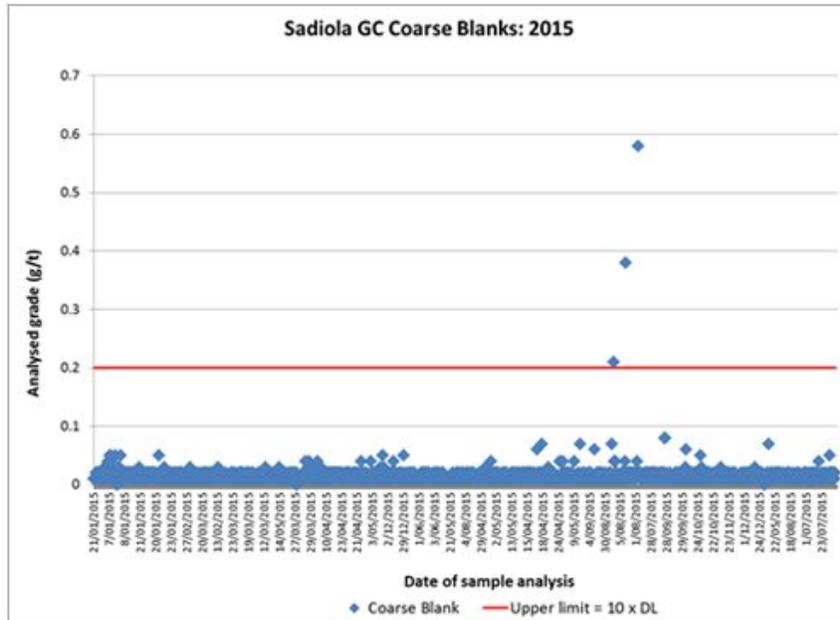
Source: Snowden, 2015

Figure 11.4: Sadiola 2015 Exploration Pulp Blank Assay Plots



Source: Snowden, 2015

Figure 11.5: Sadiola 2015 Grade Control Blank Assay Plot



Source: Snowden, 2015

## 11.7 **Field Duplicates**

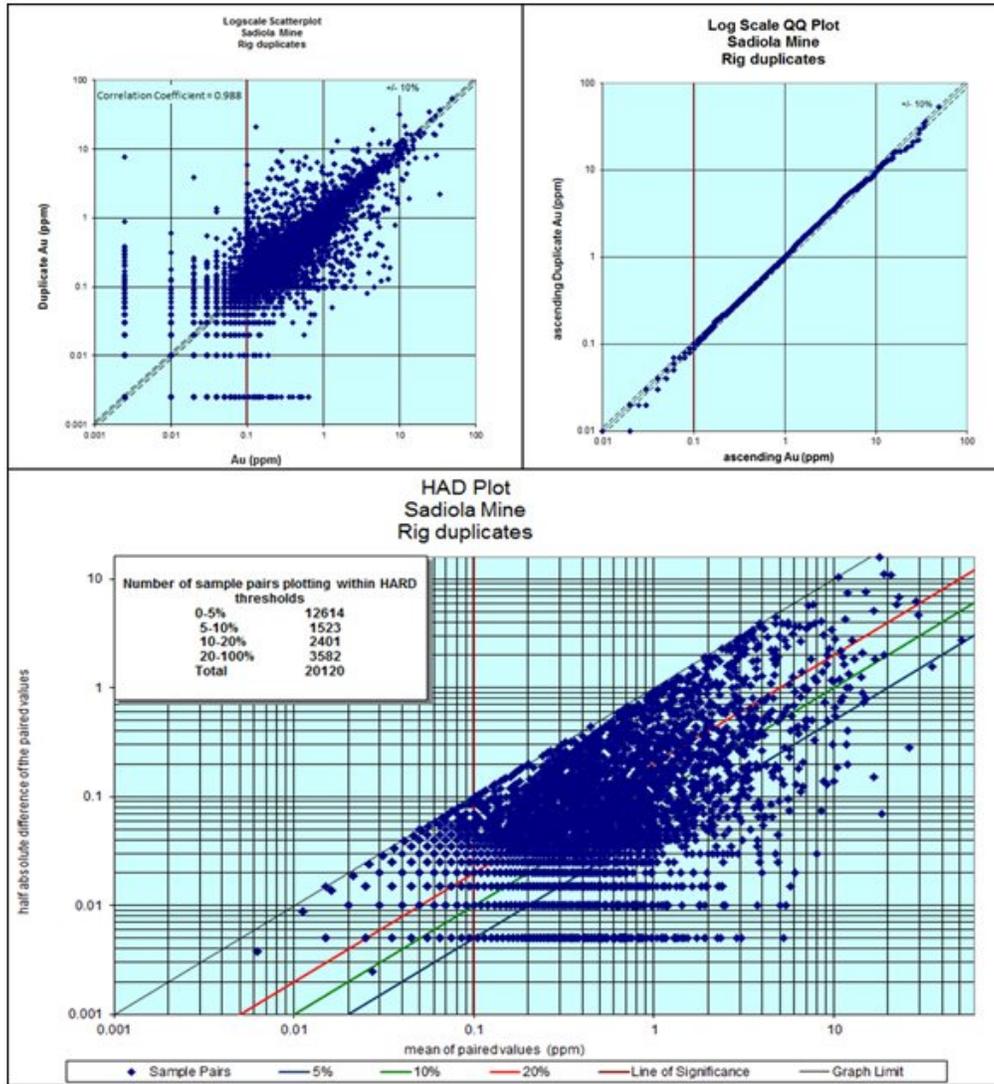
Field (coarse) duplicates are routinely collected in the field on the RC, air core and grade control RC rigs. The sampling interval is 2 m on the exploration rigs and either 2 m or 3 m on grade control rigs.

### 11.7.1 **Pre-2015**

From 2004 to 2014, a total of 20,120 field duplicates were analyzed for precision and bias. The duplicates, generally, do not show any bias.

The correlation coefficient is 0.85 indicating a strong linear correlation between the primary and duplicate samples. The half absolute difference (HAD) plot has half absolute relative difference (HARD) thresholds plotted diagonally; this plot indicates that 82% of the data has a HARD of 20% or less (Figure 11.6). Whilst reasonable for a deposit of this style, this precision is not ideal; The expectation is for 90% of the field duplicates to have a HARD of 20% or lower.

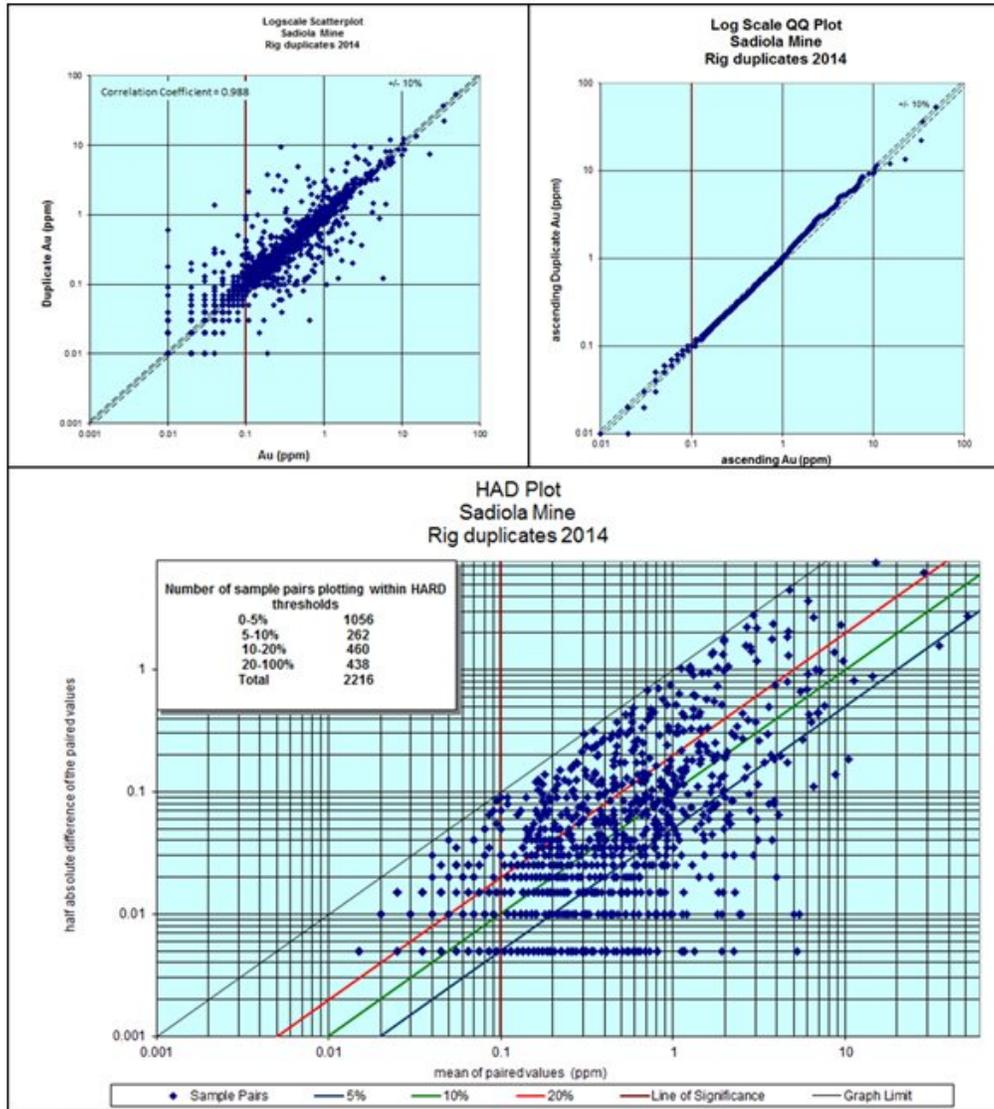
Figure 11.6: Field Duplicate Analyses



Source: Snowden, 2015

Similar analyses were undertaken for the 2014 field duplicates; the results are consistent with historical analysis. An improved correlation coefficient of 0.94 is observed with a lower level of precision, where 80% of the field duplicates have a HARD of 20% or less (Figure 11.7).

Figure 11.7: 2014 Field Duplicate Analyses

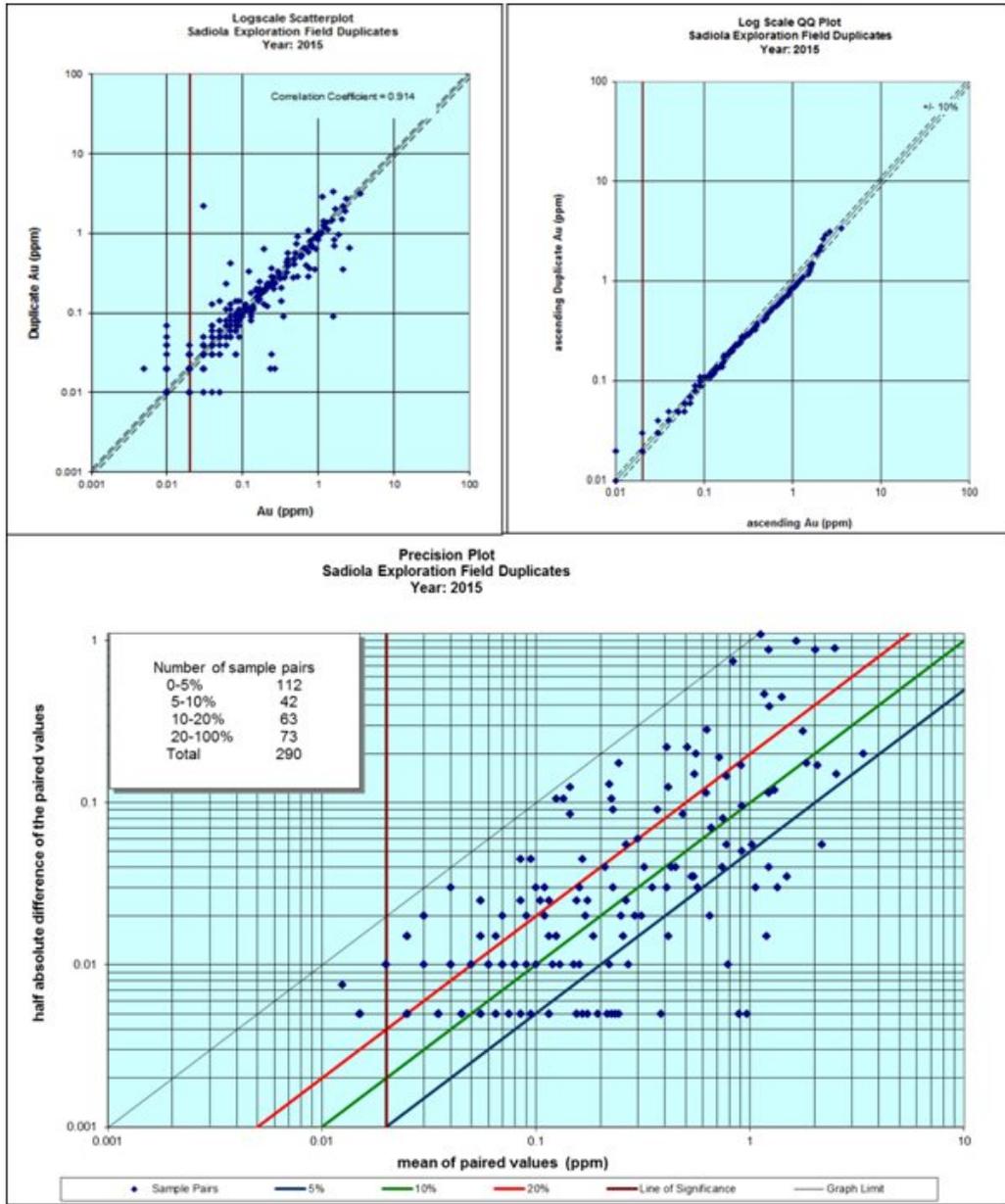


Source: Snowden, 2015

11.7.2 **2015 Analyses**

A total of 290 field duplicates were collected for the exploration QAQC program in 2015. The precision of the duplicates is poor; 75% of the samples have 20% or less HARD, which is lower than the expected precision for field duplicates of a gold deposit and is worse than that seen in 2014 (Figure 11.8 and Figure 11.9).

Figure 11.8: Sadiola 2015 Exploration Field Reject Duplicate Analyses

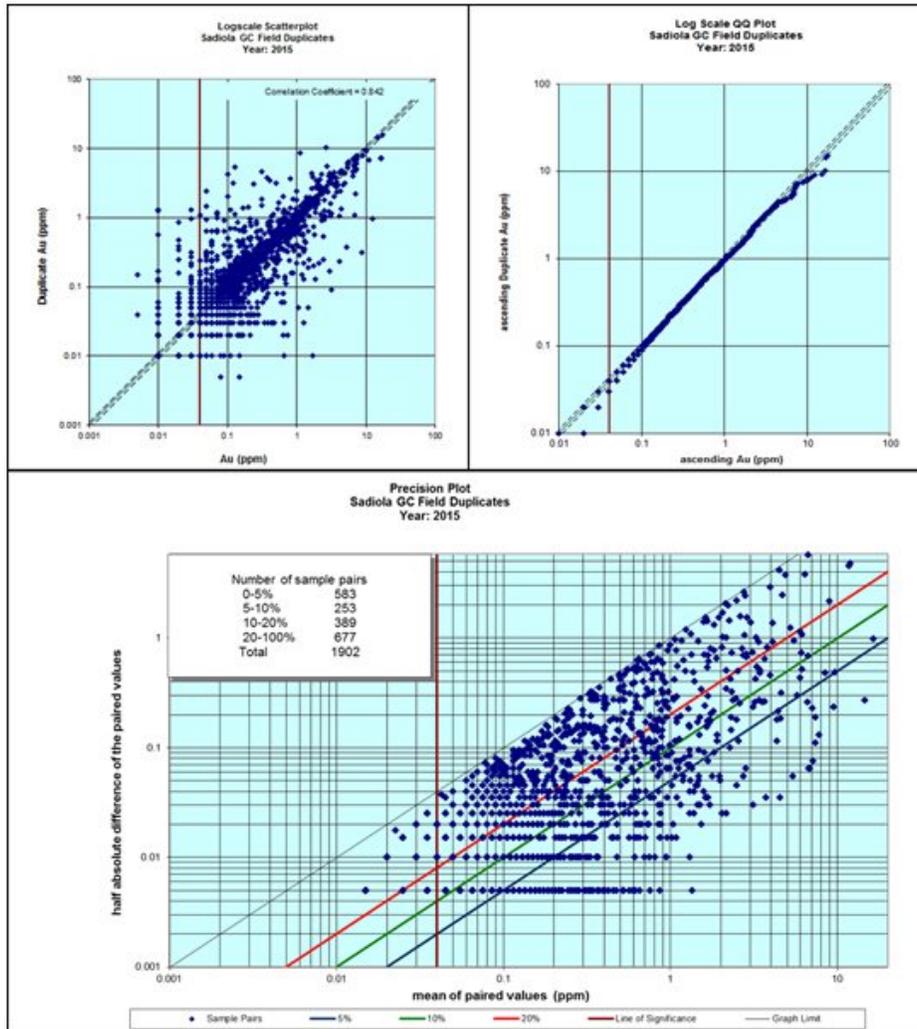


Source: Snowden, 2015

A total of 1,902 field duplicates were submitted with the grade control samples. The grade control field duplicates do not show a significant bias. The grade control precision is slightly worse than the exploration, with 68% of the field duplicates with a HARD of 20% or less.

The QP recommends that a review of the field subsampling methods be undertaken to improve the sample precision.

**Figure 11.9: Sadiola 2015 Grade Control Field Reject Duplicate Analyses**



Source: Snowden, 2015

**11.8 Pulp Duplicates**

Pulp duplicates are selected from pulp rejects from the laboratory. These are resubmitted to monitor precision of the laboratory. Coarse gold mineralization in some areas of the SEMOS deposits can result in poor precision but generally there is no coarse gold problem at Sadiola.

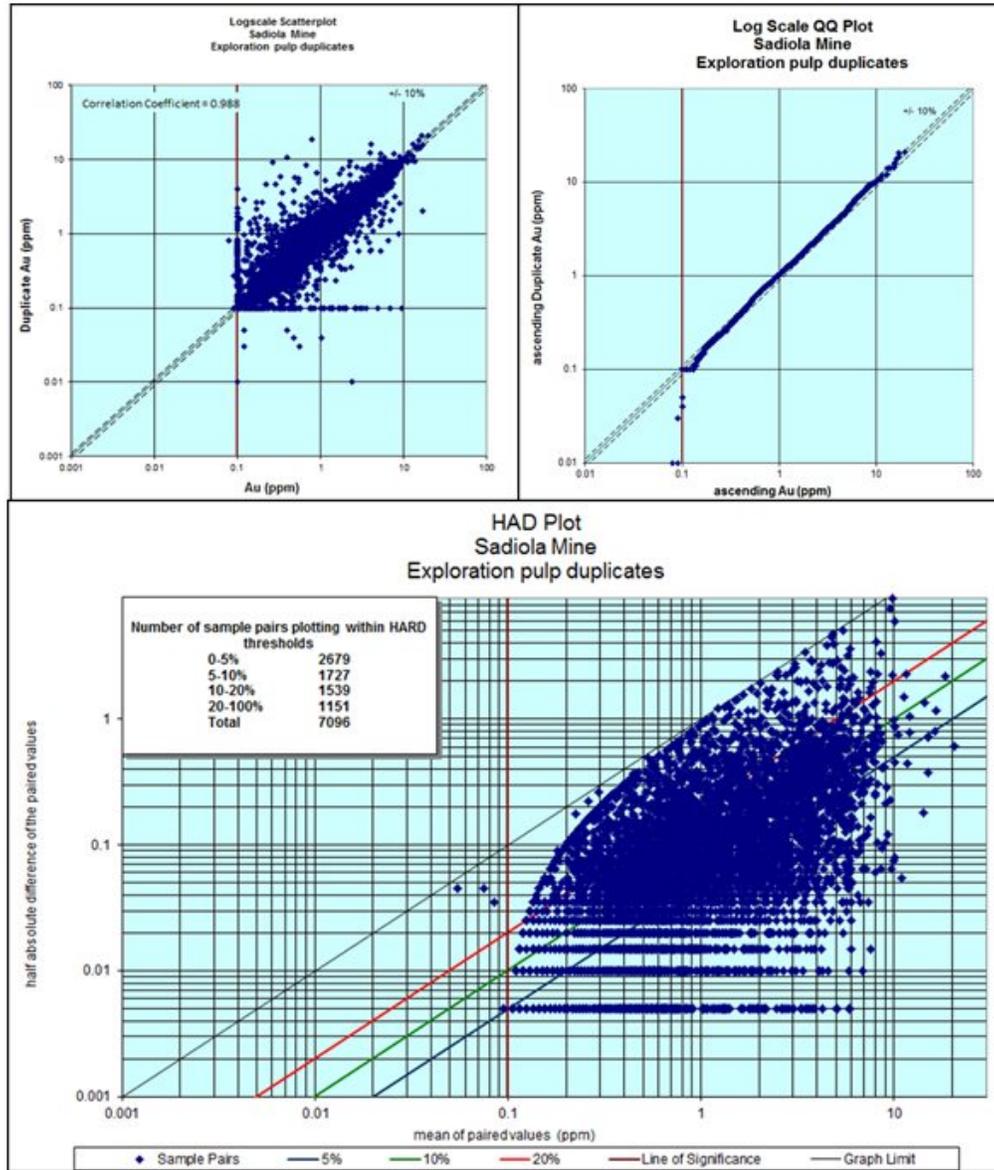
The pulps are subsampled, repackaged, renumbered and resubmitted to the laboratory for analysis within seven to 15 days of receipt of results. Pulp repeats were selected from samples collected from an ore envelope of 0.4 g Au/t, including a sample outside of the envelope.

#### **11.8.1            Pre-2015**

Pulp duplicates were assayed at SEMOS and SGS laboratories between 2010 and 2013. The first analyses are for duplicates assayed at the SEMOS laboratory. The duplicates show a linear correlation of 0.91. No bias was observed in the lower grades, however repeat assays for values >6 g/t returned higher assay values.

Precision of the pulp duplicates analyzed at SEMOS is poor, with only 62% of the pulp duplicates with a HARD value of 10% or lower (Figure 11.10), compared to the expected threshold for pulp duplicates of 90% of the data with a HARD value of 10% or less.

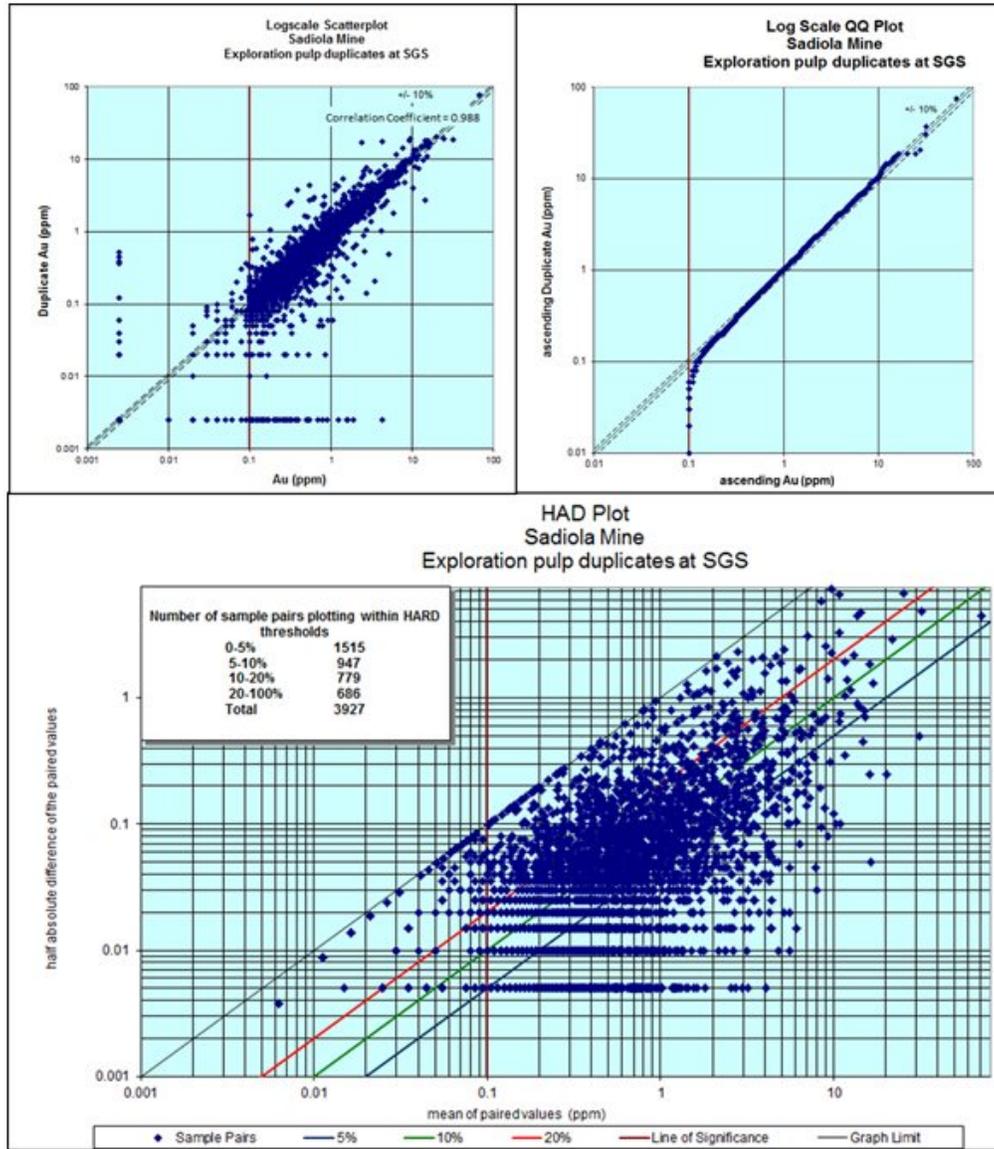
Figure 11.10: SEMOS Pulp Duplicate Analyses



Source: Snowden, 2015

The pulp duplicates analyzed at SGS Kayes display similar results as that observed at SEMOS. There is a strong linear correlation in the lower grades with a bias in the duplicates above 10 g/t. About 63% of the data have a HARD value of 10% or less, which is considered a relatively poor precision (Figure 11.11).

Figure 11.11: SGS Kayes Pulp Duplicate Analyses



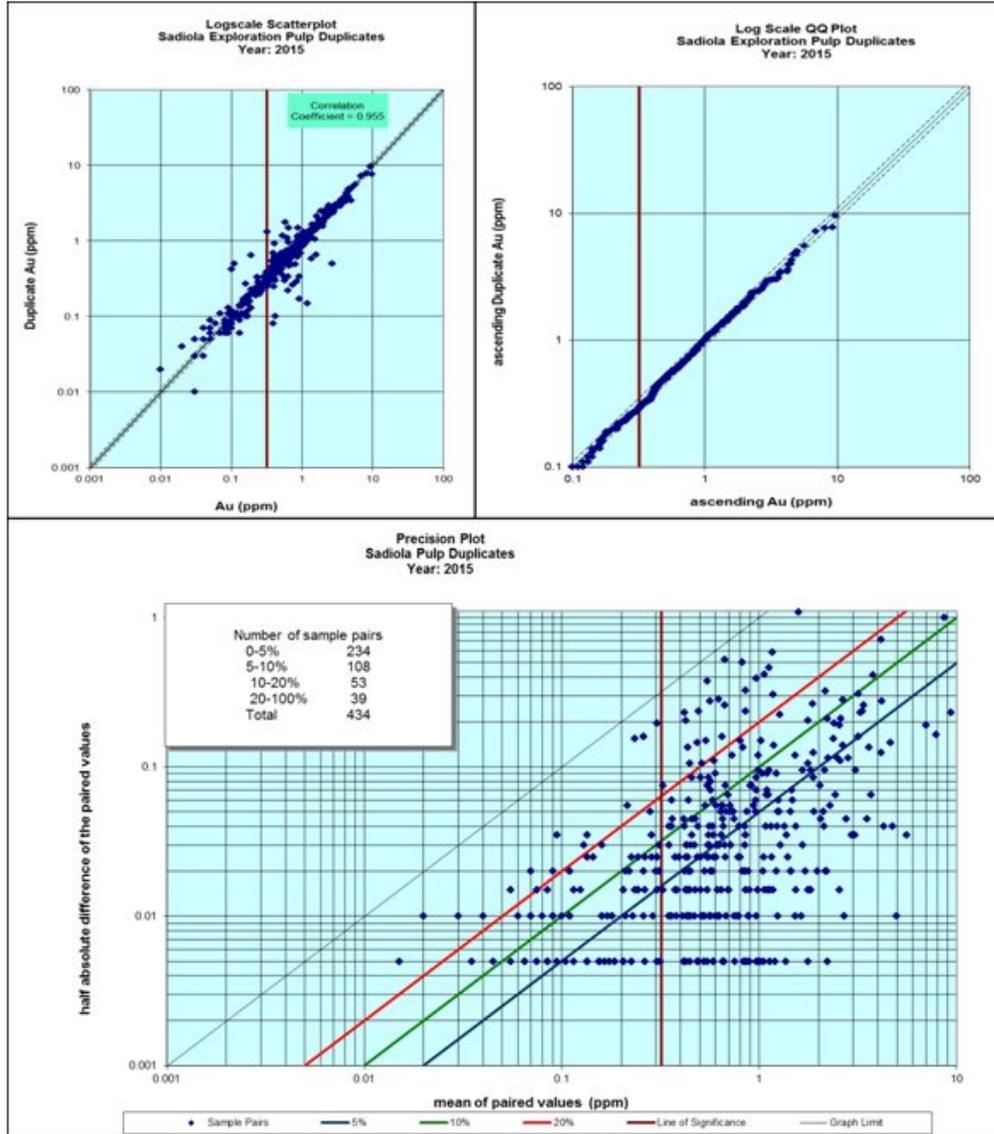
Source: Snowden, 2015

The level of precision for the pulp duplicates is worse than that for the field duplicates. In the QP’s opinion, this indicates that a significant source of the error may be during laboratory subsampling. The poor precision noted for the pulp duplicates could be due to the non-homogeneity of the material as noted by AGA (2014), where sieving analyses showed a 09% pass through a 75 µm sieve.

11.8.2 2015 Analyses

A total of 434 exploration pulp duplicates were submitted for assay. No significant bias was observed in the exploration pulp duplicates. As expected, the precision for pulp duplicates is better than that of the field duplicates; however, it is still considered to be poor for pulp duplicates, with 79% of the duplicates with a HARD of 10% or less (Figure 11.12).

Figure 11.12: Sadiola 2015 Exploration Pulp Duplicate Analyses

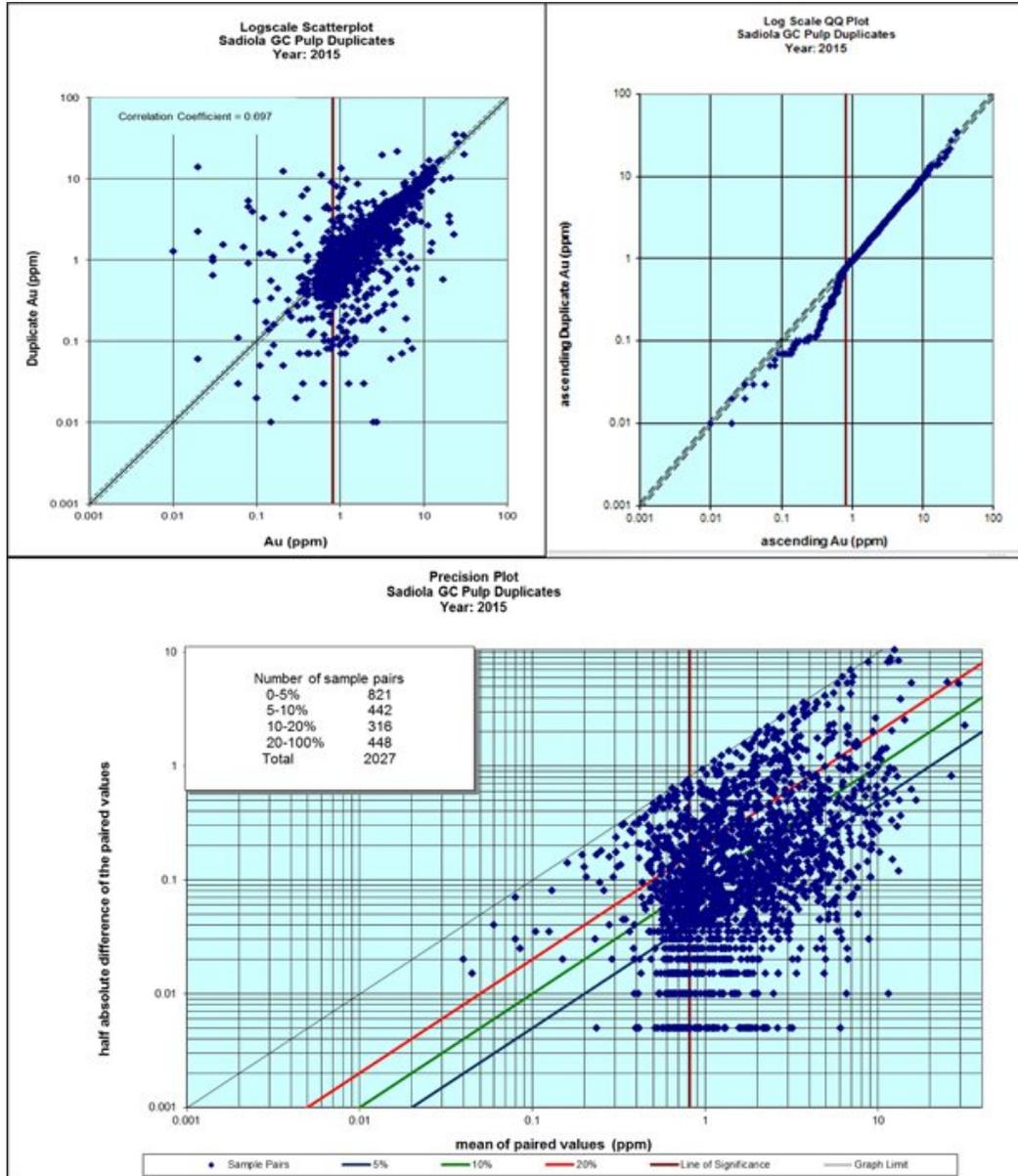


Source: Snowden, 2015

A total of 2,028 grade control pulp rejects were submitted to the primary laboratory as pulp duplicates. These samples had one pair with a duplicate assay of zero. This pair was excluded for the statistical analyses. The grade control pulp duplicates show a small constant bias towards the original assays, above 0.6 g/t.

Below 0.6 g/t, the bias is significant, with the duplicate reporting lower grade than the original assays. The precision plot shows that 62% of the samples have a HARD value of 10% or less (Figure 11.13). This level of precision is deemed poor for this type of sample. It is recommended that a review of the subsampling methods be undertaken.

Figure 11.13: Sadiola 2015 Grade Control Pulp Duplicate Analyses



Source: Snowden, 2016

**11.9 Check Assays**

Of the mineralized pulp rejects, 5% to 10% are routinely selected and resubmitted to SGS Bamako for check assaying.

Selection criteria for check assays included:

- To eliminate selection bias, pulps are derived from samples that cover a range of grades within the ore deposit;
- Selection is based on an ore envelope of 0.4 g/t. The ore envelope is extended to include a sample on either side of the ore envelope;
- Care is taken to avoid selecting unrepresentative waste material or material that has previously assayed at very low values or less than the detection limit.

The analytical procedure SGS performed matches that used by the SEMOS laboratory.

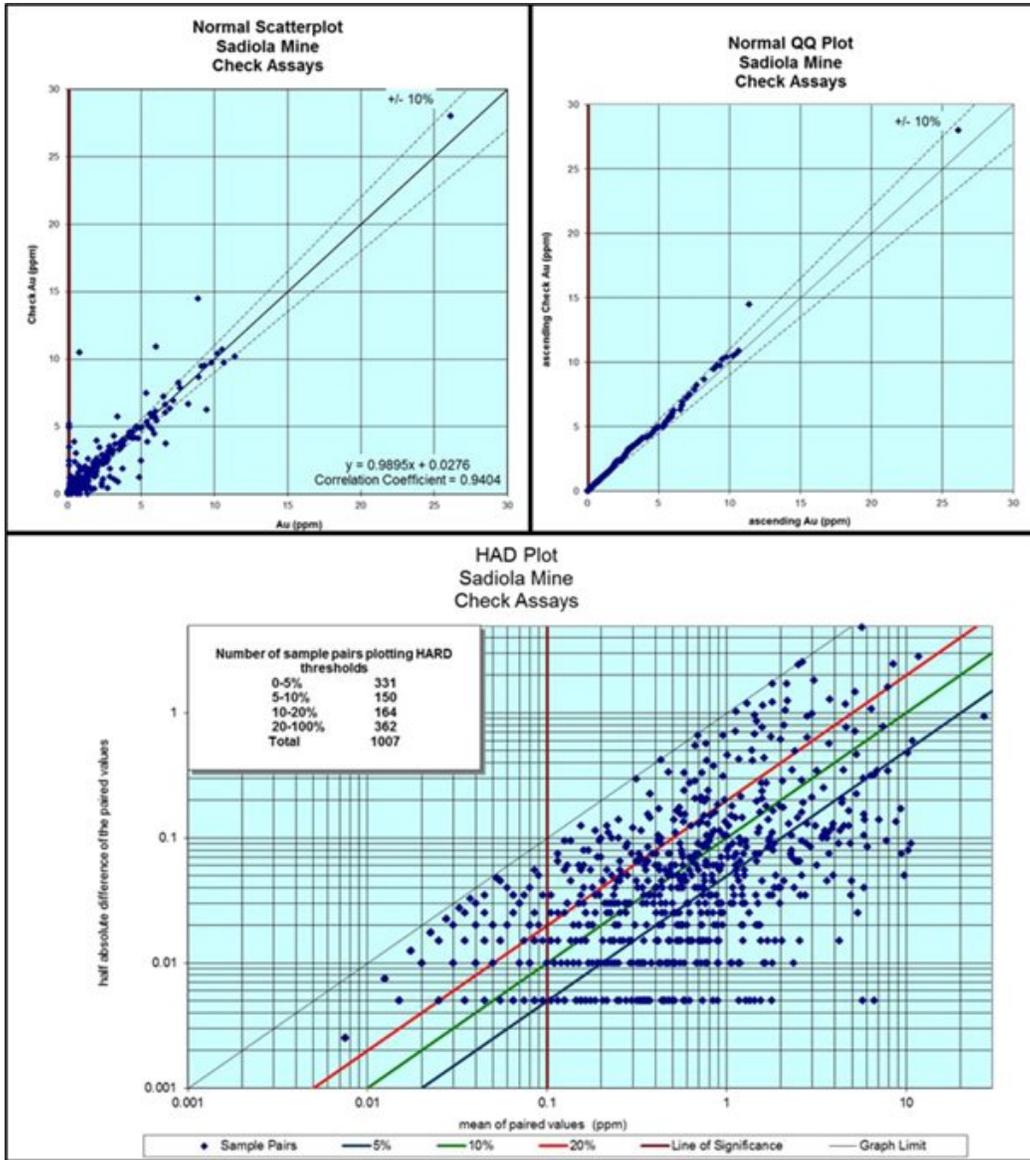
It is noted that AGA has changed the acceptable bias for pulp check assays from 5% (2014) to 10% (2015). The current relative bias is 10.5%, which was accepted by AGA.

#### **11.9.1**            **Pre-2015**

Check assays were undertaken on pulp material. The quantile-quantile (QQ) plots generally do not show any evidence for significant bias, except above 6 g/t where check assay values have higher grade than the original assays. This is in line with the negative bias observed in the analyses of CRMs with grades above 8 g/t.

The repeatability of the check assays is poor, 64% of the data have a HARD value of 20% or lower (Figure 11.14), compared to the expectation of 90% of the data to have a HARD value of 10% or less.

Figure 11.14: Check Assay Analyses



Source: Snowden, 2015

11.9.2 2015 Analyses

A total of 525 exploration and grade control pulp reject samples were submitted to SGS Bamako for check assay purposes. These submissions also included 53 CRMs and 54 blanks. The CRMs include four oxides and two sulphides, with grades ranging from 0.52 g/t to 5.758 g/t. The CRM assays show a good analytical accuracy has been achieved by SGS Bamako (Table 11.6).

A total of 53 pulp blanks were submitted to SGS Bamako with the check assays. Only one of the pulp blanks returned a grade above the upper limit of 10 times the detection limit of 0.01 g/t. It is suspected that this blank sample is likely the result of a swapped sample, rather than contamination. This indicates that there was minimum contamination during the assaying of the check assays (Figure 11.15).

AGA calculated a relative bias of 10.5% using a Reduced to major technique. This bias is deemed to be high, considering the check assays were done on pulverized samples and similar analytical techniques were used at both laboratories.

The precision of this paired data is poor with 66% of the pairs having a HARD value of 10% or less. Although this is a minor improvement from the historical check assays results, it is still significantly below industry expectation that 90% of the pairs have a HARD of 10% or less (Figure 11.16). It is recommended that a review of the laboratory subsampling methods be undertaken to ensure that the method adheres to good sampling principles.

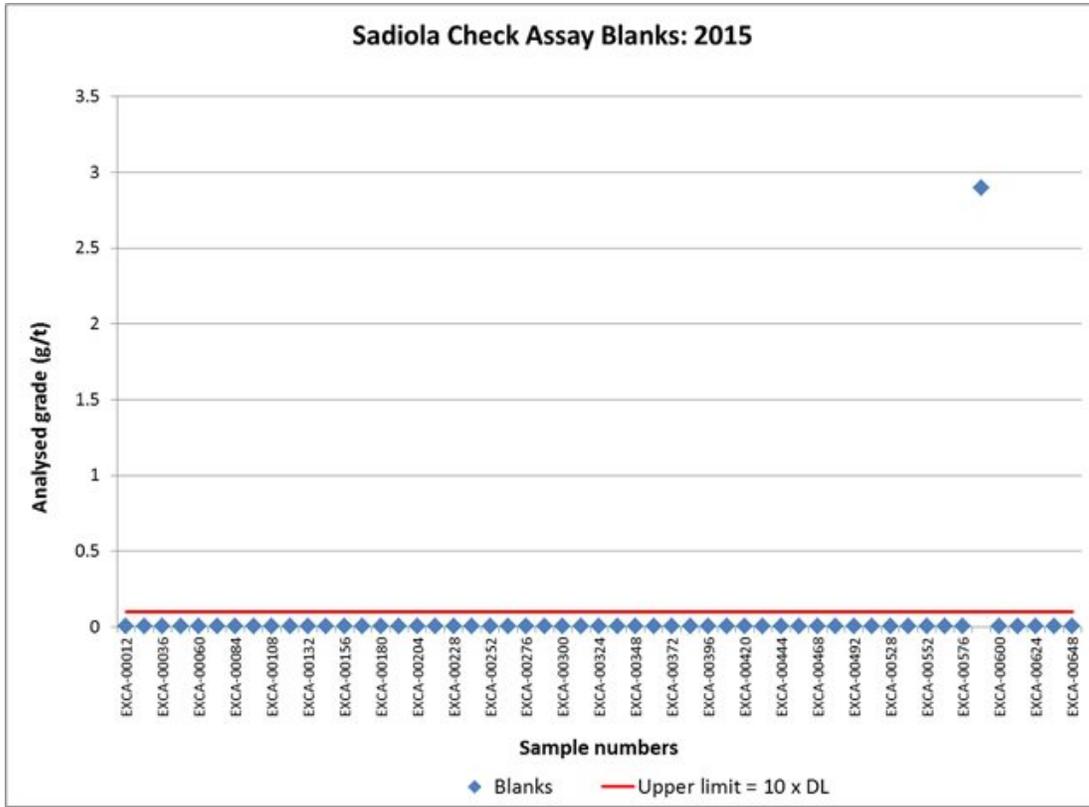
**Table 11.6: Summary of 2015 Check Assays CRM Analyses**

CRM	Certified Mean Value	Mean of Samples	No. of Samples	Bias %	% of Samples within 2STDEV	% of Samples within 3STDEV	Comments
G909-10	0.52	0.53	14	2	100	100	Good accuracy. Good precision
OXG-104	0.93	0.94	11	2	100	100	Good accuracy. Good precision
SI-15	1.81	1.85	6	2	100	100	Good accuracy. Good precision
SJ-63	2.63	2.66	10	1	100	100	Good accuracy. Good precision
OXK-110	3.60	3.54	5	-2	100	100	Good accuracy. Good precision
OXL-34	5.76	5.05	7	-12	86	86	Good accuracy, with a slight negative bias. A sample swap with a very low grade sample

Source: Snowden, 2015

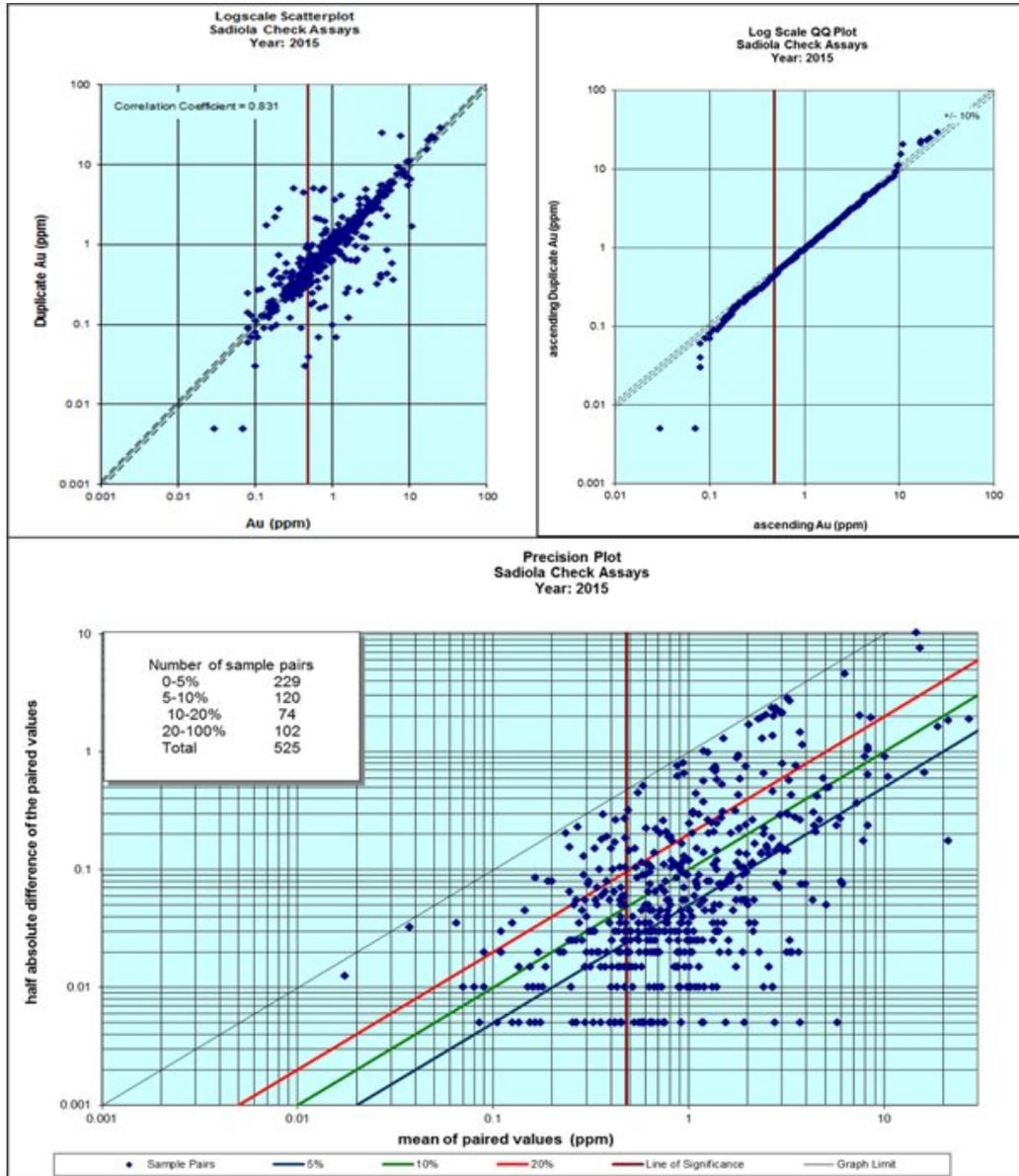
Note: The data as originally supplied to Snowden reported accuracy to four decimal places. The data was then rounded to two decimal places, as this is believed to be a better reflection of accuracy and precision.

Figure 11.15: Sadiola 2015 Check Assays Blanks Plot



Source: Snowden, 2015

Figure 11.16: Sadiola 2015 Check Assay Analyses



Source: Snowden, 2015

11.9.3 Laboratory Audits

The SGS Bamako laboratory was audited in January 2014 by the SEMOS Exploration Manager and in December 2014 by the SEMOS Database Manager. No deviations were observed that can have a material impact on the integrity of assay results (AGA, 2014b).

SGS Bamako was not audited in 2015, due to low number of samples submitted (SEMOS, 2016a). It is recommended that laboratory audits be carried out on a regular basis, regardless of the number of samples submitted.

Assay results from the SEMOS laboratory are reviewed weekly with the laboratory personnel. The entire tray is re-assayed should more than one SRM fail in the tray, i.e. mine and laboratory standards, or when standard and blank fail. The evaluation is carried out as soon as the results are available. The accepted results overwrite the previously failed results. The original and re-assay results from the laboratory are stored on the geology department server. The database maintains a history file of all multiple assays data loaded.

#### **11.10 Bulk Density**

Bulk density measurements are undertaken for each rock type and weathering horizon, as logged in the diamond core. Density is determined using the Archimedes principle, where the sample weight in air is determined relative to its weight in water. Samples are wrapped in a plastic film to limit moisture absorption in dry samples. All densities are calculated on a dry basis, with moisture correction where wet (or in situ) densities estimates are required. It is noted that this practice may result in a slight under estimation of density.

Downhole geophysical density measurements have been routinely implemented in grade control holes since 2004, using a gamma-gamma density probe which is calibrated against density readings determined using the immersion (or Archimedes) system. An external expert in downhole density logging audited the site during 2011 (Marcus Chatfield of Wireline Workshop, Technical Note 2011-09). The audit findings indicate that the current operational procedures are effective in terms of log precision but a need for some refinement of log accuracy is indicated.

Due to concerns about recent gamma-gamma probe density readings, these were not used for the 2015 Mineral Resource estimate update. Bulk densities, based on Archimedes principles were used, as they were felt to be more accurate.

Snowden is of the opinion that the gamma-gamma readings should be calibrated against direct measurements from core and twinned RC holes in order to establish a relationship between the core measurements and probe measurements, along with the comparison between the twinned DD and RC holes.

**11.11 Opinion of the QP on the Adequacy of Sample Preparation, Security and Analytical Procedures**

It is the opinion of the QP that the sampling and analytical methods and security procedures are adequate to allow for representivity in the samples collected and accuracy in the assay grades reported.

Poor precision was observed in the analyses of the field duplicates, pulp duplicates and check assays. This should be taken into account when undertaking Mineral Resource estimation and classification. The QP recommends that the subsampling methods from the drilling rigs to the laboratory be reviewed.

## 12. **DATA VERIFICATION**

All drilling data are collected, validated, managed and delivered to end users using a CAE Mining, Geographic Data Management System (GDMS). The geological and survey data are verified by the project geologist and signed off. Assays are verified by the Database Manager and the results reported to the project geologist who then signs off on the drillhole data in the database.

### 12.1 **Data Collection**

Geological and survey data are verified by the project geologist and signed off. The assay values and density (Specific Gravity (SG)) values are verified by the Database Manager. The results are then reported to the Project Geologist, who then signs off on the drillhole data in the database.

Capturing of data into the database by geological staff is achieved by:

- Direct entry using the Fusion applications;
- Data import from external data sources (.txt, .csv, Microsoft (MS) Access).

### 12.2 **Data Validation**

The GDMS has in-built validation routines that ensure that data being transferred to the central database are complete and accurate. This is achieved by:

- Capture of data into an MS Excel spreadsheet from the geologist's field logging. The spreadsheet is designed for loading data directly into Fusion. Key fields include: collar, survey, metadata, sample information, sample QAQC insertions and geological coding. A summary page, containing these fields is attached to the field log sheet. Each section is signed off by the official responsible for data capture on completion;
- Geologists check that all drillhole collar positions have been surveyed by the Survey Department, updated in the database, and that the surveyed collar positions plot in the correct location;
- Geologists validate the downhole surveys, entering dummy survey data as required and confirm the survey confidence in the database;
- Geologists validate that all metadata information is present and confirm that the drilling method, hole depth, drill rig and reason for drilling columns are all correctly populated;
- Geologists validate that all sample, QC samples and independent SRMs are captured, and that the correct sample numbers are allocated. When loading data, care is taken to ensure no duplication of sample numbers occurs;
- Checks to ensure all lithological information is correctly captured;

- Standardized pick lists.

Automatic validation of laboratory results:

- Assay results are received in electronic format from the laboratory. These are loaded directly into the database;
- A random check of 10% of the data is done by the Project Geologists to confirm data validity. Once all the information is loaded, the drillholes are authorized by the Project Geologist and locked in the database as being signed off. Further changes cannot be made without a proper audit trail.

Snowden undertook a high level review of the dataset used for the 2015 Mineral Resource update. Zero grade inputs were observed in the Main Pit North dataset. Most of these zero grades are in the waste, which is not a significant issue; however, a few were noted in the mineralized domains (KZONES; Table 12.1).

**Table 12.1: Zero Grades Input per KZONE**

KZONE	Description	Count
10	Waste	284
50	Alluvials	8
1220	Hangingwall northeast structures oxide	5
1310	Footwall north east structures oxide	10
1410	Main shear zone oxide	3

Source: Snowden, 2015

The QP observed 40 long samples, greater than 3 m in length, in the uncomposed datasets for Main Pit North. Of these 20 were in the waste with the balance in other KZONES. The samples not in waste had a maximum sample length of 8 m with a grade of 0 g Au/t, this was preceded by a 7.75 m sample with a grade of 2.1 g Au/t. The QP recommends these samples be reviewed and validated.

### 12.3 Data Organization

The GDMS is organized into pre-defined standardized structures at the mine sites, which synchronizes with AGA Head Office, Johannesburg, after hours.

The GDMS was enhanced during 2014–2015 by:

- The use of a single assay result import routine to manage all grade control assay data;

- Defining and implementing a single assay result import routine to manage all exploration data;
- All coordinate systems were standardized to equivalent Universal Transverse Mercator (UTM) coordinates;
- The drillhole naming convention was standardized and metadata was added that includes the permit name, drilling purpose, start dates, and drillhole type.

#### **12.4 Data Delivery**

The GDMS is an online database and is accessible only to predefined users. It is available to other external applications using custom export routines, queries and views, or data can be extracted directly via the Fusion Application Program Interface (API) or third party Operational Database Connectivity (ODBC) interface.

#### **12.5 Authorizing**

A strip-log is produced for each completed drillhole. The strip logs contain drilling information that has been captured. The Senior Geologist reviews and approves the strip-log hardcopy and electronically approves the drillhole in the database. An approved drillhole cannot be modified. Further changes can only be made with stated reasons, which are recorded into the database.

#### **12.6 Data Security**

The Information Technology (IT) department is accountable for performing backups and restores. The standard followed is weekly and monthly backups with the monthly backup stored off site at Yatela and a daily, off-site backup on the Head Office server, located in Johannesburg, South Africa.

#### **12.7 Opinion of the Qualified Person on the Adequacy of the Data for the Purposes Used in the Technical Report**

It is the opinion of the QP that the approach undertaken above is adequate to ensure that the data used in the estimation is valid.

## **13. MINERAL PROCESSING AND METALLURGICAL TESTING**

### **13.1 Introduction**

The December 2010 Feasibility Study of the Sadiola Deep Sulphide Project was prepared at the request of IAMGOLD Corporation and AngloGold Ashanti to extend the mine life of the Sadiola Mine beyond 2018. The Feasibility Study was prepared and compiled by IAMGOLD Project Development in a collaborative effort with resources from AngloGold Ashanti and SEMOS as well as a number of specialist external consultants. As a part of the data generated for the 2010 Feasibility Study a significant amount of testwork was conducted by a number of organizations. Since 2010, the Feasibility Study was updated and optimized, however, no additional testwork was done. This section reviews the nature and extent of the metallurgical testwork conducted.

Ore characterization was an important focus for the project due to the variability observed during the metallurgical testwork. The ore is characterized by degree of weathering, lithology and by localization within the deposit. Calcite marble is the dominant rock type that will be processed by the project.

Metallurgical testwork included the following:

- The mineralogical and geochemical characterization of 58 ore samples taken from various locations within the deposit and of the various major rock types (calcite marble, greywacke and diorite);
- A general gold deportment and sulphide liberation study was undertaken to predict gold behaviour during processing for eight composite samples;
- Organic carbon was found to be less than 0.1% and preg-robbing is not expected to be a problem;
- Heavy liquid separation reported a mass pull of below 1.7% reporting to the sinks with excellent gold upgrading of between 20% and 50% to the heavy fraction. These results were matched by the gravity tests conducted;
- A grading analysis reported a higher gold to mass ratio in the coarse material from the southern parts of the ore body;
- X-ray diffraction analyses showed that the samples were composed of quartz, feldspar, carbonates and mica and contained minor to traces of amphiboles, chlorite, scapolite and molybdenite;
- The QEMSCAN trace mineral search concluded an average gold grain size between 3 and 7 microns, or a gravity average between 4 and 12 microns.

Comminution testwork included the following tests:

- Bond Impact Work Index used to determine crushing design parameters. Results indicated 13.4 kWh/t for diorite, 10.7 kWh/t for greywacke, and 12.2 kWh/t for calcite marble;
- The JKTech drop weight test was used to determine SAG mill capability. However, the results from this testwork were not used in design as the results were not comparable with historical testwork and current SMC tests which gave more conservative results;
- The SAG mill comminution (SMC) test is a smaller scale test than the JKTech drop weight test. The lithological weighted average value of Axb was 33.4 and 0.31;
- Bond Rod Mill Work Index was determined to be 14.86 kWh/t;
- Bond Ball Mill Work Index was determined to be 13.33 kWh/t;
- Pennsylvania Abrasion test to determine the consumption of steel media. It was determined that the greywacke material will consume a lot more steel media than other rock types, the lithological weight average was a value 0.082.

Leaching and gravity testwork includes the following:

- Gravity testwork to evaluate the recoverable gold and the associated design requirements indicated that a Falcon recovery of 24.8% with an 82.1% intensive leach recovery of the gravity concentrate;
- Leaching tests were conducted on both ROM ore and the gravity tails, resulting in highly variable recovery of gold from different parts of the orebody. Cyanide consumption was estimated to be 0.632 kg/t and lime consumption 0.61 kg/t.

Although the testwork included variability testwork to the extent that ore from different zones was tested, a dedicated variability testwork programme was not included in the testwork conducted for the project.

## **13.2 Ore Classification**

Due to the high variability found in the results of the metallurgical testwork of the various ore types from the hard sulphide zones, ore classification was one of the important focus points. In this section, the ore from the Sadiola Deep Sulphide Project is classified by degree of weathering, lithology and location within the ore body. Each ore type defined by these characteristics will have an associated recovery, operating cost and throughput limitation.

### **13.2.1 Weathering**

Historically three degrees of weathering have been used and are:

- Laterite;
- Oxide:
  - Hard oxides;
  - Soft oxides;
- Sulphides:
  - Hard sulphides;
  - Soft sulphides.

The current operation has typically treated soft oxide, soft sulphide and laterite ore due to lower energy requirements and higher recoveries.

### 13.2.2 Hard Sulphide Ore

Hard sulphide ore is made of three lithologies:

- Calcite marble;
- Greywacke;
- Diorite.

Table 13.1 shows the hard sulphide ore distribution by lithology and location and Table 13.2 shows the Au distribution in the hard sulphide ore, also by lithology and location. From these two tables it is clear that 76.97% of the ore is calcite marble and that this contains 78.98% of the Au ounces.

**Table 13.1: Hard Sulphide Ore - Tonnage Distribution**

Lithology	South	Central	North	Total
Diorite	1.50%	1.17%	5.67%	8.34%
Greywacke	0.88%	3.62%	9.90%	14.40%
Calcite marble	23.10%	20.98%	32.89%	76.97%
<b>Total</b>	<b>25.48%</b>	<b>25.77%</b>	<b>48.46%</b>	<b>99.71%</b>
Undefined				0.29%

\*Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

**Table 13.2: Hard Sulphide Ore - Au Ounces Distribution**

Calcite marble	South	Central	North	Total
Diorite	1.43%	1.05%	6.18%	8.66%
Greywacke	0.62%	3.10%	8.35%	12.07%
Calcite marble	22.92%	20.77%	35.29%	78.98%
<b>Total</b>	<b>24.97%</b>	<b>24.92%</b>	<b>49.82%</b>	<b>99.71%</b>
Undefined				0.29%

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

### 13.2.3 Mineralogical Characterisation

Some 58 ore samples were analyzed by SGS South Africa for mineralogical and geochemical characterization. The samples were crushed and screened to 100% passing 1.7 mm and then split into two sub samples of which one was pulverized for X-ray diffraction and chemical analyses and the other half was mounted onto two sections for QEMSCAN bulk modal analyses to scan for mineral abundance. The samples are identified as the main zone (M), foot wall (F), hanging wall (H), calcite marble (C), greywacke (G) and diorite (D). The chemical analysis performed were package carbon species, package sulphur species, Cl, F, major oxide analysis, base metal analysis, semi-quantitative ICP and aqua regia digestion.

#### 13.2.3.1 X-Ray Diffraction

The samples were grouped into six clusters and two were left un-clustered. The minerals were quantified as predominant (>50%), abundant (20-50%), fairly abundant (10-20%), minor (3-10%), trace (<3%) and not detected. Table 13.3 shows the results of the X-Ray Diffraction analysis, with quartz, feldspar and mica making up the bulk of the gangue minerals.

**Table 13.3: X-Ray Diffraction Analysis Results**

Cluster	Results
Cluster 1	Dominated by quartz, feldspar and mica, with trace amounts of chlorite.
Cluster 2	Dominated by quartz, feldspar and lesser mica, with minor amounts of amphibole and chlorite. Trace to major amounts of calcite and/or dolomite might be present. Minor to fairly abundant amounts of scapolite may be present.
Cluster 3	Dominated by quartz and feldspar, with lesser mica, chlorite and dolomite and trace amounts of calcite. Trace amounts of pyrite and/or molybdenite may be present.
Cluster 4	Dominated by calcite, feldspar and/or quartz, with lesser mica. Minor amounts of amphibole, monticellite, chlorite, dolomite and/or pyrite may be present.
Cluster 5	The samples in this cluster can contain abundant quartz, feldspar, calcite and/or dolomite, with lesser mica. Pyrite and amphibole may be present in minor to major amounts.
Cluster 6	Calcite and feldspar are dominant in this cluster, with lesser quartz and mica and trace to minor amounts of calcite. Minor amounts of scapolite and/or amphibole may be present.
Un-clustered 1	Contained major amounts of quartz, scapolite, amphibole, mica and calcite, with minor amounts of feldspar and trace amounts of chlorite.
Un-clustered 2	Contained abundant feldspar, with lesser quartz, mica and calcite. Minor amounts of kaolinite and siderite are present.

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

### 13.2.3.2 Chemical Analysis

The major oxide analysis shows that the ore is predominately composed of SiO<sub>2</sub> and CaO, with values between 13% and 65% for SiO<sub>2</sub> and from 3% to 43% for CaO. Following these two dominant minerals there will be Al<sub>2</sub>O<sub>3</sub>, MgO, Fe<sub>2</sub>O<sub>3</sub>, Na<sub>2</sub>O and K<sub>2</sub>O. The analysis for major oxides of the six (6) clusters are shown in Table 13.4, with Table 13.5 showing sulphur and carbon analyses for the same cluster samples.

Table 13.4: Cluster Samples - Major Oxide Analyses

Mineral	Units	Cluster 1	Cluster 2	Cluster 3	Cluster 4	Cluster 5	Cluster 6
Al <sub>2</sub> O <sub>3</sub>	%	19.80	13.90	12.10	6.72	8.52	6.72
Fe <sub>2</sub> O <sub>3</sub>	%	6.64	2.82	2.61	2.58	4.18	2.97
MgO	%	3.96	4.13	5.22	1.41	8.15	4.01
CaO	%	7.22	7.47	3.66	32.90	20.00	29.00
K <sub>2</sub> O	%	4.55	0.92	3.76	0.71	2.48	1.18
Na <sub>2</sub> O	%	2.93	6.40	2.43	2.28	0.95	1.76
TiO <sub>2</sub>	%	0.42	0.42	0.36	0.26	0.32	0.26
SiO <sub>2</sub>	%	47.90	57.10	63.50	25.50	32.80	28.40
Cr <sub>2</sub> O	%	0.01	0.01	0.01	0.01	0.01	0.01
P <sub>2</sub> O <sub>5</sub>	%	0.57	0.20	0.11	0.09	0.12	0.11
V <sub>2</sub> O <sub>5</sub>	%	0.02	0.01		3.01	0.01	0.01
LOI	%	3.35	5.12	4.79	25.75	17.47	22.63
<b>Total</b>	<b>%</b>	<b>97.37</b>	<b>98.50</b>	<b>98.55</b>	<b>101.22</b>	<b>95.01</b>	<b>97.06</b>

Table 13.5: Cluster Samples - Sulphur and Carbon Analyses

Element	Units	Cluster 1	Cluster 2	Cluster 3	Cluster 4	Cluster 5	Cluster 6
Sulphide S	%	0.88	0.31	0.45	0.16	0.59	0.51
Elemental S	%	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Sulphate S	%	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
<b>Total S</b>	<b>%</b>	<b>1.11</b>	<b>0.38</b>	<b>0.62</b>	<b>0.19</b>	<b>1.07</b>	<b>0.77</b>
CO <sub>3</sub>	%	2.97	6.64	4.96	>30	25.9	28.8
Organic C	%	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Graphite C	%	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
<b>Total C</b>	<b>%</b>	<b>0.61</b>	<b>1.33</b>	<b>1.03</b>	<b>7.28</b>	<b>5.19</b>	<b>5.78</b>

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

The gold content of the cluster samples can be seen in Table 13.6, with clusters 5 and 6 showing the highest gold values. As was on average 2151 ppm but analyses varied from 60 ppm to 7,600 ppm.

**Table 13.6: Cluster Swamples - Minor Element Chemical Analyses**

Element	Units	Cluster 1	Cluster 2	Cluster 3	Cluster 4	Cluster 5	Cluster 6
Au	ppm	0.85	0.51	1.6	0.58	2.61	2.56
Ag	ppm	<1	<1	<1	<1	<1	<1
As	ppm	1517	2093	3737	640	4043	2613
Ab	ppm	12	15	72.5	10	300.5	177.5
Cu	ppm	90	50	23	30	40	39
Pb	ppm	21	13	17	7	40	46
Zn	ppm	80	70	110	20	330	370
Ni	ppm	34	26	23	34	37	33
Mo	ppm	3.1	21.3	7.3	1.2	5.8	1.1
Hg	ppm	<3	<3	<3	<3	<3	<3

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

### 13.2.3.3 Mineral Abundance (QEMSCAN)

Silicates gangue was identified as the predominant mineral in clusters 1, 2 and 3, with values between 70% and 92% mass. These minerals are mostly quartz, plagioclase and/or biotite / phlogopite. The carbonate quantity in these minerals varies from 5% to 30% mass. Minor minerals are muscovite, chlorite and kaolinite (cluster 2). The lithologies in these samples are greywacke (hanging wall and main zone) and diorites.

In clusters 4, 5 and 6 the predominant mineral is carbonate gangue ranging between 28% and 77% mass. Minor minerals in these clusters are calcic plagioclase (clusters 4 & 6) and quartz and biotite / phlogopite (cluster 5). Dolomite content will vary between clusters. The main lithology of these clusters is calcite marble.

The total sulphide was at an average of 1.6% in all of the samples and ranged between 1.34% and 2.77%. Seven sulphide species were identified in the samples (pyrite / marcasite, pyrrhotite, arsenopyrite, berthierite, chalcopyrite and molybdenite) with pyrite, pyrrhotite and arsenopyrite being the dominant minerals. The presence of relatively abundant marcasite and pyrrhotite throughout the ore indicates that oxygen consumption during cyanide leaching will be a general problem encountered in nearly all ore types and lithologies. The relatively high arsenopyrite content could indicate the presence of refractory ore, often associated with arsenopyrite.

Four distinct particulate Au-species were identified:

- Pure native gold (Au), with a silver content below detection limit;
- Gold containing variable amounts of silver, but below 20% Ag;
- Low amounts of electrum (gold with more than 20% silver);
- Austrobitite (AuSb<sub>2</sub>).

The average gold content of the gold/electrum grains was assumed to be ~90%. The austrobitite contains ~44% gold.

#### **13.2.4 Gold Deposition and Sulphide Liberation**

A general gold deposition and sulphide liberation study was done in order to predict the gold behaviour during processing. A total of eight (8) composite samples were prepared from 29 drill core samples. The samples were:

- MNS near surface;
- MNS deep;
- North near surface;
- North deep;
- South near surface;
- South deep;
- Middle near surface;
- Surface deep.

The methodology consisted of crushing & milling, head assays, heavy liquid separation, gravity concentration, grading analysis, X-ray diffraction, modal mineralogy, trace mineral (gold) search and specific mineral (sulphide) search. Head grades for the eight composite samples are shown in Table 13.7.

**Table 13.7: Head Grades of the Sadiola Composite Samples**

Sample	Avg Au grade (g/t)	Sb grade (ppm)	S grade (%)
MNS near surface	2.16	190	1.10
MNS deep	2.12	220	0.64
North near surface	2.49	420	0.52
North deep	2.17	190	0.87
South near surface	3.49	340	0.78
South deep	1.74	550	0.82
Middle near surface	1.26	240	0.26
Surface deep	2.11	280	0.80

Source: SGS, General mineralogical Characterisation, Gold Department and Sulphide Liberation Analysis: Composite Gold Ore Sample from the Sadiola Project, November 12, 2009

#### 13.2.4.1 Heavy Liquid Separation

The samples for heavy liquid separation were de-slimed at 25 microns before the test. The test was then performed with a 2.96 kg/l specific gravity heavy liquid where gold and sulphur was assayed on all fractions. The mass pull was between 25% and 30% to the slimes and 70% to 75% to the floats which left less than 1.7% reporting to the sinks or heavy fraction. There was a very good gold upgrading in the heavy fraction with gold recoveries between 20% and 50% and sulphur recoveries in the heavy fraction between 34% and 64%. The heavy fraction mass pull ranged from 0.55% to 1.68%.

#### 13.2.4.2 Gravity Concentration

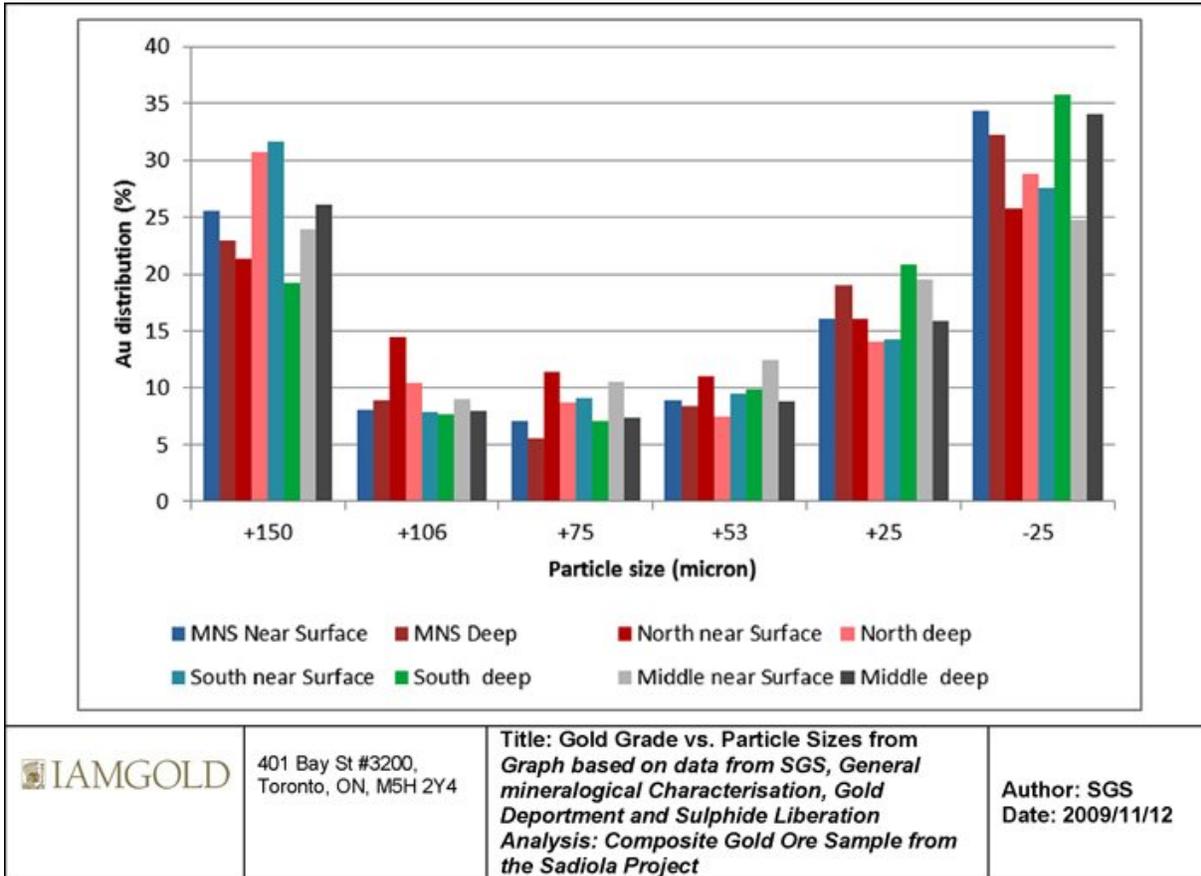
Gravity concentration tests were conducted on two 4 kg samples from every drill hole in a laboratory size falcon concentrator. The two concentrates produced for each drill hole were combined, as were the tails samples. The results matched the results of the heavy liquid separation tests with the exception of the "middle near surface" composite, where the high density media achieved higher recovery and grade in the heavy fraction. Gold recovery to the gravity fraction ranged from 26% to 57% with a mass pull between 2.26% and 3.00%.

#### 13.2.4.3 Au grade vs. Particle Sizes

A grade vs. size analysis was done on samples from every drill hole. The samples were prepared by grinding to P50 of 75 µm. General observations as seen in Figure 13.1 were that there was a higher gold

to mass ratio in the coarse material from the south near surface sample and that there was relative more gold in the -25 µm size fractions.

**Figure 13.1: Gold Grade vs. Particle Sizes**



13.2.4.4 X-Ray Diffraction Analysis

The X-ray diffraction analyses showed that the samples are composed of quartz, feldspar, carbonates and mica and contains minor to traces of amphiboles, chlorite, scapolite and molybdenite. The results are shown in Table 13.8.

**Table 13.8: X-Ray Diffraction Analysis Results**

Mineral	MNS Near Surface (%)	MNS Deep (%)	North near Surface (%)	North deep (%)	South near Surface (%)	South deep (%)	Middle near Surface (%)	Middle deep (%)
Quart	10 - 20	20 - 50	10 - 20	10 - 20	10 - 20	10 - 20	3 - 10	10 - 20
Plagioclase	20 - 50	20 - 50	10 - 20	20 - 50	10 - 20	20 - 50	10 - 20	10 - 20
K-feldspar	3 - 10	3 - 10	3 - 10	3 - 10	3 - 10	3 - 10	3 - 10	3 - 10
Scapolite	3 - 10	tr	nd	tr	tr	3 - 10	tr	3 - 10
Amphibole	3 - 10	3 - 10	nd	3 - 10	tr	3 - 10	nd	tr
Mica	10 - 20	20 - 50	10 - 20	10 - 20	10 - 20	10 - 20	10 - 20	10 - 20
Chlorite	tr	tr	nd	tr	tr	tr	tr	tr
Calcite	3 - 10	3 - 10	>50	10 - 20	>50	10 - 20	>50	20 - 50
Dolomite	3 - 10	3 - 10	nd	10 - 20	nd	10 - 20	3 - 10	10 - 20
Pyrite	tr	nd	nd	tr	tr	nd	nd	tr
Molybdenite	nd	nd	nd	nd	nd	tr	nd	nd

Source: SGS, General mineralogical Characterisation, Gold Department and Sulphide Liberation Analysis: Composite Gold Ore Sample from the Sadiola Project, November 12, 2009

#### 13.2.4.5 Gold Minerals

QEMSCAN was also used to evaluate the gold trace mineral search on samples from head samples and gravity concentrates. The samples were ground to P<sub>80</sub> of -53 microns for the analysis. Gold was classified as Gold/Electrum (90% gold) or Aurostibite (44% gold). It can be seen from the results for the head samples shown in Figure 13.2 that most of the gold is present as gold / electrum.

Gold grain exposure and association of head and gravity samples are shown in Figure 13.3, where it can be seen that between 85% and 97% of the gold in the head sample and in the gravity concentrate was exposed.

Gold grain size in the head samples was found to be coarser in almost all of the near surface samples. It is noted that although the average head gold grain size is between 3 and 7 µm, a 74 µm gold grain was found in one of the head samples. The average gold grain size in the gravity concentrate was between 4 and 12 µm.

Figure 13.2: Au Distribution per Mineral in Head Samples

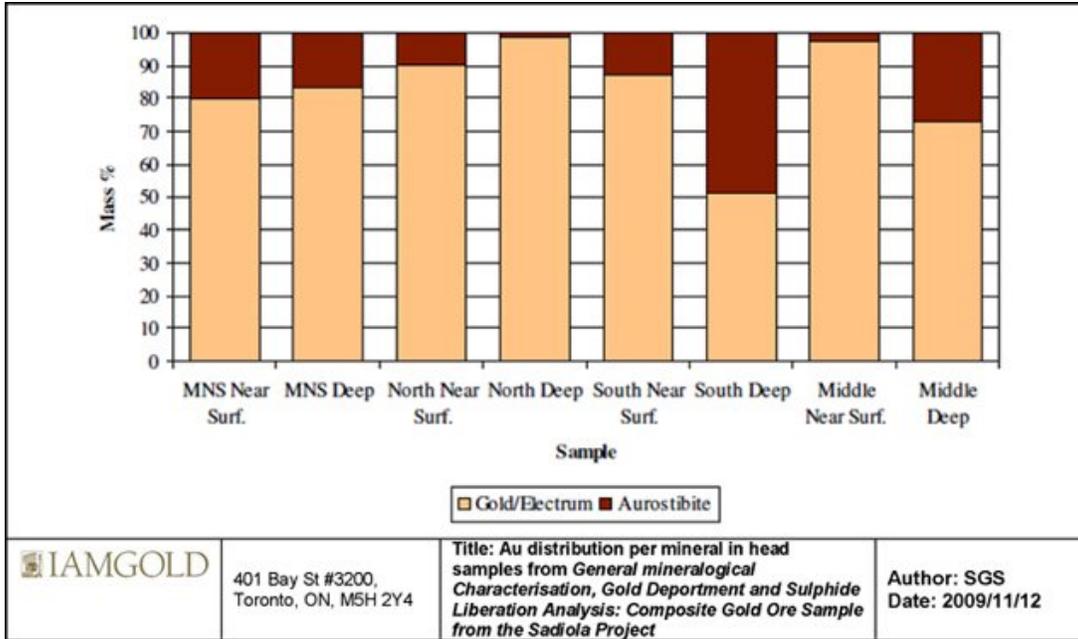
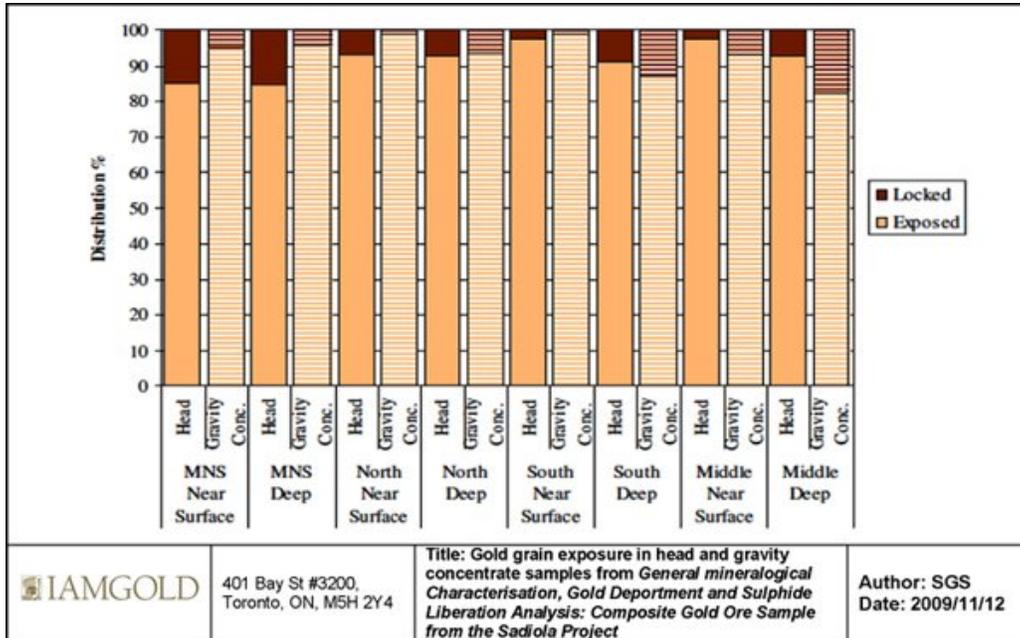


Figure 13.3: Gold Grain Exposure in Head and Gravity Concentrate Samples



#### 13.2.4.6 Sulphide Mineral Analysis

QEMSCAN analysis of the head composite samples taken from three different size fractions showed that the major sulphides are pyrite/marcasite, pyrrhotite and arsenopyrite, which is in agreement with the previous mineral abundance test results. The minor to trace minerals are berthierite, stibnite, chalcopyrite, tetrahedrite-tennantite, bornite, covelite, molybdenite, galena and ullmanite. Sulphide minerals are liberated at over 91% and only 1% is locked. The average sulphur grain size was found to be under 20 µm.

### 13.3 **Metallurgical Process Testwork**

The metallurgical testwork discussed in this section was carried out during 2009 and 2010 in support of the process design.

#### 13.3.1 **Comminution Testwork**

##### 13.3.1.1 Bond impact Work Index

The results from the Bond impact work index (BIWI) tests show the diorite sample to have the highest energy requirement at 13.4 kilowatt-hours per tonne, with the Greywacke test resulting in the lowest value at 10.7 kilowatt-hours per tonne. Calcite marble gave an average value of 12.2 kilowatt-hours per tonne. The lithological weighted average for the hard ore is of 12.08 kilowatt-hours per tonne. Generally the test indicated that the ore is soft to medium hard.

##### 13.3.1.2 JK Drop Weight Tests

The JKTech drop weight test is used to determine the SAG mill capability and is based on dropping variable steel weights mounted on guided rails onto five sized fractions of rock specimens from various heights. The results of the test provides two (2) parameters (A & B), indicating the relative hardness of the samples. As part of the test procedure, an abrasion characteristic ( $t_a$ ) is also measured in a tumbling test.

Two tests were done on calcite marble and two tests were done on greywacke. Composite samples for each lithological unit were prepared for testing, with the results the calcite marble samples shown in Table 13.9 and the results for the greywacke samples in Table 13.10. Based on these data the ore would be typically classed as soft.

**Table 13.9: JKTech Drop Weight Test Results for Calcite Marble Samples**

Sample	A	b	Axb	ta
Calcite marble 1	67.5	1.18	79.7	0.37
Calcite marble 1	68.0	1.17	79.6	0.62
<b>Average</b>	<b>67.8</b>	<b>1.18</b>	<b>79.7</b>	<b>0.495</b>

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

**Table 13.10: JKTech Drop Weight Test Results for Greywacke Samples**

Sample	A	b	Axb	ta
Calcite marble 1	75.4	0.95	71.6	0.55
Calcite marble 1	75.0	0.96	72.0	0.56
<b>Average</b>	<b>75.2</b>	<b>0.96</b>	<b>71.8</b>	<b>0.56</b>

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

### 13.3.1.3 SAG Mill Comminution

The SAG mill comminution (SMC) test is a smaller scale test than the JKTech drop weight test, but only one size fraction is tested. The results shown in Table 13.11 are the average values for the test results reported for the calcite marble, greywacke and diorite samples for the SMC tests, with very similar averages results for all three ore types, showing that the ore types can be classed as medium hard. It was found that the JK Drop Weight tests and the SMC test gave results that differed significantly, which was unexpected as the SMC test is an abbreviated version of the Drop Weight test. However, as different samples were used for the two series of tests, size fractions would have been different and may explain the differences.

**Table 13.11: SMC Test - Average Results for Different Ore Types**

Material	A	b	Axb	ta
Calcite marble	96.2	0.3	33.5	0.3
Greywacke	93.7	0.4	33.2	0.3
Diorite	100.0	0.3	32.0	0.3

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

13.3.1.4 Bond Rod Mill Work Index

The weighted average for the hard sulphide ore is 14.86 kWh/t, with the average test results shown in Table 13.12. The hardest material was shown to be the greywacke, followed by diorite, with the calcite marble shown as the softest of the three ore types.

**Table 13.12: Bond Rod Mill Work Index - Average Results for Different Ore Types**

Material	No of Samples Tested	BRMWi (kWh/t)
Calcite marble	12	14.4
Greywacke	7	16.8
Diorite	2	16.1

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

13.3.1.5 Bond Ball Mill Working Index

The average result for each of the ore types is shown in Table 13.13. The weighted average value for the three ore types is 13.33 kWh/t. As for the Bond rod mill work index test, the hardest material is the greywacke, followed by diorite and calcite marble.

**Table 13.13: Bond Ball Mill Work Index - Average Results for Different Ore Types**

Material	No of Samples Tested	BBMWi (kWh/t)
Calcite marble	17	12.28
Greywacke	8	17.44
Diorite	4	15.95

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

13.3.1.6 Pennsylvania Abrasion

The Pennsylvania abrasion (PA) test is used to determine the steel media consumption in the ball mills. The procedure consists of measuring the steel lost on a paddle in a rotating drum containing a dry ore sample. The results are shown in Table 13.14. The weighted average for the three (3) ore types is of 0.082, however it is expected that the greywacke will consume more steel, grinding media and mill liners than the other two ore types.

**Table 13.14: Average Pennsylvania Abrasion Test Results for Different Ore Types**

Material	No. of Samples Tested	Test 1 Average	Test 2 Average	Average
Calcite marble	15	0.0597	0.0480	0.06
Greywacke	5	0.2103	0.2053	0.21
Diorite	1	0.0992	0.0884	0.09

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

### 13.3.2 Gravity Concentrate Tests and Associated Intensive Leach Testwork

Gravity concentration testwork was initiated to evaluate if gravity recoverable gold was present in the ore and what the associated design requirements for centrifugal concentrators and intensive leach reactors will be. The gravity test feed material was crushed in two stages first to 100% passing 13.2 mm and then to 100% passing 3.35 mm. The resulting crushed material was then dry milled in a laboratory ball mill to 50% passing 75 µm. The samples were then combined into 10 kilogram sub samples and identified according to drill hole, lithology and spatial location. A Falcon laboratory gravity concentrator was used with 150 G acceleration and 1 psi water pressure to produce the gravity concentrates.

Each gravity concentrate was then leached under intensive leach conditions, which included:

- 10% solids content, preconditioned with lime and oxygen;
- Dissolved oxygen concentration maintained at 20 ppm;
- Addition of 2% w/v NaCN, maintained at 2% w/v for the duration of the test;
- Addition of 2 kg/t PbNO<sub>3</sub>.

Results are shown in Table 13.15 based on the gravity concentrate testwork as presented in the SGS testwork report which was provided as the appendix to the feasibility study report. The results show that recovery of gold between 12% and 32% was achieved to the gravity concentrate, with leach recoveries of this gold ranging from 74.2% and 89.6%.

**Table 13.15: Average Results – Gravity Concentrate Recovery and Intensive Leach Tests**

	Unit	Cluster 1	Cluster 2	Cluster 3	Cluster 4	Cluster 5	Cluster 6	Un-clustered 1	Un-clustered 2
<b>Gravity concentration :</b>									
No of samples		2	12	5	17	7	5	1	1
Head grade	g/t	1.07	1.43	1.53	1.42	2.32	2.23	1.91	1.06
Mass % to conc	%	0.96	1.02	1.09	1.02	1.07	1.10	0.99	1.23
Conc Au grade	g/t	37.15	24.73	14.57	36.94	37.16	52.61	24.00	88.81
Au recovery to conc	%	24.86	19.37	10.09	32.39	20.02	25.99	12.09	61.32
<b>Leach tests:</b>									
NaCN Consumption	kg/t	8.5	7.2	9.2	5.6	8.4	9.5	3.9	3.6
Au in solid tails	g/t	2.12	4.22	3.29	5.81	8.86	12.40	2.05	9.44
Au leach recovery	%	86.7	82.1	76.1	74.2	80.7	77.6	91.6	89.6

Source: SGS, Metallurgical Test Programme: Sadiola, March 2010

Based on the feasibility study, the weighted average Falcon recovery is 24.8% with an 82.1% intensive leach recovery of this concentrate. The estimated intensive leach cyanide consumption for plant design is of 7.08 kilograms per tonne of concentrate. Three samples with head grades under 0.15 gram per tonne were excluded from the analysis because they were not representative of the mine plan.

### 13.3.3 ROM and Gravity Tails Leach Tests

Leaching tests were conducted on gravity tails and ROM samples. The samples were dried and milled to 80 % passing 53 µm. There were five samples made up from the 6 previously made up clusters of samples, which in turn were made up from the 32 identified drill holes. The samples were identified IAMGold 1 (IMG1), IAMGold 2 (IMG2), AGA prefeasibility (AGA), AGA optimized conditions (AGA Optimized) and ROM.

In the IMG tests, 20 g/t of carbon was added prior to cyanide addition for a total residence time of 48 hours, as per CIL conditions. A cyanide addition rate of 2 kg/t was selected in order to ensure the test was conducted under conditions of excess cyanide.

In the AGA test pre-oxidation was included at a minimum 20 ppm oxygen and at natural pH for 2 hours, lead nitrate addition was conducted at a pH of 9.7 and the carbon addition was 30 g/L after 24 hours leaching for eight (8) hours in CIP.

The AGA Optimized and ROM test series were conducted under the same conditions as on a general test series. The leach test conditions for the various samples are shown in Table 13.16. All the tests were conducted as bottle rolls leaching tests.

**Table 13.16: Gravity Tails and ROM Leach Test Conditions**

Description	Grind p80	Res. Time (h)	Pre Ox	Leach pH	H2O2	PbNO <sub>3</sub> (g/t)	Carbon	Solids Content (%)	Cyanide (kg/t)
IMG 1	53	48h leach / CIL	No	10.5	No	No	CIL	35	2
IMG 2	53	48h leach / CIL	No	12	No	No	CIL	35	2
AGA	53	24h leach 8h CIP	Yes	9.7	No	100	CIP	50	1.5
AGA Optimized	53	48h leach / CIL	Yes	10.5	No	244	CIL	45	0.6
ROM	53	48h leach / CIL	Yes	10.5	No	244	CIL	45	0.6

Source: SGS, Metallurgical Test Programme: Sadiola, March 2010

The average results for the various clusters as reported by SGS are shown in Table 13.17.

**Table 13.17: Summary – Average Cyanide Leach Test Results**

Description	Cluster 1	Cluster 2	Cluster 3	Cluster 4	Cluster 5	Cluster 6	Un-clustered 1	Un-clustered 2
IMG 1	79.71	76.52	65.18	76.78	76.14	62.66	70.86	80.21
IMG 2	80.24	75.44	67.76	76.58	78.21	62.19	69.75	80.11
AGA	88.32	75.85	64.87	75.01	77.34	63.55	69.19	62.79
AGA Optimised	76.60	71.19	51.79	77.13	79.99	57.65	73.39	80.1

Source: SGS, Metallurgical Test Programme: Sadiola, March 2010

For each cluster, the following results were highlighted:

#### 13.3.3.1 Cluster 1

The highest dissolution for Cluster 1 was found to 87.75% using AGA prefeasibility conditions and based on the head assay, with 0.13 g Au/t in the residue. The cyanide consumption for this sample was 0.98 kg/t and a lime consumption of 0.20 kg/t was necessary to maintain the pH.

#### 13.3.3.2 Cluster 2

The highest dissolution for Cluster 2 was 94.25% using AGA optimized conditions based on the assayed head, with 0.18 g Au/t in the residue. The cyanide consumption for this sample was 0.42 kg/t and a lime consumption of 0.53 kg/t was necessary to maintain the pH.

#### 13.3.3.3 Cluster 3

The highest dissolution for Cluster 3 was 94.25% using IMG 1 conditions based on the assayed head, with 0.36 g Au/t in the residue. The cyanide consumption for this sample was 0.93 kg/t and a lime consumption of 0.40 kg/t was necessary to maintain the pH.

#### 13.3.3.4 Cluster 4

The highest dissolution for Cluster 4 was 92.22% using AGA optimized conditions based on the assayed head, with 0.01 g Au/t in the residue. The cyanide consumption for this sample was 0.18 kg/t and a lime consumption of 0.61 kg/t was necessary to maintain the pH.

#### 13.3.3.5 Cluster 5

The highest dissolution for Cluster 5 was 89.95% using AGA optimized conditions based on the assayed head, with 0.14 g Au/t in the residue. The cyanide consumption for this sample was 0.42 kg/t and a lime consumption of 0.70 kg/t was necessary to maintain the pH.

#### 13.3.3.6 Cluster 6

The highest dissolution for Cluster 6 was 68.21% using AGA optimized conditions based on the assayed head, with 0.47 g Au/t in the residue. The cyanide consumption for this sample was 0.55 kg/t and a lime consumption of 3.29 kg/t was necessary to maintain the pH.

#### 13.3.3.7 Un-clustered

The highest dissolution for the un-clustered samples was 80.21% using AGA optimized conditions based on the assayed head, with 0.13 g Au/t in the residue. The cyanide consumption for this sample was 0.87 kg/t and a lime consumption of 1.72 kg/t was necessary to maintain the pH.

Differences between the IMG and AGA test results were mainly in reagent consumption. The weighted average cyanide consumption for IMG1 was 0.54 kg/t compared to IMG2 with 0.41 kg/t and AGA with 0.78 kg/t. Cyanide consumption was lower in IMG2 tests with a higher pH. The lime consumption was on average 0.60 kg/t for IMG1, 4.71 kg/t for IMG2 and 0.78 kg/t for the AGA conditions. The IMG1 testwork conditions are considered the best with regards to reagent consumption and in minimizing the environmental detoxification requirements, should these be required.

The overall cyanide consumption calculated for the plant for intensive leaching, gravity tails leaching and elution was estimated at 0.632 kg/t. The estimated total cyanide consumption was based on 0.002 kg/t for intensive leaching, 0.54 kg/t for ROM and gravity tails leaching, 0.03 kg/t for elution and 0.15 kg/t free cyanide maintained in the leach circuit minus free cyanide recovery from the tailings thickener (estimated at 0.09 kg/t). The predicted cyanide consumption was lower than the cyanide added during the testwork. The overall cyanide consumption used in the December 2015 Feasibility Study Revision Report is 0.72 kg/t.

The lime consumption for the plant is estimated to be 0.61 kg/t as shown during the metallurgical testwork.

### **13.3.4 Overall Predicted Hard Sulphide Gold Recovery**

The feasibility study report combined the results of the gravity, intensive leach, gravity tails leach and solution and carbon gold losses to determine the overall expected recoveries as shown in Table 13.18.

The IMG1 testwork resulted in a weighted average recovery of 76.4 percent whereas the IMG2 tests yielded a recovery of 76.8%, the AGA test gave a recovery of 76.4 percent, the AGA optimized conditions gave 75.6% and the ROM samples gave 76.1%. The financial analysis is based on the plant predicted performance column which includes losses for solution and carbon.

### 13.3.5 Diagnostic Leaching

Diagnostic leaching testwork was conducted on eight gravity tail samples and one ROM sample. The samples were milled to P<sub>80</sub> of 53 µm. The diagnostic leach procedure involves the sequential solubilizing of the least-stable minerals via various pre-treatments, and extraction of the associated gold by cyanidation / CIL.

- To quantify the gold that can be extracted via direct cyanidation (i.e. free and exposed gold) a sample is subjected to a cyanide leach;
- To quantify the gold that is preg-robbed, but is recoverable via CIL processing, a second sample is subjected to a cyanide leach in the presence of activated carbon;
- To quantify the gold that can be extracted via a mild oxidative pre-leach, i.e. gold associated with calcite, dolomite, pyrrhotite, haematite, etc., the CIL residue is first subjected to hot HCl pre-treatment, followed by CIL dissolution of the acid treated residue;
- To quantify the gold associated with sulphide minerals (i.e. pyrite, arsenopyrite, etc.), the CIL residue is first subjected to a severe oxidative pre-treatment using hot HNO<sub>3</sub> followed by CIL dissolution of the acid-treated residue;
- To quantify the gold associated with carbonaceous material such as kerogen, the subsequent residue sample is subjected to complete oxidation via roasting, followed by CIL dissolution of the calcined product;
- The undissolved gold remaining in the final residue is assumed to be associated with gangue.

The average diagnostic leaching test results for the eight gravity tails samples are presented in Table 13.19. The results suggest that the expected recovery for CIL (cyanide leach plus preg-robbed CIL) is around 68%, which is lower than achieved during the gravity tails leaching testwork. The test procedure was investigated and did not reveal any issues.

Gold mineralogy and diagnostic leach testwork have proven that the hard sulphide ore contains refractory gold in the form of partially leachable aurostibite, locked in arsenopyrite, locked in marcasite, locked in pyrite and locked in other sulphides.

Table 13.18: Predicted Process Recoveries

Description	Units	Gravity	Intensive Leach	Leach Gravity Tails IMG1	Leach Gravity Tails IMG2	Leach Gravity Tails AGA	Leach Gravity Tails AGA Opt	Total IMG 1	Total IMG 2	Total AGA	Total AGA Opt	Total ROM	Predicted Plant Performance
<b>Calcite Marble</b>													
North	%	26.2	87.5	74.5	76.1	76.7	74.2	78.2	79.4	79.7	77.7	75.6	77.8
Middle	%	24.3	78.2	76.0	76.3	75.4	75.4	76.4	76.6	76.0	76.2	78.0	76.0
South	%	25.5	78.7	71.8	69.8	69.0	70.3	73.1	71.5	70.8	71.9	73.8	72.7
<b>Greywacke</b>													
North	%	18.5	92.1	81.7	83.7	82.1	83.8	83.6	85.2	83.9	85.2	85.4	83.2
Middle	%	13.9	74.7	70.3	72.3	70.6	60.2	70.9	72.6	70.9	61.9	65.1	70.5
South	%	37.6	86.3	76.7	75.6	65.5	69.6	80.0	79.1	74.3	74.1	71.5	79.6
<b>Dirorite</b>													
North	%	30.4	68.5	80.5	83.0	82.3	79.8	76.4	78.1	77.7	75.9	82.8	76.0
Middle	%	4.3	77.1	76.9	77.8	71.0	71.0	76.9	77.7	71.3	71.2	69.9	76.5
South	%	7.1	58.1	69.0	66.5	70.3	64.1	68.2	65.9	69.4	63.7	59.5	67.8

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

**Table 13.19: Gravity Tails - Average Diagnostic Leaching Test Results**

<b>Gold Association</b>	<b>Au Recovery (%)</b>
Available for direct cyanidation	61.8
Preg-robbed – CIL	6.5
HCL digestible	16.8
HNO <sub>3</sub> digestible	3.9
Carbonaceous	5.3
Quartz (balance)	5.7
<b>Total</b>	<b>100.0</b>

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

The average results for the ROM ore sample are presented in Table 13.20. The results also suggest that the expected recovery for CIL is in the 68 percent range.

**Table 13.20: ROM Diagnostic Leaching Test Results**

<b>Gold Association</b>	<b>Au Recovery (%)</b>
Available for direct cyanidation	61.8
Preg-robbed - CIL	6.0
HCL digestible	26.6
HNO <sub>3</sub> digestible	1.6
Carbonaceous	0.4
Quartz (*balance)	3.7
<b>Total</b>	<b>100.0</b>

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

### 13.3.6 Thickening and Rheology

Thickening and rheology testwork was done on the 2009 and was conducted by Golder Pastec, FLSmidth, Outotec, and Paterson and Cooke.

#### 13.3.6.1 Paterson and Cooke

Four samples were submitted to Paterson and Cooke for thickening and rheology testwork, which included slurry behaviour tests, static sedimentation tests, bench-top dynamic thickening tests and

underflow rheology tests. The samples are described as being greywacke main zone deep, calcite marble main zone deep, calcite marble foot wall near surface and a composite sample. All four samples were previously milled to a P<sub>80</sub> of 53 µm and all the tests were done pH natural and 10.5.

The slurry behaviour tests are done to determine and predict if the thickening process requires a pre conditioning. The results showed that there is ± 10 percent of clay size material (-5 microns) and none of this clay was found to be in the problematic smectite group. The natural pH was in the order of 9 and the conductivities were low (0,35 to 0,45 mS/cm).

In the static sedimentation tests, there were eight different flocculent types tested which represented different ionic natures (anionic or non-anionic) and different molecular weights (low, medium and high). The settling rate results and supernatant clarity pointed to an optimum flocculent selection of M6260 which is an anionic flocculent of medium molecular weight.

Thickener feed slurry solids concentration was also evaluated. The testwork demonstrated that to get a satisfactory settling rate above 20 m/h, a 10% solids weight percentage was needed for a natural pH and 12.5% was required for a slurry pH of 10.5. Flocculent dosage rate tests were done on the four samples. The optimum dosage rate selected is 15 g/t.

The final static sedimentation test done was a static bed compaction test to determine the underflow solids concentration. Tests were done under two test conditions. The first is a one hour static un-raked test and the second is a 24 hour raked test. The results are shown in Table 13.21.

**Table 13.21: Static Bed Compaction Tests for Base Case Conditions**

Sample Description	Underflow Solids Conc. (%m) Static Gravity Sedimentation	Underflow Solids Conc. (%m) Static 24h Raked Sedimentation
Greywacke main zone deep (nat pH)	51.3	71.5
Greywacke main zone deep (pH 10.5)	52.2	68.4
Calcite marble main zone deep (nat pH)	56.3	75.0
Calcite marble main zone deep (pH 10.5)	55.0	74.5
Calcite marble foot wall near surface (nat pH)	56.7	73.5
Calcite marble foot wall near surface (pH 10.5)	55.5	73.5
Composite (nat pH)	56.2	73.2
Composite (pH10.5)	55.5	74.4

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

Dynamic thickening tests were conducted to determine the solids flux and rise rates. The data showed similar results for calcite marble and greywacke. The expected solids flux rate for very good overflow clarity is 0.3 t/m<sup>2</sup> h.

Rheology testwork included material property tests (solids specific gravity, particle size distribution, slurry pH and particle micrographs), benchtop tests (coefficient of sliding friction, solids freely and maximum settled concentration and penetrometer test), Boger slump tests and rotational viscometer tests. The findings were as follows:

- Solids density ranged from 2,735 kg/m<sup>3</sup> to 2,581 kg/m<sup>3</sup> ;
- Particle size distribution was 73 µm on average;
- Penetrometer tests showed that these fine slurries should be easily re-suspended in pipeline;
- The calcite marble main zone sample had a yield stress of 10 Pa at 68% solids;
- The flocculent structure breaks down within 30 seconds.

#### 13.3.6.2 Golder Pastec

Samples for metallurgical testing for metallurgical testing by Golder Pastec were prepared at SGS Johannesburg. The sample was milled to 80% passing 53 µm. Parts of the sample received at Golder Pastec were shipped to FLSmidth and Outotec for testing. The specific gravity of the sample was determined to be 2.68 t/m<sup>3</sup> .

The rheological testwork conducted by Golder Pastec were slump, static yield, water bleed, plug yield, viscosity and dynamic yield stress tests.

- Slump was approximately 256 mm at 75.5% solids;
- Static yield test was 25 Pa at 68% solids;
- Water bleed was between 7% and 8% with a yield stress between 700 Pa and 800 Pa for a slump of 254 mm;
- Plug yield stress was between 200 Pa and 300 Pa at 6 cm depth for a 254 mm slump;
- Viscosity and dynamic yield stress is 0.01 Pa S and 10 Pa respectively for 68% solids.

Settling, centrifuge and vacuum filtration techniques were used for dewatering tests. The results of the tests are summarized as follows:

- Flocculent screening test resulted in the selection of AN 926 VHM as the best flocculent. This is an anionic flocculent with very high molecular weight. A feed density between 20% and 25% with a 15 g/t to 25 g/t dosage should provide good settling characteristics;
- 4 L settling static settling test resulted in 65.1% solids underflow with 25 g/t flocculent dosage and 20% solids feed;
- Centrifuge tests on the 4 L static settling test underflow gave a maximum achievable high compression underflow density of 78.5% solids for 25 g/t flocculent dosage and 20% solids feed;
- The laboratory scale paste thickener which is fed semi continuously was able to achieve close to 68% solids while giving a 120 Pa yield stress. After shearing the yield stress was reduced to 32 Pa;
- The underflow from the laboratory scale paste thickener was vacuum filtered. This method was able to reach 81.5% solids.

#### 13.3.6.3 FLSmith

Sample characterization, flocculent screening, flux testing, continuous fill deep tube and underflow rheology test were conducted by FLSmith. Results are summarized as follows:

- Flocculent screening test resulted in the selection of MF-10 as the best flocculent. This is an anionic flocculent with high molecular weight. 15 g/t to 20 g/t dosage should provide good settling rates;
- A feed density of 10% will give the best flux rate;
- The continuous test achieved 68.2% solids after 2 hours. The recommended minimum unit area is 0.04 m<sup>2</sup>/tpd;
- The rheology test provided a yield stress of 35 Pa at 68% solids.

#### 13.3.6.4 Outotec

The Outotec testwork included sample characterization, flocculent screening, feed density optimization, underflow yield stress and general batch dynamic thickening tests. The results are summarized as follows:

- Flocculent selection test resulted in the selection of MF-10 as the best flocculent. This is an anionic flocculent with high molecular weight;
- 16 g/t to 30 g/t dosage should give good settling rates;
- A feed density of 20% solids is required;

- The continuous test gave 67.5% solids. The solids loading rate was 0.5 t/m<sup>2</sup> h;
- The rheology test gave a yield stress of 110 Pa at 68% solids.

### 13.3.6.5 Thickening Testwork Conclusions

The four thickening tests done by the different laboratories did not give the same results. Based on these numbers the thickener unit settling rate varied between 0.3 and 1.263 m<sup>2</sup> /tpd. The selected unit settling rate for the thickener sizing is 0.045 m<sup>2</sup> /tpd. The different results are presented in Table 13.22

**Table 13.22: Thickening Tests Results**

Company	Recommended Flocculent	Recommended Flocculent Dose g/t	Recommended Feed Density wt%	Unit Settling Rated t/m <sup>2</sup> h	Unit Settling Rated m <sup>2</sup> /tpd	Unit Settling Rated tpd/m <sup>2</sup>
P&C	Magnafloc 6260	15	10	0.300	0.139	7.2
Outotec	Magnafloc 10	16	20	0.500	0.083	12.0
FLSmidth	Magnafloc 10	15-20	10	1.042	0.040	25.0
Golder	AN 926 VHM	20	10	1.263	0.033	30.3

Source: IAMGOLD, Sadiola Deep Sulphide Feasibility Study, December 2010

### 13.3.7 Metallurgical Testwork Conclusions

Based on the analytical and testwork results, the hard sulphide ore is made of three (3) main lithologies, i.e: calcite marble, greywacke and diorite. Calcite marble makes up approximately 76.97% of the ore body and contains about 78.98% of the gold. Greywacke makes up about 14.40% for 12.07% of the gold and the diorite 8.34% of the ore containing 8.66% of the gold.

Mineralogical examination identified seven sulphide species in the samples (pyrite / marcasite, pyrrhotite, arsenopyrite, berthierite, chalcopyrite and molybdenite) with pyrite, pyrrhotite and arsenopyrite being the dominant minerals.

The arsenopyrite suggests a level a refractoriness of the ore, which could impact on gold recovery from the ore.

Gold deportment and sulphide liberation studies classified the gold as gold/electrum (made up of 90% gold) or aurostibite (made up of 44% gold), with most of the gold present in the samples present as gold/electrum. Gold grades in the head samples ranged from 1.26 g/t to 3.49 g/t.

Although the hard sulphide circuit will process ore from existing stockpiles, no samples from these stockpiles were selected for testwork in 2010. For comminution the hard oxide and hard sulphide stockpiles were assigned the average values of the hard sulphide drill hole samples. Process recoveries for the hard sulphide stockpile were assigned the average value of the hard sulphide drill hole samples and the hard oxide recovery is the same as the soft oxide recoveries from the current processing facility. In 2014, metallurgical testing was done by AGA/SEMOS to assess the gold recovery from the hard sulphide stockpile. Two samples were taken from the stockpile. A composite made from the two samples (50-50) was pulverized to 76% passing 53 microns prior to be treated with a falcon concentrator. Falcon concentrate was treated by intensive leaching. Falcon tail was split in four and leached for 24 hours under four different reagent concentration conditions. Activated carbon was added before the leach time to simulate the CIL process. Average gold recovery for this composite sample was evaluated at 77.5%. Hard sulphide stockpile recovery estimate stays unchanged from the feasibility study at 76%.

Preliminary gravity concentration testwork demonstrated that gold recovery to a gravity concentrate ranged from 26% to 57% with a mass pull between 2.26% and 3.00%.

Comminution testwork included Bond Impact Work index tests, JK Drop Weight tests, SAG Mill comminution tests, Bond Ball and Rod Mill Work Index tests and Pennsylvania Abrasion tests. Results show that the ore can be classified soft to medium hard. Results also indicated that the greywacke was hardest, followed by the diorite the calcite marble being the softest of the three ore types.

Gravity recovery and intensive leaching tests achieved good recoveries of gold with the weighted average Falcon recovery of 24.8% and an 82.1% intensive leach recovery for this concentrate. The estimated intensive leach cyanide consumption for plant design is 7.08 kilograms per tonne of concentrate.

The overall gold recovery by lithology and by location in the deposit for the hard sulphide ore is estimated to range from 68% to 83% with a weighted average recovery for the hard sulphide estimated at 76%.

The recoveries for other rock types used in the design of the processing plant are based on actual plant results. The average gold recovery from soft oxide ore is around 94% and from soft sulphide ore around 80%.

**14. MINERAL RESOURCE ESTIMATES****14.1 Summary**

The December 2015 Mineral Resource reported by AGA is an update of the previous December 31, 2014 Mineral Resource.

The December 31, 2015 Mineral Resource estimates currently reported for the mining operations at Sadiola are summarized in Table 14.3 and Table 14.4. These Mineral Resources are based on the resource models listed in Table 14.1 and the data sources listed in Table 14.2. For economic assumptions, please refer to Section 22.

**Table 14.1: Sadiola Mineral Resource Statement December 31, 2015 Resource Models**

Area/Pit	Model Name	Model Date	Resource Estimation Completed by
SSP, including FN2 and FN3	sspfm_201510.dm	Oct 2015	E. Maritz, AGA and C Nicholls, Bloy Consultants
FE2	fe2_march2015.dm	Mar 2015	L. Chanderman, AGA Contractor
Combined FE3, FE4 and Timbabougouni	fe34_may2014.dm	May 2014	E. Maritz, AGA
Tabakoto (Sekekoto)	Tab_20jan14.dm	Jan 2014	E. Maritz, AGA
Tambali	Tamsmu_14.dm	Feb 2014	E. Maritz, AGA

Source: AGA, 2015

**Table 14.2: Sadiola Mineral Resource Statement December 31, 2014 and 2015 Data**

Area/Pit	Model Name
SSP, including FN2 and FN3	Main Pit: exploration data only FN2 and FN3: exploration and advanced grade control
FE2	Exploration and all grade control
FN2	Exploration and all grade control
Combined FE3, FE4 and Timbabougouni	Exploration and all grade control
Tabakoto (Sekekoto)	Exploration data only
Tambali	2013 portion of the model: exploration data only 2014 portion of model: exploration data and all grade control

Source: AGA, 2014d, AGA, 2015f

**Table 14.3: Sadiola Inclusive Measured and Indicated Mineral Resources by Area (December 31, 2015)**

Area	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Sadiola SSP	0.60					100,000	1.9	190,224	6,116	100,000	1.9	190,224	6,116
Area 1	0.70					1,968	2.48	4,889	157	1,968	2.48	4,889	157
Area 2	0.63					2,758	1.57	4,333	139	2,758	1.57	4,333	139
Stockpiles	-	1,462	1.68	2,451		14,155	1.09	15,405	495	15,617	1.14	17,856	574
<b>Total</b>	<b>0.60</b>	<b>1,462</b>	<b>1.68</b>	<b>2,451</b>		<b>118,881</b>	<b>1.81</b>	<b>214,851</b>	<b>6,908</b>	<b>120,342</b>	<b>1.81</b>	<b>217,302</b>	<b>6,986</b>

Source: SEMOS, 2016b

Notes: Mineral Resources are quoted inclusive of Ore Reserves. The Measured and Indicated Resources are inclusive of those Mineral Resources modified to produce the Ore Reserve. The Mineral Resources are quoted using a gold price of US\$1,400. Rounding of figures may result in computational discrepancies. Gram per ton ("g/t"); gold ("Au"); kilogram ("kg").

**Table 14.4: Sadiola Inferred Mineral Resources by Area (December 31, 2015)**

Area	Cut-off (Au g/t) weighted average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Sadiola SSP	0.60	14,652	1.82	26,725	859
Area 1	0.70	71	2.86	203	7
Area 2	0.72	802	1.76	1,408	45
Stockpiles	-	0	0	0	0
<b>Total</b>	<b>0.61</b>	<b>15,524</b>	<b>1.83</b>	<b>28,336</b>	<b>911</b>

Source: SEMOS, 2016b

Notes: The Mineral Resources are quoted using a gold price of US\$1,400. Rounding of figures may result in computational discrepancies. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**14.2 Reconciliation**

A summary reconciliation between the 2014 Mineral Resource and the 2015 Mineral Resource declaration is presented in Table 14.5.

The primary reason for the increase in the Mineral Resource is the inclusion of Mineral Resources classified at the Inferred level of confidence in the Main Pit optimized pit shell. This material was not previously included. AGA has decided to include this material to align the Mineral Resource reporting undertaken at Sadiola with other AGA operations.

**Table 14.5: Sadiola Mineral Resource Reconciliation**

	Tonnes (Mt)	Au ('000 oz)	Comments
Previous (2014)	120.64	6,887.00	Published as at December 31, 2014
Depletion	-4.26	-151.00	Depleted in 2015 from FN3 and FE2 pits and stockpiles
Gold price	-9.77	-525.00	Reduction in the resource gold price assumption from \$1,600/oz in 2014 to \$1,400/oz in 2015 resulted in reduction to the Mineral Resource
Cost	2.68	148.00	Lower fuel price and improved efficiencies resulted in lower costs for 2015 optimizations
Exploration	2.62	135.00	Infill drilling at FE2 and SSP (northern portion)
Methodology	23.48	1,392.00	Including the Inferred Mineral Resource in the optimization of the shell defining the SSP Mineral Resource (only Measured and Indicated used in 2014)
Other	0.49	11.00	Change as a result of stockpile adjustments
Current (2015)	135.87	7,897.00	As at December 31, 2015

Source: SEMOS, (2015b)

Note: The Sadiola resource reconciliation includes Measured, Indicated and Inferred resource categories.

Snowden cautions that there is no guarantee that the Inferred Resources can be converted to Mineral Reserves, given their low confidence classification.

**14.3 Disclosure**

Mineral Resources reported in Section 14.1 were prepared by Ms E Maritz, Evaluation Manager, a full-time employee of AGA; Lisa Chanderman, an ex AngloGold Ashanti employee, employed as a contractor for the duration of the resource update for FE2, who had previously worked on Sadiola, and; Carrie Nicolls, a full time employee of Bloy Resource Evaluation who had also previously worked at Sadiola. Ms Nicolls updated the Sadiola Main Pit North and FN2 models. Their work was reviewed by Mr. T. Gell and the Chairman of the AGA Mineral Resource and Ore Reserve Steering Committee.

Mr. VA Chamberlain, MSc (Mining Engineering), BSc (Hons.) (Geology), MGSSA, FAusIMM is the current Chairman of the AGA Mineral Resource and Ore Reserve Steering Committee.

Mr. Chamberlain and Mr. Gell are QPs as defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014).

Snowden reviewed the Mineral Resource Estimates provided by AGA and SEMOS.

The QP responsible for the Mineral Resources presented in this Technical Report is Mark Burnett, Principal Consultant for Snowden. He is a registered Professional Scientist with the South African Council for Scientific Professions (SACNASP Reg. No. 400361/12), a Member of the Southern African Institute of Mining and Metallurgy and a Fellow of the Geological Society of South Africa. Mark is a geologist who has worked in the minerals industry for 24 years with specific involvement in mine production and Mineral Resource estimation, mainly for gold. He has worked as a geological consultant for eight years in a technical and advisory capacity for clients covering development and mine production for a number of different mineral commodities.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

#### **14.3.1 Known Issues that Materially Affect Mineral Resources**

The QP is unaware of any issues that materially affect the Mineral Resources in a detrimental sense. These conclusions are based on the following:

- The SEMOS Exploration License has an approved environmental operating license, and SEMOS rehabilitates mine working, drill sites, and drill access roads on an ongoing basis;
- SEMOS is the holder of a Sadiola Mining Permit in good standing;
- SEMOS has represented that there are no outstanding legal issues; no legal action, and injunctions pending against the company or the mining operation;
- SEMOS has represented that the mineral and surface rights have secure title;
- There is no known marketing, political or taxation issues;
- SEMOS has represented that the Project has strong local community support;
- SEMOS has successfully mined the Main Pit, FE3, FE4, FN3, and Tambali deposits and are currently mining the FE2 and FN3 pits as well as processing mineralized waste stockpiles originating from the Main Pit;
- SEMOS has successfully treated ore from the Main Pit, FE3, FE4, FN3 and Tambali deposits.

There are no known infrastructure concerns.

#### 14.4 **Assumptions, Methods and Parameters**

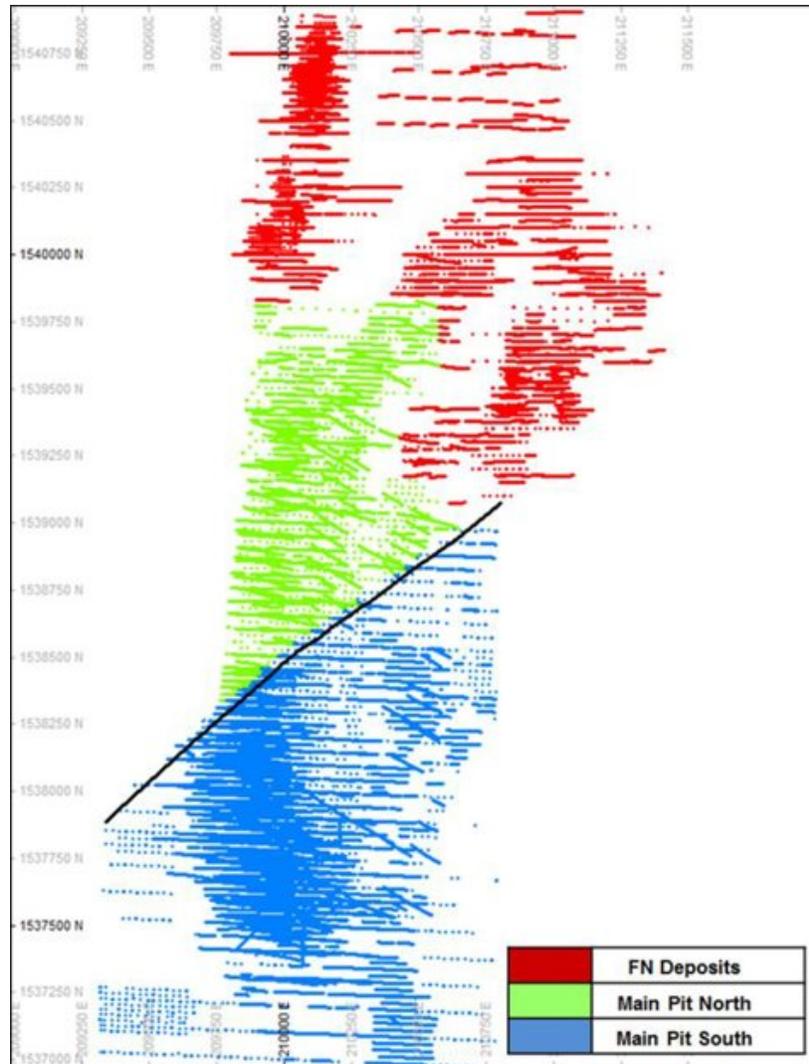
The basis of the Mineral Resource estimates for the Sadiola deposits is discussed in this section. The estimates were prepared in the following steps:

- Data validation – this was undertaken by the Sadiola Database Manager (Eric Imbeah) and reviewed by Snowden;
- Data preparation – this and subsequent steps are discussed below;
- Geological interpretation and modeling;
- Establishment of block models;
- Compositing of assay intervals;
- Exploratory data analysis of gold;
- Analysis of top cuts;
- Variogram analysis;
- Derivation of kriging plan and boundary conditions;
- Grade interpolation of gold using ordinary kriging;
- Validation of gold grade estimates;
- UC to report a recoverable resource;
- Classification into categories of Measured Resources, Indicated Resources and Inferred Resources as defined by the JORC (2012). These categories are equivalent to the definitions of Measured Resources, Indicated Resources and Inferred Resources as defined in CIM Definition Standards for Mineral Resources and Mineral Reserves (2014);
- Resource tabulation and resource reporting.

##### 14.4.1 **Drillhole Locations**

The drillhole traces shown in Figure 14.1 are for the Main Pit South, Main Pit North and FN deposits only. Since the 2013 Resource update, the FN resource models, which are extensions to the Main Pit mineralization, were joined to the Main Pit resource model to form one combined FN and Main Pit model. Three separate Leapfrog™ projects were setup for the Main Pit South, Main Pit North and FN deposits.

Figure 14.1: Plan of Drillhole Traces for the Main Pit and FN Areas



Source: AGA, 2015a

#### 14.4.2 Database

To verify that any new data contained no errors, and that all errors in the previous data had been corrected, basic data validation checks were repeated. Any remaining errors found were corrected, or eliminated, before proceeding with modeling and estimation. These included:

- Missing collar coordinates;
- Missing survey, assay or lithological data;
- Duplicate records;
- Interval errors (missing intervals, overlaps etc.);

- Zero or missing grades;
- Anomalous high or low SG values;
- Anomalous collar or downhole survey readings;
- Mis-logged lithologies;
- Discrepancies between the hole maximum depth in the collar table and maximum interval depth in interval tables.

Any errors found were corrected or eliminated before proceeding with modelling and estimation.

During previous Mineral Resource estimates, a number of errors were noted which were corrected after export from the master database and prior to estimation (in the Datamine™ files used for estimation) (AGA, 2015a; AGA, 2015b; AGA, 2014c; AGA, 2015e; AGA, 2015f; Bloy Resource Evaluation, 2015a; Bloy Resource Evaluation, 2015b).

#### **14.4.3 Geological Interpretation and Modelling**

The mineralization at Sadiola is controlled by a combination of lithology, structure, weathering and alteration. Wireframe models were generated for the topography, weathering surfaces, lithology boundaries and mineralization. A summary of the process followed is presented in this section of the Technical Report.

##### **14.4.3.1 Material Type**

Material types (also called rock types) are assigned to the resource estimation model based on surfaces modeled to delineated weathering and hardness. The surfaces were updated using logged exploration data ("EX") drillhole lithologies and hardness, which indicates the extent of blasting the material would require during mining as well as redox codes.

In order to take account of isolated, harder material, that would require blasting, "hardness probability" estimation is undertaken using inverse distance weighted estimation (IPD). The estimated probable hardness values are stored in a field named HVAL in the model. For the Mineral Resource estimation, only samples from exploration drillholes are used.

Blasting probability values are assigned to samples, according to the assigned hardness value, indicated by a D or H.

Material of hardness D4 (or H4) and D3 (or H9) were assigned blasting probabilities of one and 0.9, respectively, whereas material of hardness D2 (or H8) was assigned a blasting probability of 0.5 and free dig (soft D1 or H1 to H3; H5 to H7) material was assigned a blasting probability of zero.

All material below the hard-soft contact is assigned a value of 1 (100% blast probability). These estimated values (stored in the HVAL field) are then used to define the Blast Oxide or Blast Sulphide rock types. If a value greater than 0.9 is encountered in the soft oxides or soft sulphides, it was changed to Blast Oxide and Blast Sulphide respectively (AGA, 2015a; AGA, 2015b; AGA, 2014a; AGA, 2014b; and AGA, 2014c).

The final material types generated from the surfaces and the hardness probabilities were stored in the ROCKTYPE field (Table 14.6).

**Table 14.6: Rock Type Field Codes**

Rock type	Material type
1	Laterite and clay
2	Soft oxide
3	Hard oxide
4	Soft sulphide
5	Hard sulphide
6	Blast oxide
7	Blast sulphide
8	Transitional

Source: AGA, 2015a; AGA, 2015b; AGA, 2014a; AGA, 2014b; AGA, 2014c; AGA, 2014c; AGA, 2015e; AGA, 2015f; Bloy Resource Evaluation, 2015a; Bloy Resource Evaluation, 2015b

#### 14.4.3.2 Graphitic Alteration

Graphitic alteration was interpreted with a wireframe solid using the drillhole alteration coding. This was then used to flag the block model with a field named ALT (0 = None and 1= Graphitic). These areas are expected to be affected by graphitic alteration which may result in poorer metallurgical recoveries if not properly blended with non-graphitic material (AGA, 2015a; AGA, 2015b; AGA, 2014a; AGA, 2014b; AGA, 2014c; AGA, 2015e; AGA, 2015f; Bloy Resource Evaluation, 2015a; Bloy Resource Evaluation, 2015b).

#### 14.4.3.3 Lithology

The Main Pit deposit is primarily structurally controlled and the mineralization is related to faulting between marble and meta-graywacke and later northeast trending faults. The lithologies are mineralized along

various fault planes, but the majority of the mineralization occurs to the east of the fault and in the marbles. Lithologies have not yet been modeled for the FN pits. Lithologies have not been incorporated into the Main Pit and FN estimation domains, but have been flagged in the model.

An evaluation of the statistical characteristics of the dolerite, marble and meta-graywacke mineralization is recommended for future resource estimations, to determine if separate estimation is warranted (AGA, 2015a).

#### 14.4.3.4 Mineralization Interpretation

Most of the mineralized envelopes were modeled in Leapfrog Geo™ software using the automatic grade interpolation technique. The methodology employed is iterative, whereby a series of grade shells are created and visually assessed and compared to any available exploration and grade control data in order to determine which grade shell best defines the mineralization and limits the amount of internal waste.

Some manual adjustments of the envelopes were required to fine tune the result. This manual adjustment involved using “dummy” high or low grade points to either extend mineralization where continuity was less than desired or restrict mineralization where it was more than desired (or where there were unreasonable extensions beyond data support – often termed Leapfrog™ “blowouts”) (AGA, 2015a; AGA, 2015b; AGA, 2014a; AGA, 2014b; AGA, 2014c, AGA, 2015e, AGA, 2015f, Bloy Resource Evaluation, 2015a, Bloy Resource Evaluation, 2015b).

Mineralization envelopes at Tambali Central and South were modeled in Datamine Studio 3™, using a 0.4 g Au/t cutoff while the Tambali North mineralization was modeled in Leapfrog™ using a threshold of 0.35 g Au/t. A similar methodology was employed as described above (AGA, 2014c).

The FE3 and FE4 deposit is divided into three domains (based on mineralization style and structural controls): FE3 and Timbabougouni are defined as Domain 1; FE4 West and the “Gap Area” as Domain 2; and FE4 Main (the shear) as Domain 3 (AGA, 2014b).

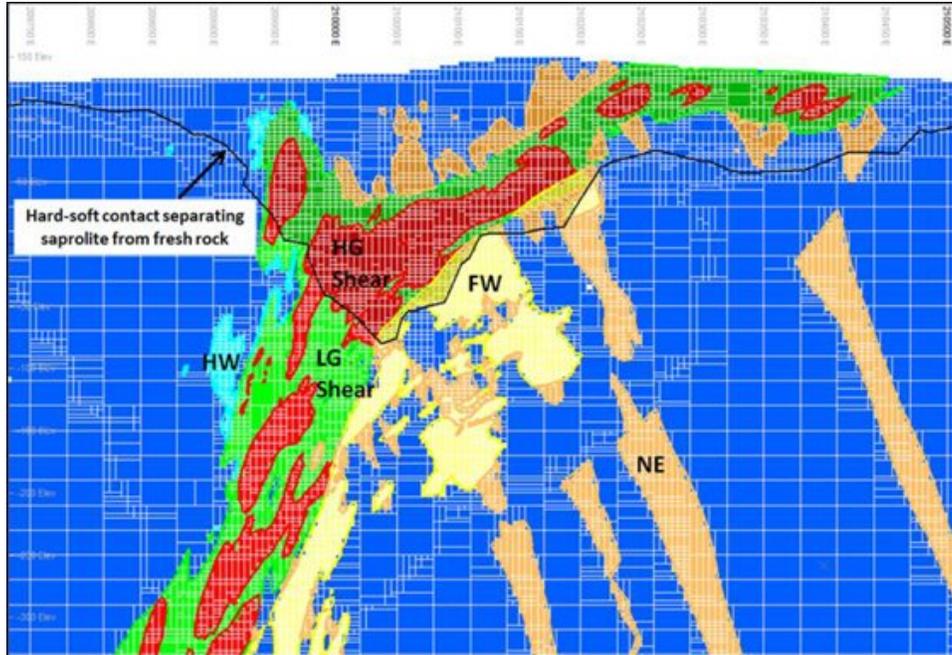
The FE2 and FN2 updates were based on revised mineralization envelopes created by AGA using Leapfrog Geo™ software. Minor changes were made to the FE2 model domains (AGA, 2015f). When modelling the FN2 area, Bloy (Bloy Resource Estimation, 2015b), modelled the alluvial deposits as a separate domain, based on the revised interpretation provided by AGA. In addition, the mineralized “halo” cut-off was reduced to 0.3 g Au/t for the oxides and the deep northeast footwall sulphide structures were removed from the model, due to a reinterpretation of their continuity.

Mineralization at FE2 strikes north-south (for 1.1 km) and dips at 50° towards the east. A break in mineralization occurs near its center where a dolerite dyke has intruded along a fault which separates and displaces the mineralization (AGA, 2015b).

The Main Pit South wireframes were updated in 2013. The envelopes for the main shear, footwall splays and northeast trends were based on visual analysis of the assay values together with assessment of a series of grade shells at different thresholds. The minimum threshold is 0.7 g Au/t, with a further high grade inner core envelope based on a 2 g Au/t threshold. The hangingwall (HW) envelope was based on a 0.5 g Au/t threshold.

The updates showed a more constrained HW interpretation, slightly narrower foot wall (FW) mineralization and a less steep main shear in the south of the Main Pit. The Main Pit South shear is generally less steep and in some instances slightly narrower than the Main Pit North shears (Figure 14.2) (AGA, 2015a).

**Figure 14.2: West-East Section – Main Pit South Mineralization Domains**

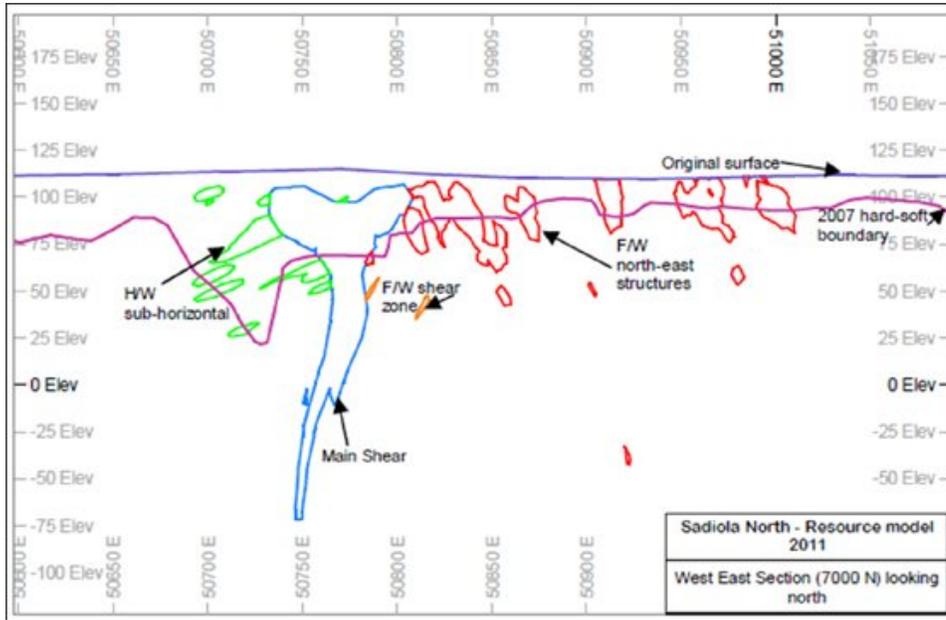


Source: AGA, 2015a

The Main Pit North model was updated in 2013, incorporating additional drilling data (Figure 14.3), but the domaining approach for the Main Pit North has remained unchanged. Mineralized envelopes for the following units were modeled using a threshold of 0.7 g Au/t, based on visual examination of the data and by running a series of iterative grade shells (AGA, 2015a):

- Main shear;
- Hangingwall splays;
- Hangingwall sub-horizontal;
- Hangingwall northeast trends;
- Footwall splays;
- Footwall northeast trends.

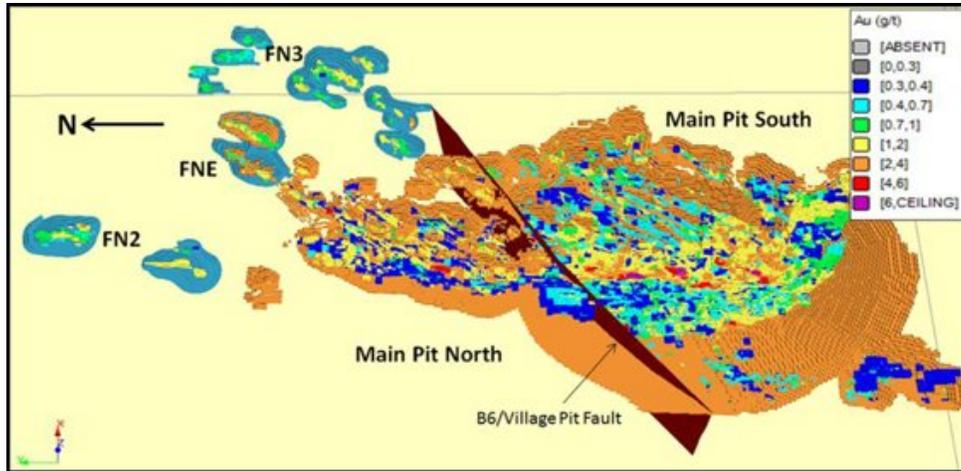
**Figure 14.3: West-East Section – Main Pit North Mineralization Domains**



Source: AGA, 2015a

The two major faults within the Sadiola Main Pit area are shown in Figure 14.4 and include the B16 and B6 (Village Pit) faults. Trend analysis across these faults showed a significant change in grade across the B6/Village Pit fault, but no significant change across the B16 fault. As such, the Main Pit was separated into a northern and southern domain along the B6/Village Pit fault (AGA, 2015a).

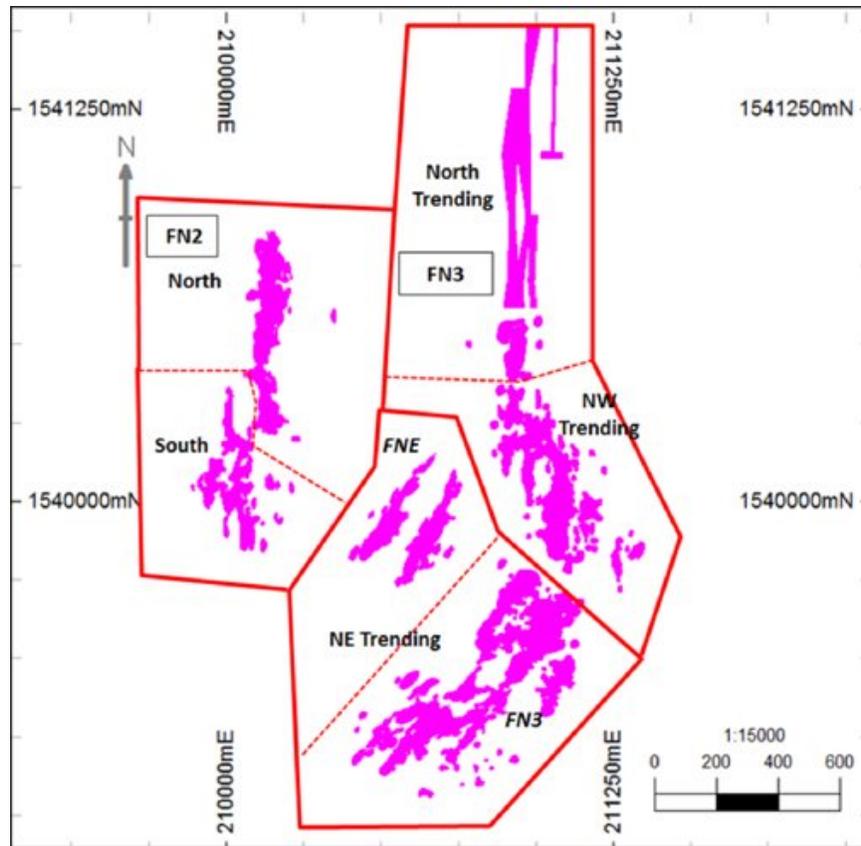
Figure 14.4: Main Pit Subdivision along the Village Pit Fault



Source: AGA, 2015a

The FN mineralization comprises five main trends which were further sub-domained into laterite, saprolite and fresh mineralization (Figure 14.5). The envelopes were based on a 0.30 g Au/t threshold.

Figure 14.5: Plan View of FN Mineralization Trends and Domains



Source: AGA, 2015a

#### 14.4.4 Compositing of Assay Intervals

Drillhole samples were coded according to mineralization, lithology, weathering and structure in the kriging zone ("KZONE") field prior to compositing.

The drillholes were generally sampled at either 1 m or 2 m intervals. Samples selected within individual mineralization wireframes were composited to 2 m intervals while honoring the domain codes. In order to maintain equal sample support and avoid discarding of samples, the composite length within a domain was set to be as close to 2 m as possible without forming a residual sample (MODE=1 in the Datamine™ "COMPDH" process). Validations of the composites confirmed nearly no sample loss or gain in terms of length and metal accumulation (AGA, 2015a; AGA, 2015b; AGA, 2014a; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

#### 14.4.5 Declustering

Most of the datasets were not declustered, as the data is not considered to be significantly clustered and, it was considered that this would not have a significant impact on the domain statistics.

Only the FE3 and FE4 data were declustered (AGA, 2015a). Declustering was carried out using cell weighting with a cell size of 10 m by 10 m by 2 m, based on drillhole spacing. The declustered data was used for statistical analysis.

#### **14.4.6 Exploratory Data Analysis**

The drillhole samples were coded according to mineralization, lithology, weathering and structure in the EZONE or KZONE field. Statistical analysis was undertaken on the coded, composited data for gold for each domain (AGA, 2015a; AGA, 2015; AGA, 2014b; and AGA, 2014c) (Table 14.7).

Snowden independently reviewed the statistical analysis for the FN2 area (Bloy Resource Estimation, 2015b) and was able to reproduce Bloy's results.

#### **14.4.7 Top Cuts**

As a result of the long, high grade tails of the distributions, coefficients of variation (CV = standard deviation/mean) were large (>1.5) in some estimation domains. For estimation, the influence of extreme grades was minimized by making use of top capping. Top capping was applied to prevent over-estimation in small subsample sets due to disproportionate high grade samples, through constraining extreme grades. The influence of the extreme grade is controlled by resetting those extreme grades to a more stable or realistic grade, (AGA, 2015a; AGA, 2015f, Bloy Resource Evaluation, 2015b).

The domain histograms, log probability plots and mean and variance plots were assessed to determine appropriate grade caps which were kept at less than about 2% of the composite data, except in the FN2 South Fresh; where a small number of top caps (four) resulted in a high percentage of values being capped. Table 14.8 and Table 14.9 show the top cap grades used for the primary deposits at Sadiola.

**Table 14.7: Gold Grade Summary Statistics – Main Pit and Related FN Estimation Domains**

<b>KZONE</b>	<b>Domain</b>	<b>No.</b>	<b>Minimum</b>	<b>Maximum</b>	<b>Mean</b>	<b>CV</b>
<b>FN domains</b>						
10	Waste	44,968	0.00	84.17	0.16	3.89
13	FN2 South – Laterite and saprolite	933	0.01	235.00	1.73	4.90
15	FN2 South – Hard	33	0.06	126.00	12.39	2.10
21	FN2 HW (fresh)	48	0.09	2.79	0.83	0.77
31	FN2 FW (fresh)	53	0.15	5.24	0.91	0.93
41	FN2 SW (fresh)	49	0.16	2.46	0.76	0.71
50	Alluvials	1,065	0.00	17.11	0.63	2.46
53	FN3 NW – Laterite and saprolite	1,634	0.02	58.50	0.77	2.30
55	FN3 NW – Hard	261	0.04	9.17	0.86	1.40
121	FN 2 HW (oxide)	245	0.01	15.32	1.33	1.37
131	FN2 FW (oxide)	199	0.02	84.91	2.42	3.51
141	FN 2 SZ (oxide)	388	0.10	8.13	0.88	1.06
<b>Main Pit South domains</b>						
100	Main Pit South FW (fresh)	5,336	0.003	75.13	1.91	1.45
200	Main Pit South HW (oxide and fresh)	3,990	0.004	718.00	1.40	9.96
300	Main Pit South NE trend (fresh)	3,637	0.002	27.63	0.66	2.45
421	Main Shear South low grade (fresh)	7,750	0.003	87.94	1.06	1.31
422	Main Shear South high grade (fresh)	4,282	0.050	147.21	3.94	1.39
1100	Main Pit South FW (oxide)	685	0.017	63.25	2.64	1.71
1300	Main Pit South NE trend (oxide)	6,999	0.006	87.32	1.65	2.01
1421	Main Shear South low grade (oxide)	9,320	0.002	67.00	1.05	1.15
1422	Main Shear South high grade (oxide)	12,283	0.004	281.00	5.16	1.43

KZONE	Domain	No.	Minimum	Maximum	Mean	CV
<b>Main Pit North domains</b>						
110	Main Pit North FW splays (fresh)	281	0.08	12.5	1.81	0.99
210	Main Pit North HW (fresh)	349	0.13	18.10	1.62	1.06
220	Main Pit North HW NE (fresh)	93	0.21	5.09	1.39	0.73
230	Main Pit North HW sub – hor. (fresh)	109	0.16	12.83	1.35	1.00
310	Main Pit North NE trend (fresh)	1,261	0.05	33.57	2.37	1.35
410	Main Pit North Main SZ (fresh)	744	0.00	54.81	2.11	1.67
1110	Main Pit North FW splays (oxide)	145	0.17	42.99	2.40	1.89
1210	Main Pit North HW (oxide)	30	0.11	2.97	0.83	0.91
1220	Main Pit North HW NE (oxide)	833	0.00	15.00	0.90	1.06
1230	Main Pit North HW sub – hor. (oxide)	763	0.03	11.99	1.13	1.02
1310	Main Pit North FW NE trend (oxide)	5,009	0.00	154.00	2.41	2.57
1410	Main Pit North Main SZ (oxide)	1,940	0.00	49.84	1.85	1.46
<b>FE2 domains</b>						
<b>1</b>	<b>Laterite and Saprolite</b>	<b>3,418</b>	<b>0.00</b>	<b>54.86</b>	<b>1.74</b>	<b>1.53</b>
<b>2</b>	<b>Hard</b>	<b>157</b>	<b>0.02</b>	<b>16.50</b>	<b>1.45</b>	<b>1.61</b>
<b>3</b>	<b>Waste</b>	<b>20,239</b>	<b>0.00</b>	<b>1.66</b>	<b>0.06</b>	<b>1.41</b>

Source: AGA, 2015a; AGA, 2015f; Bloy Resource Evaluation, 2015b

Note: CV – coefficient of variation

Note: The QP has updated those areas where additional drilling and/ or a change in domaining has occurred. If no additional drilling, or a change in geological interpretation has occurred, the summary statistics will remain unchanged.

Table 14.8: Grade Capping – Main Pit and Related Estimation Domains

Domain	Top Cap (g Au/t)	No. Capped (%)	Mean before Capping (g Au/t)	CV before Capping	Top Capped Mean (g Au/t)	Top Capped CV
Waste	1.0	1.5	0.16	3.89	0.14	1.32
FN2 South – Oxide	16.0	0.6	1.34	5.4	1.06	1.9
FN2 South – Fresh	21.0	8.9	9.20	2.5	4.27	1.6
FN 2 HW (fresh)	-	-	0.83	0.77	0.83	0.77
FN 2 FW (fresh)	-	-	0.91	0.93	0.91	0.93
FN 2 SZ (fresh)	-	-	0.76	0.71	0.76	0.71
Alluvials	-	-	0.63	2.46	0.63	2.46
FN 2 HW (oxide)	-	-	1.33	1.37	1.33	1.37
FN 2 FW (oxide)	16	2.5	2.42	3.51	1.66	1.74
FN 2 SZ (oxide)	-	-	0.88	1.06	0.88	1.06
FN3 NW – Oxide	7.0	0.6	0.77	2.3	0.72	1.2
FN3 NW – Fresh	-	-	0.86	1.4	-	-
FN3 North – Oxide	5.0	1.1	0.58	2.3	0.51	1.5
FN3 North – Fresh	-	-	0.41	0.7	0.41	0.7
FN NE trending	22.5	0.7	1.92	2.4	1.78	1.7
South FW (fresh)	20.0	0.2	1.66	1.6	1.64	1.4
South HW (oxide and fresh)	10.0	0.2	0.77	11.7	0.62	1.4
South NE trend (fresh)	10.0	0.2	0.45	2.6	0.43	2.2
South Shear low grade (fresh)	50.0	0.0	1.36	1.5	1.35	1.4
South Shear high grade (fresh)	90.0	0.0	2.98	1.6	2.97	1.5
South FW (oxide)	25.0	0.5	2.47	1.8	2.40	1.6
South Shear low grade (oxide)	43.0	0.1	1.63	2.4	1.58	1.5
South Shear high grade (oxide)	100.00	0.10	4.28	1.60	4.25	1.40
Main Pit North FW splays (fresh)	-	-	1.81	0.99	1.81	0.99
Main Pit North HW (fresh)	-	-	1.62	1.06	1.62	1.06
Main Pit North HW NE (fresh)	-	-	1.39	0.73	1.39	0.73

Domain	Top Cap (g Au/t)	No. Capped (%)	Mean before Capping (g Au/t)	CV before Capping	Top Capped Mean (g Au/t)	Top Capped CV
Main Pit North HW sub – hor. (fresh)	-	-	1.35	1.00	1.35	1.00
Main Pit North NE trend (fresh)	17.00	0.90	2.37	1.35	2.29	1.12
Main Pit North Main SZ (fresh)	26.00	0.30	2.12	1.66	2.05	1.38
Main Pit North FW splays (oxide)	22.00	0.70	2.40	1.89	2.25	1.53
Main Pit North HW (oxide)	-	-	0.83	0.91	0.83	0.91
Main Pit North HW NE (oxide)	-	-	0.90	1.06	0.90	1.06
Main Pit North HW sub – hor. (oxide)	-	-	1.13	1.02	1.13	1.02
Main Pit North FW NE trend (oxide)	33	0.50	2.41	2.57	2.25	1.72
Main Pit North Main SZ (oxide)	30	0.20	1.85	1.46	1.83	1.36

Source: AGA, 2015a; AGA, 2015f, Bloy Resource Evaluation, 2015 b

Note: CV – coefficient of variation

Note: The QP has updated those areas where additional drilling and/ or a change in domaining has occurred. If no additional drilling, or a change in geological interpretation has occurred, the summary statistics will remain unchanged.

Table 14.9: Grade Capping – Sadiola Satellite Deposits

	Description	Top Cap (g Au/t)	No. Capped (%)	Mean before Capping (g Au/t)	CV before Capping	Top Capped Mean (g Au/t)	Top Capped CV
<b>FE2</b>							
1	Soft ore	10	1.40	N/A	N/A	1.65	1.18
2	Hard ore	7	3.20	N/A	N/A	1.23	1.25
<b>FE34</b>							
14	Domain 1, laterite waste	1.68	0.17	0.12	0.12	1.69	0.96
15	Domain 1, saprolite waste	1.33	0.63	0.14	0.13	2.51	1.14
16	Domain 1, hard waste	1.21	1	0.15	0.14	2.51	1.26
24	Domain 2, laterite waste	0.97	0.22	0.1	0.1	1.97	0.72
25	Domain 2, saprolite waste	1.3	0.64	0.14	0.14	2.33	1.14
26	Domain 2, hard waste	0.84	0.84	0.08	0.07	3.2	1.68
34	Domain 3, laterite waste	0.97	0.76	0.13	0.12	2.99	1
35	Domain 3, saprolite waste	1.26	1	0.14	0.12	3.96	1.27
36	Domain 3, hard waste	0.4	1.42	0.05	0.05	2.36	1.57
<b>Tambali</b>							
121	North LG soft	16	0.40	1.31	1.9	1.25	1.4
221	North HG soft	15	0.40	1.64	2.1	1.57	1.3
22	North hard	16.5	0.50	1.52	1.75	1.47	1.5
51	South NE soft	7	0.30	0.87	2.7	0.79	1.1
52	South NE hard	-	-	0.91	1.1	-	-
61	South Main soft	9.5	1.60	1.7	1.58	1.58	1.3

Source: AGA, 2015b; AGA, 2014b; and AGA, 2014c; AGA, 2015f; Bloy Resource Estimation, 2015b

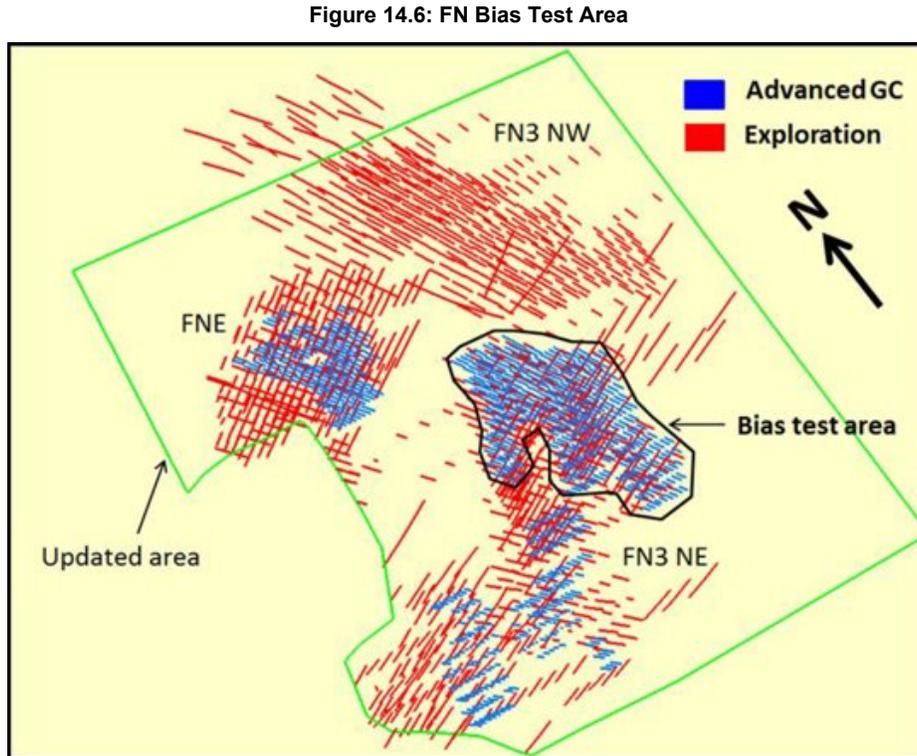
Note: CV – coefficient of variation

Note: The QP has updated those areas where additional drilling and/or a change in domaining has occurred. If no additional drilling, or a change in geological interpretation has occurred, the summary statistics will remain unchanged.

**14.4.8 Bias Testing**

Bias testing for the Main Pit South, showed no significant bias between RC and DD samples or between grade control and exploration (“EX”) samples. Review of QQ plots comparing the different datasets showed that the RC samples had slightly higher grade than the diamond hole samples, with the mean of the RC holes being 7% higher than that of the diamond holes. This was not deemed to be significant and the two datasets were combined for the estimation.

For the FN area, only mineralized samples (within the ore envelopes) from the selected bias test area were used for the bias testing between exploration and advanced grade control (Figure 14.6) (AGA, 2015a).



Source: AGA, 2015a

Histograms of the two datasets show similar statistics and shapes at the lower end of the distributions with good comparison between the median and 50<sup>th</sup> percentile values (up to approximately 2 g Au/t). However, differences start emerging at the higher ends of the distributions (from around the 75<sup>th</sup> percentile).

Overall, the advanced grade control mean is 10% higher than the EX sample mean. This could be related to the fact that the advanced grade control holes are closer drilled than the EX holes and therefore, the

high grade tail of the advanced grade control dataset was better sampled and represented. Another potential reason could be related to the drillhole orientations – all the advanced grade control drillholes were oriented towards the east; however, some of the earlier EX drillholes were oriented towards the west (as can be expected, there may not have been a good understanding of the mineralization trends at the earlier stages of exploration) (AGA, 2015a).

The observed difference of 10% between the EX and advanced grade control sample means that it is deemed to be at the limit of acceptance and is an issue that may need further consideration once mining of the FN3 deposit commences (AGA, 2015a).

Bias testing between EX and grade control data at Tambali showed no significance difference, therefore the two datasets were combined for estimation (AGA, 2014c).

#### **14.4.9 Variogram Analysis**

AGA used Snowden Supervisor™ (Version 8) geostatistical software to calculate and model the variograms. Variogram contours on the horizontal, across-strike and dip planes were evaluated to attain a direction of maximum continuity, from which the continuity of the mineralization in three dimensions was also modeled. The calculated experimental variograms were modeled using spherical models. The nugget effect was modeled from the downhole variogram. All variograms were standardized to a sill of one, representing the sample variance. The variogram models in Table 14.10 are for the main deposits at Sadiola.

The variogram axes rotations for each domain are shown in Table 14.11 in terms of a Datamine™ ZXZ rotation, where the X axis is aligned to the direction of maximum continuity.

Table 14.10: Normalized Variogram Parameters – Main Pit and Related FN Domains

Domain	Nugget Effect	Structure 1				Structure 2			
		Sill	Range Dir1 (m)	Range Dir2 (m)	Range Dir3 (m)	Sill	Range Dir1 (m)	Range Dir2 (m)	Range Dir3 (m)
Waste	0.30	0.44	20	50	12	0.26	150	70	100
FN2 South	0.5	0.15	35	2	4	0.39	45	20	4
FN 2 HW (fresh)	0.5	0.18	35	2	4	0.32	45	20	4
FN 2 FW (fresh)	0.5	0.18	35	2	4	0.32	45	20	4
FN 2 SZ (fresh)	0.5	0.18	35	2	4	0.32	45	20	4
Alluvials	0.24	0.31	30	25	5	0.45	145	48	9
FN 2 HW (oxide)	0.5	0.18	35	2	4	0.32	45	20	4
FN 2 FW (oxide)	0.5	0.18	35	2	4	0.32	45	20	4
FN 2 SZ (oxide)	0.5	0.18	35	2	4	0.32	45	20	4
FN3 N and NW	0.4	0.22	10	10	4	0.38	55	55	4
FN NE trending	0.3	0.2	65	14	3	0.5	65	50	4
South FW	0.56	0.25	5	5	5	0.19	35	35	35
South HW	0.20	0.43	130	15	6	0.37	130	130	12
South NE trend	0.29	0.30	7.5	7.5	7.5	0.41	32	32	32
SZ South – low	0.56	0.33	20	10	5	0.11	50	55	45
SZ South – high	0.55	0.30	15	5	5	0.15	25	20	5
North FW	0.64	0.2	22	9	6	0.16	75	25	10
North HW SZ	0.35	0.65	65	40	5	-	-	-	-
North HW NE	0.5	0.26	18	20	5	0.24	75	40	10
North HW subhz.	0.35	0.65	70	70	4	-	-	-	-
North NE	0.47	0.48	8	8	8	0.05	45	45	45
Shear North	0.23	0.38	20	20	18	0.39	80	60	35
Main Pit North FW splays (oxide)	0.64	0.2	22	9	6	0.16	75	25	10
Main Pit North HW (oxide)	0.35	0.65	65	40	5	-	-	-	-
Main Pit North HW NE (oxide)	0.50	0.26	18	20	5	0.24	75	40	10
Main Pit North HW sub – hor. (oxide)	0.35	0.65	70	70	4	-	-	-	-
Main Pit North Main SZ (oxide)	0.23	0.38	20	20	18	0.39	80	60	35
FE 2 Soft	0.35	0.34	18	10	3	0.31	40	28	5
FE 2 Hard	0.35	0.34	18	10	3	0.31	40	28	5
FE 2 Waste	0.49	0.06	57	29	23	0.45	89	53	49

Source: AGA, 2015a; AGA, 2015f; Bloy Resource Estimation, 2015b

The “fresh” variograms were also applied to the oxide domains (for the Main Pit domains); the variogram rotations were individually adapted to fit the generally shallower dipping oxide and steeper dipping fresh mineralization. This was done as a large part of the oxides are already depleted and the previous oxide variograms appeared to show similar grade continuity to that of the new “fresh” variogram models (AGA, 2015a).

**Table 14.11: Variogram Axes Rotations – Main Pit and Related FN Domains**

Domain	Datamine™ Z	Datamine™ X	Datamine™ Z
All waste	-100	60	0
FN2 South	-85	75	0
FN2 North	-85	80	0
FN3 NW	-105	35	0
FN 2 FW (fresh)	-65	-75	-25
FN 2 SZ (fresh)	-75	80	0
Alluvials	37	0	0
FN 2 HW (oxide)	-85	80	0
FN 2 FW (oxide)	-65	-75	-25
FN 2 SZ (oxide)	-75	80	0
FN3 North	-90	70	0
FN NE trending	-55	-45	0
FW oxide	-100	45	0
NE trend oxide	-66	-70	0
SZ South – low oxide	-100	50	10
SZ South – high oxide	-100	50	10
FW fresh	-100	60	0
HW oxide and fresh	-90	75	0
NE trend fresh	-66	-70	0
SZ South – low fresh	-100	70	10
SZ South – high fresh	-100	70	10
North FW (fresh)	-75	65	-25
North HW shear zone (fresh)	-85	90	-25
North HW northeast structures (fresh)	-65	115	40
North HW sub-horizontal structures (fresh)	-75	25	20
North NE trend (fresh)	-65	105	0
Main shear north (fresh)	-80	90	15
North FW (oxide)	-75	65	-25
North HW shear zone (oxide)	-85	90	-25
North HW northeast structures (oxide)	-65	115	40
North HW sub horizontal structures (oxide)	-75	25	20
North East Trend (fresh)	-65	105	-25
FW NE trend (oxide)	-65	105	-25
North NE trend (oxide)	-65	105	0
Main shear north (oxide)	-80	90	15
FE 2 Soft	100	30	0
FE 2 Hard	100	30	0
FE 2 Waste	-105	15	160

Source: AGA, 2015a; AGA 2015f; Bloy Resource Estimation, 2015b

**14.4.10 Review of FN2 Variography**

The QP independently completed variography for the FN2 zone and compared this to the variography of Bloy (Bloy Resource Evaluation, 2015b). Results show a slightly higher nugget but otherwise the models are not materially different to the FN2 variograms used for estimation.

**14.4.11 Block Model Setup**

A parent block size of 15 mE by 30 mN by 10 mRL, based on a kriging neighborhood analysis (KNA), was used in the estimation of the FE, FN and Main Pit South HW domains; a 30 mE by 30 mN by 20 mRL parent block size was used for the rest of the Main Pit domains (AGA, 2015a; AGA, 2015f). The FN2 alluvials were modeled using a parent block size of 30 mE by 30 mN by 10 mRL (Bloy Resource Evaluation, 2015b).

The minimum sub-cell size was 5 mE by 5 mN by 3.33 mRL was used for both the FN and Main Pit models, the Z length of 3.33 m was used to accommodate flitching (sub-benching). The block model parameters are shown in Table 14.12.

**Table 14.12: Main Pit and FN block model parameters**

Direction	Minimum	Maximum	Increment
Easting	209,000	213,800	30
Northing	1,536,580	1,541,530	30
Elevation	-750	1,090	20

Source: AGA, 2015a

The secondary satellite deposits used different origins, but all used a parent block size of 25 m by 25 m by 10 m (AGA, 2015a; AGA, 2014a; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

**14.4.12 Grade Interpolation and Boundary Conditions**

To determine which of hard or soft boundaries during grade estimation would be most appropriate, the grade variations across boundaries were investigated, which included the grade change across the hard-soft contact and across the mineralized envelope boundaries. The result is expressed as a search threshold measured in distance (m) from the contact. A “soft boundary threshold” refers to the distance that samples, falling outside of a particular domain, will still be seen when estimating the domain (when a boundary is gradational). For hard boundaries, this distance will be zero (sharp contact). A summary of the boundary analyses for the Main Pit and related FN domains is presented in Table 14.13 (AGA, 2015a).

The domain boundaries for FE3/4, FE2 and Tambali deposits were deemed to be hard based on the boundary analyses (AGA, 2015b; AGA, 2014b; and AGA, 2014c). Bloy Resource Evaluation (Bloy Resource Evaluation, 2015b) reviewed the boundary analysis undertaken previously by AGA and determined that all the boundaries for FN2 could be regarded as hard.

**Table 14.13: Boundary Analysis – Main Pit and Related FN Domains**

Surface/Wireframe	Soft Boundary
<b>FN domains</b>	
FN2 South	Soft boundary of 2 m
FN2 North	Hard boundaries
FN NE trending	Hard boundary
FN3 NW and North trending	Soft boundary of 2 m
<b>Main Pit South</b>	
Main shear low grade and waste	Hard boundary for estimation of waste. Soft boundary for mineralization estimation reduced to 2 m
FW splays and waste	Hard boundary for estimation of waste. Soft boundary for mineralization estimation reduced to 2 m
NE trends and waste	Hard boundary for estimation of waste. Soft boundary for mineralization estimation kept at 5 m
Hangingwall and waste	Hard boundary for estimation of waste. Soft boundary for mineralization estimation set to 4 m
Main shear low grade and main shear high grade	Soft boundary reduced to 4 m on either side of the contact
<b>Main Pit North</b>	
Shear and waste	Hard boundaries
Footwall splays and waste	Hard boundaries
NE trending footwall and waste	Hard boundary for estimation of waste. Soft boundary of 2 m for mineralization
HW shear and waste	Hard boundaries
HW northeast trending and waste	Hard boundary for estimation of waste. Soft boundary of 2 m for mineralization
HW sub-horizontal and waste	Hard boundaries

Source: AGA, 2015a

The estimation and search parameters used for ordinary kriging estimation were optimized through a KNA by reviewing the kriging efficiency and slope of regression (“Pslope”) as well as the negative weights.

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Ellipse orientations were based on the continuity observed in the variograms (AGA, 2015a; AGA, 2014b; AGA, 2014b; AGA, 2014c; Bloy Resource Evaluation, 2015b).

Parameters for the Main Pit and related FN estimation domains are summarized in Table 14.14.

**Table 14.14: Estimation and Search Parameters – Sadiola Main Deposit Estimation Domains**

Domain	Block Size (mE by mN by nRL)	Rotation			Search Radii			No. of Samples	
		Z	X	Z	X	Y	Z	Minimum	Maximum
Waste	30 x 30 x 20	-100	60	0	120	120	50	10	80
FN2 South – Oxide	15 x 30 x 10	-85	75	0	110	80	20	8	100
FN2 South – Fresh	15 x 30 x 10	-85	75	0	110	80	20	8	100
FN2 North – Oxide	15 x 30 x 10	-85	80	0	110	110	20	8	100
FN2 North – Fresh	15 x 30 x 10	-85	80	0	110	110	20	8	100
FN3 NW – Oxide	15 x 30 x 10	-105	35	0	120	120	20	8	100
FN3 NW – Fresh	15 x 30 x 10	-105	35	0	120	120	20	8	100
FN3 North – Oxide	15 x 30 x 10	-90	70	0	120	120	20	8	100
FN3 North – Fresh	15 x 30 x 10	-90	70	0	120	120	20	8	100
FN NE trending	15 x 30 x 10	-55	-45	0	120	120	20	8	100
South FW (fresh)	30 x 30 x 20	-100	60	0	120	120	120	10	80
North FW (fresh)	30 x 30 x 20	-75	65	-25	110	110	25	10	80
South HW (oxide and fresh)	15 x 30 x 10	-90	75	0	160	160	40	10	50
North HW Shear (fresh)	30 x 30 x 20	-85	90	-25	110	110	25	10	80
North HW NE (fresh)	30 x 30 x 20	-65	115	40	110	110	25	10	80
North HW horiz. (fresh)	30 x 30 x 20	-75	25	20	110	110	25	10	80
South NE trend (fresh)	30 x 30 x 20	-66	-70	0	120	120	120	10	80
North NE trend (fresh)	30 x 30 x 20	-65	105	0	110	110	25	10	80
North Shear (fresh)	30 x 30 x 20	-80	90	15	110	110	50	10	80

Domain	Block Size (mE by mN by nRL)	Rotation			Search Radii			No. of Samples	
		Z	X	Z	X	Y	Z	Minimum	Maximum
South Shear low grade (fresh)	30 x 30 x 20	-100	70	10	120	120	50	10	80
South Shear high grade (fresh)	30 x 30 x 20	-100	70	10	120	120	50	10	80
South FW (oxide)	30 x 30 x 20	-100	45	0	120	120	120	10	80
North FW (oxide)	30 x 30 x 20	-75	65	-25	110	110	25	10	80
North HW Shear (oxide)	30 x 30 x 20	-85	90	-25	110	110	25	10	80
North HW NE (oxide)	30 x 30 x 20	-65	115	40	110	110	25	10	80
North HW horiz. (oxide)	30 x 30 x 20	-75	25	20	110	110	25	10	80
South NE trend (oxide)	30 x 30 x 20	-66	-70	0	120	120	120	10	80
North NE trend (oxide)	30 x 30 x 20	-65	105	0	110	110	25	10	80
North Shear (oxide)	30 x 30 x 20	-80	90	15	110	110	50	10	80
South Shear low grade (oxide)	30 x 30 x 20	-100	50	10	120	120	50	10	80
South Shear high grade (oxide)	30 x 30 x 20	-100	50	10	120	120	50	10	80

Source: AGA, 2015a

To limit the influence of isolated high grade samples in poorly informed estimates, a kriging variance constraint was applied to the waste and northeast trend estimates only (Table 14.15). The grades of blocks with poor kriging variances were set to absent (not determined). The threshold was visually selected (AGA, 2015a).

**Table 14.15: Kriging Variance Thresholds for Poorly Informed Estimates**

Domain	Threshold
Waste (KZONE 10)	0.18
South NE trends (KZONES 300 and 1300)	0.16
South NE trends (KZONES 310 and 1310)	0.125

Source: AGA, 2015a

#### 14.4.13 Uniform Conditioning

UC was undertaken in Isatis™. UC is used to estimate the recoverable resources. Recoverable resources are a function of the selectivity that can be achieved by the equipment being used. Large equipment is more productive than small equipment, but selectivity is reduced. The selective mining unit (SMU) is the smallest volume of material on which ore-waste classification is determined, hence it is a clear measure of the equipment selectivity and has a large impact on the resource estimation (Neufeld *et al.*, 2005).

Recoverable resources and reserves are calculated using the SMU distribution that has been derived from the panel estimate and the change of support model. UC uses the discrete Gaussian model to accomplish the change of support (Neufeld *et al.*, 2005).

The panel and SMU sizes selected depend on the nature of the mineralization and the size of the equipment. It is planned that the current smaller scale equipment will be used for the oxide and narrower mineralization whereas larger bulk mining equipment will be used for the sulphides. This is reflected in the SMU sizes selected for recoverable resource estimation – more selective units have a smaller SMU (10 mE by 10 mN by 3.33 mRL) than the bulk mining units (10 mE by 10 mN by 10 mRL). The selection of the 10 mE by 10 mN by 10 mRL SMU for the Main Pit sulphides is supported by a study completed by ZStar Mineral Resource Consultants in 2010 (ZStar, 2010). The selection of the 10 mE by 10 mN by 3.33 mRL SMU for the FN deposits was determined following discussions with site Mining Engineers before commencing with the update. It was believed that this was a representative SMU size given the current mining equipment (which will be used to exploit the FN oxides).

The tonnage adjustment factor was calculated at half the SMU size and is expressed as a percentage of the panel size. Any proportions smaller than this are removed as these would not practically be recoverable during mining.

Some of the domains lacked a sufficient amount of data for UC and were therefore excluded (FN2 North fresh and FN2 South fresh; FN3 North fresh and the northern HW Oxide Shear Zone). UC on the waste areas was also not completed.

The SMU model is represented by the fields Gxxxx and Pxxxx which is the grade (“G”) and proportion (“P”) of a block above a particular cut-off grade (xxxx). For example, P050 is the proportion and G050 the grade of the block above a cut-off grade of 0.5 g Au/t.

The overall UC curves compared well with the theoretical curves (generated at both panel and SMU support with and without information effect), especially in the well informed domains, and the degree of selectivity achieved appeared reasonable (AGA, 2015a; AGA, 2015b; AGA, 2014b; AGA, 2014c; AGA, 2015f, Bloy Resource Evaluation, 2015b).

Snowden notes that if the grade tonnage curves for the ordinary kriged estimate align with the recoverable resource, there is little benefit in carrying out a change of support.

#### 14.4.14 Density

The density information for grade control samples is collected using a density probe and for EX samples using the immersion technique (Archimedes’ Principle). All density data collected is used to inform the estimation of global bulk densities assigned to each rock type (AGA, 2015a; AGA, 2015b; AGA, 2014a; AGA, 2014b; AGA, 2014c; AGA, 2015f, Bloy Resource Evaluation, 2015b).

It must be noted that no density measurements have been taken for the FE2 deposit. FE3 deposit densities were applied to the FE2 deposit; these two deposits are considered to be similar in rock type, alteration and mineralization style (AGA, 2015b). Average bulk densities were assigned to each ROCKTYPE (material type) as summarized in Table 14.16. Both Bloy Resource Evaluation (Bloy Resource Evaluation, 2015b) and AGA (AGA, 2015f), used slightly different bulk densities than that noted in earlier reports. This is not deemed material, but Snowden recommends that the reasons for these changes are clearly documented and explained.

**Table 14.16: Bulk Average Densities**

Rock Type	FN pits	Main Pit	Tambali	FE3	FE4	FE2	Sekokoto
1 Laterite	1.97	1.97	1.93	2.22	2.08	2.11	1.97
2 Saprolite (soft oxide)	1.82	1.82	1.95	1.97	1.92	1.97	1.70
3 Silicified oxide	2.56	2.56	2.40	2.16	1.99	2.05	2.50
4 Soft sulphide	1.98	1.98	2.00	2.46	1.95	2.46	1.70
5 Hard sulphide	2.70	2.70	2.65	2.40	2.56	2.70	2.70

Rock Type	FN pits	Main Pit	Tambali	FE3	FE4	FE2	Sekokoto
6 Blast oxide	2.16	2.16	2.40	2.10	2.10	2.05	2.10
7 Blast sulphide	2.37	2.37	2.40	2.10	2.10	2.70	2.10
8 Transitional	2.26	2.26	Soft (hard=0 or hval>=0.8) 2.10 Hard (hard=1) 2.40	2.23	1.94	Soft (hard=0) 2.21 Hard (hard=1) 2.46	Retains original density value of rock type 1-7

Source: AGA 2015a, AGA 2015b; AGA, 2014a; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Estimation, 2015b

#### 14.4.15 Model Validation

The estimates were validated using the following methods:

- Visual comparison of the input grades with the model estimates;
- Comparison of the global model and input means;
- Sectional plots comparing the number of composites, model grades and composite grades occurring within a specified distance on either side of a section;
- Grade-tonnage curves comparing the estimates to the theoretical model (block anamorphosis of the input point data computed with and without the information effect).

Visual validation showed that the model grades represented the input data well.

Globally, model means for most Sadiola domains were within 10% of the input data means which is considered to be within acceptable limits. Domains showing differences greater than 10% were generally poorly informed as a result of limited data in the domains. The FN waste domain showed a difference between model mean and composite mean of 65%, this is as a result of the kriging variance constraint applied to the estimates (AGA, 2015a; AGA, 2015b; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

Slice (or sectional) plots were generated for each domain comparing the mean of the estimates against the mean of the input data within a particular slice. Slices were generated by both northing and elevation. In well informed areas, the means compared well with the estimates and the estimates followed the trends present in the input data (AGA, 2015a; AGA, 2015b; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

Theoretical grade-tonnage curves (at panel support) were generated for the mineralized zones and were compared with the kriged grade-tonnage curves. The theoretical grade-tonnage curves were computed with and without the information effect and by performing a change of support on the input data (using a Discrete Gaussian Model). This allowed an assessment of the degree of “smoothing” in the ordinary kriged estimates. The degree of smoothing present in an estimate is a function of the amount, distribution and spacing of data available and the variogram model, and also of the choice of neighborhood samples used to make the estimate (AGA, 2015a; AGA, 2015b; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

#### **14.4.16 Mineral Resource Classification**

After all items specified within the JORC Code (2012) such as sampling techniques, data quality and estimation techniques were considered, the Mineral Resource was classified according to the AGA standard “15% rule” as described in the Mineral Resource and Ore Reserve Guidelines. This classification technique provides an average grade above cut-off estimate with less than 15% relative error at 90% confidence. For an Indicated Resource, annual production and for a Measured Resource, quarterly production should meet these criteria (SEMOS, 2014).

No resources were classified at the Measured Resource level of confidence (AGA, 2015a; AGA, 2015b; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

The Main Pit North and FN mineralization are extensions to the Main Pit South mineralization, but are significantly narrower with poorer geological and grade continuity. To take account of this, it was considered that the drill spacing used to define Indicated Resources in these areas should be closer, and a spacing of 25 m by 25 m was deemed appropriate (AGA, 2015a).

Areas outside the Indicated Resource with drilling space of up to 100 m by 100 m were classified as Inferred Resources.

Areas covered by drill spacing in excess of 100 m by 100 m (or which did not contain a sufficient amount of samples to produce a reliable kriged estimate) were not classified as a Mineral Resource, but were considered blue sky potential (AGA, 2015a; AGA, 2015b; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

For the secondary satellite deposits, areas covered with a drill spacing of 25 m by 25 m were classified at the Indicated Resource level of confidence and areas with 50 m by 50 m drillhole spacing at the Inferred Resource level of confidence. No resources were classified at the Measured Resource level of confidence (AGA, 2015a; AGA, 2015b; AGA, 2014b; AGA, 2014c; AGA, 2015f; Bloy Resource Evaluation, 2015b).

All Mineral Resources are quoted inclusive of Mineral Reserves.

#### **14.4.17 Mineral Resource Reporting**

The SEMOS Mineral Resources, as presented in this Technical Report, are reported in accordance with the guidelines of the JORC Code (2012), and the AGA reporting guidelines (2014), above the Mineral Resource cut-off grade.

The classification categories of Measured, Indicated and Inferred Mineral Resources classified in accordance with JORC (2012) are equivalent to the CIM categories of Measured, Indicated and Inferred Mineral Resources (CIM, 2014).

After assessing all items specified within the JORC Code (2012) such as sampling techniques, data quality and estimation techniques, the Mineral Resource was classified according to the AGA standard “15% rule”, as described in the Mineral Resource and Ore Reserve Guidelines. This classification technique provides an average grade above cut-off estimate with less than 15% relative error at 90% confidence. For an Indicated Mineral Resource, annual production, and for a Measured Mineral Resource, quarterly production, should meet these criteria.

During 2015, a total of 151,000 oz were processed. Reconciliation of the resource can be reviewed in Table 14.5.

The oxide resources are made up of eight discrete in-situ areas (current and future open pits) and stockpiles located in Zone A of the SEMOS Mining Permit (around the main ROM pad and the FE3/4 Satellite ROM pads). Of the eight areas, two were mined during 2015: FE2 and FN2.

A stockpile re-evaluation was completed in 2014. This resulted in a decrease of 145,000 oz (or 68%) in 2014 compared to 2013. This was mostly to restating the hard sulphide stockpile tonnages, following a drilling program, and a review of the survey data.

Table 14.17 and Table 14.18 show the cut-off grades used for the following Mineral Resource tabulations. The cut-off grades are based on a gold price of US\$1,400/oz. The deposits are economically viable at the Mineral Resource gold price of US\$1,400/oz. However, the sulphide (hard rock) resources will require a new processing plant.

The tonnages were estimated and are stated on a dry basis. These are shown on a 100% ownership basis.

**Table 14.17: Mineral Resource Gold Cut-off Grades – Main SSP and Main Satellite Pits**

Material type	SSP	FE4	FE3
Laterite/Clay (ox)	0.60	0.65	0.65
Saprolite	0.60	0.65	0.65
Siliceous oxide	0.60	0.65	0.65
Sulphide – soft	0.60	0.95	0.95
Hard sulphides	0.60	0.95	0.95
Blast oxide	0.60	0.65	0.65
Blast sulphide	0.60	0.95	0.95
Transition	0.60	0.65	0.65
Graphite	0.60	1.15	1.15

Source: SEMOS, 2014

**Table 14.18: Mineral Resource Gold Cut-off Grades by – Secondary Satellite Pits**

Material type	Tambali	FN2	FN3	FE2	Sekokoto
Laterite/Clay (ox)	0.55	0.60	0.60	0.55	0.65
Saprolite	0.55	0.60	0.60	0.55	0.65
Siliceous oxide	0.65	0.65	0.65	0.65	0.65
Sulphide – soft	0.90	0.9	0.90	0.90	0.65
Hard sulphides	0.90	0.95	0.95	0.90	0.95
Blast oxide	0.60	0.60	0.60	0.60	0.95
Blast sulphide	0.90	0.95	0.95	0.90	0.65
Transition	0.60	0.60	0.60	0.60	0.95
Graphite	1.10	1.15	1.15	1.10	0.65

Source: SEMOS, 2014

The Sadiola Inclusive Measured and Indicated Mineral Resources, by area, are presented in Table 14.19 to Table 14.37.

**Table 14.19: Sadiola Inclusive Measured and Indicated Mineral Resources by Area (December 31, 2015)**

Area	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Sadiola SSP	0.60	-	-	-	-	100,000	1.9	190,224	6,116	100,000	1.9	190,224	6,116
Area 1	0.70	-	-	-	-	1,968	2.48	4,889	157	1,968	2.48	4,889	157
Area 2	0.63	-	-	-	-	2,758	1.57	4,333	139	2,758	1.57	4,333	139
Stockpiles	-	1,462	1.68	2,451	79	14,155	1.09	15,405	495	15,617	1.14	17,856	574
<b>Total</b>	<b>0.60</b>	<b>1,462</b>	<b>1.68</b>	<b>2,451</b>	<b>-</b>	<b>118,881</b>	<b>1.81</b>	<b>214,851</b>	<b>6,908</b>	<b>120,342</b>	<b>1.81</b>	<b>217,302</b>	<b>6,986</b>

Source: SEMOS, 2016b

Notes: Mineral Resources are quoted inclusive of Ore Reserves. The Measured and Indicated Resources are inclusive of those Mineral Resources modified to produce the Ore Reserve. The Mineral Resources are quoted using a gold price of US\$1,400. Rounding of figures may result in computational discrepancies

**Table 14.20: Sadiola Inferred Mineral Resources per Area (December 31, 2015)**

Area	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Sadiola SSP	0.60	14,652	1.82	26,725	859
Area 1	0.70	71	2.86	203	7
Area 2	0.72	802	1.76	1,408	45
Stockpiles	-	-	-	-	-
<b>Total</b>	<b>0.61</b>	<b>15,524</b>	<b>1.83</b>	<b>28,336</b>	<b>911</b>

Source: SEMOS, 2016b

Notes: The Mineral Resources are quoted using a gold price of US\$1,400. Rounding of figures may result in computational discrepancies. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.21: Sadiola Main Pit – SSP Inclusive Measured and Indicated Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.60	-	-	-	-	4,623	1.47	6,804	219	4,623	1.47	6,804	219
Transition	0.60	-	-	-	-	3,056	1.93	5,889	189	3,056	1.93	5,889	189
Sulphide	0.60	-	-	-	-	92,321	1.92	177,531	5,708	92,321	1.92	177,531	5,708
<b>Total</b>	<b>0.60</b>	-	-	-	-	<b>100,000</b>	<b>1.9</b>	<b>190,224</b>	<b>6,116</b>	<b>100,000</b>	<b>1.9</b>	<b>190,224</b>	<b>6,116</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.22: Sadiola Main Pit – SSP Inferred Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.60	636	1.47	937	30
Transition	0.60	500	2.00	1,000	32
Sulphide	0.60	13,516	1.83	24,788	797
<b>Total</b>	<b>0.60</b>	<b>14,652</b>	<b>1.82</b>	<b>26,725</b>	<b>859</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.23: Area 1 – FE4 Inclusive Measured and Indicated Mineral Resource by Material Type (December 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.65	-	-	-	-	192	2.24	431	14	192	2.24	431	14
Transition	0.95	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide	0.74	-	-	-	-	64	2.32	147	5	64	2.32	147	5
<b>Total</b>	<b>0.67</b>	-	-	-	-	<b>256</b>	<b>2.26</b>	<b>578</b>	<b>19</b>	<b>256</b>	<b>2.26</b>	<b>578</b>	<b>19</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.24: Area 1 – FE4 Inferred Mineral Resource by Material Type (December 2015)**

Material	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.65	32	2.77	89	3
Transition	-	-	-	-	-
Sulphide	0.65	5	2.5	13	-
<b>Total</b>	<b>0.65</b>	<b>37</b>	<b>2.73</b>	<b>102</b>	<b>3</b>

Source: SEMOS, 2015c

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.25: Area 1 – FE3 inclusive Measured and Indicated Mineral Resource by Material Type (December 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.65	-	-	-	-	611	2.24	1,367	44	611	2.24	1,367	44
Transition	0.95	-	-	-	-	278	3.06	849	27	278	3.06	849	27
Sulphide	0.65	-	-	-	-	824	2.54	2,095	67	824	2.54	2,095	67
<b>Total</b>	<b>0.70</b>	-	-	-	-	<b>1,713</b>	<b>2.52</b>	<b>4,311</b>	<b>139</b>	<b>1,713</b>	<b>2.52</b>	<b>4,311</b>	<b>139</b>

Source: SEMOS, 2015c

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.26: Area 1 – FE3 Inferred Mineral Resource by Material Type (December 2015)**

Material	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.65	8	2.51	21	1
Transition	0.95	11	3.34	36	1
Sulphide	0.65	14	3.05	44	1
<b>Total</b>	<b>0.75</b>	<b>34</b>	<b>3.01</b>	<b>101</b>	<b>3</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.27: Area 2 – Tambali Inclusive Measured and Indicated Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.55	-	-	-	-	188	1.5	281	9	188	1.5	281	9
Transition	0.9	-	-	-	-	1	1.48	1	-	1	1.48	1	-
Sulphide	0.6	-	-	-	-	58	1.59	93	3	58	1.59	93	3
<b>Total</b>	<b>0.57</b>	-	-	-	-	<b>247</b>	<b>1.52</b>	<b>375</b>	<b>12</b>	<b>247</b>	<b>1.52</b>	<b>375</b>	<b>12</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.28: Area 2 – Tambali Inferred Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) weighted average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.55	93	1.23	114	4
Transition	0.95	2	2.15	5	-
Sulphide	0.60	18	1.89	34	1
<b>Total</b>	<b>0.57</b>	<b>113</b>	<b>1.35</b>	<b>153</b>	<b>5</b>

Source: SEMOS, 2015c

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.29: Area 2 – FN2 Inclusive Measured and Indicated Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.60	-	-	-	-	778	1.31	1,018	33	778	1.31	1,018	33
Transition	-	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide	1.10	-	-	-	-	5	1.59	7	3	5	1.59	7	3
<b>Total</b>	<b>0.60</b>	-	-	-	-	<b>783</b>	<b>1.31</b>	<b>1,026</b>	<b>33</b>	<b>783</b>	<b>1.31</b>	<b>1,026</b>	<b>33</b>

Source: SEMOS, 2015c

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.30: Area 2 –FN2 Inferred Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.60	55	1.76	97	3
Transition	-	-	-	-	-
Sulphide	-	-	-	-	-
<b>Total</b>	<b>0.60</b>	<b>55</b>	<b>1.76</b>	<b>97</b>	<b>3</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.31: Area 2 – FN3 Inclusive Measured and Indicated Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.60	-	-	-	-	543	1.54	838	27	543	1.54	838	27
Transition	0.95	-	-	-	-	224	2.08	466	15	224	2.08	466	15
Sulphide	0.60	-	-	-	-	176	1.85	325	10	176	1.85	325	10
<b>Total</b>	<b>0.68</b>	-	-	-	-	<b>943</b>	<b>1.73</b>	<b>1,629</b>	<b>52</b>	<b>943</b>	<b>1.73</b>	<b>1,629</b>	<b>52</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.32: Area 2 – FN3 Inferred Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.60	101	1.36	137	4
Transition	0.95	236	2.25	533	17
Sulphide	0.60	24	2.79	68	2
<b>Total</b>	<b>0.83</b>	<b>361</b>	<b>2.04</b>	<b>737</b>	<b>24</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.33: Area 2 – FE2 Inclusive Measured and Indicated Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.55	-	-	-	-	247	1.46	361	12	247	1.46	361	12
Transition	0.90	-	-	-	-	4	1.82	7	-	4	1.82	7	-
Sulphide	0.61	-	-	-	-	40	1.59	64	2	40	1.59	64	2
<b>Total</b>	<b>0.56</b>	-	-	-	-	<b>291</b>	<b>1.48</b>	<b>431</b>	<b>14</b>	<b>291</b>	<b>1.48</b>	<b>431</b>	<b>14</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.34: Area 2 – FE2 Inferred Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.55	1	1.24	2	-
Transition	-	-	-	-	-
Sulphide	-	-	-	-	-
<b>Total</b>	<b>0.55</b>	<b>1</b>	<b>1.24</b>	<b>2</b>	<b>-</b>

Source: SEMOS, 2015c

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.35: Area 2 – Tabakoto (formerly Sekekoto) Inclusive Measured and Indicated Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.65	-	-	-	-	476	1.76	835	27	476	1.76	835	27
Transition	0.95	-	-	-	-	-	2.32	1	-	-	2.32	1	-
Sulphide	0.65	-	-	-	-	18	2	36	1	18	2	36	1
<b>Total</b>	<b>0.65</b>	-	-	-	-	<b>494</b>	<b>1.76</b>	<b>872</b>	<b>28</b>	<b>494</b>	<b>1.76</b>	<b>872</b>	<b>28</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

**Table 14.36: Area 2 – Tabakoto (formerly Sekekoto) Inferred Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Inferred Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Oxide	0.65	270	1.55	419	13
Transition	0.95	-	-	-	-
Sulphide	0.65	-	1.95	1	-
<b>Total</b>	<b>0.65</b>	<b>271</b>	<b>1.55</b>	<b>420</b>	<b>13</b>

Source: SEMOS, 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Due to the uncertainty that may be attached to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Resource after continued exploration.

**Table 14.37: Stockpile\* Inclusive Measured and Indicated Mineral Resource by Material Type (December 31, 2015)**

Material	Cut-off (Au g/t) Weighted Average	Measured Resource				Indicated Resource				Measured + Indicated Resource			
		Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)	Tonnes ('000)	Au (g/t)	Metal (kg)	Metal (koz)
Stockpile	-	1,462	1.68	2,451	79	14,155	1.09	15,405	495	15,617	1.14	17,856	574
<b>Total</b>		<b>1,462</b>	<b>1.68</b>	<b>2,451</b>	<b>79</b>	<b>14,155</b>	<b>1.09</b>	<b>15,405</b>	<b>495</b>	<b>15,617</b>	<b>1.14</b>	<b>17,856</b>	<b>574</b>

Source: SEMOS; 2016b

Notes: Minor discrepancies may occur due to rounding. The Mineral Resources are quoted using a gold price of US\$1,400. Figures are reported to five significant figures. Resources are reported inclusive of Ore Reserves. The Measured and Indicated are inclusive of those Mineral Resources modified to produce the Ore Reserve.

\*Stockpiles did not report Inferred Resource.

**15. MINERAL RESERVE ESTIMATES****15.1 Summary**

The mineral reserve estimated at December 31, 2015 includes reserves in Satellite pits, Stockpiles and the SSP pit. For economic assumptions, please refer to Section 22.

**Table 15.1: Sadiola December 31, 2015 Mineral Reserve**

SSP Reserves	Tonnage (kt)	Gold (oz)	Grade (g/t)
Satellites Pits	-	-	-
Stockpiles	-	-	-
SSP Pit	-	-	-
<b>Total Proven</b>	-	-	-
Satellites Pits 2	1,503	108,595	2.25
Stockpiles 2	5,262	331,279	1.96
SSP Pit 1	63,030	3,916,724	1.93
<b>Total Probable</b>	<b>69,795</b>	<b>4,356,598</b>	<b>1.94</b>
<b>Total Proven &amp; Probable</b>	<b>69,795</b>	<b>4,356,598</b>	<b>1.94</b>

<sup>1</sup> Mineral Reserve was estimated by Louis Pierre Gignac, Vice-President Engineering at G Mining Services inc. He has been involved in mining engineering and financial evaluation for 14 years. He fulfills the requirements as a qualified person for the Purpose of NI 43-101.

<sup>2</sup> Mineral Reserve was estimated by Andrew Bridges, has a minimum of 5 years relevant experience to the type and style of mineral deposit under consideration and to the activity which is being undertaken to qualify as a Competent Person (or Recognized Mining Professional) as defined in the 2012 Edition of the JORC Code and the 2009 edition of the SAMREC code.

<sup>3</sup> Satellites pits and stockpile are estimated at \$ 1,100/oz and the SSP pit is estimated at \$ 1,200/oz.

<sup>4</sup> Cut-off-grade for satellites pits is 0.85 g/t Saprolite Oxide and 1.10 g/t Hard Sulphides. For the SSP pit, the cut-off-grade is 0.70 g/t Hard Sulphide.

**15.2 SSP In-Pit Mineral Reserve Statement**

The in-situ SSP mineral reserve is estimated at 63.03 M t of ore at 1.93 g/t for 3.9 M oz considering the 7.2 Mtpa Scenario. The SSP pit reserves are all in the probable category. The SSP in-pit mineral reserve does not include a dilution factor and assumes 100% mining recovery.

**15.3 Ore Stockpile Mineral Reserve Statement**

Information on stockpile inventories was provided by SEMOS. The ore stockpile inventor at December 31, 2015 is estimated at 5,262 kt at an average grade of 1.96 g/t for 331 k oz.

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Of this amount a 2,672 kt at an average grade of 2.42 g/t for 208 k oz is hard sulphide material which is best suited for processing through the new plant conceived for the SSP Project. The milling schedule of the SSP project includes this hard sulphide ore stockpile.

## **16. MINING METHODS**

### **16.1 Introduction**

IAMGOLD Corporation (“IAMGOLD”) commissioned G Mining Services Inc. (“G Mining”) to prepare a feasibility level mining study and mineral reserve estimate for the Sadiola Sulphide Project (“SSP”) expansion.

Mining is currently being conducted in a number of oxide pits which are expected to be depleted in the next two years. The SSP project involves a significant push-back to the existing Sadiola main pit to recover hard sulphide mineralization beneath the existing depleted oxide pit.

Mining of the SSP pit is to be carried out using conventional open pit techniques with hydraulic shovels and mechanical drive trucks in a bulk mining approach with 10 m benches compared to the small bench height flitches (3.33 m) practiced in the past. An important stockpiling and grade segregation strategy was practiced in the past to maximize mill head grades. For the SSP project it is required that all ore grade material be sent to the mill in the initial years due to the initial high stripping and given that there is a larger mill to at the rate of 7.2 million tonnes per year.

Contract mining at Sadiola has been performed by Moolmans since 1996. Moolmans currently performs all drilling, blasting (excluding charging), loading, hauling and associated support mining activities. The plan is for Moolmans to continue mining in the Satellite pits until they are depleted. For the Sadiola main pit expansion an owner mining approach has been taken with a majority of the initial fleet already purchased. Some equipment is present on site while other equipment is either in ports ready for export or stored with suppliers.

#### **16.1.1 Resource Description**

Gold mineralization occurs along the Sadiola Fracture Zone (SFZ) over a drilled strike length of approximately 2,500 m. Mineralization occurs in all of the four major rock types (marble, greywacke, diorite, and quartz-feldspar porphyry) and is associated with a complex alteration. Sadiola has been classified as a “Mesothermal - Shear Hosted” gold deposit.

Deposits of this type worldwide exhibit good continuity of mineralization both along strike and extend to great depth. The deposit is structurally controlled, where high grade shoots typically occur within a more pervasive lower grade halo.

The Sadiola Sulphide Model was updated in 2014 and includes all material below the hard-soft contact. The new undepleted model is called SSP2014.

The main shear zone occurs between the hangingwall marble and footwall meta-sandstone and these are cross-cut by various diorite dykes.

### 16.1.2 Geological Parameters

An updated geotechnical report was prepared by SRK of South Africa to complement the SRK Pre-Feasibility Study report from October 2009.

#### **The main conclusions from the study are:**

- The saprolite strength properties were reviewed on the basis of the historic (2000) and supplementary (2010) tri-axial testing results and the strength parameters  $c = 40$  kPa and  $\phi = 25^\circ$  corresponding to wet samples tested under effective stress conditions were confirmed as the appropriate values to use for the stability analysis;
- The results of the shear tests from the supplementary testing program provided the required data to confirm the strength characteristics along joints, used for the analysis of stability with structural control mechanisms;
- Bench scale stability was reviewed for wedge failure in the west slope and planar failure in the east slope through a series of sensitivity analyses to assess design factors such as bench height and bench face angle. The results of these analyses together with the evaluation of requirements of catch berm width for rock fall control were used to confirm that the 20 m high benches with battered faces at  $75^\circ$  prescribed in the current design correspond to the more appropriate configuration to balance the benefits of wider benches with those of battered faces;
- The observed performance of the slopes in saprolite and weathered rock was used to guide the analysis of these slopes and the results of the analysis indicate that the saprolite slope design needs to be extended from elevation +100 m down to elevation +80 m into the weathered rock transition to ensure acceptable stability conditions in this area;
- The results of the stack analysis in the west slope indicate that the current design stack angle might result in unacceptably high probabilities of failure for stack slopes more than 60 m high in a pit with a single ramp access. Options to correct this situation include flattening the stack angle to  $53^\circ$ , maintaining the current ramp configuration, or replacing the geotechnical berm in the current configuration with a second ramp, maintaining both the stack and overall slope angles;
- The results of the stability analysis in the south slope indicated the need for adjustments in the stack angle configuration of this slope in order to achieve acceptable factors of safety of the slope. Main adjustments include a  $35^\circ$  stack angle in the saprolite as recommended in the west slope, flattening the stack slope in weathered diorite from the current  $55^\circ$  to  $50^\circ$  and steepening of the stack slope in the marble-shear zone at the toe of the slope from current  $46^\circ$  to  $50^\circ$ ;

- The overall slope stability was reviewed with limit equilibrium and finite element analysis along selected sections of the slope with the results confirming the adequacy of the design. However, some changes of these angles might result from implementing the recommended adjustments in the stack angles.

**The main recommendations are:**

- Confirm the bench configurations used in the current design consisting of 20 m high benches with 75° battered faces;
- Extend the saprolite slope design, consisting of 10 m high benches stacked at 35° toe to toe, from elevation +100 m down to elevation +80 m;
- Possible replacement of the geotechnical berms on the west slope with an additional access ramp, keeping the same stack (55° toe to toe) and overall ( $\approx 49^\circ$ ) slope angles. (the Geotechnical berms were however retained due to practical design issues);
- Adjust the stack angles of the south slope implementing 35° (toe to toe) in the saprolite slope and 50° (toe to toe) in the rest of the slope with a maximum stack height of 100 m.

**Table 16.1: Geotechnical Parameters vs. Weathering Profile**

Aspect	West Wall		East Wall	
	Saprolite & Weathered Rock	Hard Rock (MGWK)	Saprolite & Weathered Rock	Hard Rock (CAMB)
Batter angle (°)	70	75	70	75
Bench height (m)	10	20	10	20
Stack angle (°)	35	55	35	60
Bench width (m)	10.65	8.65	10.65	6.2
Geotechnical berm width (m)		30		30
Maximum stack height (m)		100		100

**16.1.3 Hydro Geological Considerations**

The hydrogeological study of the area was carried out by SRK and subsequently by the SEMOS hydrogeology department, based on piezometer information and ground water measurements on site. Seepage mapping in the pit and observed performance of the slopes has confirmed the existence of a groundwater regime having a moderate effect on the pit excavations. The objective of the hydrogeological

program is to define the strategies required to achieve dewatered slopes using wells and sub-horizontal drainage.

The Sadiola Project area lies within the “tropical wet-dry” climatic zone of Mali between the 700 mm and 900 mm isohyets. The mean annual precipitation on site is 825 mm (1994-2009). Maximum annual rainfall recorded was 1,131 mm (1996) and the minimum was 712 mm (1994).

There are two distinct seasons (wet and dry). The wet season occurs between July and October, when 96% of the total annual rainfall occurs, and consists of high intensity convective thunderstorms which are frequently accompanied by strong winds. The dry season occurs between November and May and consists of fine clear weather.

The mean average temperatures range from a minimum of 27°C in December to 33°C in April-May. The Sadiola air temperature rises sharply in February and March, bringing in a brief, very hot, air temperature reaching 44°C to 46°C.

The largest values of mean monthly evaporation occur between February and May, ranging from 231 to 274 mm. The mean annual evaporation for Sadiola is 1,949 mm.

The regional catchment area is drained by a series of ephemeral watercourses that flow in a general north-west direction and form part of the tributary network associated with the Falémé River located to the north-west of Sadiola Mine. Surface water east of the escarpment drains eastwards towards the Senegal River. A river diversion has been cut at the northern end of the main pit (around cut-back 4) to allow for the extension of the pit in this direction.

The Senegal River basin, within which the Sadiola Region falls, lacks surface water during the dry season. During the wet season, which lasts from July to October, heavy rainfall combined with poor ground cover and hard surfaces (laterite and ferricrete), leads to rapid run-off from land surfaces and poor penetration. The deposition of fine silt in shallow river valleys during flood events impairs moisture penetration and groundwater recharge.

The surface water catchment area associated with the Sadiola Mine covers some 70.5 km<sup>2</sup>, of which 58 km<sup>2</sup> is located upstream of the open pit mine. In the immediate vicinity of the mine, some streams flow north toward the Falémé River / Farabakouta stream. These ephemeral streams are important sources of water for local subsistence agriculture. Flow in the Farabakouta stream is not monitored on a regular basis, but peak flow on September 9, 2003, after 81.2 mm of rainfall, was calculated at 18.82 m<sup>3</sup>/sec.

The hydrostratigraphy is controlled by the geology (lithology and structure) as well as the impact of weathering and other geomorphological processes on the host material. At a regional level, the hydrostratigraphy can be simplified to a three layer system. The main hydrostratigraphic units are an upper aquifer consisting of transported and weathered material, a saprolite layer consisting of in situ weathered material and the underlying fractured aquifer.

The aquifers present in the Main Pit area are identified as follows:

- Upper Weathered Zone (Decalcified Marbles (top aquifer) in the pit varying to saprolite (aquitard) in other areas);
- Weathered/Fresh contact (Decalcified Marbles – Marble Contact (basal aquifer) in pit, and saprolite/fresh contact in other areas);
- Fresh Rock (Marbles in main pit area as well as Greywacke to the west).

The main aquifer zone at the Sadiola pit is termed the Basal aquifer. The Basal aquifer occurs at the contact between the soft (saprolite) and hard rocks and therefore has a trough-like distribution because of the depth of weathering within the deposit. The contact zone between the decarbonated marble and the unaltered marble forms the most important section of this main aquifer and is 4 to 6 m thick. Locally, the contact zone is karstic with wide open surfaces that transmit significant quantities of groundwater. The weathering trough (and therefore the main aquifer) and the decarbonated marble is deeper to the south of the main deposit.

The recharge percentage utilized in 2001 (SRK report) was 8% of mean annual precipitation (MAP) based on four different techniques for the recharge assessment. However, if the chloride techniques are discounted, this appears to be the upper range of recharge.

## **16.2 Pit Optimization**

Open pit optimization was conducted to determine the optimal economic shape of the open pit in three dimensions. This task was undertaken using the Whittle software which is based on the Lerchs-Grossmann algorithm.

For this feasibility study, measured and indicated resource blocks were considered for optimization purposes and excluded inferred resources.

### **16.2.1 Optimization Parameters**

Plant throughput has been set at 7.2 Mtpa (Million Metric Tonnes per annum) and mining fleet capacity at 36 Mtpa to accelerate initial stripping, while retaining bench drop down rate below 60 m/yr.

Gold selling cost includes a 6% royalty and 1% local tax plus a transport and refining costs of \$2/oz; at a \$1,200/oz gold price this represents \$86/oz.

**Table 16.2: Main Economics Parameters for Pit Optimization**

Economic Parameters		
Gold price (P)	US\$/oz	1,200
Long term oil price	US\$/bbl	75
Site diesel price	US\$/litre	0.83
Euro exchange rate	EUR/USD	1.15
CFA exchange rate	CFA/USD	570
Transport & refining cost	US\$/oz	2.00
Power cost	CFA/kWh	70.0
Power cost	US\$/kWh	0.123
Royalty (3+3)+ Local tax (1%)	US\$/oz	84.00
Cost of selling (Cs)	US\$/oz	86.00
Discount rate	%	6.00

Unit reference mining costs are used for a “reference mining block” usually located near the pit crest (surface) as to enable simplified cost calculation and auditing. Mining costs of blocks located elsewhere within the modeled volume are calculated through the optimisation run via a set of referential factors (material and depth based – “mine cost adjustment factors” or MCAF). The reference mining cost is estimated at \$2.53/t with a depth incremental cost of 0.031\$/t per 10 m bench.

Geotechnical parameters for the value optimization exercise are based on the SRK inter-ramp angle recommendations with provisions for ramp sections in the overall slope. The slope in the saprolite weathering profile is 30 degrees uniformly around the pit and is 42.2 degrees in rock on the West slope and 45.6 degrees in rock on the East slope.

### 16.2.2 Cut-Off Grades

Ore is determined through the calculation of cut-off grades using the simple assumption that the cut-off grade is the grade that allows revenue to cover costs.

Mining costs are excluded from the cut-off calculation, because under the “optimum pit shell” requirements, all of the material contained within the shell must be mined. Consequently, the only decision to be made is whether to send this material to the waste dump or to the processing plant.

Table 16.3 numbers are indicative of FGO cut-off grades based on the SSP cost assumptions.

Table 16.3: Ore Based Cost &amp; COG by Ore Type

Ore Based Cost & COG by Ore Type	Units	Laterite	Soft Oxide	Silicified Oxide	Soft Sulphide	Hard Sulphide	Blast Oxide	Blast Sulphide	Tran.
Rock type		1	2	3	4	5	6	7	8
Density		1.97	1.82	2.56	1.98	2.70	2.16	2.37	2.26
Metallurgical recovery (rFGO)	%	88.0%	94.0%	85.0%	80.0%	76.0%	85.0%	76.0%	75.0%
Processing rate	Mt/yr	7.25	7.25	7.25	7.25	7.25	7.25	7.25	7.25
Plant availability	%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%
Avg. Processing rate	t/hours	900	900	900	900	900	900	900	900
Avg. power consumption	kWh/t	21.4	21.4	34.5	21.4	43.3	23.9	47.7	34.5
TSF	US\$/t treated	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38
Reagents	US\$/t treated	6.39	2.90	3.53	6.39	4.16	3.53	4.16	5.28
Power	US\$/t treated	2.63	2.63	4.24	2.63	5.32	2.93	5.86	4.24
General	US\$/t treated	4.72	4.72	4.72	4.72	4.72	4.72	4.72	4.72
<b>Total processing cost (Cp)</b>	US\$/t treated	14.12	10.63	12.87	14.12	14.58	11.56	15.12	14.62
Mining dilution (d)	%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Ore premium mining cost (Com)	US\$/t treated	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ore Feed	US\$/t treated	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total fixed G&A costs	MUS\$/yr	27.26	27.26	27.26	27.26	27.26	27.26	27.26	27.26
G&A cost (Ca)	US\$/t treated	3.76	3.76	3.76	3.76	3.76	3.76	3.76	3.76
Rehabilitation (Cmc)	US\$/t treated	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Stay-in-business capital (Csibc)	US\$/t treated	0.57	0.57	0.57	0.57	0.57	0.57	0.57	0.57
<b>Total Ore Based Cost</b>	<b>US\$/t treated</b>	<b>18.65</b>	<b>15.16</b>	<b>17.40</b>	<b>18.65</b>	<b>19.11</b>	<b>16.09</b>	<b>19.65</b>	<b>19.15</b>
<b>Full Grade Ore in-situ COG</b>	<b>g Au/t</b>	<b>0.59</b>	<b>0.46</b>	<b>0.57</b>	<b>0.65</b>	<b>0.70</b>	<b>0.53</b>	<b>0.72</b>	<b>0.71</b>

**16.2.3 Optimization Results & Pit Shell Selection**

Pit optimizations were performed with only the measured and indicated resources valued and excluded the inferred resources for purposes of selecting an optimal shell to guide the final pit design and mineral reserve estimates. However, other sensitivity runs were performed with the inferred resources to estimate their possible contribution and identify those resources of interest to upgrade through additional definition drilling.

Pit shell No. 23 has been selected as the optimum pit shell for the Feasibility Study (Figure 16.1), to account for a reasonable drop down rate, peak mining capacity and plant feed requirements. This selected shell corresponds to a \$940/oz shell which contains a total of 272 million tonnes including 63.7 million tonnes of ore at an average grade of 1.88 g/t for 3.74 M ounces of contained gold.

This pit shell selection made for \$1,200/oz is quite robust and is an appropriate size pit between \$1,100/oz and \$1,300/oz as shown by the pit size versus value curves presented in Figure 16.2.

Table 16.4: SSP M&amp;I Whittle Results

Pit Shell	Rev. Factor	Gold Price	Total (kt)	Ore (kt)	Strip Ratio	Au k oz	Au g/t
16	0.6667	800	114,196	32,702	2.49	2,052	1.95
17	0.6833	820	124,781	35,257	2.54	2,191	1.93
18	0.7000	840	149,455	40,023	2.73	2,469	1.92
19	0.7167	860	153,290	41,333	2.71	2,528	1.90
20	0.7333	880	167,660	44,328	2.78	2,686	1.88
21	0.7500	900	192,226	48,387	2.97	2,924	1.88
22	0.7667	920	228,385	55,116	3.14	3,297	1.86
23	0.7833	940	271,607	61,852	3.39	3,696	1.86
24	0.8000	960	280,434	63,301	3.43	3,776	1.86
25	0.8167	980	301,774	66,280	3.55	3,954	1.86
26	0.8333	1,000	312,461	67,794	3.61	4,040	1.85
27	0.8500	1,020	330,496	70,417	3.69	4,186	1.85
28	0.8667	1,040	337,891	71,622	3.72	4,247	1.84
29	0.8833	1,060	350,372	73,126	3.79	4,337	1.84
30	0.9000	1,080	356,917	73,941	3.83	4,384	1.84
31	0.9167	1,100	416,385	79,287	4.25	4,749	1.86
32	0.9333	1,120	430,678	80,933	4.32	4,839	1.86
33	0.9500	1,140	443,627	82,354	4.39	4,924	1.86
34	0.9667	1,160	446,408	82,780	4.39	4,944	1.86
35	0.9833	1,180	447,165	82,958	4.39	4,951	1.86
36	1.0000	1,200	455,178	83,981	4.42	5,003	1.85
37	1.0167	1,220	455,878	84,175	4.42	5,009	1.85
38	1.0333	1,240	463,972	85,002	4.46	5,056	1.85
39	1.0500	1,260	477,635	86,326	4.53	5,135	1.85
40	1.0667	1,280	488,744	87,469	4.59	5,197	1.85
41	1.0833	1,300	497,388	88,353	4.63	5,246	1.85
42	1.1000	1,320	499,283	88,691	4.63	5,259	1.84
43	1.1167	1,340	511,904	89,738	4.70	5,322	1.84
44	1.1333	1,360	513,251	89,982	4.70	5,330	1.84
45	1.1500	1,380	514,850	90,251	4.70	5,341	1.84
46	1.1667	1,400	525,894	91,063	4.78	5,391	1.84
47	1.1833	1,420	529,373	91,513	4.78	5,411	1.84
48	1.2000	1,440	533,082	91,890	4.80	5,429	1.84
49	1.2167	1,460	542,685	92,656	4.86	5,473	1.84
50	1.2333	1,480	547,540	92,987	4.89	5,494	1.84
51	1.2500	1,500	547,879	93,105	4.88	5,497	1.84

Figure 16.1: M&I Pit by Pit Graph

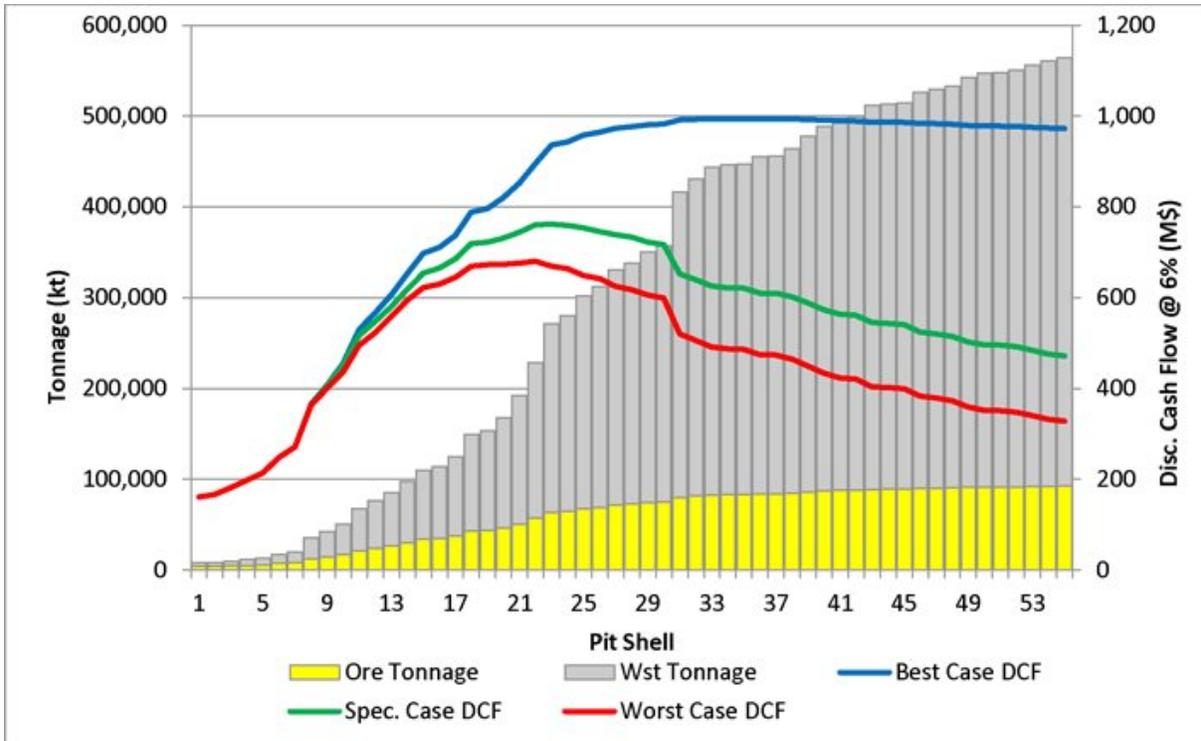
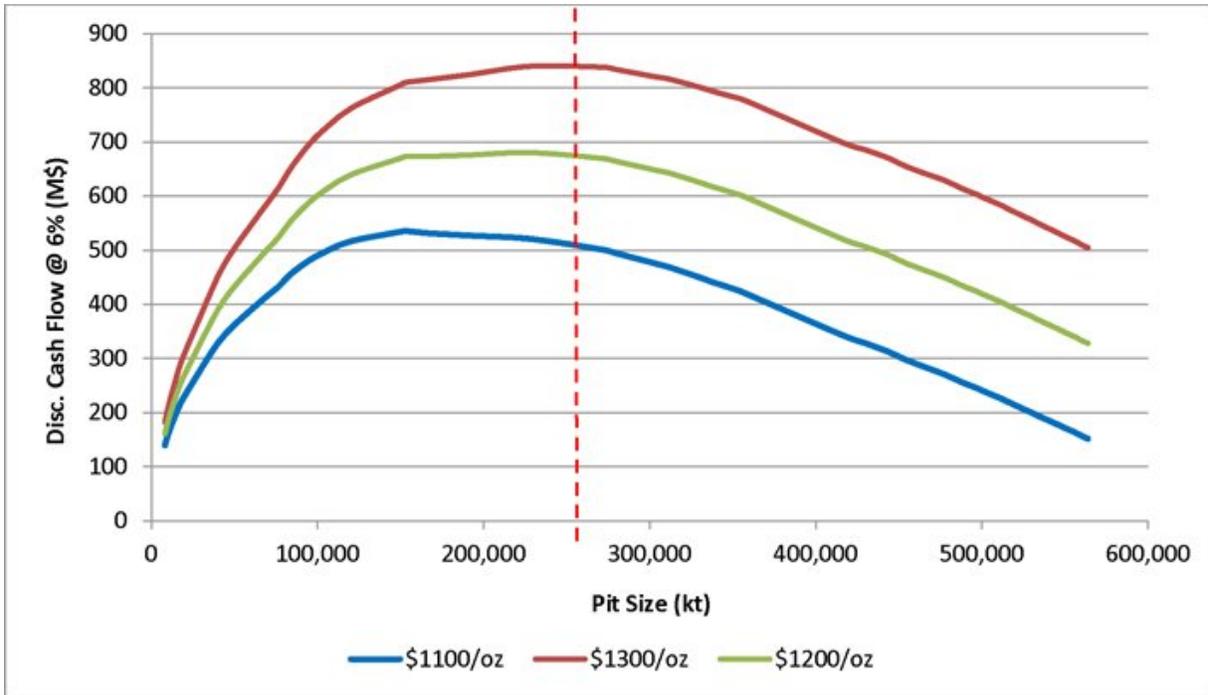


Figure 16.2: Pit Size vs. Value at Various Gold Prices



**16.3**      **Open Pit Mine Designs****16.3.1**      **Final Pit & Phase Designs**

A two-phase mining approach for the SSP pit is proposed. The second phase is an expansion on the West-side of the pit. With the existing actual mined out open pit many adjustments were required to minimize waste stripping but still allow access to all the ore and allow for minimum mining widths for efficient and safe operations.

The current excavation, Phase 1 and the final open pit design all share a common final wall on the East side. Phase 1 and Phase 2 will share a common wall on the North and South sectors of the open pit.

The total tonnage to be mined out is 311.6 M t at with average strip ratio of 3.94:1. The second phase has a slightly higher grade but a higher strip ratio. The first phase will mine out 60% of the contained ounces. Phase 1 reaches a mining depth of 270 m and is 635 m wide and 1,560 m along the North-South strike direction (Figure 16.3). The final pit reaches a depth of 400 m, is 840 m wide and 1,625 m along strike (Figure 16.4).

**Table 16.5: Mine Phase Design Summary**

<b>Phase Design Content</b>	<b>Units</b>	<b>Phase 1 Pit</b>	<b>Phase 2 Pit</b>	<b>Grand Total</b>
Total tonnage	kt	166,067	145,509	<b>311,576</b>
Waste rock	kt	127,433	121,113	<b>248,546</b>
Ore tonnage	kt	38,634	24 396	<b>63,030</b>
Avg. Ore Grade	g/t	1.89	2.00	<b>1.93</b>
In-situ gold	Oz s	2,344,816	1,571,908	<b>3,916,724</b>
Strip ratio	W:O	3.30	4.96	<b>3.94</b>
% of gold	%	60%	40%	<b>100%</b>

**16.3.2**      **Dump Design Criteria**

Waste dumps are designed with 10 m lift heights. An 11 m wide catch bench is incorporated at every lift in order to achieve an overall slope profile of 2.5H:1V. This gentle slope allows easy rehabilitation activities. The dump face is designed at an angle of repose of 35 degrees.

Figure 16.3: Scenario 7.2 Mtpa, Phase 1 Design

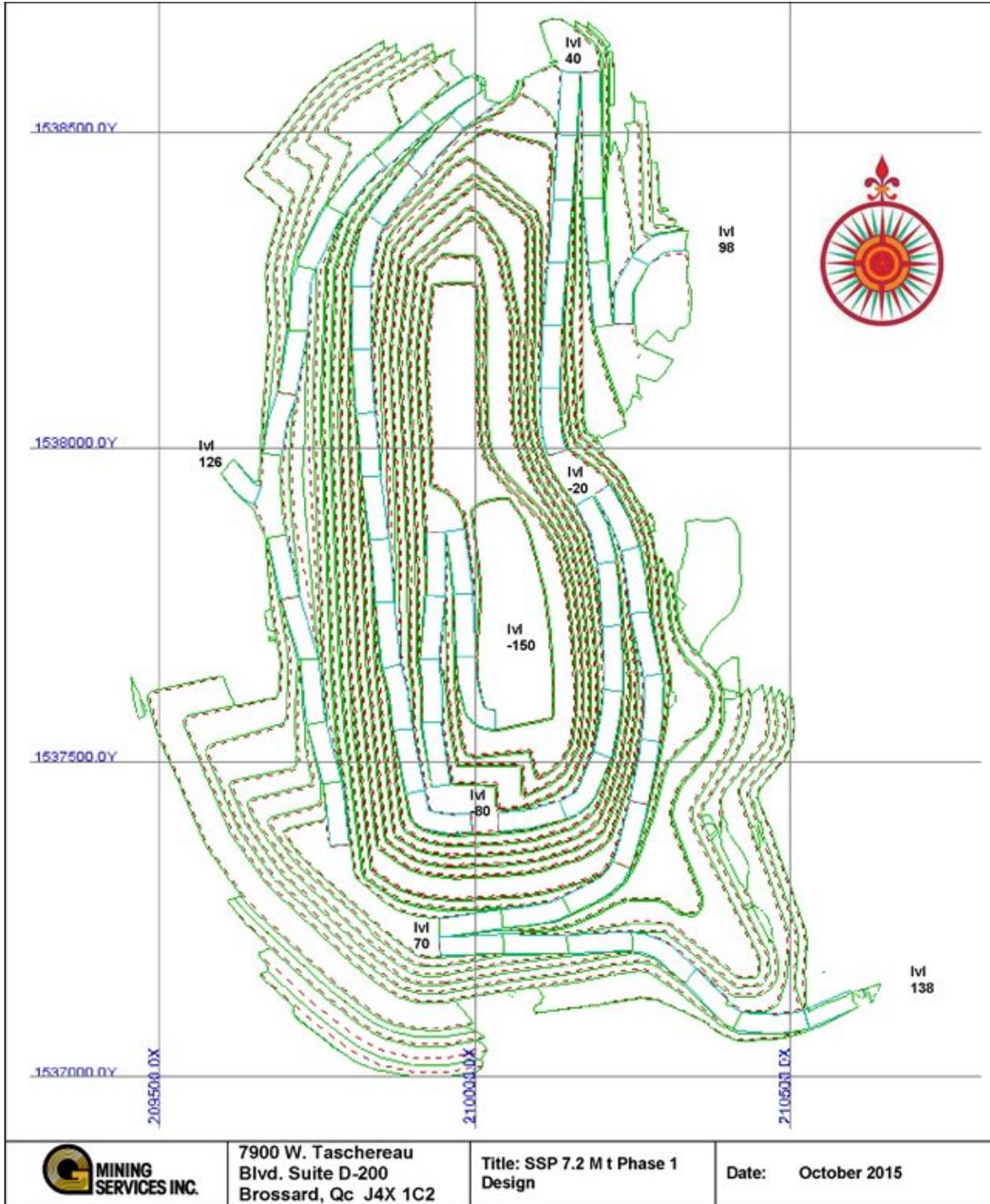
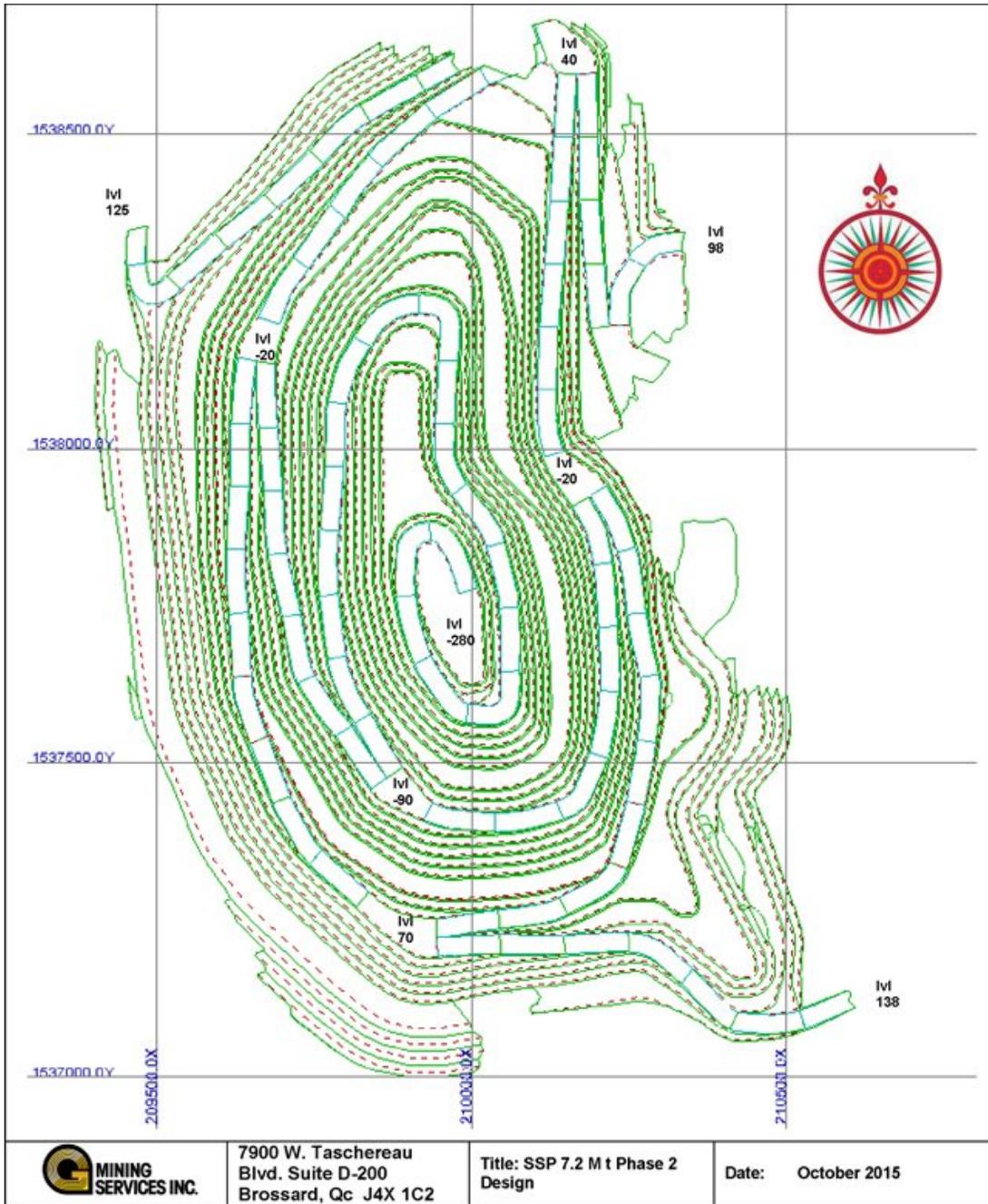


Figure 16.4: Scenario 7.2 Mt, Phase 2 Design (Final Pit)



**16.4 Waste Dump Designs**

Waste rock material will be stockpiled on both the North and South ends of the Open Pit. The North dump will be built over the top of satellite pits planned in this area that will be depleted by the end of 2016. Consequently, this area will require sterilization in order to establish dump capacity for the North dump. When Open pit mining reaches the level from which the backfill zone is accessible, in-pit dumping will happen without having to haul outside of the pit therefore reducing the haulage distance.

Before in-pit dumping is possible, the north Phase 1 dump, which is located on the west side of the actual pit, will serve as the dumping area in the north.

The north waste dump is constrained with infrastructure all around it while the south waste dump is not submitted to the same constraints and has more flexibility in height and footprint.

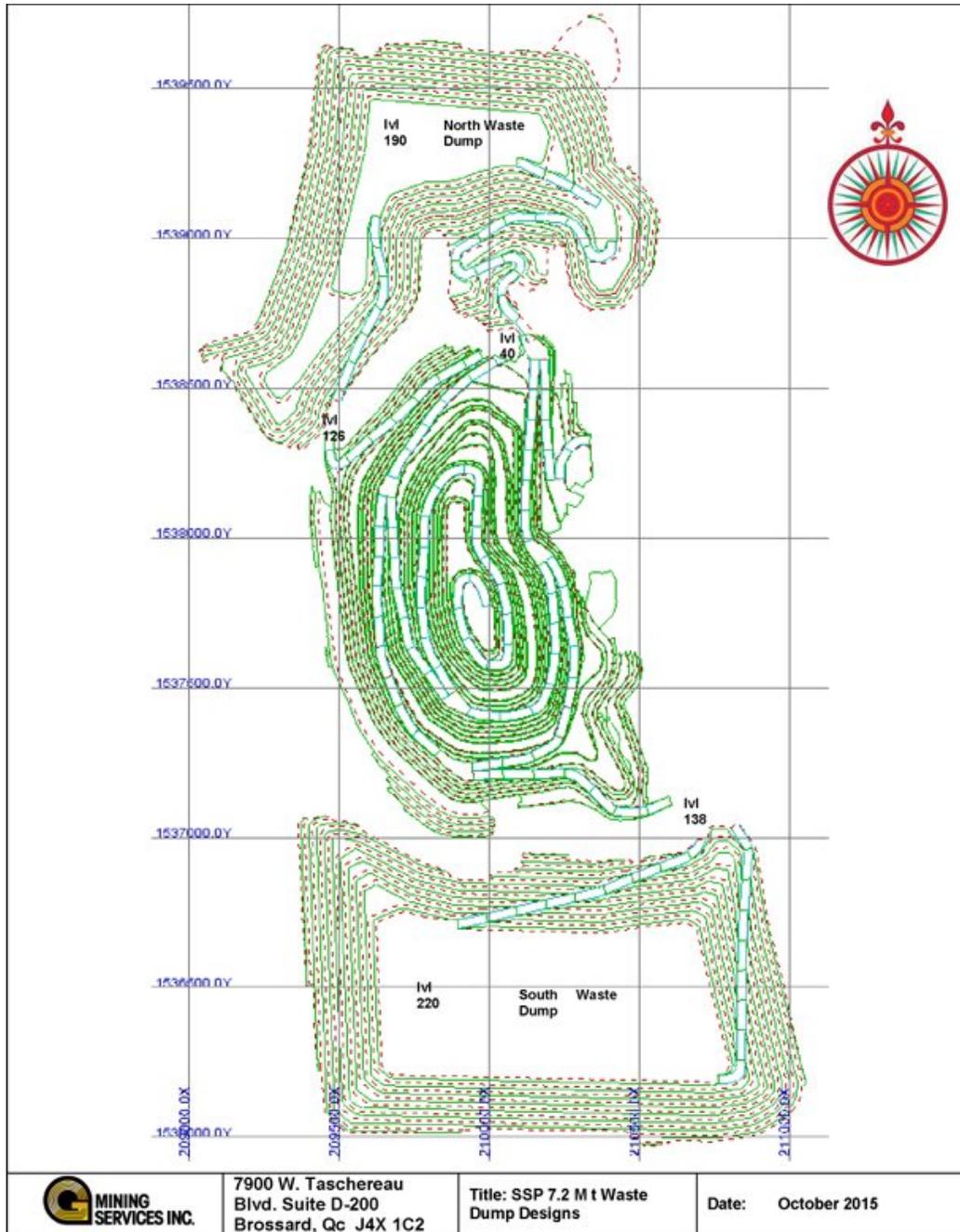
A total of 245.8 M t of waste rock has to be stored, which equates to a loose volume of 117.3 M m<sup>3</sup>. The total waste dump capacity is 129.6 M m<sup>3</sup>.

The dumping sequence will follow a simple “shortest distance first” rule. The north dumps will be topped up to their full capacity; the rest will be stored on the South dump.

**Table 16.6: Waste Storage Capacities**

Dump Name	Volume (km <sup>3</sup> )	Footprint (ha)	Maximum Elevation
Ph 1 North Waste Dump	22,011	64	190
North Waste Dump	24,639	113	190
South Waste Dump	82,909	150	220
<b>Total</b>	<b>129,558</b>		

Figure 16.5: Scenario 7.2 Mt, Waste Dump Designs



### **16.4.1**            **Mine Haul Road Designs**

Mining haul roads have been designed to suit the largest equipment being a 136 metric tonne class haul truck, Caterpillar 785C model. The haul truck has a width of 6.7 meters, for double lane traffic, the industry's best practice recommends to have a hauling width of a minimum 3.5 times the haul truck width, leading to an overall double lane ramp of 30 meters. The ramp gradient will be established at 10%.

The safety berm will be constructed from crushed rock and on the ramp outside edge. The berm height of 1.45 m is equivalent to half of the tire height. A safety berm is required whenever a drop-off greater than 3 m exists. In this case, the berm will be built with a 1.1H:1.0V slope and a top width of 0.4 m. Water drainage will be assured by a 2% cross slope from the outer edge towards the inner edge where a 3.3 m wide ditch is planned.

Operations will have two (2) options for double ramp design that will not impact the open pit design. The ramp width will always be a total of 30 m, however, depending on the site conditions, a distance up to 3 m from the wall edge can be kept to prevent from crest break. The hauling distance between trucks and the berm or the open pit wall will decrease by 1 m for each side to accommodate for the Crest Break possibility.

If the site conditions prove to be more stable and do not require a crest break offset, the hauling width of 3.5 times the haul truck width will be respected.

## **16.5**            **Production Scheduling**

### **16.5.1**            **Scheduling Criteria**

The main objectives that were explored in developing the LOM schedule included:

- Accessing high-grade hard sulphide material in sufficient quantities supply a steady state milling rate of 7.2 Mtpa;
- Limiting pre-production or stripping in the SSP pit during the construction period to reduce the initial capital cost for the project;
- Maximizing the processing of soft ore through the existing plant early on to maintain the productivity;
- Delay processing of hard sulphides through the existing plant as long as sufficient quantities of soft ore remain available from satellite pits and stockpiles to utilize the existing plant capacity;
- Limiting the peak mining rate to about 36 Mtpa with a start-up of SSP pit stripping in the sixteenth month;
- Plant ramp-up period is planned in Q4 Year 2.

**16.5.2 Schedules**

An owner fleet is scheduled to mine the SSP pit starting in Year 2 until end of the mine life in Year 12. The maximum mining rate is reached in Year 3 with 36 M t and is maintained over six years and then decreases when significant mineralization is reached in Phase 2.

The actual mill capacity is 4.8 Mtpa with little capacity to process hard ore. With the expansion, it will increase to 7.2 Mtpa from Year 6. Mill ramp-up occurs from Year 3 to Year 5 during which final connexion between soft and hard circuits is made and maximum throughput is limited to 6.8 Mtpa during that year.

Hard Sulphide represents 92% of the material feed to the mill from SSP Pit.

Annual gold production is below 300,000 ounces before Year 5. Then gold production goes above 300,000 ounces except in year Year 9 where a large amount of low grade ore from the stockpile is processed. The total gold production from Year 2 to the end of mine life is estimated at 3.15 M ounces.

The SSP milling schedule includes the 63.0 M t of ore from the SSP pit plus 2.7 M t of hard sulphide ore that has been stockpiled that the current plant is not well suited to process for a total of 65.7 M t of ore to be processed through the SSP plant or as a result of the project.

Table 16.7 Summary of Sadiola Sulphide Pit Mining Schedule

Total All Phases SSP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
<b>Total Tonnage (kt)</b>		<b>18 291</b>	<b>35 879</b>	<b>36 121</b>	<b>35 812</b>	<b>36 157</b>	<b>36 116</b>	<b>35 995</b>	<b>32 491</b>	<b>21 049</b>	<b>17 021</b>	<b>6 614</b>	<b>311 547</b>
<b>Waste (kt)</b>		<b>18 060</b>	<b>34 256</b>	<b>31 506</b>	<b>27 609</b>	<b>29 164</b>	<b>28 012</b>	<b>28 574</b>	<b>27 522</b>	<b>12 790</b>	<b>8 508</b>	<b>2 515</b>	<b>248 517</b>
Soft		13 728	7 823	2 045	6 767	1 011	163	32	2	0	0	0	31 570
Transitional		1 410	3 518	1 332	1 382	1 308	81	15	55	4	0	0	9 104
Hard Sulphide		2 922	22 915	28 130	19 461	26 845	27 768	28 527	27 466	12 786	8 508	2 515	207 843
<b>Ore Tonnage (kt)</b>		<b>230</b>	<b>1 622</b>	<b>4 615</b>	<b>8 203</b>	<b>6 993</b>	<b>8 104</b>	<b>7 422</b>	<b>4 969</b>	<b>8 259</b>	<b>8 514</b>	<b>4 099</b>	<b>63 030</b>
Soft		227	1 295	808	424	287	167	80	22	3	0	0	3 312
Transitional		3	219	361	351	393	229	187	31	3	0	0	1 776
Hard Sulphide		0	109	3 446	7 428	6 314	7 709	7 155	4 916	8 253	8 514	4 099	57 941
<b>Gold (Oz)</b>		<b>7 729</b>	<b>63 052</b>	<b>280 708</b>	<b>489 338</b>	<b>421 022</b>	<b>507 531</b>	<b>482 895</b>	<b>299 121</b>	<b>544 902</b>	<b>537 368</b>	<b>283 060</b>	<b>3 916 724</b>
Soft		7 627	49 340	62 966	28 626	20 006	16 852	6 540	1 438	267	0	0	193 663
Transitional		102	10 114	20 169	25 753	27 468	17 654	15 554	2 367	260	0	0	119 442
Hard Sulphide		0	3 598	197 573	434 959	373 548	473 024	460 801	295 315	544 374	537 368	283 060	3 603 619
<b>Avg. Grade (g/t)</b>		<b>1.04</b>	<b>1.21</b>	<b>1.89</b>	<b>1.86</b>	<b>1.87</b>	<b>1.95</b>	<b>2.02</b>	<b>1.87</b>	<b>2.05</b>	<b>1.96</b>	<b>2.15</b>	<b>1.93</b>
Soft		1.04	1.19	2.42	2.10	2.17	3.14	2.56	2.02	3.02	0.00	0.00	1.82
Transitional		1.13	1.44	1.74	2.28	2.17	2.40	2.58	2.38	2.36	0.00	0.00	2.09
Hard Sulphide		0.00	1.03	1.78	1.82	1.84	1.91	2.00	1.87	2.05	1.96	2.15	1.93

Table 16.8: Summary of Plant Feed Tonnes and Grade by Material

Milling Schedule	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
<b>Ore Tonnage (kt)</b>		<b>727</b>	<b>3 915</b>	<b>4 495</b>	<b>6 800</b>	<b>7 200</b>	<b>6 562</b>	<b>65 700</b>					
Soft		227	1 295	808	424	287	167	80	22	3	0	0	3 312
Transitional		0	220	361	334	410	229	187	31	0	0	3	1 774
Hard Sulphide		500	2 401	3 326	6 042	6 503	6 804	6 934	7 147	7 197	7 200	6 559	60 614
<b>Gold (Oz)</b>		<b>49 579</b>	<b>239 788</b>	<b>270 288</b>	<b>449 969</b>	<b>427 486</b>	<b>482 567</b>	<b>476 774</b>	<b>360 605</b>	<b>515 409</b>	<b>495 719</b>	<b>356 708</b>	<b>4 124 891</b>
Soft		7 627	49 340	62 966	28 626	20 006	16 852	6 540	1 438	267	0	0	193 663
Transitional		29	10 186	20 169	24 504	28 717	17 654	15 554	2 367	0	0	260	119 442
Hard Sulphide		41 923	180 261	187 153	396 839	378 763	448 061	454 680	356 800	515 141	495 719	356 447	3 811 787
<b>Avg. Grade (g/t)</b>		<b>2.12</b>	<b>1.90</b>	<b>1.87</b>	<b>2.06</b>	<b>1.85</b>	<b>2.08</b>	<b>2.06</b>	<b>1.56</b>	<b>2.23</b>	<b>2.14</b>	<b>1.69</b>	<b>1.95</b>
Soft		1.04	1.19	2.42	2.10	2.17	3.14	2.56	2.02	3.02	0.00	0.00	1.82
Transitional		0.00	1.44	1.74	2.28	2.18	2.40	2.58	2.38	0.00	0.00	2.36	2.09
Hard Sulphide		2.61	2.33	1.75	2.04	1.81	2.05	2.04	1.55	2.23	2.14	1.69	1.96
Gold Production (Oz)		<b>39 043</b>	<b>190 368</b>	<b>210 171</b>	<b>344 446</b>	<b>326 434</b>	<b>367 830</b>	<b>362 883</b>	<b>274 096</b>	<b>391 721</b>	<b>376 746</b>	<b>271 095</b>	<b>3 154 832</b>
Avg. Recovery		86.4%	91.4%	68.7%	76.5%	76.4%	76.2%	76.1%	76.0%	76.0%	76.0%	76.0%	76.5%

## **16.6** Mine Operations & Equipment Selection

### **16.6.1** Grade Control

Reverse Circulation (RC) drilling is preferred to blast hole sampling allowing the geology team to interpret the results followed by better mine planning. Two RC drill rigs are planned for the SSP pit.

Grade control will be done over the LOM for both ore and waste. RC drill holes will be done in a 10 m by 10 m grid and inclined at 60 degrees with a vertical depth of 20 m to cover two benches. The grid is widened out in waste sections.

### **16.6.2** Production Drilling

Drill and blast specifications are adjusted to the various rock type harnesses resulting from the weathering profile. Saprolite material has historically being drilled and blasted to increase digging productivity due to the presence of a hard duricrust encountered in the saprolite horizon.

A 203 mm (8-inch) blast hole diameter is selected regardless of rock type but burden, spacing, subdrill and stemming vary according to the rock type. Productivity decreases from 3.2 holes per hour in saprolite to 1.9 holes per hour in rock.

### **16.6.3** Pre-Split

Pre-split drill and blast is planned to maximize stable bench faces and to maximize inter-ramp angles along pit walls as prescribed by the geotechnical pit slope study. Pre-split along the Phase 1 and Phase 2 pit walls when in fresh rock with a hole spacing of 1.50 m. Pre-split holes, unlike production holes, are 20 m in length with a smaller diameter of 102 mm (4 inches).

Two drill rigs have been purchased. This rig has flexibility for drilling RC drill holes if configured accordingly, angled hole drilling capability and the possibility of drilling production drill holes.

### **16.6.4** Blasting

A high energy emulsion is planned regardless of wet or dry conditions encountered in the pit. The emulsion density will be 1.20 g/cm<sup>3</sup> and will be initiated by 400 g boosters. In the case of pre-split blasting, special packaged pre-split explosives will be used to facilitate loading and proper energy distribution in the hole. Powder factors range from 0.38 kg/bcm for the saprolite to 0.90 kg/bcm for the rock.

Blasting activities on site are currently outsourced to BME which are responsible for operating the explosives plant, and deliver explosives in the holes as well as connecting and firing the blast. BME therefore provide a turnkey solution for SEMOS or Moolmans. A continuity of this operating mode has been planned for the SSP pit.

**16.6.5**      **Loading**

The majority of the loading operations in the pit will be done by three (3) 20-22 m<sup>3</sup> hydraulic face shovels. Two of the shovels have been purchased for the project and were O&K RH170 at that time and have since been rebranded by Caterpillar as 6040 hydraulic shovel. The shovels with a bucket capacity of 20 m<sup>3</sup> are matched with Caterpillar 785C mine trucks with a 136 t payload and will require 4 bucket passes. The loading fleet is sized to achieve a peak of 42.2 M t when taking into account stockpile re-handle and allowing for some flexibility with the use of wheel loaders.

In addition to the primary loading fleet, the two (2) 14.5 m<sup>3</sup> front-end wheel loaders will assist in loading waste rock in the open pit as well as stockpile re-handling activities. One Caterpillar 993 front-end loader unit has already been purchased for the project.

The shovel productivity is estimated to range between 1,900 to 2,325 tonne per hour (tph) with an OEE (Overall Equipment Effectiveness) of 61.9% which represents 5,425 productive loading hours a year. This translates to annual production of 10.3 to 12.6 M t per year per shovel.

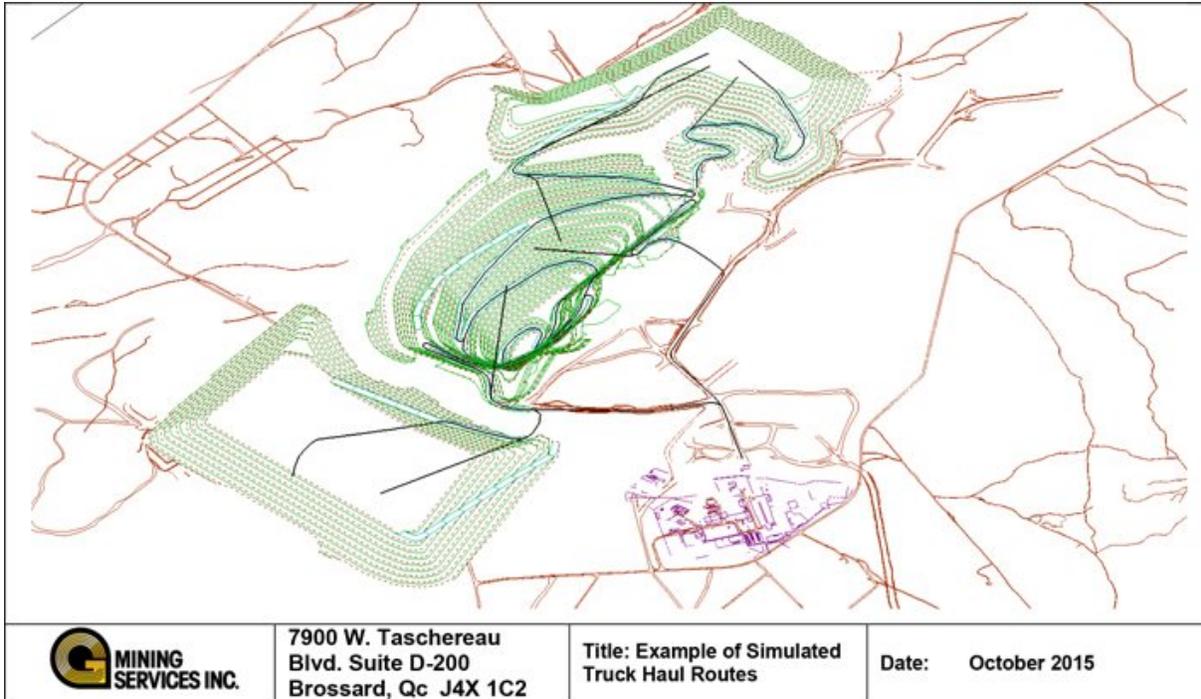
The estimated productivity of the Caterpillar 993 front-end loader is between 1,155 to 1,400 tph, depending on the rock type. With a planned OEE of 60.3% the annual productivity is between 6.1 and 7.4 M t per year per unit.

**16.6.6**      **Hauling**

Mine haulage will be performed by 136 metric tonne Caterpillar 785C mine trucks. The equipment's hauling productivity was estimated in Talpac software, developed by Runge. Annual haulage profiles have been digitalized with the haul routes exported to Talpac to estimate cycle times. Cycle times have been estimated for each bench and for each destination by interpolating between simulated results where necessary. Average cycle times have been weighted based on tonnage or number of cycles. Cycle times were also estimated for each dump location which uses different ramps in the pit. The number of haul trucks required is based on the hauling hours needed per year for ore and waste transportation, as well as on estimated ready hours available per unit.

The peak truck requirement is estimated at 22 units by Year 5; there are currently 17 trucks purchased for the project.

Figure 16.6: Example of Simulated Truck Haul Routes



#### 16.6.7 Dewatering

The pit area has to be dewatered to allow for dry conditions in the pit. A total four (4) pumps will be running throughout Year 2 to dewater the open pit and during the following years, from Year 3 to Year 9, six (6) pumps will be running at the same time. At the end of the mine life, four pumps will be left.

In addition to pit sumps three dewatering boreholes are planned in the pit bottom as well as ten pit perimeter dewatering wells. The hydrogeological monitoring will require eighteen new and replacement piezometer holes. Timing of drilling of the dewatering boreholes is planned in Year 3. The dewatering boreholes drilled in the pit have typically been located in the middle of Sadiola Fracture Zone where good hydraulic conductivity is achieved and these holes are re-drilled every 80-100 m drop down in mining level.

Since the pit operations have ceased in July 2009, a pit lake has developed at the pit bottom. A groundwater inflow rate of 31 m<sup>3</sup>/h was estimated. Piezometers will be installed and exploration drill holes will be used for dewatering purpose.

**16.6.8 Road and Dump Maintenance**

Road maintenance plays a significant role in reducing truck unit costs and allowing for optimal truck haulage productivity. The road maintenance fleet consists of one (1) wheel dozer, three (3) 16ft blade motor graders and two (2) water trucks for dust control with one in operation at all times. The wheel dozer is a Caterpillar 834 and has already been purchased as well as two of the three (3) Caterpillar 16 graders. The two (2) Caterpillar 777 trucks with mega water tanks have also been purchased.

Dump maintenance consists of a total of six (6) Caterpillar D9 dozers on waste dumps and stockpiles pushing piles and maintaining proper dumping areas. Four (4) of the six (6) units are presently on site.

**16.6.9 Support Equipment**

Support equipment are necessary for various tasks such as wall scaling, building berms, cleaning ditches, building mine roads, transporting employees, lighting, fuel and lube servicing of track equipment, transporting supplies, changing tires in the field, and performing field mechanical maintenance and repairs.

The list of support equipment required to support the production fleets is presented in Table 16.9. Only the two compactors have been purchased at this time and all other units must be purchased as required over time.

**Table 16.9: Support Equipment Peak Requirements**

<b>Equipment</b>	<b>Max Quantity</b>
Small Excavator (349)	2
Wheel Loader (6.1 m <sup>3</sup> )	1
Fuel Truck	2
Light Plants	8
Pickups	30
Pit Bus	2
Mech. Service Truck	2
Tire Service Truck	1
Crane	1
Boom Truck	1
Forklift	1
Welding Machines	4
IT Carrier	2
Compactor	2

**16.7 Mine Equipment & Manpower Requirements**

Table 16.10 shows the equipment requirements to meet the production schedule. Many of the units have already been purchased for the project.

The mine will have 532 employees at the peak in Year 8, of which 96% will be nationals and the remaining 4% will be expatriates. About 44% of the employees will be in the mine operations department, 22% will be in geology which includes hydrogeology, health and grade control technicians and samplers. Approximately 28% of employees in mining will be in mechanical maintenance and 6% in the engineering support department.

Table 16.10: SSP Equipment Requirement Schedule

Requirements	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
<b>Major Equipment</b>												
Pit Viper 235		4	5	5	5	5	5	5	5	3	3	1
FlexiRoc D65		-	2	2	2	2	2	2	1	1	1	1
FlexiRoc D65 RC		2	2	2	2	2	2	2	2	1	1	1
RH170/CAT 6040		3	3	3	3	3	3	3	3	2	2	1
CAT 993		2	2	2	1	1	1	1	1	1	1	1
CAT 785C		13	19	21	22	22	22	22	22	19	17	9
CAT D9T		6	6	6	6	6	6	6	6	4	3	2
CAT 834K		1	1	1	1	1	1	1	1	1	1	1
CAT 16M		3	3	3	3	3	3	3	3	2	2	2
CAT 777 WT		1	1	1	1	1	1	1	1	1	1	1
D65 RC		2	2	2	2	2	2	2	2	1	1	1
CAT 349F		2	2	2	2	2	2	2	2	1	1	1
CAT 980		1	1	1	1	1	1	1	1	1	-	-
<b>Sub-Total</b>		<b>38</b>	<b>47</b>	<b>49</b>	<b>49</b>	<b>49</b>	<b>49</b>	<b>49</b>	<b>48</b>	<b>37</b>	<b>33</b>	<b>21</b>
<b>Support Equipment</b>												
10 in Pump		4	6	6	6	6	6	6	6	4	4	4
Fuel Truck		2	2	2	2	2	2	2	2	2	1	1
Light Plant		8	8	8	8	8	8	8	8	8	4	4
Hilux Pickup		30	30	30	30	30	30	30	30	30	20	20
Pit Bus		2	2	2	2	2	2	2	2	2	1	1
Mech. Service Truck		2	2	2	2	2	2	2	2	2	1	1
Tire Service Truck		1	1	1	1	1	1	1	1	1	1	1
Crane 60 t		1	1	1	1	1	1	1	1	1	1	1
Boom Truck		1	1	1	1	1	1	1	1	1	1	1
Forklift 4 t		1	1	1	1	1	1	1	1	1	1	1
Welding Mach.		4	4	4	4	4	4	4	4	4	2	2
IT Carrier	IT Carrier	2	2	2	2	1	1	1	1	1	1	1
CS56		2	2	1	-	-	-	-	-	-	-	-
<b>Sub-Total</b>		<b>60</b>	<b>62</b>	<b>61</b>	<b>60</b>	<b>59</b>	<b>59</b>	<b>59</b>	<b>59</b>	<b>57</b>	<b>38</b>	<b>38</b>

## **17. RECOVERY METHODS**

### **17.1 Introduction**

The Sadiola Sulphide Project will be developed to mine the hard sulphide orebody that lies below the exhausted weathered horizon. This will be achieved with a pushback to the existing main pit, and it will require construction of a processing facility capable of processing hard rock ore.

The processing plant, which includes current and new equipment, will be designed to process 900 tph of hard ore for the Sadiola Deep Sulphide Project. The design of the processing plant is based on the transition of processing soft ore to hard ore over the life of the mine. The flexibility of the combined existing and new processing plants to process the two types of ore in parallel will be maintained in the design for the first two (2) years after the commissioning of the new plant. The combined plants will be able to process 600 tph of hard ore with 600 tph of soft ore.

When the oxide ore stockpile will be exhausted and/or when the full ramp up of the main pit sulphide mining to 7.2 Mtpa of ore, the existing soft plant will be modified to help process a higher throughput of hard rock. It is envisaged that the new processing circuit will provide the crushing and SAG milling requirements for both plants, with the feed split between the old and the new circuits after the SAG mill to achieve 900 tph of hard ore.

### **17.2 Hard Sulphide Ore Process Description**

A simplified hard sulphide ore process flowsheet is shown in Figure 17.1 and Figure 17.2.

Figure 17.1: Simplified Flowsheet of the Hard Sulphide Ore Process – New Plant

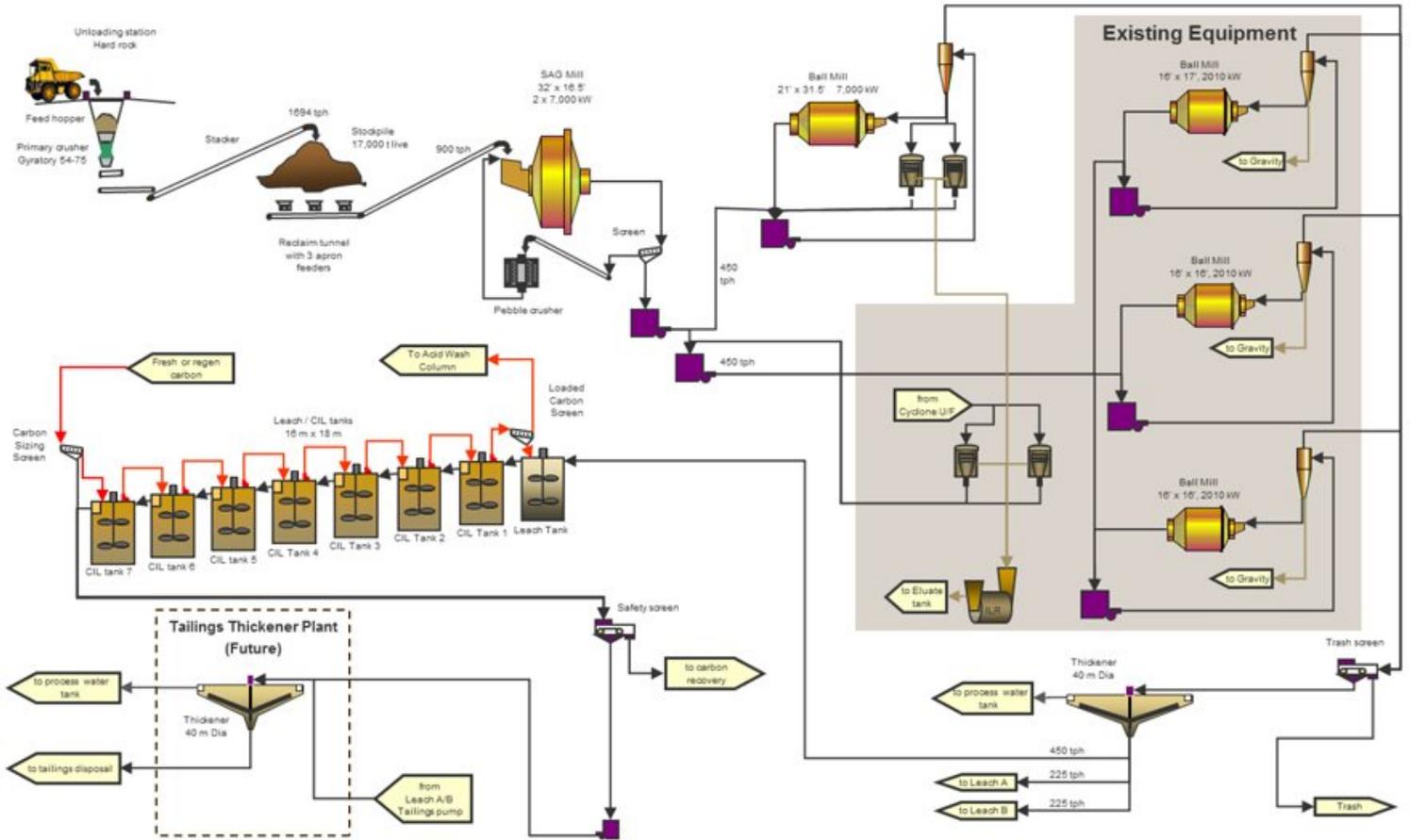
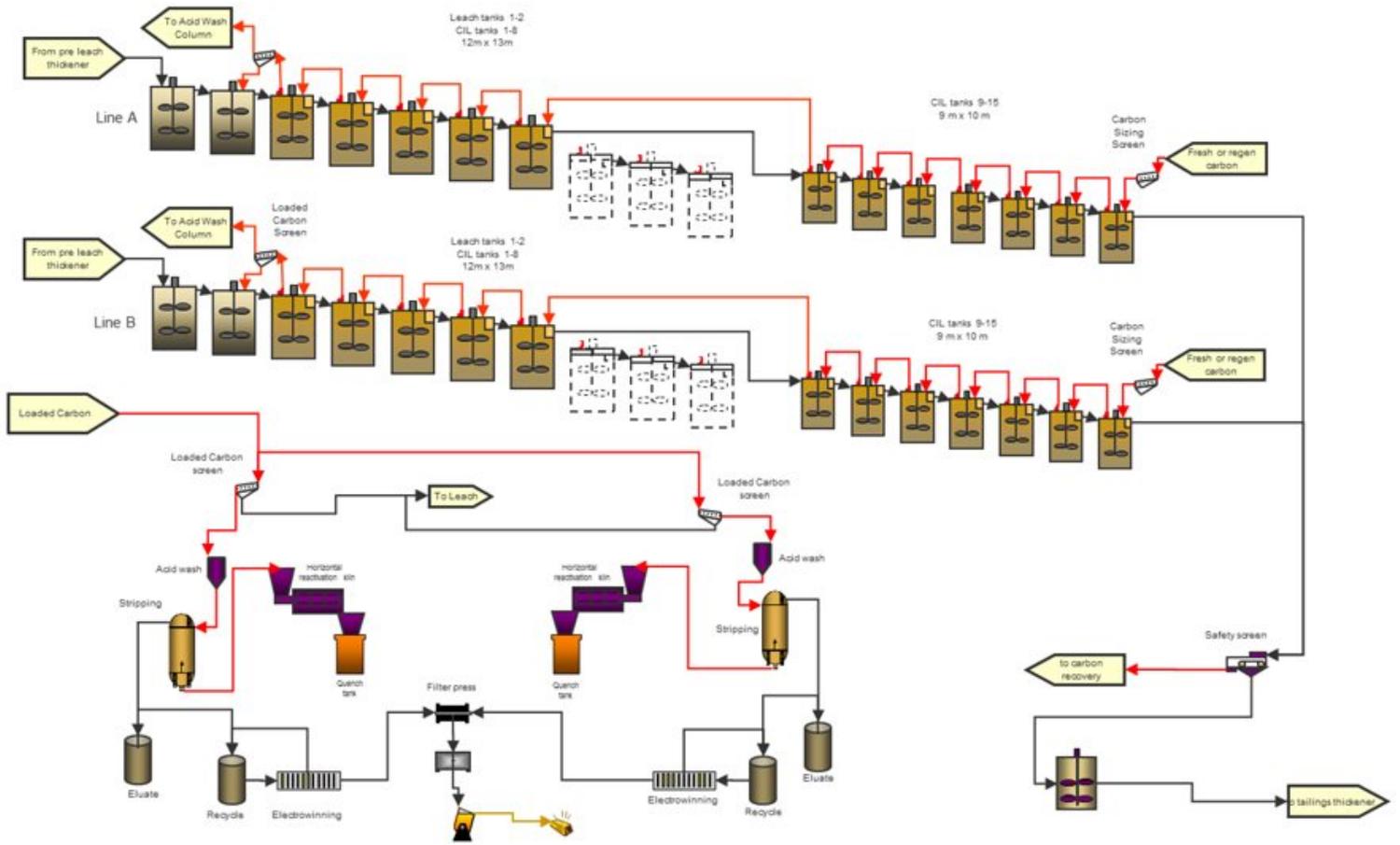
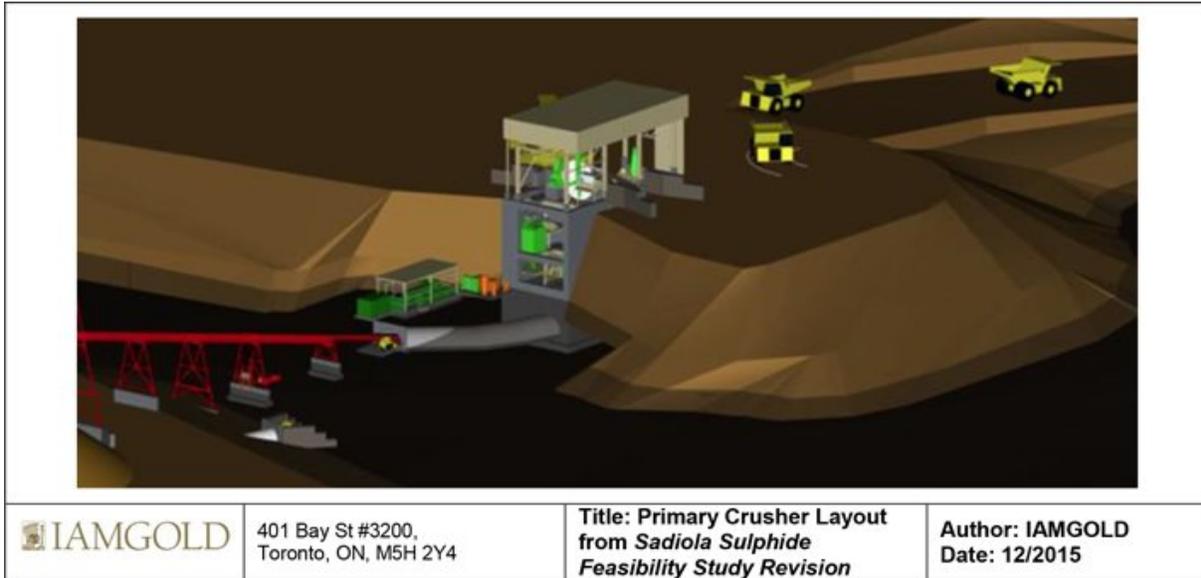


Figure 17.2: Simplified Flowsheet of the Hard Sulphide Ore Process – Existing Plant



**17.2.1 Ore Receipt and Primary Crusher**

Mill feed will consist of a combination of direct mine feed, delivered by 140 t haul trucks directly to the unloading station, or alternatively can be recovered from new stockpiles from the deep sulphide project ore from the existing stockpiles. From the 210 t ROM ore feed hopper the ore will be discharged to a gyratory crusher for primary crushing. Feed size of the crusher is expected to have a F<sub>80</sub> of 750 mm, with the discharge size expected to have a P<sub>80</sub> of 127 mm. Figure 17.3 shows the primary crusher layout. A hydraulic rock breaker will break up oversized rocks in the feed to the crusher. The crushed ore will be transferred with a belt conveyor to a crushed ore stockpile, with live storage of about 10,800 t and total storage capacity of 36,000 t.

**Figure 17.3: Primary Crusher Layout****17.2.2 Stockpile Reclaim**

Three variable speed apron feeders will extract the ore from the crushed ore stockpile and can be operated individually or simultaneously, with each feeder having the capacity to feed the SAG mill at 900 tph. The three (3) feeders transfer the ore via the SAG Mill feed conveyor, fitted with weightometers and metal detectors, to the SAG Mill feed chute. After initial grinding in the SAG mill the mill discharge is screened on one SAG Mill discharge screen. Oversized material is discharged via conveyors to the pebble crusher, with the discharge from the pebble crusher returned to the SAG mill.

### 17.2.3 Grinding Circuit

The purpose of the grinding circuit is to produce P<sub>80</sub> slurry of 53 µm and to deliver the slurry to the carbon-in-leach circuit. The grinding circuit is made up of a 10,000 kW SAG mill followed by four (4) ball mills (one 7,000 kW and three 2,070 kW) installed in parallel.

The SAG mill feed conveyor discharges into a retractable chute feeding the SAG mill. Water is added to the mill to produce 70% solids (% w/w). Lime is also added to the SAG feed chute to adjust the slurry pH to 10.5.

The SAG mill is installed in a closed circuit comprising of one vibrating screen and a pebble crusher (cone type crusher). The SAG mill circuit product is designed to produce a final product of slurry for transfer to the ball mills with a P<sub>80</sub> of 1,197 µm.

The SAG mill discharges into a box that feeds one double deck screen. Water is added to the box to lower the screen undersize slurry solids content to 46.8% to facilitate pumping.

The screen oversize (F<sub>80</sub> of 34.25 mm) is transferred via a series of conveyors to the pebble crusher. Two self-cleaning magnets and a metal detector are installed on the conveyors to prevent tramp metallic pieces from falling into the cone crusher. A diverter chute enables the bypass of the pebble crusher when metal is detected or during maintenance.

The pebble crusher discharge (P<sub>80</sub> of 15.85 mm) is returned to the SAG mill feed conveyor for return to the SAG mill.

SAG Mill discharge screen is equipped with a pump box to transfer the underflow stream to the ball mill circuit. From the pump box the screen undersize slurry is transferred by a variable speed pump to a diversion box where the flow is split equally between the new plant (450 tph) and the current plant (450 t/h).

From the diversion box, the slurry transfers by gravity to the new ball mill pump box (ball mill #2A), while the transfer to the three current ball mills (ball mills #1A, #1B and #1C) is done via a pumpbox and one of two (2) variable speed pumps (one operating and one stand-by).

#### 17.2.3.1 Current Plant

In the existing plant, the slurry transferred from the SAG mill discharge is pumped to the new ball mills distribution box that splits the flow equally between the existing three (3) ball mill circuits, now operating in

parallel. The three (3) ball mills are installed in a closed circuit with hydrocyclones and the following description applies to all three (3) circuits.

From the distribution box, the slurry gravitates into the mill discharge sump and is pumped by one of two variable speed pumps to the hydrocyclone cluster for classification. Water is added to the pump box to adjust the hydrocyclones feed slurry density to about 57.7% solids. The level in the pump box is maintained by adjusting the speed of the pump with the variable speed drive.

A large portion of the cyclones underflow (coarse fraction) is directed by gravity to the ball mill feed chute. The remaining smaller portion of the cyclone underflow transfers under gravity into a common underflow collection launder that flows to the gravity concentration circuit.

The overflow (fine fraction, at a P<sub>80</sub> of 53 µm) from the three (3) cyclone clusters is transferred by a common launder to the pre-leach thickener.

The mill discharges through a trommel screen to the discharge sump from where the slurry returns to the cyclones.

#### 17.2.3.2 New Plant

The slurry diverted to the new ball mill gravitates to the ball mill discharge pumpbox, from where the slurry is pumped by one of two variable speed pumps to the hydrocyclone cluster for classification in a closed circuit with the ball mill. Water is added to the pump box to adjust the hydrocyclones feed slurry density to about 57.7% solids, by weight. The level in the pump box is maintained by adjusting the speed of the pump with the variable speed drive.

The underflow (coarse fraction) from nine operating cyclones in the cluster is directed by gravity to the ball mill retractable feed chute. Lime can be added in the ball mill feed chute for pH adjustment. The underflow from the four (4) remaining operating cyclones in the cluster flows to the gravity concentration circuit.

The overflow, at a P<sub>80</sub> of 53 µm, is directed by gravity to the pre-leach thickener where it joins the overflow from the current plant hydrocyclones.

The mill discharges through a trommel screen by gravity back into the pump box.

The ball mill and SAG mill are equipped with inching drives to facilitate maintenance and liner change. The grinding area is serviced by an overhead crane to facilitate maintenance and a sump pump to collect and dispose of any accidental spill.

## 17.2.4 **Gravity Circuit**

### 17.2.4.1 **Current Plant**

In the existing plant, approximately 25% of the three cyclone clusters underflow is sent to the existing gravity concentration circuit for gold recovery. The installation and operation of the existing gravity concentration circuit will remain unchanged.

### 17.2.4.2 **New Plant**

The new plant expansion includes a gravity concentration circuit consisting of two gravity concentrators installed in parallel, each concentrator being fed by the underflow from two (2) cyclones. Approximately 30% of the total cyclone underflow is sent to the gravity concentrator circuit for gold recovery.

In each line a screen is installed ahead of the gravity concentrator to return the oversize material (approximately 15% of the feed) to the ball mill pump box for reclassification and grinding. The screen undersize feeds the gravity concentrator where fine free gold is recovered into a gravity concentrate. The concentrate from both concentrators is collected in a single pump box and is pumped to the intensive leach reactor (ILR) located in the current plant. On an annual basis, approximately 1,860 kg of gold will be recovered through the new gravity concentration circuit.

The tailings from the gravity concentrators will join the screen oversize and flow by gravity back to the ball mill pump box.

## 17.2.5 **Pre-Leach Thickening**

In the pre-leach thickening area (Figure 17.4) the solids content of the slurry feeding the leach and carbon-in-leach (CIL) circuits is increased from approximately 30% solids to 52% solids. The water recovered at the thickener overflow is used as process water throughout the plant.

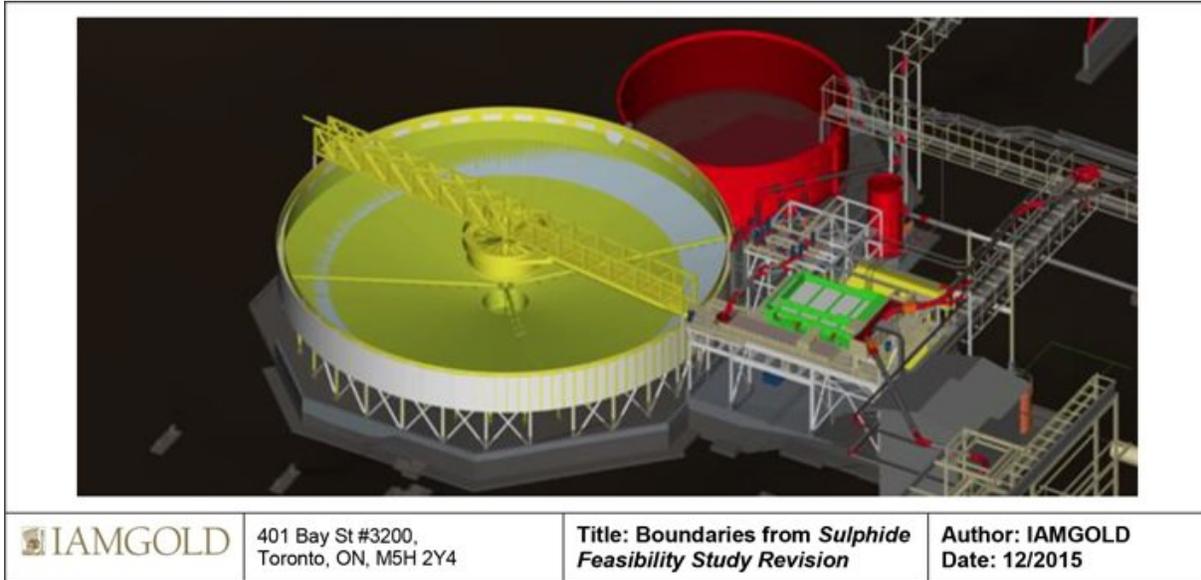
The current plant cyclones overflow collection launder and the overflow from the new cyclones are combined on the linear trash screen installed ahead of the pre-leach thickener. The screen is equipped with a spray system to facilitate screening and for screen cloth cleaning on a continuous basis.

The screen undersize transfer to a collection launder that feeds the thickener feed box. Flocculent is added to the feed box to promote particles sedimentation in the thickener.

Three underflow pumps are installed below the 40 m diameter thickener. One pump is used to transfer the thickened slurry to the current circuit leach tanks, the second pumps to the new circuit leach tank and the third is a standby pump.

The thickener overflow is directed by gravity to a process water tank. Two (2) pumps deliver process water to the users located in the current and in the new plant. Clean water is also delivered to a gland seal water tank located in the area.

**Figure 17.4: Pre-Leach Thickener Area**



### 17.2.6 Leach and Carbon-In-Leach

In the leach and CIL circuit, gold is recovered using cyanide and is adsorbed onto activated carbon.

The plant operates three parallel leach and CIL circuits. One pre-leach thickener underflow pump transfers about 50% of the slurry to the current plant where it is handled by two converted leach and CIL circuits. The other underflow pump transfers the other 50% of the slurry to a new leach and CIL circuit in the new plant.

The gold recovery from the process is estimated at 76.0% and the total retention time in each leach and CIL circuit is approximately 48 hours.

#### 17.2.6.1 Current Plant

In the current plant, the leach feed from the thickener underflow is pumped into a distribution box where it is split between two parallel leach and CIL circuits.

With the new design for the processing of sulphides, the first two leach tanks in each circuit continue to serve as leach tanks while five of the last eight leach tanks are converted to CIL tanks. The overflow from the last CIL tank bypasses the last three installed tanks to feed the existing CIP tanks converted to CIL. The total retention time provided in each circuit is about 47 hours.

Pure oxygen from a dedicated 20-tonne capacity oxygen plant is injected at the bottom of both first leach tanks through an EDR system. In the second leach tanks and all the CIL tanks low pressure compressed air is used instead of oxygen and is injected through the agitator shaft.

The slurry flows by gravity from one tank to the next passing through an interstage screen to retain carbon in the tank. The design of the circuit allows the bypass of any tank in case of maintenance.

Screened activated carbon is added to last CIL tank. A diversion box allows carbon addition to tank 11 in case the last tank is bypassed.

The activated carbon is pumped counter-current to the flow of slurry using vertical pumps installed in each tank. The loaded carbon is pumped from the first CIL tank in each circuit to a new common loaded carbon recovery screen. The loaded carbon screen undersize returns into the first CIL tank. The loaded carbon (oversize) falls into the grit separator tank before transfer to the loaded carbon tank and is pumped to the loaded carbon screen.

The barren slurry overflowing from both the final CIL tanks transfer to the linear screens in the residue disposal area.

#### 17.2.6.2 New Plant

In the new plant the leach feed from the thickener underflow is sampled and transferred to the leach feed distribution box. The distribution box allows the bypass of the leach tank to the first CIL tank.

Cyanide is added to the distribution box for gold leaching. Lime can also be added to maintain the pH level at 10.5.

Pure oxygen from a new and dedicated 20-tonne capacity oxygen plant is injected at the bottom of the leach tank through a reversed cone type of sparging system. Air is used instead of oxygen in the CIL tanks.

The slurry flows by gravity from one tank to the next passing through an interstage screen to retain carbon in the tank. The design of the circuit allows the bypass of any tank in case of maintenance.

Activated carbon is added to the final (seventh) CIL tank. A diversion box allows carbon addition in the sixth tank in case the final tank is bypassed.

The activated carbon is pumped counter-current to the flow of slurry using vertical pumps installed in each tank. The loaded carbon is pumped from the first CIL tank to the loaded carbon recovery screen. The loaded carbon screen undersize returns into the first CIL tank. The loaded carbon is recovered in the loaded carbon tank and is pumped to the loaded carbon screen located in the current plant.

The barren slurry overflowing from the final CIL tank transfers to the safety screen where fine carbon is recovered.

#### **17.2.7**            **Fine Carbon Recovery**

The fine carbon recovery circuit is necessary to recover fine carbon from the leach and CIL circuit tailings to limit gold losses associated with the fine carbon particles.

The overflow from the final CIL tanks (current and new plants) flows by gravity to their respective linear belt type safety screen. The screen oversize (mostly carbon) is recovered onto a second screen, the fine carbon recovery screen from where it is discharged to the carbon recuperation hopper that fills fine carbon bags. The fine carbon bags are to be sold to a refiner to be credited for the gold content.

The undersize from the safety screen is transferred to the tailings thickener for final discharge to the tailings storage facility.

#### **17.2.8**            **Tailings Transfer**

During sulphides operation, tailings from both the new plant and the existing plant report to the tailings storage facility.

#### 17.2.8.1 Current Plant

In the current plant, for the oxide ore processing two parallel trains (one in operation and one stand-by) of six centrifugal pumps installed in series were used to pump the final tailings to the tailings storage facility.

For the sulphide ore processing, only one pump will be required to transfer the tailings to the tailings storage facility. The necessary piping is to be installed to by-pass the ten unused pumps.

#### 17.2.8.2 New Plant

In the new plant, two (2) parallel trains (one (1) in operation and one (1) stand-by) of two (2) centrifugal pumps installed in series are necessary to pump the final tailings to the tailings storage facility.

### 17.2.9 Carbon Stripping and Regeneration

The carbon acid wash, carbon elution, carbon handling and regeneration and solution electrowinning circuits located in the current plant remain unchanged and can process the loaded carbon from the new expansion and modified existing CIL circuits.

Each of the two (2) current parallel and identical lines is designed to process seven (7) tonnes of carbon daily. The process conditions generated by the new expansion and the current plant modifications require the processing of two (2) six (6) tonnes batches daily, one (1) batch per line.

### 17.2.10 Tailings Thickening Plant (TTP)

The tailings management strategy put forward considers deposition of the “Hard Sulphides” stream in a New TSF area fully lined with a geo-synthetic membrane. Tailings are to be deposited from the dykes as slurry at 50% solids at the start of the New TSF operation. Towards the end of the operation, tailings will be thickened to a targeted 68% solids content and deposition will take place from service roads built directly on the tailings surface inside the New TSF area. Changing from slurry to thickened tailings will increase the storage capacity of the New TSF and shape its deposition surface to promote surface runoff and prevent water accumulation in the TSF.

Tailings will be delivered from the mill via the mill tailings pipeline. Prior to entering the plant, valves will be installed to allow for the diversion of the tailings to the emergency discharge point, by-passing the TTP during downtime.

Tailings will enter the TTP shown in Figure 17.5 and be discharged into the Thickener Feed Box. The tailings will be mixed with diluted flocculent, TSF reclaim water, plant sump liquor and any thickener underflow returns. The Thickener Feed Box offers some buffering against fluctuations in the feed of

tailings and allows for homogenisation of the slurry should segregation occurs during transfer from the mill.

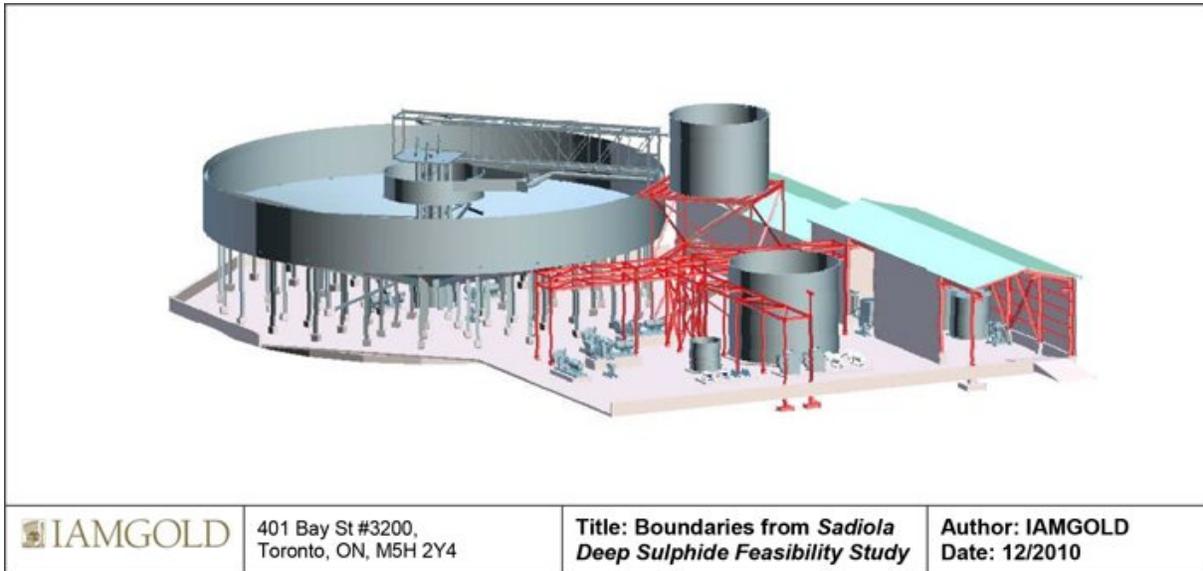
The Thickener Feed Box will also allow the tailings cyanide dilution with the TSF reclaim water. This will aid in the tailings cyanide recovery and its reuse in the mill allowing a faster return through the thickener overflow and excess water return system.

The combined tailings, TSF reclaim water and flocculent mix will be delivered to the High Rate Thickener at a solids concentration of about 42.6 wt% and will produce an underflow of about 68 wt% solids. Thickener overflow water will be transferred to the process water tank.

The density of the discharged tailings will be controlled by density and flow meters downstream of the Thickened Underflow Pumps. Dilution water will be added to the thickener underflow material before it enters the pumps if the density is too high. Over diluted tailings will be diverted back to the thickener feed box if the ideal density range cannot be achieved.

The thickened tailings will be pumped to the TSF through a series of in-line Thickened Tailings Disposal Pumps.

**Figure 17.5: Tailings Thickening Plant**



Process water feeds from the thickener overflow to a Process Water Tank located in the TTP. The process water will be distributed by Process Water Pumps for use in the plant. Excess process water will

be returned to the mill plant through the Excess Water Pumps. Flush water for the discharge pipelines will be fed directly to the Thickened Tailings Disposal Pumps.

The plant fresh water tank will be filled with reclaim water from the TSF, which will be filtered prior to entering the fresh water tank. Fresh water pumps will deliver water from the tank to the flocculent mixing system and the gland seal water pumps will be fed from the fresh water tank.

#### **17.2.11 Water Management**

During the course of the Feasibility Study Revision December 2015, the water management scheme evolved under the direction of Golder and Associates. In the newly developed water management scheme, the existing tailings storage facility is re-habilitated and the Upper and Lower Water Basis are eliminated. Also, the installation of the new TTP is postponed to the eighth year of operation.

In this scheme, during the first seven (7) years, water is re-circulated to the new sulphide process water tank and the existing process water reservoirs by floating pumps installed directly in the TSF internal water basis.

After the installation of the new TTP during the eighth (8th) year of the project, water is re-circulated to the new sulphide process water tank via TTP overflow water tank, and to the existing process water reservoirs by floating pumps installed directly in the TSF internal water basis.

Figure 17.6 summarizes the water management schemes developed for the first seven (7) years of Scenario #2. In the case, approximately 3,010,000 m<sup>3</sup>/y of fresh water from the Senegal River is required for the processing operation. A large proportion (88%) is required for the process. The remaining 12% is treated to supply potable water to the village and to the plant site.

Figure 17.6: Water Management Scheme

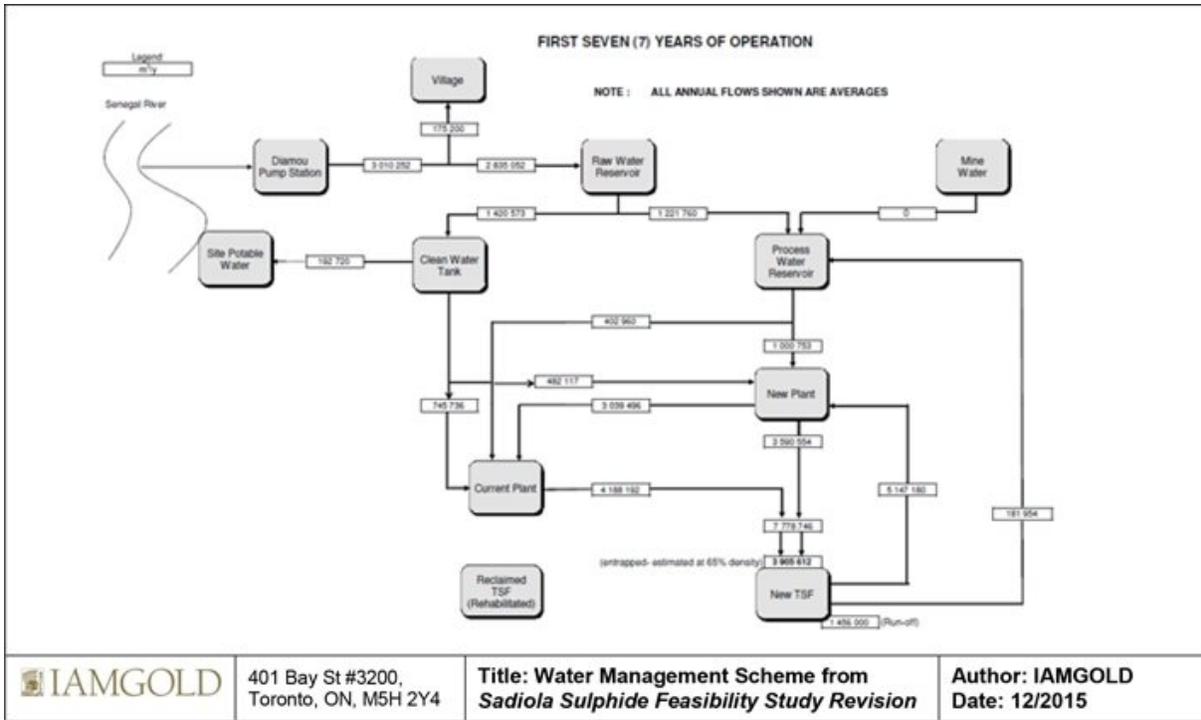
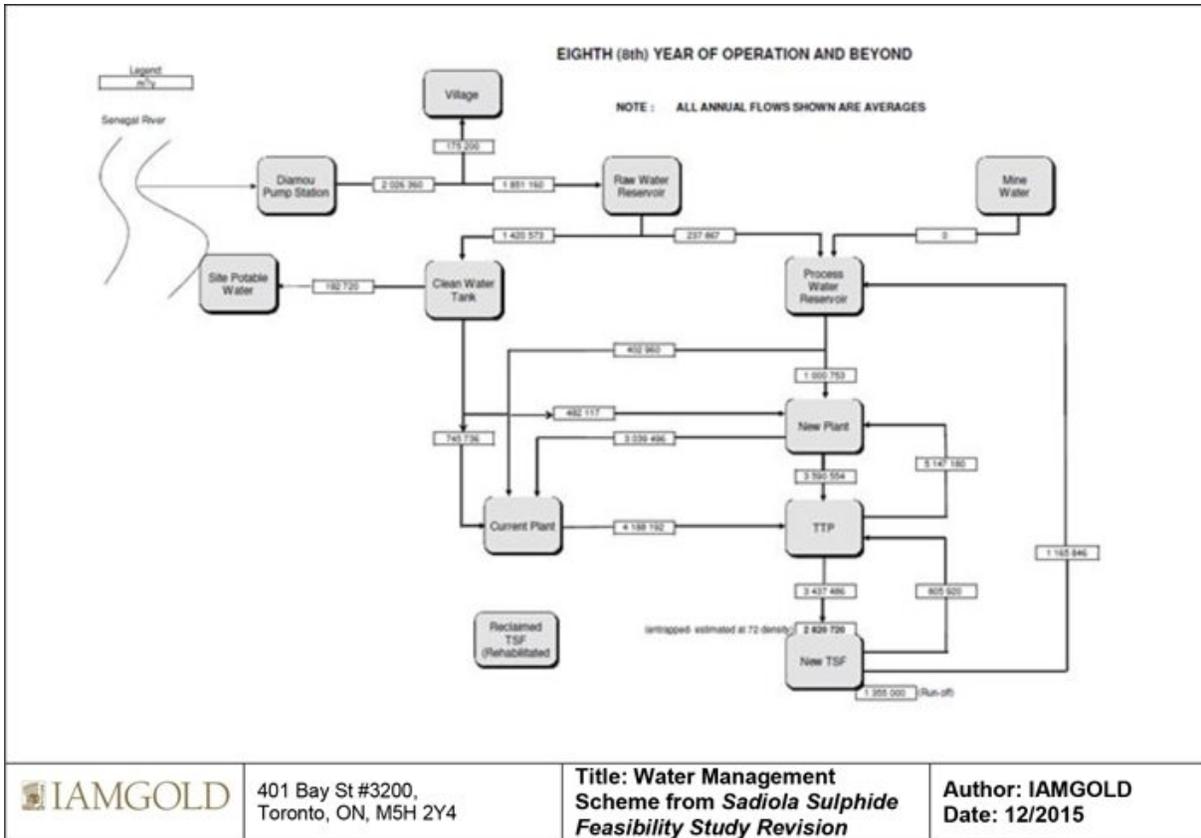


Figure 17.7 summarizes the water management schemes developed for the eighth (8<sup>th</sup>) year of operation and beyond. In the instance, approximately 2,026,000 m<sup>3</sup>/y of fresh water from the Senegal River is required for the processing operation. The process requires a larger proportion (82%) of the river. The remaining 18% is treated to supply potable water to the village and to the plant site.

For both scenarios, most of the water used by the process is re-circulated water from the pre-leach thickener overflow, the tailings treatment plant and the rainfall run-off in the tailings storage facility.

Figure 17.7: Water Management Scheme



**17.2.12 Reagent Preparation**

No major modifications are required in the current plant reagents area to suit the new sulphides process. A brief description of the modifications is listed below.

**17.2.12.1 Lime**

The lime pumping system, including two (2) pumps, is modified to deliver the milk of lime solution in a closed-loop system.

One (1) pump delivers the milk of lime to the leach and CIL circuits via a closed loop. The other pump delivers the milk of lime to the grinding circuits, also via a closed loop.

#### 17.2.12.2 Cyanide

In the cyanide preparation area, two new pumps are installed to deliver sodium cyanide to the new grinding and leach area and the CIL area. One (1) metering pump per addition point is added.

#### 17.2.12.3 Lead Nitrate

The lead nitrate preparation and distribution equipment is not used for the hard sulphide ore processing.

#### 17.2.12.4 Caustic (Sodium Hydroxide)

No modifications are required in the current sodium hydroxide receiving, storage and distribution facility.

#### 17.2.12.5 Hydrochloric Acid

No modifications are required in the current hydrochloric acid receiving, storage and distribution system.

#### 17.2.12.6 Copper Sulphate

During oxides processing, copper sulphate was added to the detoxification tanks for cyanide destruction. The copper sulphate preparation and distribution equipment is no longer used for the hard sulphide ore processing.

#### 17.2.12.7 Sodium Metabisulphite

During oxides processing, sodium metabisulphite was added to the detoxification tanks for cyanide destruction. The sodium metabisulphite preparation and distribution equipment is no longer used for the hard sulphide ore processing.

#### 17.2.12.8 Viscosity Modifier

The viscosity modifier system is currently inactive. For the new sulphides process expansion, the viscosity modifier is not used and remains decommissioned.

#### 17.2.12.9 Flocculent

The flocculent system used to deliver flocculent solution to assist particles sedimentation and promote water clarity in the water recovery tank located in the fine carbon recovery area is no longer in service.

#### 17.2.12.10 New Plant

Flocculent is used to assist particles sedimentation and to promote overflow clarification in the pre-leach thickener. It will be delivered on site in dry form in one (1) tonne super bags and must be dissolved with water and diluted before it will be injected in the pre-leach thickener feed box.

The flocculent consumption is estimated at 15 g/t of mill feed and the flocculent system is designed to deliver approximately 18 kg/h of flocculent (dry basis).

### 17.2.13 Oxygen Plant, Compressors and Services (Areas 1640 & 1670)

#### 17.2.13.1 Air Compressors

Three low pressure air compressors (two operating and one stand-by) are installed in the new plant to provide compressed air to the new CIL tanks and to the second leach and CIL tanks in the existing plant. One compressor is dedicated for each plant and the spare one can be used as a back-up unit for either one of the operating compressors.

Two (2) high pressure air compressors are also installed in the new plant to provide service air and dry instrument air.

#### 17.2.13.2 Oxygen Plant

A new 20-tonne per day capacity and dedicated oxygen plant is installed in the new plant expansion. Oxygen is injected at the bottom of the leach tank. The air separation plant uses the oxygen VSA technology (vacuum swing adsorption). The plant is sized to accommodate another equivalent leach and CIL circuit expansion.

### 17.3 Process Design Criteria

The process plant design was developed by SNC Lavalin and the plant is designed to treat up 7.2 Mtpa of hard sulphide ore. The plant availability is assumed to be 92% for an hourly production rate of 900 tph. The design gold head grade is 2.21 g Au/t with an overall recovery of 76%.

An abbreviated summary of the main process design criteria is shown in Table 17.1.

Table 17.1: Summary of Main Process Design Criteria

GENERAL INFORMATION	UNITS	DATA
<b>Ore throughput</b>		
Hard ore design throughput	Tpa	7,200,000
Process plant availability	%	92
<b>Overall Ore grade</b>		
Gold	g/t	2.21
Silver	g/t	1.00
<b>Metal Recovery</b>		
Gold - Gravity circuits	%	20.7
Gold - Overall	%	76.0
Silver - Overall	%	40.0
<b>Crushing &amp; Grinding parameters</b>		
Bond Impact work index	kWh/t	12.1
Bond Ball mill work index	kWh/t	13.1
Abrasion resistance index		0.08
<b>JK SimMet -Parameters</b>		
SAG mill A*B		33.5
SAG mill ta		0.31
<b>Crushing</b>		
Operating time	d/week	7
Operating time	hpd	16
Hourly production - design	tph	1,671
Crusher feed size (F80)	mm	750
Crusher product size (P80)	mm	127
Crushed ore live storage	hours	12
<b>Grinding</b>		
Mill type		SAG + 4 Ball mills
SAG mill feed size F80	mm	127
SAG mill product size after screening T80	µm	1,197
SAG mill Ball charge	%	12
SAG mill Ball size	mm	125

GENERAL INFORMATION	UNITS	DATA
Ball mill circuit (cyclone O/F) product size P80	µm	53
<b>Gravity circuit</b>		
Gravity Concentrator feed (each)	tph	225
Number of gravity concentrator	#	2
<b>Dewatering – Pre-leach thickener</b>		
Thickener type	-	Hi-Rate
Thickener solids feed rate - average operation	t/h	900
Thickener unit area at design feed rate	t/(m <sup>2</sup> /h)	0.93
Thickener underflow density	% solids by wt.	52.0
<b>CIL</b>		
New Leach - C.I.L. feed rate	tph solids	450
New Leach tanks	#	1
New C.I.L. tanks provided	#	7
Residence time	h	48.6
<b>Carbon Loading and Movement</b>		
Carbon concentration - New CIL	g/L	15
Net carbon loading – Au	g/t	2,131
Net carbon loading – Ag	g/t	864
<b>Carbon Elution</b>		
New CIL elution column capacity	tpd	6.0
<b>Tailings Thickener (By Golder PasteTec)</b>		
Thickener solids feed rate - average operation	tph	900
Tailings disposal	% solids by wt.	68.0

Source: Sadiola Deep Sulphide Feasibility Study (December 2010)

**Table 17.2: Design Criteria Forecast Reagent and Consumables Consumption**

Reagents and Consumables	Unit	Consumption
Grinding Media	kg/t	1.60
Lime - (Quicklime basis)	kg/t	0.65
<b>Cyanide</b>		
Leach & CIL	kg/t	0.65
Elution	kg/t	0.03
Carbon	kg/t	0.07
Sodium Hydroxide	kg/t	0.14
Hydrochloric acid	kg/t	0.14
<b>Flocculent</b>		
Pre-leach thickener	kg/t	0.015
Tailings thickener	kg/t	0.015

Source: Sadiola Sulphide Project Feasibility Study 2015 (December 2015)

The process design criteria appears to be reasonable for the proposed operation and expects a reasonable amount of flexibility in the design due to the allowances made for the different ore types and ore characteristics.

#### 17.4 **Process Operating Costs**

The process operating costs include labour, maintenance, power consumption, reagents / consumable costs, tailings and water management costs. There was significant work done to revise the operating cost in the Sadiola Sulphide Feasibility Study Revision (December 2015). Labour compliment and consumables were developed from first principles, while the most up to date contractual rates for reagents was applied to determine an updated operating cost as outlined in Table 17.3. These operating costs reflect the configurations encountered in the first seven (7) years of operation (without TTP) and were obtained with a cost of purchasing diesel at US \$0.78/liter and using an exchange rate of 595 XOF/ US\$ Exchange.

**Table 17.3: Updated Process Operating Costs**

<b>Parameter</b>	<b>Unit</b>	<b>Rate</b>
Engineering	US\$/t	3.08
Labour	US\$/t	2.05
Reagents	US\$/t	4.58
Energy	US\$/t	4.92
TSF	US\$/t	0.13
<b>Total</b>	US\$/t	<b>14.77</b>

Source: Sadiola Sulphide Feasibility Study Revision (December 2015)

## 18. PROJECT INFRASTRUCTURE

Sadiola has operated several years with infrastructure to support plant, administration and warehouse needs as well as a mine village to accommodate both local and expatriate workers. The infrastructures included in this section are related to the site expansion and specifically relate to: electrical substations, construction camp and offices, truck shop, lube and wash bays, and warehouse buildings.

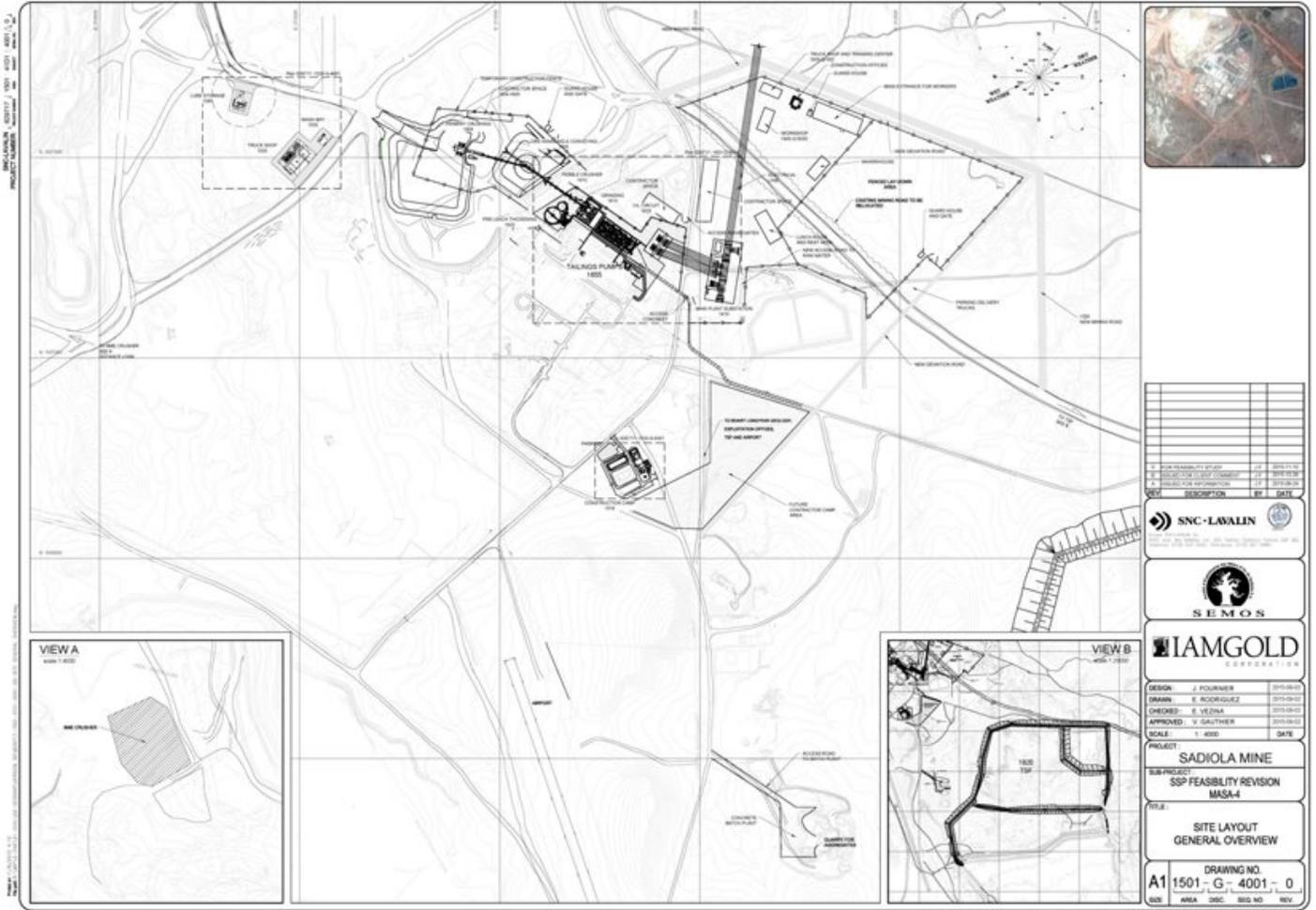
### 18.1 Overall Site Layout

The *Site Layout - General Overview* is shown in Figure 18.1

The key features shown on this layout are:

- The following temporary facilities added during the project:
- A 20-meter wide deviation gravel road for mining operations linking the existing mine pad to the Tailings Storage Facility (TSF);
- A 10-meter wide deviation gravel road to isolate mine operation traffic from construction traffic;
- Fencing to isolate the construction facilities including IAMGOLD's construction offices (1905);
- Laydown areas reserved for the contractors for their construction setup (storage and trailers). These are located to mainly the north-east of the existing plant and near the new crushing and handling area (1604-1605);
- A concrete batch plant is planned, about 1 km away along the road to the TSF beside the original quarry. Aggregates for the concrete batch plant will be produced there. See report on test results for concrete aggregates (Appendix 18.12);
- Construction camp (1518) to accommodate workforce during construction is a temporary facility;
- The new surface facilities to support mining operations include:
- A new Truck Shop Building (1530);
- A Wash Bay area adjacent to the Truck Shop (1530);
- A lube storage facility (1540).
- New electrical substation (1411 and 1415):
- The new electrical substations are fed by an aerial power line powered by EDM and replace the actual diesel power plant. The new electrical substations are connected to electrical grid and feed all facilities.

Figure 18.1: General Project Site Overview



- The electrical grid:
- Extensions to the existing electrical grid to feed permanent installations such as the Truck Shop, Lube Storage, Construction Camp and TSF.
- The Tailing Storage Facility (TSF) (1820):

New Tailing Storage Facilities will be located on the North side of existing TSF adjacent to the existing North Dyke. The new TSF will be constructed in phases following the containment requirements.

### **18.2 Water Services in the Mill Area (1600)**

The northern expansion of the mill impacts the following existing networks:

- Sanitary sewer: Portion of the existing 200 mm UPVC network requires to be relocated;
- Firefighting network: The firefighting network is extended to the mill extension;
- The potable water system is extended to the mill extension and a pipe to the water treatment plant needs to be relocated;
- A diversion ditch bordering the northern limits of the existing mill, which directs the run-off outside the site, needs to be partially relocated.

Run-off along the mill's roadways is collected by a series of drains towards a ditch which carries the collected water to the Pollution Control Dam (610). These drains are to be intercepted and directed to the existing pollution pond.

### **18.3 Electrical Services**

The electrical services for the new plant and the electrical feed to the existing plant are connected to the electrical distribution system fed by the new network power connection.

### **18.4 Construction Camp (1518)**

The Construction Camp (1518) will house IAMGOLD's personnel during the construction phase. The camp is built south of the plant and north of the airport runway. It is accessible via an existing gravel road. The camp has the following services:

- Fire water network;
- Potable water network;
- Sewer network;
- Electricity;

- Exterior lighting;
- Security fencing.

The camp accommodates 140 persons. The proposed camp includes two housing blocks and one service block.

#### 18.4.1 Water Services

The existing potable water network is extended to the temporary camp. The HDPE water line is underground. The existing fire protection network is extended to ensure fire protection at the camp with fire hydrants. It was evaluated that the existing firefighting network is able to supply the required flow at the necessary pressure. The construction camp sewage is collected by an underground sewer system.

#### 18.4.2 Electrical

Camp electrical feed is 400/230 V, 50 Hz, 3-phases, 4 wires + Ground. Estimated electrical loads are presented in Table 18.1.

**Table 18.1: Construction Camp Estimated Electrical Loads**

Description	Estimated Electrical Load (kW)
Camp B	280
Service building	591
Exterior lighting	20
<b>Total</b>	<b>891</b>

#### 18.4.3 Roadway, Access and Fencing

The construction camp is secured with a 2.45 m high barb wire fence. Access to the camp is via the east side. A gravel parking area is available before access through the main gate. In order for suppliers to have access to the food preparation area, access to the unloading dock is possible via the north side. This access is located outside the security fencing to facilitate delivery.

To ensure security lighting, 12 lamp posts with 250 W lighting are planned. Camp units are supplied with exterior lighting next to exits.

## **18.5 Truck Shop and Wash Bay (1530)**

### **18.5.1 Location**

The Truck Shop and Wash Bay are located northwest of the loading pad for crushing (1604) at the center of mining operations. The chosen location is at an elevation of approximately 140 m in a backfilled area southwest of the existing plant. The area is not currently serviced by water or electrical networks.

### **18.5.2 Building Description**

The building dimensions are 61.3 m x 46.1 m (2,820 m<sup>2</sup>) and it is divided into three areas:

- Change room and training rooms of 670 m<sup>2</sup>;
- Warehouse of 610 m<sup>2</sup>;
- Truck Shop area of 1,600 m<sup>2</sup> with workshop, and five (5) garage bays for equipment fleet maintenance.

The building is a structural steel building covered by pre-painted metal siding and roofing and surrounded by a concrete apron on two sides and is open on the garage façade. Each area is composed of single level buildings but height varies from 3 metres to 15 metres.

The Truck Shop bays are serviced by a 25-tonne electrical overhead crane. Clearance under the overhead crane is 12.0 metres to allow for maintenance of the Caterpillar 785C truck which has an 11.207 m with box up.

The five (5) garage bays are equipped with a total of three (3) reels to dispense oil and grease; compressed air, clean water hose reels, and welding stations. All these systems are centralized. The Truck Shop is the main consumer of oil stored in the oil tanks at the lube station.

### **18.5.3 Wash Bay**

The Wash Bay is located next to the Truck Shop to wash equipment prior to maintenance. The Wash Bay consists of a concrete pad of about 21.0 m by 18.0 m as shown to accommodate 6.28 m wide Caterpillar 785C Trucks. The system is designed for:

- Wash frequency is four (4) trucks per day based on eight (8) hour operation;
- Manual wash by one operator.

The system includes a water treatment system which allows for recycling of wastewater. This system provides:

- Easy capture and accessibility of sludge for handling and disposal;
- Reduction of SS and pH control to ensure proper functioning of the system and to avoid clogging and wear, to ensure durability of mechanical components.

#### 18.5.4 Water Services

An existing water line passes in the area and connects the mill area to the main pit. Its location, diameter and type need to be clarified in the detailed engineering phase. The existing fire protection system is extended to the Truck Shop. The building is protected by two fire hydrants.

The existing potable water system is extended to the Truck Shop. Sewage for the dry is collected and directed to the existing sewer system. The fresh water network is extended to the Wash Bay area (make-up water) and to the Truck Shop for general floor maintenance of the garage bays. Both the Wash Bay and Truck Shop are equipped with oil/water separators of the following capacity:

- Wash Bay: 150 GPM (570 LPM);
- Truck Shop: 100 USG (380 LPM).

The effluent is discharged in the environment after passing through the oil/water separators.

#### 18.5.5 Electrical Services

The Truck Shop electrical feed is 400/230 V, 50 Hz, 3 Phases, 4 Wires + Ground. Estimated electrical loads are presented in Table 18.2.

**Table 18.2: Truck Shop Estimated Electrical Loads**

Description	Estimated Electrical Load (kW)
Base building	81.2
Building services	37.3
Hot water	7.5
Workshop services	64.0
Warehouse	10.0
Exterior lighting	12.0
HVAC	55.4
Wash Bay pumps and blowers	37.0
<b>Total</b>	<b>304.4 kW</b>

The Truck Shop is connected to the electrical grid.

**18.5.6 Roadway, Access and Fencing**

The Truck Shop and Wash Bay area are next to the existing mining road. Pads surfaces are covered with gravel to allow for circulation around the facilities.

**18.6 Lube Storage Facility (1540)****18.6.1 Location**

The lube and oil storage facility is located next to the truck-shop at the center of trucking operations. Given the mine's geographic location, oil and diesel and lubricants are delivered in bulk to the mine site. The planned support facilities for the mine include a lubricant and fuel storage facility to meet the daily needs (for diesel) and the maintenance needs (for oil and lubricants) of the fleet. The goal aims at limiting transport to the service point from the fuel depot.

Mine trucks fuel daily at the lube storage facility which requires daily deliveries of diesel. The lube, diesel and lubricant storage facility is also equipped to allow the operators to lubricate and add oil daily. No maintenance is performed in this area. Truck maintenance is performed in the adjacent truck-shop garage (1530).

**18.6.2 Description of Facility**

The facility consists of a concrete pad with containment wall to encompass 110% of the volume of the largest tank and a concrete apron with a fueling station.

The fuel and oil tanks are listed in Table 18.3.

**Table 18.3: List of Fuel Tanks**

Area	Activity Code	Equipment Type	Sequential Number	Description (tank complete with remote fill station)	Fluid Code according to piping material specification
1540	3410	605	001	25,000 l waste oil storage tank	WOL
1540	3410	605	003	25,000 l axle oil storage tank	
1540	3410	605	004, 005	25,000 l transmission oil storage tank (2 units)	PHD
1540	3410	605	002	10,000 l transmission Oil tanks	PHD
1540	3410	605	006	40,000 l Engine Oil tanks	PLO
1540	3410	605	007	60,000 l Diesel Tank	PFO

The supply includes:

- Each product tank is supplied with its own remote transfer station to allow for transfer of the product from the delivery truck to the tank. The remote transfer station includes pumps, valves, hoses from the transfer station to the tank, and meters for billing of received product;
- Two (2) complete diesel fueling stations are connected to the 60,000 liter diesel tank. These fueling stations serve mining trucks having 2,500 l fuel tanks. The fueling stations must have a flow rate between 400 and 800 lpm;
- The stations include temperature compensator, flow metering, feed lines, motorized valves, level gauges, high level sensors, overfill valves, and volume meters for the diesel tanks

### **18.6.3**            **Services**

The fire water network is extended to the lube storage facility. Fire Hydrants ensure fire protection. No other water services are planned.

Run-off captured in the containment area is collected in a sump which requires occasional emptying by pumping. Run-off on the diesel fuel delivery pad is drained and collected into an oil separator prior to discharge in the environment.

### **18.7**                **Power Line**

The new power line is a 225 kV single-circuit, three-phase transmission line between the existing Kayes substation and the Sadiola Mine. It also includes modifications to the Kayes substation and the construction of three (3) substations along the line route (Diamou, Sadiola and SEMOS).

The line connecting the Kayes substation and the Sadiola facilities will have a total length of 89 km and will be separated in two (2) sections:

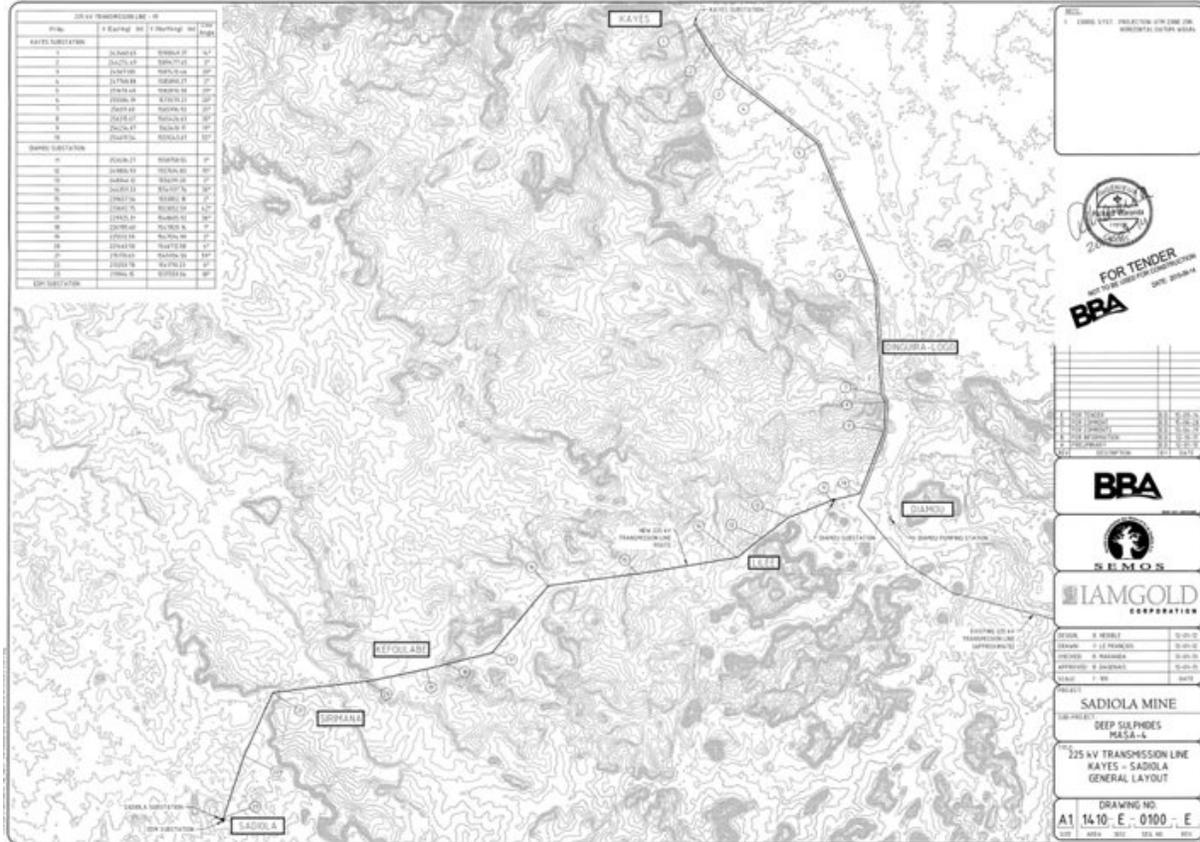
- From Kayes substation to Diamou substation (38 km);
- From Diamou substation to Sadiola EDM substation (51 km).

#### **18.7.1**            **Line Route**

Figure 18.2 below shows the routing of the line. The starting point of the first line section will be the Kayes substation. The line will run alongside the existing line going toward the Manantali dam over a length of 36 km. It will then branch off to the West for 2 km to reach the Diamou substation.

The second section will start at the future Diamou substation and will then head southwest to reach the Sadiola mining site. The line route will remain at a limited distance from the road going to the mine avoiding the few communities located along the road as well as rock formations in this sector. This line section will be connected to the future Sadiola EDM 225 kV substation.

Figure 18.2: Electrical Power Line



Line supports will be galvanized steel lattice towers, between 35 and 50 meters high. The average distance between each tower will be 430 meters. The 225 kV transmission line is a single circuit line with an optical ground wire.

The line right-of-way will be 50 meters wide. A road will be built inside the right-of-way to connect the construction sites, as required

**18.7.2 Technical Description – Substations**

Connection of the new 225 kV transmission line to the national network and the SEMOS Sadiola Mine will be performed by completing work at the following substations:

- Existing Kayes substation (Sector 1405) - Modifications to the existing substation by dismantling the old 90 kV line bay and replacing it by a new 225 kV line bay;
- New 225-30 kV substation at Diamou (Sector 1412) - This substation will then be transferred to EDM;
- New 225 kV substation at Sadiola (Sector 1411) - This substation will then be transferred to EDM;

- New 225 kV-11 kV substation at the SEMOS Sadiola Mine (Sector 1415).

#### 18.7.2.1 Modifications to the Existing 225-90 kV Kayes Substation (Sector 1405)

The modifications to the Kayes substation consist of:

- Removing the electrical apparatus from the bay of the 90 kV line previously planned towards Sadiola;
- Adding a 225 kV line bay for the new line planned towards Diamou.

The protection and control equipment will be installed in the existing control building.

#### 18.7.2.2 New 225 -30 kV Diamou Substation (Sector 1412)

The construction of the 225-30 kV Diamou substation will be carried out in two separate phases. The first phase to be performed within the current project framework will consist only in installing 225 kV incoming and outgoing line bays, a 225 kV busbar, a 225-30 kV power transformer and a 30 kV GIS inside the Diamou substation. This new substation will include:

- One (1) 225 kV busbar with three (3) single-circuit steel gantries;
- Two (2) 225 kV line bays;
- One (1) line bay towards the 225 kV air-core reactors associated with the outgoing line to Kayes;
- A line bay associated with a 225-30 kV.

The protection and control equipment will be installed in a new control building.

#### 18.7.2.3 New 225 kV Sadiola EDM Substation (Sector 1411)

The construction of this substation will be carried out in two separate phases. The first phase to be performed within the current project framework will consist only in installing a 225 kV incoming line bay and two 225 kV outgoing line bays as well as a 225 kV busbar, inside the EDM Sadiola substation. This new substation will include:

- One (1) 225 kV busbar with two (2) single-circuit steel gantries;
- One 225 kV line bay coming from Diamou;
- One (1) line bay towards the 225 kV air-core reactors associated with the outgoing line to Diamou;
- Two (2) 225 kV line bays towards SEMOS Sadiola;

- Control building.

#### 18.7.2.4 Construction of a New 225-11 kV Substation at SEMOS Sadiola (Sector 1415)

Finally, a new 225 kV-11 kV substation will be installed on the SEMOS Sadiola Mine property. This new substation includes:

- Two (2) 225 kV line bays from the EDM Sadiola substation;
- One (1) oil separator tank for the two (2) power transformers;
- One (1) steel lightning tower.

The protection and control equipment will be installed in the mine electrical room 1421. The basic characteristics of the power transformers are shown in Table 18.4:

**Table 18.4: Transformers Characteristics**

	<b>Diamou</b>	<b>Sadiola</b>
Type of equipment	Three-phase	Three-phase
Rated capacity	20 MVA	42/56/70 MVA
Cooling	ONAN	ONAN/ONAF/ONAF
Connection	YNyn0d1	YNyn0d1
Voltage	225 – 30 kV ±15% of 225 kV OLTC	225 – 11 kV ±15% of 225 kV OLTC
Impedance	7.7%	7.7 %

## 18.8 Tailings, Water & Waste Management

### 18.8.1 General

The expansion of the Tailings Storage Facility (TSF) has been designed to accommodate the additional 75 M t of tailings planned in the Sadiola Sulphides project. The tailings from the “Hard Sulphides” zone milling process will be deposited over a period of approximately 10 years at an average production rate of 7.2 M tpa.

Studies preceding 2009 have considered the “Hard Sulphides” tailings stream deposition in the current TSF, either by raising it using the current footprint or expanding it to the south. Several major issues were raised during these studies, i.e. the rate of rise, water pond control and overall stability of the TSF. Since 2009, studies started considering the deposition of the “Hard Sulphides” tailings stream in a separate facility to the north of the current TSF and referred to as the New TSF. The New TSF will be located directly downstream of the current TSF in a valley about 1.5 km of the mill. It will be abuts to the

dam (referred to as the Wall) that constitutes the northern limit of the current TSF and confines the “Oxides/Soft Sulphides” tailings.

The tailings management strategy put forward during the 2010 Sulphides project feasibility study (Golder, 2010) considered thickened tailings deposition of the “Hard Sulphides” stream in the New TSF area confined by a perimeter dyke with a low permeability core, while the current TSF would continue to receive “Oxides/Soft Sulphides” tailings for a short period of time at the beginning of the New TSF operation. Hence, water basins would have to be built to replace the Return Water Dam (RWD) basin used by the current TSF for water management. The New TSF operation would also rely on the existence of a low permeability stratigraphic unit under the study area to limit seepage rate to the underlying aquifers from the New TSF and water basins footprint.

Since the 2010 Feasibility Study, evaluations have been carried out bringing modifications to the tailings management strategy, namely the storage capacity increase from 69 M t to 75 M t, the incorporation of a geosynthetic liner in the design, the deferment of the construction of the thickening plant in time and the transfer of deposition operations from the current TSF to the New TSF upon its commissioning. These modifications have been integrated in the New TSF design as part of the Sadiola Sulphide Project Feasibility Study 2015 (December, 2015).

Tailings are to be deposited from the dykes as slurry at 50% solids at the start of the New TSF operation. Towards the end of the operation, tailings will be thickened to a targeted 68% solids content and deposition will take place from service roads built directly on the tailings surface inside the New TSF area. Changing from slurry to thickened tailings will increase the storage capacity of the New TSF and shape its deposition surface to promote surface runoff and prevent water accumulation in the TSF.

### **18.8.2 Field Investigation Summary**

The proposed New TSF location is in the valley downstream of the current TSF. This area consists of a large valley approximately 30 m deep and opening to the North. The valley is encased between two ridges on the East and the West sides. In the valley, the terrain is generally flat with some undulating surfaces and rocky outcrops.

The proposed site was the subject of field investigations in 2009, 2010 and 2013 to assess the following elements in the study area:

- Soil stratigraphy;
- Geotechnical parameters for the dyke foundations;
- Hydrogeological parameters of targeted stratigraphic horizons;
- Groundwater level and water quality; and

- Borrow material availability and properties at the main pit and waste piles.

### 18.8.3 New TSF Area

A relatively consistent stratigraphy was observed in the boreholes at the New TSF, although there is a great variability in the horizon thicknesses observed along the dyke alignments. Typically, the soil stratigraphy is as follows in descending order:

- Organic soil (referred to as topsoil in the logs): A layer of dark brown organic and silty or clayey material with some sand was locally found at ground surface. This layer is generally non cohesive and loose to compact;
- Laterite and alluvium: layer composed of sand and gravel within a clayey or silty matrix, considered generally non cohesive and compact to very dense. The unit thickness generally ranged between 1.0 and 4.0 m; however, this layer was found to be greater than 6 m thick at some locations (maximum thickness of 11 m encountered during investigation), and absent at other locations. Locally, the laterite layer can form a thick and very dense cuirass at the ground surface and present a very porous connected honeycomb structure;
- Saprolite: layer composed mainly of silt or silty clay with minor amounts of sand and gravel, considered generally as cohesive and is stiff to hard. The top of the saprolite was generally encountered at depths ranging between 1 m to 4 m below ground surface. This unit thickness varied between 0 m and 29 m and typically ranged between 2 m and 10 m in thickness;
- Saprock: layer of moderately- to highly-weathered bedrock encountered underneath the saprolite unit with a thickness generally ranging from about 5 m to 30 m. The soil portion of the saprock unit is typically composed of sand and gravel within a silty clay to clayey silt matrix;
- Fresh bedrock: typically meta-sediments, diorite intrusive and marble encountered at depths ranging between 4.5 m and 53 m below ground surface.

The foundation preparation for the dykes will require the removal of the highly weathered surface layers and organics. Locally, these layers appear to be quite thick particularly the weathered surfaces at the topographical high points. These layers will have to be removed as they do not constitute a suitable foundation for dyke construction. Thus, an increase in earthworks and material placement should be anticipated at these locations. Locally, subgrade preparation for liner placement will also require more efforts to provide a level surface due to the presence of highly weathered surface soil and rock layers.

Standard penetration testing (SPT) values obtained were generally above 20 for the laterite and the saprolite layers. SPT testing showed good results, as pressure meter test profiles did. The interpretation of the net pressure limit measured for the laterite and the saprock units gave friction angles of 32° and 38° respectively. As for the saprolite unit, interpretation of the pressuremeter test results gave undrained

shear strength ranging from 95 kPa to 270 kPa, with a low value of 68 kPa encountered at the PM10-10 location at 3 m deep. In general, results from pressuremeter testing indicated that undrained shear strength was rapidly increasing with depth. If stability analyses in effective stress conditions are required, a friction angle of about 28° could conservatively be used. Meeting the overall stability of the dyke structure should not be of concern.

Permeability tests were conducted during the field investigation in piezometers installed in both the saprolite and the saprock layers. Generally, estimated hydraulic conductivity is higher in the piezometers installed closer or within the transition zone (saprock). Tests performed in the saprock yielded values varying between  $7 \times 10^{-6}$  m/s and  $4 \times 10^{-8}$  m/s for an average hydraulic conductivity of  $3 \times 10^{-7}$  m/s. The hydraulic conductivity value was also high ( $2 \times 10^{-6}$  m/s) in the only laterite installed piezometer (RC-12-44B). Lower hydraulic conductivity values were found in the saprolite, varying between  $4 \times 10^{-8}$  m/s and  $2 \times 10^{-9}$  m/s with a geometric mean estimate of  $4 \times 10^{-8}$  m/s.

Water level measurements were taken in the RC open holes and in the monitoring wells installed by Golder in April 2010. These measurements, along with the ones from SEMOS monitoring well network located in the study area and its vicinity as well as the pit area, taken in April and May 2010, were used to generate a map of the piezometric surface. The general groundwater flow direction in the study area is toward the north. The groundwater table is located about 15 to 20 m below the surface in the saprock layer.

#### 18.8.3.1 Borrow Sources

The primary sources for borrow materials that are considered for the project are:

- The pushback area of the Sadiola pit, mainly the western push back of the pit wall for clayey fill material; and
- The existing waste material piles for both clayey fill and granular materials.

For the pushback area, grab samples were collected from the upper part of the main pit walls. Exploration boreholes transecting the pushback area of the Sadiola pit were also used to select core samples representing potential construction material generated through mining the pushback area of the existing open pit. As for the waste material piles, the potential borrow sources investigated for fine grained material were SPF1 near FE3 pit and SP1F northwest of Sadiola pit. The potential borrow sources investigated for rockfill material were SP1S, SP1W and SP1N near Sadiola pit. Samples were tested for both their geotechnical and their geochemical properties.

Test results indicated that both waste pile sources for fine grained material (SPF1 and SP1F) are considered suitable material from a geotechnical point of view for dyke construction. Granular material

from the SP1W and SP1N waste piles are also considered suitable from a geotechnical point of view for dyke construction. Material requiring specific sizes could be obtained by crushing and/or sieving the rock borrowed in the hard waste dumps. From a geochemical standpoint, static testing indicates that material in the sampled waste piles has generally very low sulphide content which yields no potential for ARD and low leaching potential. Borrow source material availability should not be of concern for construction needs.

Samples from the pushback report no bulk potential to generate ARD and low sulphur content. However, material from the pushback has potential for arsenic leaching. All samples reported arsenic leachate concentrations within the same order of magnitude of the IFC effluent guideline (0.1 mg/L), except for one sample which reported a concentration (1.2 mg/L) that is one order of magnitude above the IFC effluent guideline. The pushback material could be used with restrictions as construction material such as limiting its use in areas designed for water containment or areas directly opening to the surrounding environment because of its potential for arsenic leaching. Specific restrictions will be defined during the detailed engineering phase. These restrictions could be reviewed if additional test results or kinetic testing indicate an overall low potential for leaching.

#### **18.8.4 Tailings Characterisation**

##### **18.8.4.1 Geotechnical**

Tailings were generated from a batch metallurgical circuit using 50 kg of sulphide ore provided by SEMOS. The testing program completed included characterization of geotechnical properties for various aspects of the TSF design. Tailings were characterized for particle size, specific gravity, soil water characteristic curve (SWCC), consolidation with hydraulic conductivity and for shear strength by consolidated undrained triaxial test. Laboratory test results are presented in the New TSF Geotechnical and Hydrogeological Investigation report (Golder, 2014c).

The tested tailings sample is a non-cohesive well-graded fine-grained material with 98% of the tailings mass passing 80 µm and a specific gravity of 2.80. The consolidation test gave a compression index (Cc) of 0.07 and an average coefficient of consolidation (cv) of 0.09 cm<sup>2</sup>/s. Values of hydraulic conductivity were measured near 1 x 10<sup>-7</sup> m/s at low confining stress (0.9 kPa) and near 4 x 10<sup>-8</sup> m/s at higher confining stress (1,000 kPa). Evaluation of the SWCC indicates an air entry value (pressure at which a material begins to desaturate) of 9 kPa. Evaluation of the tailings sample shear strength gave a cohesion C of 0 kPa and a friction angle of 34°.

#### 18.8.4.2 Geochemical

Geochemical testing was also carried out on the tailings sample generated in metallurgical testing to support the design of the New TSF. The objectives of the testing program were the characterization of the process water to evaluate seepage water quality from the new TSF and the characterization of chemical release from the solid phase of tailings to evaluate TSF surface runoff water quality. The testing program included chemical composition, acid generation potential, leaching potential and aging tests for process water chemistry evaluation.

Results from the testing program indicate that tailings solids demonstrate no potential to generate ARD. However, tailings report a short-term flush of total cyanide, dissolved arsenic and dissolved iron at concentrations above the IFC effluent guidelines. Free and WAD cyanide concentrations are below IFC effluent guidelines. Therefore, tailings could impart a chemical load to the TSF surface runoff water quality.

Process water contains elevated concentrations of arsenic, copper, iron, mercury, and nickel, and aging tests indicate that cyanide concentrations are high and do not decrease with time. The sustained elevated cyanide concentrations are likely a result of the volatilization rates of free cyanide being equal to or slower than the continual recharge of cyanide from pore water in the settled tailings solids into the supernatant water column. Sustained cyanide concentrations in tailings pore water and in pond water should be expected throughout the operation of the New TSF.

#### 18.8.5 New TSF Design

##### 18.8.5.1 New TSF Development Phase

The development of the New TSF is divided in four main phases described below.

- Phase 1 (Year 1 and Year 2): Construction of part of Perimeter dyke E1 and Intermediate dyke E2 is planned in the southern area of the New TSF (at the toe of the existing TSF) to accommodate the first two (2) years of operation. Tailings will be deposited as slurry along the dyke perimeter. The water pond will form gradually in the western sector of the Phase 1 area. During this period, the RWD basin will be dried to allow liner placement and dyke construction in that sector. It is assumed that water from the RWD basin will be pumped back to the mill via the New TSF basin to be used as reclaim water. Therefore, the RWD pond volume has been taken into account in the water management plan of the New TSF. For this phase, an event corresponding approximately to the 1:100 years peak runoff event will be contained by the Intermediate dyke E2. An event larger than the 1:100 years peak runoff and up to 1:1,000 years peak runoff, if it ever occurs during this short period, will be contained by both the Intermediate dyke E2 and the RWD.

- Phase 2 (Year 3 to Year 5): First stage of construction of the full length of Perimeter dyke E1 is planned to accommodate three years of operation. Construction of Internal Dyke E3 to its first stage elevation will also be carried out at the same time. Tailings will be deposited as slurry along the dyke perimeter and the Intermediate dyke. Pounded water in Phase 1 area will be transferred to the Internal Basin by pumping. Water transfer may also occur by gravity through dyke E2 spillway. Pumping infrastructure will be transferred afterwards to the Internal Basin. It is understood that the current TSF will undergo reclamation during this period. It is assumed that water from the current TSF pond will be pumped back to the mill via the Internal Basin to be used as reclaim water. Therefore, the current TSF pond volume has been taken into account in the water management plan of the New TSF.
- Phase 3 (Year 6 to Year 8): Dykes E1 and E3 will be built to their final elevation during a second stage of dyke construction to accommodate slurry tailings deposition until Year 8 of operation. Tailings will be deposited as slurry from the perimeter dyke to promote drainage towards the Internal Basin. The water management plan for this phase does not account anymore for the current TSF runoff as it is assumed that the current TSF area is reclaimed and the runoff, if any, is treated, if required, and then discharged to the environment.
- Phase 4 (Year 9 to Year 11): All dykes are already at their final elevations. The thickening plant will be commissioned for thickened tailings discharge from inside the TSF area. Tailings will be discharged from service roads built in the southern sector of the New TSF area, over the tailings surface, to promote drainage towards the Internal Basin. Progressive reclamation of the southern area, where the tailings surface will have reached its final configuration, is planned for the last two (2) years of operation (Year 10 and Year 11) to reduce the volume of surface runoff reporting to the Internal Basin.

#### 18.8.5.2 Monitoring

Instruments will be installed to monitor dyke and site performance during construction and throughout the operating life of the New TSF. Performance will be monitored in terms of:

- Pore water pressure evolution in fill material and foundation;
- Dyke settlement;
- Horizontal deformation and displacement;
- Seepage water quantity and quality.

Monitoring instruments will be incorporated at strategic locations in or around the facilities. Their number and location recommended for the New TSF will be defined with the detailed plans at a later stage of the project.

### 18.8.6 Design Analyses

#### 18.8.6.1 Tailings Deposition Plan

A deposition plan was developed to evaluate the storage capacity required at each phase of construction. Storage capacity definition took also into account results from the water balance described below. The following parameters were used to develop the deposition plan:

- Slurry deposition at 50% solids;
- Tailings dry density assumed at 1.4 t/m<sup>3</sup> ;
- Tailings beach slope above water of 1.5%;
- Tailings beach slope under water of 3%;
- Thickened tailings deposition at 68% solids;
- Tailings dry density assumed at 1.5 t/m<sup>3</sup> ;
- Average tailings beach slope of 2%.

Tailings will be discharged through multiple discharge points using spigots. Deposition planning should aim at locating the pond water in the northwest sector for Phase 1. During Phases 2 and 3, tailings deposition should aim at promoting drainage towards the Internal Basin to minimize water ponding over the tailings surface and circumscribe water in a localized area delimited by the Internal Dyke. The same objectives will be maintained during Phase 4 when deposition of thickened tailings will essentially take place from service roads located within the TSF area. These service roads will have to be raised in stages, as the tailings are deposited in the New TSF. The main deposition points will be located along northeast - southwest axes, slightly dipping to the northeast to create a series of cones that will progressively overlap to direct surface runoff towards the Internal Basin.

A yearly deposition plan was simulated based on the assumed parameters and management approach described here.

#### 18.8.6.2 Water Balance

A numerical water balance model was developed using the GoldSim software (GoldSim Technology Group, 2015) in order to:

- estimate the water storage capacity of the New TSF that will be required to limit the risk of spill to the environment over 10.4 years of operation to 1; and
- estimate the volumes of water available for mine processes.

Provided local rainfall data were utilized to generate a 1,000-year long stochastic rainfall time series, which is representative for the site. This time series formed the basis for the probabilistic approach of the analysis. It is important to note that the short length of the historical records for daily rainfall leads to uncertainty in the estimation of extreme events.

The water balance model incorporated the relevant hydrologic components: bleed water released by consolidating tailings, the rainfall runoff production on the tailings surface, direct precipitation and evaporation at the water basin, and pumping from the TSF towards the process water reservoir. Pump capacity from New TSF varies according to tailings properties (slurry or thickened), based on the Iamgold site water balance model (IAMGOLD, 2015).

Other than during the reclamation works for the current TSF (Year 3 and Year 4), no inflow from current TSF towards the reservoir of the New TSF is considered in the water balance model. It's assumed that the emergency spillway of Current TSF directs all overflow towards the environment.

The critical periods, in terms of required storage capacity, are at the end of Phase 1 and at the end of Phase 4 of the New TSF's lifetime when the facility is almost filled by tailings. It is assumed that progressive reclamation on approximately 0.8 km<sup>2</sup> of the tailings surface area will take place in the last two and half years of operation (Phase 4). Runoff from reclaimed surfaces will be diverted directly to the environment as clean water or previously to an effluent treatment station if required.

#### 18.8.6.3 Stability Analysis

Stability analyses for the following conditions were performed to confirm the stability of the New TSF dikes for various loading conditions applicable to the project such as end of construction conditions, long-term conditions, water pond level during an extreme event, seismic and post-seismic conditions.

The position of the piezometric surface in the natural ground was based on the piezometric map generated from the groundwater level readings on site, and is well below the surface near the contact between the saprock and the bedrock. A piezometric surface has been included in the model to simulate the perched water table created by the presence of the geosynthetic liner underneath the tailings hindering drainage. The water table was fixed according to the expected median annual maximum pond water level, and the spillway sill for simulating the extreme event conditions. Stability analyses in pseudo-static conditions were done using a bedrock acceleration coefficient of 0.04 g corresponding to the PGA

(peak ground acceleration) with a probability of exceedance of 10% in 50 years (1 in 476 years). The post-liquefaction analyses were simulated by reducing the friction angle of the tailings to an assumed residual value of 5°. Material properties were based on site investigation and laboratory results.

The results of the stability analyses indicate that calculated factors of safety meet the minimum safety factors defined in the Canadian Dam Association (CDA) specifications for tailings dam (CDA, 2014). Having the geosynthetic liner laid underneath both the Intermediate (E2) and Internal (E3) dikes requires the addition of stability berms. Both upstream and downstream toe berms are needed to ensure the stability of Intermediate dike E2. The berms could be downsized if the material used to build the dike is changed to rockfill. As for the Internal Dike E3, a small upstream toe dike is planned and its downstream slope has been lowered to 3 H:1V to ensure long term stability.

#### 18.8.6.4 Quantity Estimate

Table 18.5 presents a summary of the quantities of material required per construction phase. The quantity estimate is based on the New TSF development phases presented in drawing 1820-G-4205 and the typical cross-sections presented in drawings 1820-G-4206 to 4209. Phase 4 consists mainly of the construction and commissioning of thickening plant and service roads for tailings deposition. Quantities associated to these items are not presented in Table 18.5. Material grades are given in Table 18.5 as indicators for cost estimate purposes. Specifications for these materials will be defined at the detailed engineering design phase of the project.

The estimate is based on a feasibility level design. The quantities presented are subject to changes following the completion of the detailed design that will be carried out at a later stage. The quantity estimate presented here does not cover access road to the site, service roads for deposition inside the TSF area, tailings thickening plant and tailings delivery system, or other facility that may be needed for the construction, the operation or closure of the New TSF. Structures and decommissioning activities associated to closure of New TSF and the current TSF are not presented in this estimate.

**Table 18.5: Quantity Estimate per Construction Phase**

Item	Quantity				Unit
	Phase 1	Phase 2	Phase 3	Total	
Clearing of trees and vegetation	84	203	0	287	ha
Topsoil stripping (layer thickness of 0.2 m)	196,000	414,200	0	610,200	m <sup>3</sup>
Subgrade preparation (base)	840,000	2,031,000	0	2,871,000	m <sup>2</sup>
Non-woven protection geotextile installation (400 g/m <sup>2</sup> )	1,208,000	2,380,500	51,000	3,639,500	m <sup>2</sup>

Item	Quantity				Unit
	Phase 1	Phase 2	Phase 3	Total	
Textured geomembrane liner installation (white, 1.5 mm HDPE)	987,000	2,260,000	51,000	3,298,000	m <sup>2</sup>
Zone 2 fill placement and compaction (saprock)	1,102,000	2,401,500	1,465,000	4,968,500	m <sup>3</sup>
Zone 3A crushed rock (0-150 mm) placement - erosion protection layer	6,500	76,500	82,000	165,000	m <sup>3</sup>
Zone 3B crushed rock (0-50 mm) placement - liner protection layer	221,100	169,000	33,500	423,600	m <sup>3</sup>
Zone 5 clean rockfill (20-300 mm) placement	6,500	2,000	0	8,500	m <sup>3</sup>
Zone 6 Run-of-mine rockfill (0-900 mm) placement and compaction	315,000	1,057,000	172,000	1,544,000	m <sup>3</sup>
Zone 7 fill placement and compaction - Laterite road capping and safety berm	7,500	0	20,000	27,500	m <sup>3</sup>
Zone 8 Run-of-mine rockfill (0-300 mm) placement and compaction	353,000	125,000	23,500	501,500	m <sup>3</sup>
Subgrade Preparation (slope)	82,000	113,000	0	195,000	m <sup>2</sup>
Overburden excavation	24,200	6,100	0	30,300	m <sup>3</sup>
Anchor trench excavation and backfilling	1,250	2,000	900	4,150	m <sup>3</sup>
Non-woven separation geotextile installation (250 g/m <sup>2</sup> )	28,500	44,500	5,200	78,200	m <sup>2</sup>
Excavation	2,000	600	600	3,200	m <sup>3</sup>

### 18.8.7 New TSF Reclamation

The closure concept is a tributary of the requirements set for site closure, construction material characteristics and tailings geochemical characteristics. Currently, several key factors that would allow the development of a detailed closure plan are subject to uncertainty and will not be determined before a few years of operation. Therefore, it is recommended to develop and update this plan throughout the life of the New TSF operation. However, general guidelines can be given as to the main elements.

Closure of the tailings area will probably include placement of a cover and possibly revegetation to prevent, at the least, surface erosion of the tailings. Thickened tailings deposition inside the TSF area will help shape the tailings deposition surface to promote surface runoff and prevent water ponding in the TSF. The addition of a cover made of combined layers of granular and fined grained material would further promote runoff, prevent tailings salts reaching the surface and diminish water infiltration. The surface area of the tailings at their final configuration, as currently modeled, represent about 226 ha. If

surface runoff meets water quality criteria, it could be discharged through a series of spillways built into the Perimeter dike at regular intervals.

In the Years 10 and 11, in areas where the tailings surface has reached its final configuration, the external slopes should undergo progressive reclamation to reduce the volume of surface runoff reporting to the Internal Basin. The runoff from reclaimed areas could be discharged to a collection system downstream of the New TSF via multiple spillways built into the Perimeter dike E1. Water quality would be monitored prior to discharge to the environment. Initial estimates indicate that early reclamation of about 0.8 km<sup>2</sup> of the New TSF area (30% of the total area) would be enough to limit the required storage capacity.

The Internal basin could either be filled with tailings up to a level where water could flow freely to the environment or it could be breached. In the first case, the emergency spillway in the north east would probably have to be modified to serve as a closure spillway. In both cases, the Internal Basin will be emptied. Surface water quality modeling results suggest that upon closure, the water contained in the Internal could require treatment prior to release to the environment. Water quality in the basin should be monitored during the operation to evaluate and prepare for the need for possible treatment at closure.

#### **18.8.8            Current TSF**

Seepage from the current TSF is being collected through a series of ditches and sumps along its western and eastern sides and pumped back to the existing RWD basin. This system will be maintained throughout the New TSF operation, by pumping the seepage to the pond area in Phase 1 first, and then to the internal basin, until:

- either water quality monitoring indicates that, following reclamation of the current TSF, the seepage meets water quality criteria; or
- an independent water treatment system (either passive or active) is set in place.

Details on quality and quantity of the seepage are not currently available. It is assumed that the overall volume collected from this seepage is relatively small compared to the bleed and water volume from the New TSF and can safely be managed within the New TSF operation.

The current TSF will undergo reclamation during the operation of the New TSF. The current closure plan, developed by AGA, is to maintain the current TSF topography which has a central depression where runoff may accumulate to later be evaporated or infiltrated through the tailings, possibly maintaining a pond throughout the year (Schlumberger, 2009). An overspill structure is planned to convey safely water out of the tailings pond in case of an extreme event. At the moment, the overflow water would be directed in the New TSF area. It is recommended that the location of this structure and water channeling be

reviewed to convey water safely to a natural drainage rather than the New TSF area. This will contribute to a better risk management control.

## 19. MARKET STUDIES AND CONTRACTS

### 19.1 Market Price/Long-Term Price Outlook

Gold is the principal commodity at Sadiola mine and is freely traded at prices that are widely known, so that prospects for sale of any production are virtually assured. For this study, IAMGOLD has used gold price and exchange rates aligned with the IAMGOLD corporate strategy as listed in the Table 19.1.

All gold produced by Sadiola is in the form of doré bars, which is then shipped to a refiner who refines the doré into bullion. There is a standard off-take agreement in place with Rand Refineries, since the inception of the Sadiola Mine.

All contractual fees have been accounted for in G&A direct operating cash costs.

The bullion is then sold directly on the open market to gold trading institutions at prevailing market prices.

**Table 19.1: Gold Price Assumptions**

	Year 1	Year 2	Year 3	Year 4	Year 5+
Gold price (US\$/oz.)	1,150	1,225	1,250	1,250	1,275
Exchange rate (XFO/US\$)	595	570	546	524	524

### 19.2 Material Contracts

The Sadiola mine (SEMOS) has material contracts which are directly linked to the operations. Some contracts are managed by either SEMOS, AngloGold Mali S.A. or AngloGold Ashanti Ltd. as stated in the Table 19.2. Contracts are normally negotiated by going on tenders. The contracts higher than US\$5 M per year are listed in Table 19.2. As the reader will notice most of these contract were negotiated on a long term basis.

Contracts that have been negotiated by SEMOS and AGA follow proper governance and have been competitively bid and awarded.

Table 19.2: SEMOS Material Contracts as at December 2014

Vendor	Contract Value (US\$ M)	Description of Service/Goods	Resp.	Contracting Party AGA or Sadiola	Long-Term Contract ("LT") or Once-off Orders ("OO")
Total Gapmos	± 45	Fuel and oil supply	GSC	AngloGold Mali S.A. acting on behalf of Sadiola S.A	LT
Moolman's	± 40	Mining	GSC	SEMOS	LT
LTA Moolman	± 15	Mining	GSC	SEMOS	LT
Outotec	± 5	Mill supply/repair	Reg.	AGA with SEMOS	LT and OO
Société de Forage de Trav.P	± 5		Mine	SEMOS	OO

Source: Sadiola Mine, 2015

Note: Reg. – regional; GCS – Global group; AngloGold Ashanti Ltd

## **20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT**

### **20.1 Introduction**

SEMOS is bound by the original prospecting and exploitation agreement and its subsequent legal modifications (April 15, 1990) between AGEM and the Mali Government. The mining license is valid for the original mineral commodities until April 15, 2020. The identity number of the current exploitation area is “DECRET No 00-080/PM-RM DU 06 MARS 2000” and is a modification of all previous exploitation areas. The surface area is defined by “DECRET No 00-063/PM-RM DU 25 FEV 2000”. The Malian legislation requires the compilation of an Environmental and Social Impact Assessment (ESIA) for the Sadiola Sulphide Project. The Direction *Nationale de l’Assainissement et du Contrôle des Pollutions et des Nuisances* (DNACPN) manages the process. Subsequently, terms of reference for the ESIA were submitted to DNACPN by SEMOS in March 2010. On May 13, 2010, approval was received from DNACPN to proceed with the ESIA.

The construction, operation and closure phases of the Sadiola Sulphides Project will be governed by the relevant environmental and social legislation of the Malian Government, the International Finance Corporation (IFC) Performance standards and Environmental, Health and Safety Guidelines, and relevant company procedures and performance standards. SEMOS will ensure that the legislative requirements as well as the commitments included in the ESIA environmental management plans are effectively implemented and responsibly executed to support a sustainable economy.

A comprehensive and detailed environmental and social impact assessment study (ESIA) was conducted in 2010 for a previous version of the SSP project (2010 ESIA). This ESIA was conducted by Digby Wells & Associates. Numerous environmental and social baseline studies, including geochemical characterization of the waste rock and the tailings were conducted for the 2010 ESIA by Digby Wells & Associates, Lorax Environmental and Environmental and Social Development Company (ESDCO) – SARL. This ESIA and some of the baseline studies have not been updated since 2010. As a result, most of the information presented in this Section are highlights from the 2010 ESIA and associated baseline studies.

Changes have occurred at the mine site since the 2010 ESIA. Some important changes include new pits, new waste rock piles and modification of the surface drainage pattern. These changes may have an influence on the environmental and social impacts as assessed in the 2010 ESIA.

Some of the SSP project components have changed significantly between the project considered in the 2010 ESIA and the current SSP project. Some of these elements are favourable to improving the impact on the environment and social elements while other may generate new impacts that were not identified in the 2010 ESIA. The most significant changes are the following:

- Modification of the surface water flow path of the stream located North of the main pit;
- Change in the location of the waste rock dumps from a North-East location to North-West and South location along with in-pit dumping;
- Installation of an impermeable liner underneath the new tailing storage facility (TSF);
- Decision to delay the construction and the operation of the tailing thickening plant (TTP) and to dispose of un-thickened tailings in the new TFS before the TTP is in operation;
- Change in the location of buildings and infrastructure such as the truck shop and the construction camp;
- Modification of a mining haul road;
- Minor changes in the layout of the plant, bunded areas and use of catchment areas rather than double lined pipes on the tailings line.

Most of the environmental and social impacts identified and assessed in the 2010 ESIA are still considered valid for the revised project. SEMOS committed to implementing numerous mitigation measures in the 2010 ESIA. This Section presents a summary of the 2010 ESIA impact assessment and of the main mitigation measures SEMOS committed to.

However, as they have not been properly assessed by the 2010 ESIA update process, the environmental and social impacts associated with the significant changes mentioned above are not discussed in detail in this Section. The level of effort required to update the environmental and social impact assessment based on changes will be influenced by the Malian authorities requirements and expectations, considering the fact that permits had already been issued in 2011 and 2012 for the SSP project (see below).

## 20.2 **Permitting Process**

Based on the 2010 ESIA, the *Direction Nationale de l'Assainissement et du Contrôle des Pollutions et des Nuisances* (DNACPN) of the Government of Mali issued various environmental and construction permits for the Sulphide plant and infrastructures as follows:

- Sulphide project environmental permit, issued on August 22, 2011;
- Sulphide project construction permit, issued on October 24, 2011;
- Powerline construction permit, issued on May 28, 2012.

As the project has not been implemented to date, the SSP construction permit expired October 23, 2012 and must be reissued prior to construction. The Sulphide project environmental permit expired on August 21, 2014.

Community consultation is an integral part of the project environmental permitting process. Some community consultation activities were conducted during the 2010 ESIA process. However, the final step of the consultation process was not conducted. This final step involves presenting the permit requirements, the mitigation measures and the environmental and social management plan to various stakeholders.

### **20.3 Environmental Baseline Studies**

Various environmental and social baseline studies were conducted for the 2010 ESIA. These baseline studies focused on the following environmental and social aspects: 1) Surface water; 2) Groundwater; Air and greenhouse gases emissions; 4) Noise and vibrations; 5) Fauna and flora; 6) Land use; 7) Archeology; 8) Public health; and 9) Tailings and waste rock geochemistry, acid rock drainage and metal leaching potential.

Due to the changes that have occurred at the mine site since the 2010 ESIA (new pits, new waste rock piles), potential changes which could have occurred in the surrounding communities in terms of land use and population growth / migration, and due to the changes in some SSP project components (change of location of the waste rock pile, modification of mine haul roads), some of these baseline studies may need to be updated and the associated project environmental and social impacts may need to be re-assessed to support the new environmental permit requests or permit renewals.

### **20.4 Environmental and Social Impact Assessment – 2010 ESIA**

As part of the 2010 ESIA, the potential environmental and social impacts associated with the construction, the operation and the closure phases were assessed for the 2010 version of the SSP project. The assessment prior to mitigation measures and with the mitigation measures in place have been documented and identified.

### **20.5 Acid rock Drainage and Metal Leaching Potential**

Previous acid rock drainage (ARD) and metal leaching (ML) studies, as well as water quality monitoring conducted on waste rock dumps, stockpiles, the existing TSF, the Sadiola Main Pit, etc., indicated that levels of arsenic and antimony could become elevated in surface and groundwater systems as a result of mining associated with the sulphide deposit. Test work conducted between 2010 and 2012 supports the results of previous studies that weathering has a strong effect on the ARD potential of rocks at Sadiola. Soft sulphides exhibit the greatest ARD potential. In contrast, the vast majority of hard sulphide would not be expected to generate ARD. The same holds true for tailings generated from soft sulphide and hard sulphide processing. The tailings generated by soft sulphide processing are considered potentially acid generating while those produced by hard sulphide processing would be non-acid generating.

The test work has also indicated that the waste rock and the tailings for both soft and hard sulphide have a moderate to high metal leaching potential. However, due to the expected low water flux rate in the waste rock dumps, it was concluded in the 2010 ESIA process that the waste rock dumps had an overall low metal leaching potential.

No new geochemical test work has been conducted for the 2015 SSP. However, no significant changes are expected with the 2015 SSP project and monitoring of surface water will be included in the environment management plan.

## **20.6 Environmental and Social Management Plan and Mitigation Measures**

The actual SEMOS mining activities are conducted and monitored according to an environmental management system (EMS) certified as compliant with ISO 14001:2004. This EMS does not account for the new infrastructures and activities associated with the SSP project.

No formal environmental and social management plan (ESMP) was submitted and approved by the Malian authorities for the SSP project during the 2010 ESIA process. No ESMP has been prepared yet for the current version of the SSP project.

A formal ESMP will have to be developed to the authorities along with the new environmental permit request. This ESMP will include most of the mitigation measures SEMOS committed to in the 2010 ESIA and the new mitigation measures which will be identified to control the impacts associated with the projects changes. The current project will impose further work to bridge previous commitments.

An important commitment that will require further work is the planned 70 m high waste rock dump North-West of the main pit. This could be considered by the community and the authorities as a failure to meet the commitment to have the Western rock dump (which would fill the gap between two existing waste rock dumps) with the same height as the current waste rock dumps (which are about 25 m high). This commitment was originally made to mitigate the impacts on air quality, on the visual, landscape and topography aspect as well as on noise and vibration. This is considered an important risk as this commitment was made following concerns expressed by the community during the 2010 ESIA process.

Another commitment that will require consultation is the postponement of the TTP by a few years. This commitment was made to mitigate the impact on surface water quantity / availability. The amount of water pumped from the Senegal River remains limited to the amount authorized by the Government.

The manner in the way process water management in the mill will be done varies from the 2010 SSP study but it is considered appropriate as all potential spills from the mill will be collected and discharged in the existing pollution pond (rather than being contained in the mill bunded area).

The present version of the SSP project will meet all the commitments made by SEMOS to protect the groundwater quality in the 2010 ESIA. Also, a significant new mitigation measure is considered in the current version of the SSP project with the construction of an impermeable liner underneath the new TFS. This liner will provide a very efficient way to protect the groundwater flowing under and around the new TFS.

## **20.7 Closure, Decommissioning and Reclamation**

Sadiola Gold Mine first prepared a mine closure plan in 2002 and updated it early in 2003. In addition to the mine generated closure plans Sadiola commissioned Schlumberger Water Services to compile a mine closure plan. With the assistance of external consultants such as Schlumberger Water Services, Sadiola Mine has been reviewing its mine closure plan over the years. The current closure plan which consists mainly of biophysical aspects was compiled by Schlumberger Water Services in 2009 and continues to be reviewed against LOM changes.

The existing mine site closure costs are updated by SEMOS every year but these cost estimates do not consider the new land disturbances and infrastructures that would be associated by the SSP project. Closure costs were reviewed and updated in the 2015 SSP project and are included in the financial section.

## **20.8 Key Permitting, Environmental, Social and Community Issues**

A number of key issues need to be addressed for the SSP project to proceed further. The key permitting, environmental, social and community issues are the following:

- Expired environmental permits:
- Discussions are required with the Malian authorities to determine whether these permits can be renewed or extended, or if new permit requests are required. If new permit requests are required, further discussions with the authorities are required to determine if the 2010 ESIA must be updated to assess the impacts of the changes made between the 2015 SSP and the 2010 SSP (for example changes in location of various project components);
- To date, no formal discussions have taken place yet with the Malian authorities to determine whether these permits can be extended, renewed or if wholly new permit requests are required;
- Change in the location of the North-East waste rock dump projected in 2010 to the projected North-West location:
- This change of location could be considered by the community and the authorities as an important change in a previous commitment not build waste rock dumps on the West side of the main pit

(other than to fill the gap between two existing dumps); the community opposed this type of waste rock location during the 2010 ESIA process;

- The surrounding communities have not been informed, consulted or engaged on this new location;
- Community pushback should be expected from this change of location and waste dump height and reputation may be impacted by previous commitments;
- Environmental and social impacts associated with building a new waste rock dump at this location have not been assessed. Some potential impacts that must be assessed include noise, dust, visual impact, air flow modification, loss of crop / grazing land;
- Incomplete environmental and social baseline studies, impact assessment and mitigation measures:
- As environmental and social baseline studies may need to be updated to account for the dynamic nature of the ecosystems and of the community development;
- The mitigation measures that are required to eliminate, control or manage environmental and social impacts may need to be updated or modified to account for the new baseline conditions, the project changes and the associated impacts;
- The updated baseline studies and impact assessment could be presented as an addendum to the 2010 ESIA;
- Incomplete community consultation process:
- As indicated, the community consultation process was not fully completed during the 2010 ESIA process;
- This final step of community consultation will likely be required by the Malian authorities. However, some of the earlier steps of the community consultation process will also be likely required to engage the community members on the most significant SSP changes, especially the change in the waste rock dump locations;
- Depending on the extent of the required community consultation process, delays in the permitting process could occur;
- Following the community consultation process, the SSP project may have to be modified to account for community / authorities concerns, which could again delay the SSP permitting and project development process;
- The community consultation process may not proceed until the environmental and social baseline studies are conducted and the impact assessment process has been completed;
- In light of recent downturn in the mining industry and closure of Yatela eminent, there might be considerable social and political pressure to have project approved;

- Absence of an environmental and social management plan:
- As indicated, a formal ESMP has never been submitted and approved by the Malian authorities during the 2010 ESIA process;
- A formal ESMP needs to be developed and included in the new environmental permit requests or the permit renewals.

## **21. CAPITAL AND OPERATING COSTS**

### **21.1 Capital Costs – Introduction**

The CAPEX estimate includes all Project direct and indirect costs for the implementation of the Project, inclusive of the execution phase. The CAPEX is deemed to cover the period starting from the Project approval date and finishing at start-up of operations, i.e. including pre-commissioning (also named mechanical completion), commissioning, as well as transfer to operations, but excluding costs pertaining to performance testing, start-up and ramping-up to full production as these costs are treated with the OPEX.

#### **21.1.1 Assumptions**

Following is a list of assumptions:

- Estimate is based on 6 days @ 10 hours workweek for construction contractor;
- Estimate is based on 6 weeks in / 2 weeks out rotation schedule for construction contractor;
- Estimate assumes that labor skills will range from medium to high, i.e. no unskilled nor low skill labor;
- Estimate assumes that the origin of skilled workers will be Senegal and Mali;
- Estimate assumes overburden disposal within plant limits, i.e. within a 5.0 km radius;
- Estimates for Mining OPEX in this Section were calculated based on \$0.84/l and 524 XOF/USD. In the total operating cost Section 21.7, fuel price and exchange rate were in accordance with the corporate 2016 assumptions published in September 9<sup>th</sup>, 2015. See Table 21.1;
- Estimates for milling hard rock material are obtained by taking account of cost of purchasing diesel at US\$.078/liter and using an exchange rate of 595 XOF/ USD Exchange.

Table 21.1: Study Assumptions

**Sadiola Assumptions****Metal Price**

	<u>Year 1</u>	<u>Year 2</u>	<u>Year 3</u>	<u>Year 4</u>	<u>Year 5</u>	<u>Year 6</u>	<u>Year 7</u>	<u>Year 8</u>	<u>Year 9</u>	<u>Year 10</u>	<u>Year 11</u>	<u>Year 12</u>	<u>Year 13</u>	<u>Year 14</u>	<u>Year 15</u>
Gold Price (US\$/oz)	\$1,150	\$1,225	\$1,250	\$1,250	\$1,275	\$1,275	\$1,275	\$1,275	\$1,275	\$1,275	\$1,275	\$1,275	\$1,275	\$1,275	\$1,275
Silver Price (US\$/oz)	\$16.00	\$18.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00

**Financial and Operating Assumptions**

Fuel (\$/bbl)	\$65	\$75	\$80	\$80	\$85	\$85	\$85	\$85	\$85	\$85	\$85	\$85	\$85	\$85	\$85
Diesel on Site (\$/l)	\$0.78	\$0.83	\$0.86	\$0.86	\$0.89	\$0.89	\$0.89	\$0.89	\$0.89	\$0.89	\$0.89	\$0.89	\$0.89	\$0.89	\$0.89
Exchange Rate															
CFA (USD-CFA)	XOF 595	XOF 570	XOF 546	XOF 524											
USD (EURO-USD)	\$1.10	\$1.15	\$1.20	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25	\$1.25
Power (XOF/Kwh)	N/A	XOF 70													

**21.1.2 Exclusions**

Following is a non-exhaustive list of exclusions:

- Escalation;
- Risk;
- Risk mitigation plan;
- Currency fluctuation;
- Hazardous waste;
- Financing charges;
- Delays caused by community relation, permitting issues, project financing, etc.;
- Carry-over work;
- All costs beyond commissioning, i.e. start-up, ramp up and operations;
- Sunk cost.

**21.2 Capital Cost Summary**

A summary of the capital expenditures is presented in Table 21.2. As discussed, capital expenditures were evaluated using a fixed exchange rate; in the financial evaluation, the numbers were factored according to the Iamgold assumptions.

**Table 21.2: Capital Expenditures Summary**

Capital Expenditures	USD	
	Fixed Exchange	Variable Exchange
03 – Mining	78,136,000	78,136,000
04 - Transmission Line	37,169,204	37,169,244
05 - Other Infrastructure	12,313,640	12,336,871
06 – Plant	70,943,760	71,619,565
08 - Tailings Facilities	33,175,034	33,570,973
09 - Construction Management	89,189,971	89,203,151
Fuel	1,511,980	1,560,441
Existing Commitments on purchased long-lead items	14,161,000	14,161,000
10 - Owner Costs	2,360,233	2,363,235
998 - Contingency	24,515,184	24,515,184
999 - Management Fees	13,974,681	13,974,681
<b>Grand Total</b>	<b>377,450,687</b>	<b>378,610,345</b>

The Capital cost estimate is based on information received from suppliers and local contractors. For major mechanical equipment, three suppliers were asked to provide a budget price. Following the reception of the budget quote documents, a technical and commercial recommendation was performed by the project team and the final equipment cost was added to the estimate. For construction, the project team met with several local contractors to present the project scope of work. Following these meetings, a formal request for budget quote was sent for each construction packages. This included the power line package which was assumed as a turnkey contract. For indirect cost, the estimation was based mainly on historical cost and in house data. These data are accurate and reliable considering the recent construction experience of IAMGOLD in western Africa.

### 21.3 Initial Capital Costs

#### 21.3.1 Mining

Major equipment items were already purchased in previous years. Table 21.3 shows on-hand equipment where dozers and water trucks are already on site. Table 21.4 summarizes initial mining capex to carry on the 7.2 Mtpa expansion.

**Table 21.3: On-hand Equipment**

Major Equipment	On-Hand
FlexiRoc D65	1
RH170/CAT 6040	2
CAT 993K	1
CAT 785C	17
CAT 785C BODY	17
CAT D9R	4
CAT 834H	1
CAT 16M	2
CAT 777 WT	2
FlexiRoc D65 RC	2
CAT 349D W. Bucket	1
CAT 349D - HAMMER	1
Support Equipment	On-Hand
Integrated Toolcarriers CAT IT14G	1
CAR Utility Compactor CB32	1
CS56	1

Table 21.4: Initial Capital for Mining

Initial Capital Schedule \$M	Total	1-12	13-24
		Year 1	Year 2
<b>Major Equipment</b>			
Pit Viper 235	8.51	-	8.51
FlexiRoc D65	-	-	-
RH170/CAT 6040	7.10	-	7.10
CAT 993	2.80	-	2.80
CAT 785C	-	-	-
CAT D9T	1.80	-	1.80
CAT 834K	-	-	-
CAT 16M	0.98	-	0.98
CAT 777 WT	-	-	-
D65 RC	-	-	-
CAT 349F	-	-	-
CAT 980	0.66	-	0.66
<b>Sub-Total</b>	<b>21.85</b>	-	<b>21.85</b>
<b>Support Equipment</b>			
10in Pump	0.34	-	0.34
Fuel Truck	0.66	-	0.66
Light Plant	0.10	-	0.10
Hilux Pickup	1.16	-	1.16
Pit Bus	0.13	-	0.13
Mech. Serv. Truck	0.58	-	0.58
Tire Truck	0.28	-	0.28
Crane 60t	0.49	-	0.49
Boom Truck	0.17	-	0.17
Forklift 4t	0.02	-	0.02
Welding Mach.	0.05	-	0.05
IT Carrier	0.13	-	0.13
CS56	-	-	-
<b>Sub-Total</b>	<b>4.09</b>	-	<b>4.09</b>
<b>Other</b>			
Pre-Strip		-	52.19
<b>Grand-Total</b>	<b>78.14</b>	-	<b>78.14</b>

### 21.3.2 Power Supply

The power line was considered as a Lump Sum Contract. A summary of Power Supply Expenditures is presented in Table 21.5.

**Table 21.5: Power Supply Capital Expenditures**

<b>Area</b>	<b>USD</b>
1401 - General – Power Supply	91,030
1405 - Kayes Substation	1,629,594
1410 - Transmission Line 225 kV	20,372,720
1411 - Sadiola EDM Substation	4,705,496
1412 - Diamou Substation	5,631,462
1415 - Sadiola SEMOS Substation	4,738,902
<b>Grand Total</b>	<b>37,169,204</b>

### 21.3.3 **Process and Infrastructures**

A summary of the Infrastructure capital expenditures is presented in Table 21.6.

**Table 21.6: Infrastructures Capital Expenditures**

<b>Capital Expenditures</b>	<b>4Q 2015 USD</b>
1250 - Exploration Office	546,359
1501 – General Infrastructure	744,669
1530 - Truck Shop & Warehouse	9,424,966
1540 - Lube & Fuel Storage	1,526,411
1550 - Potable Water	71,235
<b>Grand Total</b>	<b>12,313,640</b>

A summary of the Process capital expenditures is presented in Table 21.7.

**Table 21.7: Processing Capital Expenditures**

<b>Processing Capital Costs</b>	<b>USD</b>
<b>Existing Plant</b>	<b>2,874,788</b>
382 - Electrowinning & Smelting	98,718
391 - Residue Disposal	829,101
392 - Tailings Dam	14,080
550 - Reagent Handling & Storage	730,485
650 - Water Supply & Treatment	5,191
850 - Plant Generator Station	1,197,213
<b>New Plant</b>	<b>68,068,972</b>
1421 - Main Plant Electrical Room	6,139,929
1422 - Crushing Electrical Room	498,500
1423 - TTP Electrical Room	279,513
1425 - Compressor Electrical Room	288,068
1601 – General	2,812,651
1604 - Primary Crushing	10,996,420
1605 - Ore Handling & Conveying	4,461,595
1610 – Grinding	18,571,591
1612 - Pebble Crusher	1,355,299
1615 - Gravity Circuit	1,035,462
1620 - Pre-Leach Thickening	7,646,694
1625 - CIL Circuit	8,474,229
1635 - Reagents Handling	341,432
1640 - Mill Services (Compressors, Process Air)	2,330,544
1655 - Tailings Pumps and Pipeline	2,431,322
1670 - Oxygen Plant	405,723
<b>Grand Total</b>	<b>70,943,760</b>

#### 21.3.4 **Tailings and Water Management Facilities**

The following table summarizes the TSF initial capital cost which includes an HDPE liner. The TSF will be built by phases and it is expected to install the Thickened Tailings Plant later in the life of mine.

**Table 21.8: Tailings & Water Capital Expenditures**

<b>Area</b>	<b>4Q 2015 USD</b>
1426 - Booster Station Electrical Room	370,781
1805 - Tailings Booster Station	1,438,582
1815 - Reclaim Water	8,643,125
1820 - Tailings Storage Facility (TSF)	22,722,546
<b>Grand Total</b>	<b>33,175,034</b>

### 21.3.5 Indirect Costs and Contingencies

A capital expenditures summary for Indirect Costs and contingencies is presented in Table 21.9.

**Table 21.9: Construction Indirect Capitals**

<b>Indirect Costs and Contingency</b>	<b>USD</b>
<b>Construction Management</b>	<b>89,189,971</b>
1905 - Construction Facilities	4,206,804
1910 - Construction Equipment & Tools	12,619,756
1915 - Construction Equipment Maintenance	500,000
1920 - Construction Engineering	9,425,980
1925 - Construction Management	19,822,329
1930 - Construction Freight	20,515,932
1935 - Construction Room & Board	7,172,845
1940 - Construction Transportation	7,492,114
1960 - Construction HSS	2,376,000
1980 - Commissioning	2,178,902
1985 - Corporate Administration	2,879,309
<b>Contingencies</b>	<b>24,515,184</b>
998 - Contingency	24,515,184
<b>Grand Total</b>	<b>113,705,155</b>

### 21.4 Working Capital Costs

The working capital required for the SSP project with an owner mining strategy and considering a production increase is \$32 M of additional inventory.

**21.5 Sustaining Capital Costs****21.5.1 Introduction**

Sustaining capital is estimated to \$257 M, most of which come from the capitalized waste, the upgrade of the existing plant, the Thickened Tailings Plant, and the staging of the Tailings.

Sustaining capital is presented in Table 21.10.

**Table 21.10: Sustaining Capital Costs**

Year	1	2	3	4	5	6	7	8	9	10	11	12
Sustaining Capital (M\$)	4	4	137	40	9	7	14	8	26	5	3	-

**21.5.2 Reclamation and Closure Costs**

The reclamation and closure costs for SSP are estimated to be \$20.4 M. Closure costs would cover the following activities:

- Waste dumps reclamation;
- Tailing dams reclamation;
- Surface infrastructure;
- Social and community;
- Workforce retrenchment;
- Operational costs during demolition and rehabilitation;
- Plant decommissioning;
- Monitoring and post closure period security.

**21.6 Operating Costs****21.6.1 Introduction**

Operating cost for processing and general and administration are based on Sadiola's actual data. The current mining is conducted by a contractor. Thus, mining operating costs, for an owner mining set-up, were calculated from a zero base.

### 21.6.2 Power and Fuel

The local fuel price delivered on site was approximated from Sadiola's history. The derived equation is shown below.

$$\text{Fuel price (US\$/liter)} = 0.0056 * \text{Brent Crude Oil (US\$/bbl)} + 0.4127$$

Electricity cost is based on a government agreement and applicable for a 10-year period after the project start. The unit cost is 70 XOF/kWh. It represents \$ 0.13/kWh based on the long term exchange rate at 524 XOF/USD.

### 21.6.3 Mining Operating Costs

Mine operating costs have been estimated from first principles based on equipment hourly operating costs, equipment usage models and productivity assumptions. The first principle estimates are benchmarked in particular with the Essakane mine operation in Burkina Faso operated by IAMGOLD which operate similar size equipment. For this reason an identical activity based accounting cost structure has been developed for the estimate.

The average LOM operating cost for the Sadiola main pit is estimated at \$2.96/t mined which includes costs associated with re-handling from stockpiles.

Haulage represents 35.1% of the mine operating costs followed by blasting (12.8%), loading (10.4%) and drilling (7.2%). Maintenance supply parts and fuel are the main cost items (24.0%) followed closely by manpower costs (21.0%). Fuel price and exchange rate were fixed respectively at \$ 0.84/L and 524 XOF/\$1US.

#### 21.6.3.1 Mine Manpower Requirements

Manpower forecast is for 532 employees at the peak in Year 8, of which 96% are nationals and the remaining 4% are expatriates. About 44% of the employees are in the mine operations department, 22% are in geology which includes hydrogeology, health and grade control technicians and samplers. Approximately 28% of employees in mining are in mechanical maintenance and 6% in the engineering support department. Manpower labour costs represent 20% of the total mining operating cost.

Table 21.11: Mine Operating Cost Summary per Year (k US\$)

<b>Total Mining Costs</b>													
<b>Activity</b>	<b>Total</b>	<b>Year 1</b>	<b>Year 2</b>	<b>Year 3</b>	<b>Year 4</b>	<b>Year 5</b>	<b>Year 6</b>	<b>Year 7</b>	<b>Year 8</b>	<b>Year 9</b>	<b>Year 10</b>	<b>Year 11</b>	<b>Year 12</b>
Operation Admin.	<b>31,678</b>	-	3,040	3,423	3,344	3,189	3,189	3,189	3,189	3,189	3,189	1,981	756
Drilling	<b>66,361</b>	-	2,585	7,628	8,294	7,264	7,619	7,682	8,176	6,781	4,769	4,080	1,482
Blasting	<b>118,517</b>	-	5,345	12,885	14,095	12,925	13,805	14,008	14,333	12,659	8,747	6,979	2,738
Loading	<b>95,801</b>	-	6,445	11,425	11,120	11,288	11,002	10,895	10,843	9,652	6,145	5,010	1,976
Hauling	<b>323,436</b>	-	8,404	23,723	32,836	36,124	35,335	37,337	38,946	37,031	31,547	29,426	12,727
Dewatering	<b>9,406</b>	-	581	1,009	1,009	1,011	1,009	1,009	1,009	1,009	762	665	333
Dump Maintenance	<b>32,206</b>	-	2,303	3,587	3,587	3,596	3,587	3,587	3,587	3,587	2,392	1,794	598
Road Maintenance	<b>30,727</b>	-	2,353	3,123	3,123	3,130	3,123	3,123	3,123	3,123	2,603	2,603	1,301
Geology Admin	<b>49,475</b>	-	5,029	5,170	5,029	5,029	5,029	5,076	5,076	4,670	4,670	3,748	950
Grade Control Drilling	<b>18,919</b>	-	1,180	2,210	2,186	2,188	2,181	2,167	2,159	1,968	1,244	1,026	410
Rehandling	<b>26,851</b>	-	3,948	5,694	3,733	1,183	1,392	1,250	1,250	2,800	1,338	1,342	2,922
Maint. Admin	<b>33,386</b>	-	3,252	3,573	3,471	3,471	3,471	3,471	3,471	3,471	3,117	2,013	604
Support Equipment	<b>21,738</b>	-	1,855	2,445	2,355	2,271	2,266	2,266	2,266	2,266	1,909	1,227	613
Engineering	<b>52,502</b>	-	5,591	5,585	5,498	5,498	5,498	5,498	5,498	4,810	4,029	3,845	1,153
Extra Crushing	<b>11,636</b>	-	1,077	5,626	4,932								
<b>Total</b>	<b>922,640</b>	<b>-</b>	<b>52,987</b>	<b>97,106</b>	<b>104,613</b>	<b>98,167</b>	<b>98,505</b>	<b>100,558</b>	<b>102,926</b>	<b>97,017</b>	<b>76,461</b>	<b>65,738</b>	<b>28,564</b>

Table 21.12: Unit Mine Operating Cost Summary per Year (\$/t Mined)

<b>Total Mining Costs</b>													
<b>Activity</b>	<b>Total</b>	<b>Year 1</b>	<b>Year 2</b>	<b>Year 3</b>	<b>Year 4</b>	<b>Year 5</b>	<b>Year 6</b>	<b>Year 7</b>	<b>Year 8</b>	<b>Year 9</b>	<b>Year 10</b>	<b>Year 11</b>	<b>Year 12</b>
Operation Admin.	0.10	-	0.17	0.10	0.09	0.09	0.09	0.09	0.09	0.10	0.15	0.12	0.11
Drilling	0.21	-	0.14	0.21	0.23	0.20	0.21	0.21	0.23	0.21	0.23	0.24	0.22
Blasting	0.38	-	0.29	0.36	0.39	0.36	0.38	0.39	0.40	0.39	0.42	0.41	0.41
Loading	0.31	-	0.35	0.32	0.31	0.32	0.30	0.30	0.30	0.30	0.29	0.29	0.30
Hauling	1.04	-	0.46	0.66	0.91	1.01	0.98	1.03	1.08	1.14	1.50	1.73	1.92
Dewatering	0.03	-	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.04	0.04	0.05
Dump Maintenance	0.10	-	0.13	0.10	0.10	0.10	0.10	0.10	0.10	0.11	0.11	0.11	0.09
Road Maintenance	0.10	-	0.13	0.09	0.09	0.09	0.09	0.09	0.09	0.10	0.12	0.15	0.20
Geology Admin	0.16	-	0.27	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.22	0.22	0.14
Grade Control Drilling	0.06	-	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Rehandling	0.09	-	0.22	0.16	0.10	0.03	0.04	0.03	0.03	0.09	0.06	0.08	0.44
Maint. Admin	0.11	-	0.18	0.10	0.10	0.10	0.10	0.10	0.10	0.11	0.15	0.12	0.09
Support Equipment	0.07	-	0.10	0.07	0.07	0.06	0.06	0.06	0.06	0.07	0.09	0.07	0.09
Engineering	0.17	-	0.31	0.16	0.15	0.15	0.15	0.15	0.15	0.15	0.19	0.23	0.17
Extra Crushing	0.04	-	0.06	0.16	0.14								
<b>Total</b>	<b>2.96</b>	<b>-</b>	<b>2.89</b>	<b>2.71</b>	<b>2.90</b>	<b>2.74</b>	<b>2.72</b>	<b>2.78</b>	<b>2.86</b>	<b>2.99</b>	<b>3.63</b>	<b>3.86</b>	<b>4.32</b>

**Table 21.13: SSP Mine Manpower Summary per Year**

<b>Manpower</b>	<b>1</b>	<b>2</b>	<b>3</b>	<b>4</b>	<b>5</b>	<b>6</b>	<b>7</b>	<b>8</b>	<b>9</b>	<b>10</b>	<b>11</b>	<b>12</b>
Mine Operations		198	234	238	233	233	233	233	229	185	157	106
Geology		116	116	116	116	116	117	117	116	116	86	56
Maintenance		115	135	143	147	143	147	151	147	118	105	61
Engineering		32	31	31	31	31	31	31	31	30	23	23
<b>Grand Total</b>		<b>461</b>	<b>516</b>	<b>528</b>	<b>527</b>	<b>523</b>	<b>528</b>	<b>532</b>	<b>523</b>	<b>449</b>	<b>371</b>	<b>246</b>
EXPAT		22	22	22	22	22	22	22	20	18	11	8
LOCAL		439	494	506	505	501	506	510	503	431	360	238
<b>TOTAL</b>		<b>461</b>	<b>516</b>	<b>528</b>	<b>527</b>	<b>523</b>	<b>528</b>	<b>532</b>	<b>523</b>	<b>449</b>	<b>371</b>	<b>246</b>
STAFF		129	128	128	127	127	128	128	127	125	99	74
HOURLY		332	388	400	400	396	400	404	396	324	272	172
<b>TOTAL</b>		<b>461</b>	<b>516</b>	<b>528</b>	<b>527</b>	<b>523</b>	<b>528</b>	<b>532</b>	<b>523</b>	<b>449</b>	<b>371</b>	<b>246</b>

**21.6.4 Processing Operating Costs**

The processing operating costs were reviewed in order to account for the cost escalation since the 2010 feasibility study, optimization work and the processing plant production schedule. The processing operating costs are developed in order to account for the different types of ore and the processing plant production schedule. The different types of ore are: soft oxide, hard oxide, soft sulphide, hard sulphide, laterite and mineralized waste. In this Section, the operating, maintenance and assay labs are covered. It is important to mention that a proportional part of the assay lab costs based on the number of assays are transferred to the mining costs and that part of the energy costs are transferred to the general and administration costs .

**21.6.4.1 Plant Manpower Requirements**

The processing manpower requirements are summarized for the met plant, maintenance, TSF and assay lab in Table 21.14. The manpower requirements are based on the production scenarios as mentioned previously .

**Table 21.14: Manpower Requirement**

Department	Without TTP	With TTP
Eng - Plant	123	123
Met - Plant	175	175
Met - Laboratory	142	142
Met - TSF	23	31
<b>Total</b>	<b>463</b>	<b>471</b>

**21.6.4.2 Power**

The calculated energy requirements take into account motor efficiency, load factor and operating time. The estimated energy requirements to process the sulphide ore when the process plant will be at design capacity is 40.8 kWh/t. Electrical energy is supplied from the existing power generators until the end of the construction at 20.36¢ per kilowatt-hour and after from grid power at 11.8¢ per kilowatt-hour. These prices are obtained by taking into account the cost of purchasing diesel at US\$0.78/liter and using an exchange rate of 595 XOF/US\$ Exchange. Grid power cost is 70 XOF/kWh and is also converted with the exchange rate of 595 XOF/US\$. The estimated energy requirements by operating with the soft ore are 22.0 kWh/t as per the present consumption.

**21.6.4.3 Consumable Requirements**

Reagent consumption for the soft ore was based on actual data from the current operation and from metallurgical test work for the hard sulphide ore. The detailed reagent consumption is found in Table 21.15.

**Table 21.15: Plant Consumables**

Consumable Rates (k/t)	Soft Oxide	Hard Oxide	Soft Sulphide	Hard Sulphide	Laterite	LG Soft Oxide
Steel balls	0.6	1.6	0.6	1.6	0.6	0.6
Lime	1.5	1.5	3.1	0.61	3.1	1.5
Hydrogen peroxide						
Cyanide	0.423	0.423	1.223	0.72	1.223	0.423
Lead nitrate						
Carbon	0.06	0.06	0.06	0.045	0.06	0.06
Caustic soda	0.147	0.147	0.147	0.147	0.147	0.147
Hydrochloric acid	0.156	0.156	0.156	0.156	0.156	0.156
Flocculant	0.015	0.015	0.015	0.015	0.015	0.015

**21.6.4.4 Total Processing Operating Costs**

The process operating costs are summarized in Table 21.16, and include labour, maintenance, power consumption, reagents / consumable costs, tailings and water management costs. These operating costs reflect the different configurations encountered in the LOM. These prices are obtained by taking account of cost of purchasing diesel at US \$ 0.78/liter and using an exchange rate of 595 XOF/ US\$ Exchange.

Table 21.16: Processing Operating Costs

Plant Operating Costs		Actual	4.8 Mtpa	7.2 Mtpa	Transition 9.6 Mtpa
Hourly throughput	tph	580	580	900	1180
Annual throughput	Mt/yr	4.8	4.8	7.2	9.6
Fixed Costs					
Engineering (Maintenance)	\$/t	3.35	3.49	3.08	2.55
Labour	\$/t	2.86	2.99	2.06	1.55
<b>Total Fixed Costs</b>	<b>\$/t</b>	<b>6.21</b>	<b>6.48</b>	<b>5.14</b>	<b>4.10</b>
Variable Costs					
Reagents	\$/t	3.13	4.61	4.58	3.21
Power	\$/t	4.50	5.51	4.92	4.06
TSF	\$/t	0.14	0.14	0.13	0.11
<b>Total Variable Costs</b>	<b>\$/t</b>	<b>7.77</b>	<b>10.26</b>	<b>9.63</b>	<b>7.38</b>
<b>Total Processing Costs</b>	<b>\$/t</b>	<b>13.98</b>	<b>16.74</b>	<b>14.77</b>	<b>11.48</b>

### 21.6.5 G&A Operating Costs

General and Administration Services include operating departmental cost of: general management, general administration, accounting and finance, IT, human resources, training, supply chain, environment, community relation, maintenance mine property, transport, security, health safety and clinic, patent tax and others services. In most cases, these services represent fixed costs for the site as a whole. The General Services costs exclude certain costs such as refining, retrenchment, environmental rehabilitation and closure costs. Total GA employees is estimated to 436, Table 21.18.

A summary of G&A costs and head counts are presented below.

Table 21.17: General Services Expenditures

Yearly average cost	(\$M USD)
Departments	
General Administration	4.9
General Management	1.6
Accounting, Finance & Legal	2.9
Information & Technology	1.6
Human Resources & Training	1.5
Supply Chain	1.4
Environment + Closure Team	1.9
Community Relation & Foundation	1.6
Maintenance & Mine Property	2.1
Transport	2.1
Security	2.1
Health, Safety & Clinic	3.7
Patent Tax	1.4
Other services: Recreation Center, Schools, Shop, Guest Accommodation, Parks and Village Utilities Tax	2.8
<b>Grand Total</b>	<b>31.8</b>

**Table 21.18: General Services Head Counts**

<b>Yearly Average Head Count</b>	
Departments	
General Administration	16
General Management	2
Accounting, Finance & Legal	27
Information & Technology	14
Human Resources & Training	24
Supply Chain	39
Environment + Closure Team	27
Community Relation & Foundation	8
Maintenance & Mine Property	60
Transport	35
Security	86
Health, Safety & Clinic	65
Other services: Recreation Center, Schools, Shop, Guest Accommodation, Parks and Village Utilities Tax	33
<b>Grand Total</b>	<b>436</b>

**21.6.6 Royalties**

Royalty payable to Government of Mali is comprised of two elements: CPS (contribution for service delivery) and advalorem. Both are calculated at 3% on revenue respectively.

**21.6.7 Transportation and Refining**

Based on historical data, total refining and transport cost are estimated at \$4.88 per ounce produced.

**21.7 Total Operating Costs**

Fuel price and exchange rate XOF/USD are based on the IAMGOLD corporate assumptions published on September 9<sup>th</sup>, 2015 and summarized in Table 21.1.

Power cost is included within the processing cost. Refining and local tax are within the G&A. Mining includes stockpile rehandling cost. The average cash cost including royalties and management fees is estimated at \$735/oz.

Total manpower is estimated at its peak to 1,431 employees with 532 in mining, 463 in processing and 436 in general administration.

Total operating costs are presented in Table 21.19.

Table 21.19: Total Operating Costs

Description	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mining	-	-8*	-15*	99	100	100	102	104	98	78	67	29
Processing	-	8	26	82	109	113	113	112	112	112	112	106
General Services	-	1	3	23	34	33	34	34	33	34	28	23
<b>Total Opex M\$</b>	-	2	15	203	242	246	248	250	244	223	207	158
Royalties	-	2	12	18	26	25	28	28	21	30	29	22
Management Fees	-	0	2	3	4	4	5	5	3	5	5	4
<b>Total Cost M\$</b>	-	4	29	224	273	275	281	283	268	258	240	183
<b>\$/ft. milled</b>	-	7.11	11.02	37.43	40.08	38.19	39.05	39.28	37.23	35.83	33.37	27.96
<b>Total Cash Cost \$/oz</b>	-	111	177	923	791	842	764	779	978	659	638	677

\* Capitalized pre-strip period

## 22. ECONOMIC ANALYSIS

The economic analysis was built to support the 7.2 Mtpa SSP project. During the feasibility study, focus remains on the project since the actual mine starts reaching its final years of operation. The reader will notice a small discrepancy between reserves and financial statement (5%). This is due to stockpiles and the satellites pits production strategy that involves a dynamic cut-off-grade. Over the next three (3) years the NCF from the current operation is expected to be mostly neutral under the current assumptions. The following section presents the cash flow model created to evaluate the hard sulphide expansion project SSP without the current satellite mine plan.

No adjustment for future increases in costs is planned, which means that the financial analysis remains in constant dollars. Cash flows were estimated on an annual basis and include debt financing. A discount rate of 6% was applied to determine the present value of the project. The internal rate of return was used to determine the economic viability. All the amounts are in constant 2015 dollars and expressed in United States dollars, unless otherwise indicated.

### 22.1 Assumptions

The scenario presented in this document was prepared according to the assumptions in Section 21.1 and the following:

Tax	30% as per the mining convention between SEMOS and the Government A 5 years tax holiday was applied to the model.
Royalties	3% CPS Royalties 3% Ad valorem tax
Management fees	1% is considered as a royalty for tax calculations

#### 22.1.1 Metal Price

For the study, gold price ranges from US \$1,150/oz Au to US \$1,275/oz Au. It is aligned with the IAMGOLD's long-term corporate assumptions. The average gold price over the life of mine is \$1,271/oz Au.

#### 22.1.2 Income Tax

Income taxes are calculated in accordance with the mining convention between SEMOS and the Malian Government. A five-year tax exemption period from year 3 to year 7 was applied to the model according to an agreement with the Malian Government. The income tax rate is 30%. The calculations were reviewed by IAMGOLD Corporation's tax department. The estimated income taxes account for \$133 M.

**22.1.3**      **Royalties**

Royalties and legislated taxes of 6% have been applied to gross revenues. The 6% rate comprises a 3% royalty and a 3% ad valorem tax. No withholding taxes have been applied. The calculations were reviewed by IAMGOLD Corporation's tax department. The Royalties account for \$241 M.

Operator management fees of 1% (\$40 M) are treated as a royalty for tax calculation purposes.

**22.1.4**      **Other Special Taxes**

Special taxes are associated with electrical generation, water use charges and patent tax and other taxes. They were estimated in this study and are part of the general administrative costs. This practice is consistent with the methodology currently used at the Sadiola mine. In the project assessment, these taxes account for approximately \$28 M.

**22.1.5**      **Production**

The SSP mining starts in Q2 Year 2 and end in Year 12. A pre-strip period takes place until the end of Year 3. Since the current plant does not have the capacity to process the current hard sulphide stockpile, the mill is fed with this material during the pre-stripping period. The peak mining period is scheduled in Year 3 with a 36 M t and continues over six (6) years. The annual production in this study is set at 7.2 Mtpa. With this tonnage, the intent is to reach a production level above 300 koz per annum with a total gold production of 3.15 M ounces.

**22.1.6**      **Working Capital**

The working capital required for the project is established taking into consideration that the increase of production (sales) meets the increase in payables. The same logic applies to the inventory level. The total amount is set at \$32 M representing an increase in HME and plant inventory, HME capital spare parts and plant consumables.

**22.1.7**      **Interest Rate on Shareholder Loan**

The interest rate applied to the loan is the estimated LIBOR rate plus a 2% premium. The total interest on expenses is \$108 M.

**22.2**      **Work Schedules**

Before beginning the construction of surface infrastructures, a period must be allotted to complete the legal authorization and permitting processes, which includes public hearings and updates to the ESIA. Detailed engineering starts soon after the study approval. Additionally, the construction camp is part of the

critical path to deliver the project on time hence it will have to be addressed as a priority for purchasing purposes.

Construction starts in Q2 Year 1 and will be finished in Q4 Year 2. It is followed by a commissioning and a ramp-up period that extends from Q4 Year 2 to Q1 Year 3.

### 22.3 Operating Schedules

In order to fulfill the operating obligations presented in this feasibility study, the main pit pre-stripping must begin during the Q2 Year 2. Moreover, in the owner mining perspective, key players in the mine department are hired before the pre-stripping starts. Maintenance training and equipment erection is scheduled to begin in Q1 Year 2. On-hand equipment bought is shipped on site in Q4 Year 1. See Table 22.1

**Table 22.1: Operating Schedule**

SSP project	Year 1				Year 2				Year 3				Year 4
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Yr
<b>Operational Readiness</b>													
Equipment Shipping													
Personal hiring													
Personal Training													
Equipment Erection													
<b>Pre-Prod</b>													
Pre-stripping													
<b>Production</b>													
Waste stripping													
Operation													
<b>Milling</b>													
Commissioning/Ramp-up													

### 22.4 Cash Flows

The cash flow obtained for the 7.2 Mtpa SSP project was discounted at 5%, 6%, 8% and 10%. A rate of 6% was selected as the discount rate used for the study final evaluation. Table 22.2 shows the results after taxes.

**Table 22.2: After Tax Discounted Cash Flows in Millions Dollars (M\$)**

Discount Rate	7.2 Mtpa SSP
0%	\$740
5%	\$378
6%	\$325
8%	\$234
10%	\$158

**22.5 Internal Rate of Return**

The internal rate of return, presented in Table 22.3, is the result of the difference between the hard sulphides project and the actual mine budget. In other words, the total cash flow obtained is subtracted from the cash flow forecast of the current Sadiola mine plan, prior to calculating the return. Table 22.3 shows the return after taxes.

**Table 22.3: Internal Rate of Return**

7.2 Mtpa SSP	
Internal Rate of Return (%)	16.0%

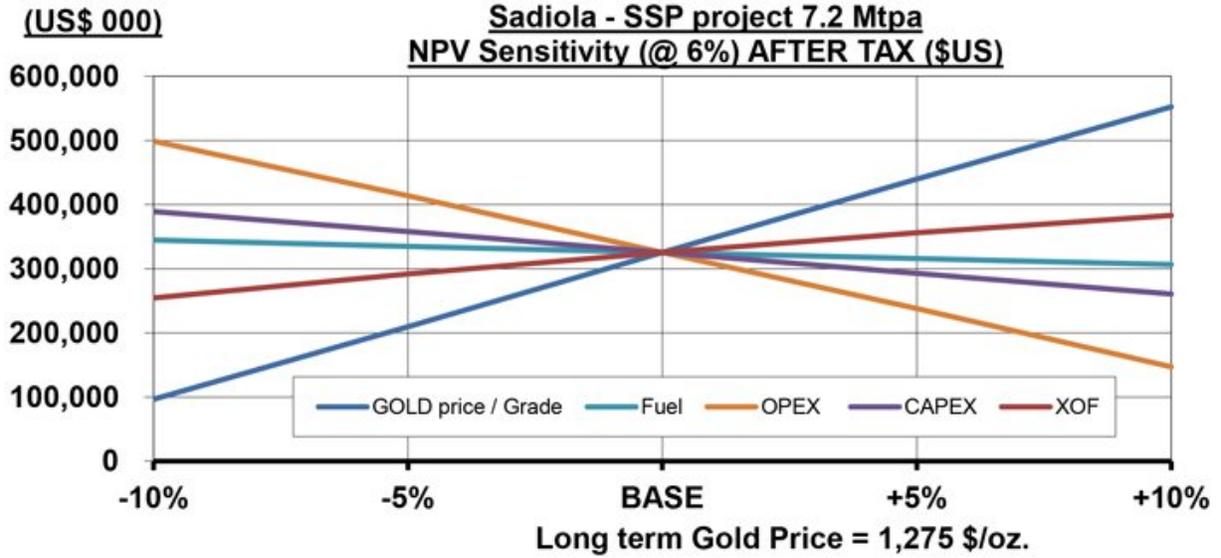
**22.6 Sensitivity**

Sensitivity analysis of the gold price, exchange rate XOF/USD, Capex, Opex and fuel price show that the project is relatively robust in an environment ranging between -10% to +10%. Table 22.4 shows the 7.2 Mtpa study parameters considered. Figure 22.1 illustrates the net present value calculated using a 6% discount rate after taxes and royalties.

**Table 22.4: 7.2 Mtpa Study Parameters**

Study Parameters	Year 1	Year 2	Year 3	Year 4	Year 5 & (+)
Average Gold Price (\$/oz)	\$ 1,150	\$ 1,225	\$ 1,250	\$ 1,250	\$ 1,275
Exchange Rate (USD-XOF)	XOF 595	XOF 570	XOF 546	XOF 524	XOF 524
Fuel (\$/l delivered)	\$ 0.78	\$ 0.83	\$ 0.86	\$ 0.86	\$ 0.89
Power (XOF/Kwh)	N/A	XOF 70	XOF 70	XOF 70	XOF 70

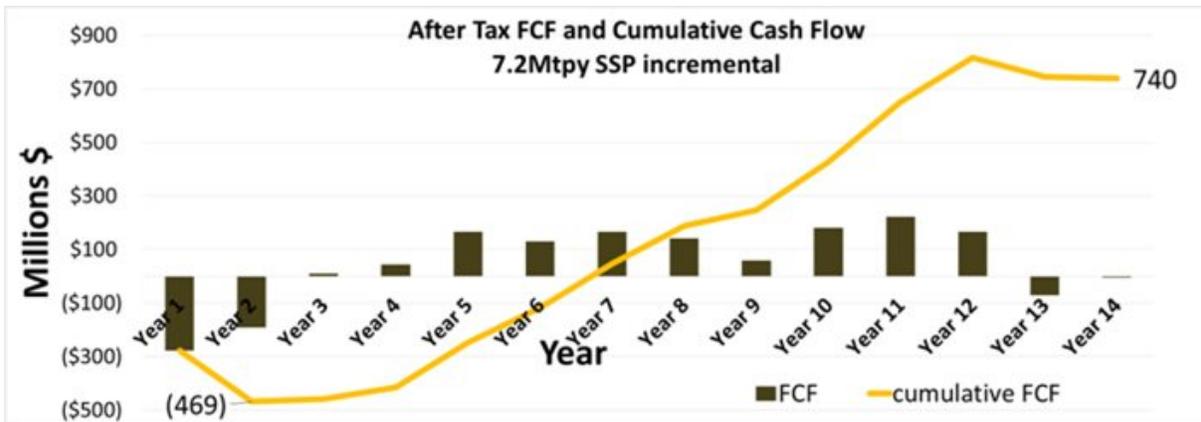
Figure 22.1: Sensitivity Analysis



22.7 Investment Payback Period

Figure 22.2 illustrates the time period required to repay the investment of the 7.2 Mtpa hard sulphide project. Five (5) years is necessary from the start of the production.

Figure 22.2: After Tax FCF and Cumulative CF for 7.2 Mtpa Incremental



## **22.8**            **Project Financial Summary**

The scenario presented shows a robust project under the financial environment considered for this study. The mine would operate for 10 years at a throughput of 7.2 Mtpa averaging an annual production above 300 koz Au.

Net revenues would be \$4,012 M over the LOM. The operating costs over the same period would be \$2,037 M. Initial capital expenditures of \$379 M on which \$57 M for sale of equipment in base case should be added and sustaining of \$257 M over the LOM would complete the expenditures. Operating Management fees and royalty accounts respectively for \$40 M and \$241 M. Interest expense represents \$108 M and rehabilitation for \$20 M. A pre-tax cash flow of \$873 M would be generated. Income taxes accounts for \$133 M. This would result in a net cash flow after taxes of \$740 M. At the selected discount rate of 6%, this represents a net present value after taxes and interest expense of \$325 M. Table 22.6 summarizes the project results.

### **22.8.1**            **Detailed Cash Flows**

Table 22.5 shows the various costs estimated in this study.

**Table 22.5: Detailed Cash Flow**

<b>Sadiola</b>	<b>7.2 Mtpa</b>
Tonnes (Mt)	312
Strip Ratio	3.9
Milled Tonnes (Mt)	65.7
Recoverable Gold (Moz)	3.2
Max. Throughput (Mt)	7.2
Average Recovery (%)	76.5%
Mine Life (yr)	10
<b>Gross revenue (\$M)</b>	<b>\$4,012</b>
Direct Cash Cost (\$/oz)	\$646
Cash Cost (\$/oz)	\$735
AISC (\$/oz)	\$816
Initial Capital (\$M)	\$379
Resale of Equipment (\$M)	\$57*
Sustaining Capital (\$M)	\$257
<b>Total Capital</b>	<b>\$693*</b>
Closure Costs (\$M)	\$20
<b>SADIOLA (100%)</b>	
After tax NPV 0	\$740
After tax NPV 6%	\$325
After tax NPV 8%	\$234
After tax NPV 10%	\$158
<b>After tax IRR</b>	<b>16.0%</b>
<b>Payback Period (yr)</b>	<b>5.0</b>

(\*) Assuming equipment already acquired would be disposed in the base case.

### 22.8.2 Annual Cash Flows

Annual cash flows of hard sulphides project was derived from the difference between the current mine plan and the combination of the 7.2 Mtpa expansion with the current mine plan. Results are shown in Table 22.6.

Table 22.6: Summary of Cash Flow

Sadiola		1\$US=XOF	595	570	546	524	524	524	524	524	524	524	524	524	524	524	
		\$/bbl	58	65	75	80	80	85	85	85	85	85	85	85	85	85	85
		\$/oz	1,150	1,225	1,250	1,250	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	
<b>7.2Mtpa Incremental</b>		<b>Total</b>	<b>1</b>	<b>2</b>	<b>3</b>	<b>4</b>	<b>5</b>	<b>6</b>	<b>7</b>	<b>8</b>	<b>9</b>	<b>10</b>	<b>11</b>	<b>12</b>	<b>13</b>	<b>14</b>	<b>15</b>
Mined tonnage (Mt)		311.5	-	18.3	35.9	36.1	35.8	36.2	36.1	36.0	32.5	21.0	17.0	6.6	-	-	-
Milled tonnage (Mt)		65.7	-	0.5	2.6	6.0	6.8	7.2	7.2	7.2	7.2	7.2	7.2	6.6	-	-	-
Produced oz (koz)		3,155	-	35	161	243	344	326	368	363	274	392	377	271	-	-	-
Revenus (M\$)		4,012	-	29	202	300	439	416	469	463	350	500	480	363	-	-	-
Direct Operating cost (M\$)		(2,037)	-	(2)	(15)	(203)	(242)	(246)	(248)	(250)	(244)	(223)	(207)	(158)	-	-	-
Royalty (M\$)		(241)	-	(2)	(12)	(18)	(26)	(25)	(28)	(28)	(21)	(30)	(29)	(22)	-	-	-
Management Fees (M\$)		(40)	-	(0)	(2)	(3)	(4)	(4)	(5)	(5)	(3)	(5)	(5)	(4)	-	-	-
Rehabilitation (M\$)		(20)	-	6	20	31	32	7	-	-	-	-	(36)	(75)	(6)	-	-
EBIDTA (M\$)		1,673	(0)	32	194	107	199	148	188	180	81	242	240	144	(75)	(6)	-
Interest Expense (M\$)		(108)	(6)	(17)	(23)	(22)	(20)	(13)	(6)	(1)	-	-	-	-	-	-	-
Incomes tax (M\$)		(133)	0	(0)	2	1	-	-	-	(30)	(3)	(44)	(46)	(11)	-	-	-
Development capital (M\$)		(436)	(235)	(201)	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital (M\$)		(257)	(4)	(4)	(137)	(40)	(9)	(7)	(14)	(8)	(26)	(5)	(3)	-	-	-	-
Changes in working cap (M\$)		0	(32)	(1)	(25)	(2)	(4)	2	(3)	1	5	(10)	32	34	3	-	-
FCF (M\$)		<b>740</b>	<b>(277)</b>	<b>(192)</b>	<b>10</b>	<b>44</b>	<b>166</b>	<b>130</b>	<b>165</b>	<b>142</b>	<b>58</b>	<b>182</b>	<b>223</b>	<b>166</b>	<b>(72)</b>	<b>(6)</b>	-
Cumulative FCF (M\$)			(277)	(469)	(458)	(415)	(249)	(119)	46	188	246	428	651	817	746	740	740
DCF (M\$)		5%	378														
		6%	<b>325</b>														
		8%	234														
		10%	158														

## 23. ADJACENT PROPERTIES

The Yatela Gold Mine is an adjacent small open pit operation located approximately 30 km north of the Sadiola Mine. Construction of Yatela Gold Mine commenced in 2000 and the processing plant was commissioned in 2001. The Shareholders of Yatela S.A. include AGA (40%), IAMGOLD (40%) and the Government of Mali (20%).

Yatela comprises an open-cast pit, a smaller satellite pit, a heap leach facility, waste dumps, and a small mining village. Activated carbon (containing gold) is trucked to the Sadiola processing plant for final processing. Although it shares some infrastructure with SEMOS, the Yatela operation is run as a separate company.

The Yatela operation is scheduled to close in 2015 with the input of the National Closure Commission ("NCC").

A broader view of other mineral properties surrounding the Sadiola property is shown in Figure 23.1.

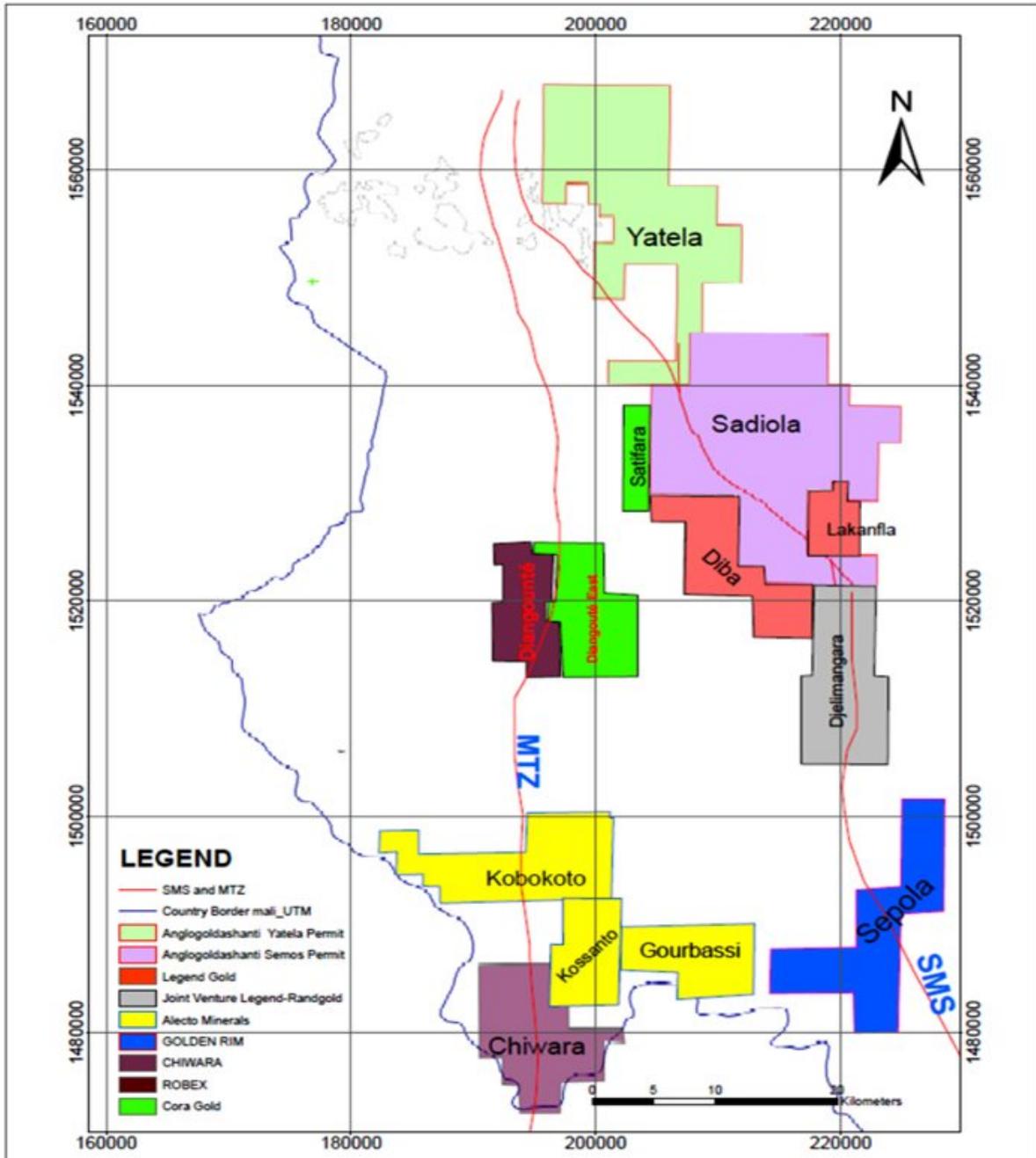
Legend Gold Corporation ("LGC") has a 100% interest in two advanced stage exploration projects; the Lakanfla Project (formerly the Kantela Project) and the Diba Project, both of which are situated adjacent to the Sadiola Permit. The projects comprise 83 km<sup>2</sup> and 48 km<sup>2</sup> of mineral permits respectively. At Lakanfla carbonate-hosted, karst-enriched mineralization, similar to the Yatela Mine, is reported while at Diba the mineralization is reported as being a sediment-hosted, disseminated epigenetic, replacement gold deposit.

A NI 43-101 Technical Report filed in 2013 reported that approximately 7 M t of Indicated and Inferred Mineral Resources have been declared at Diba, based on exploration drilling, containing approximately 300,000 oz of gold. A Mineral Resource has not yet been declared at Lakanfla.

LGC has reported that the Diba permits have been combined with other exploration permits (Badiazila) into a new permit application under the name "Korali Sud".

It was not possible to validate the information above which is not necessarily indicative of the mineralization at the Diba Project.

Figure 23.1: Active Properties Adjacent to Sadiola Gold Mine



Source: SEMOS, 2015

## 24. OTHER RELEVANT DATA AND INFORMATION

The Sadiola Sulphide Project execution will be directly managed by IAMGOLD project management team which will include SEMOS people on site. The engineering, procurement and construction works will be contracted out to qualified firms. The construction work will be mainly contracted out to local (regional) contractors under the supervision of the Project team. Project control functions such as scheduling, cost control, project logistics and site supervision will be executed directly by the IAMGOLD project team.

Major project milestones are presented in Table 24.1.

**Table 24.1: Major Project Milestones**

Description	Start Date	Completion Date
Project approval	-	Q1 - Year 1
Detailed engineering	Q1 - Year 1	Q4 – Year 1
Permitting	-	Q2- Year 1
Truck shop	Q3 – Year 1	Q2 – Year 2
Plant construction	Q3 - Year 1	Q4 - Year 2
225 kV transmission line	Q3 - Year 1	Q4 – Year 2
Pit pre-stripping	Q2 –Year 2	Q4 – Year 2
TSF construction	Q3 – Year 1	Q4 – Year 2
Commissioning and Ramp Up		Q4 – Year 2
Plant commercial production	-	Q1 - Year 3

### 24.1 Risk Management

A risk session was organized during the Feasibility study. The following presents the main risks and opportunities:

- The main negative risks associated with the project include:
- The extension of environmental and construction permits and any delays that this may cause to the project. Some changes to the project have occurred such as the addition of the North West Dump, the South Dump and the postponement of the TTP that will require more work. This could also increase project costs if the North dump needs to be relocated;
- Lack of accuracy in the financial model for the recovery estimate. The recovery estimates are based on spacial location data. The overall stated recovery for the project of 76% is accepted but is based on a wide range of results which vary from lows at around 50% to highs at around 90%.

Inability to model the recovery more accurately will lead to period of high cashflow variation which may not be accurately forecasted;

- The comminution circuit was designed at 50th percentile rather than the 75th which could impact the grind size and the production;
- The ability to attract the required personnel for construction and operations. This could be reduced by the current market downturn.
- The positive risks which may improve the project include:
  - Potential of finding Sulphide Ore in satellite pits to be processed early in the LOM. The full capacity of the new plant will not be achieved in years 4-5 and more sulphide could have a significant impact on the project financial KPIs;
  - Accelerate pre-production mining schedule in order to access ore earlier and fill the mill to its maximal capacity sooner;
  - Power line length could be reduced to 54 km instead of 89 km. This will need some negotiations with the Government but could represent a good saving for the project but also for EDM who could save up to \$20 M of power losses;
  - The mineralized waste piles yielded a slightly better head grade but have not been factored in at this time.

## 25. INTERPRETATION AND CONCLUSIONS

### 25.1 Mineral Resources and Mining Operations

The QP is satisfied that the processes and procedures used by the SEMOS and AGA team to estimate the Mineral Resources are in line with industry practice and no fatal flaws were noted.

It is understood why Leapfrog™ is used to model the deposits.

#### 25.1.1 Risk Assessment

The QP has summarized identified real risks pertaining to the Sadiola Mineral Resource in Table 25.1. The risk values have been assessed from a mitigation point of view.

It is the QP's opinion that all risks listed are manageable and can be effectively mitigated if adequate consideration and active planning is established.

#### 25.1.2 Risks

**Table 25.1: Risk Assessment – Major Perceived Risks**

Risk	Risk Assessment	Comment
<b>Property, mineral rights and Mineral Resources</b>		
Mining rights	Low	Mineral rights secured by legal licenses. Expires prior to completion of mining by one year.
Mineral Resources	Low	Significant Measured, Indicated and Inferred Resource.
Surface rights	Low	Surface rights are tied to mining rights.
Land restitution	Low	Limited land cases have occurred in the past. No significant litigation expected.

### 25.2 SSP Feasibility Project Risks

Sadiola gold mine is currently processing soft ore coming from satellite pits and existing stockpiles. The soft ore mineral reserve is now depleting and modifications to the current processing plant and related infrastructures are needed to pursue the actual operation. The Sadiola Sulphide Project - Feasibility study of the exploitation of an important hard sulphide ore deposit, presented in this technical report, has demonstrated attractive financial results.

The main risks associated with the project are related to the permitting process and recent security in Mali. Environmental permits are currently expired and some changes in the project engineering will require work to update the current baseline.

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The financial results of the 7.2 Mtpa scenario indicates that the Sadiola Sulphide Project is financially attractive to all the stakeholders and joint-venture partners and this project could assure the continuity of the Sadiola Mine operation.

## 26. **RECOMMENDATIONS**

The QP is of the opinion that the insertion rate of blanks and SRMs is low and the Sadiola team should consider increasing the insertion rate to 10% (from the current 5%). It is also recommended to reconfirm that all Leapfrog™ models need to be validated against geological reality.

Before final project approval, it is recommended to initiate the following work to mitigate projects risk and capture some opportunities:

- Expand sulphide exploration to satellite pits to maximise the processing rate of hard sulphide during years 3 and 4 of the SSP project;
- Initiate discussion with GoM and begin environmental work to update the ESIA and to proceed with the renewal of permits;
- Initiate discussion with GoM regarding the opportunity to modify the routing of the power line and reduce the length;
- Initiate the engineering for the construction camp and power line;
- Refine project execution plan and initiate equipment inspection for equipment that are already purchased and presently in storage.

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**28. DATE AND SIGNATURE PAGE**

This report titled "IAMGOLD Sadiola Sulphide Project (SSP) 2015 NI 43-101 Report effective December 31, 2015" and dated March 15, 2016, was prepared and signed by the following authors:

Dated at Longueuil, QC  
March 15, 2016

**(Signed and Sealed) "Philippe Gaultier"**

Philippe Gaultier, Ing. MASC (OIQ # 130381)  
Director of Development Projects  
IAMGOLD Corporation

Dated at Johannesburg, Au  
March 15, 2016

**(Signed and Sealed) "Mark Burnett"**

Mark Burnett, MSc (MRM), B.Sc.(Hons); PGDTE; CBM; CAG; GCG; Pr. Sci.  
Nat; FGSSA: MSAIMM  
Snowden

Dated at Longueuil, QC  
March 15, 2016

**(Signed and Sealed) "Daniel Vallières"**

Daniel Vallières, Ing. (OIQ # 107203)  
Director, Mining engineering  
IAMGOLD Corporation

Dated at Longueuil, QC  
March 15, 2016

**(Signed and Sealed) "Jérôme Girard"**

Jérôme Girard, Ing., P. Eng (OIQ # 116471) (PEO # 100160489)  
Manager, Metallurgy  
IAMGOLD Corporation

Dated at Longueuil, QC  
March 15, 2016

**(Signed and Sealed) "Luc-B Denoncourt"**

Luc-B Denoncourt, Ing. (OIQ # 129874)  
Project Director  
IAMGOLD Corporation

Dated at Longueuil, QC  
March 15, 2016

**(Signed and Sealed) "Louis-Pierre Gignac"**

Louis-Pierre Gignac, Eng. (OIQ 132995)  
Co President  
G Mining Services Inc.

**29. CERTIFICATE OF QUALIFIED PERSON S**

Section 29

March, 2016

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# Certificate of Qualified Person

PHILIPPE GAULTIER

I, Philippe Gaultier, Ing., as an author of this report entitled "IAMGOLD Sadiola Sulphide Project (SSP) 2015 NI 43-101 Report" prepared for IAMGOLD and dated and effective March 15<sup>th</sup>, 2016, do hereby certify that;

1. I am the Director of Projects Development with IAMGOLD Corporation, 1111, St. Charles Street West, Longueuil, QC, Canada, J4K 5G4;
2. I am a graduate of University of Manitoba, Manitoba; in 1988 with a BSc. Mechanical Engineering and University of Toronto, Ontario with MSc Mechanical Engineering
3. I am registered as a Mechanical Engineer in the Province of Quebec (OIQ #130381). I have worked as a mechanical engineer for a total of twenty eight years since my graduation. My relevant experience for the purpose of the Technical Report are in the Sections:
  - a. Infrastructure and Plant Engineering for internal Feasibility Report for Sadiola Deep Sulfide 2010
  - b. Update of subsequent documents and engineering related to aforementioned report
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Sadiola last on August 28<sup>th</sup>, 2015.
6. I am responsible for Sections 1.2.4, 1.12, 1.13.1, 1.13.2, 1.14, 1.15, 5 (prior to 5.1), 5.1, 5.6.1, 5.6.2, 18 (except for 18.8) and 20 of the Technical Report.
7. I have been working for IAMGOLD from 2008 to 2016. As a Director Development Projects, since 2014, I am a full-time employee of IAMGOLD Corporation, Canada and I own shares of IAMGOLD Corporation;
8. I am not independent of IAMGOLD Corporation as set out in Section 1.5 of National Instrument 43-101 -as per NI 43-101 s.8.1(2)(f) and I did receive from my employer participation incentive securities ("options") and company shares in 2009 - 2015 ;
9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and NI 43-101F1.
10. At the effective date of the technical report, to the best of the my knowledge, information and belief, the sections contained in the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated 15th day of March, 2016

**(Signed and Sealed) " Philippe Gaultier "**

Philippe Gaultier, Ing. MSc (OIQ # 130381)  
Director of Development Projects  
IAMGOLD Corporation

# Certificate of Qualified Person

## MARK BURNETT

I, Mark Burnett, Pri. Sci. Nat. as an author of this report entitled "IAMGOLD Sadiola Sulphide Project (SSP) 2015 NI 43-101 Report" dated and effective March 15, 2016, do hereby certify that:

1. I am a Principal Consultant for Snowden Johannesburg Technology House, Greenacres Office Park, Cnr. Victory and Rustenburg Roads, Victory Park, Johannesburg 2195, SOUTH AFRICA;
2. I am a graduate of The University of the Free State in South Africa and hold an MSc in Mineral Resource Management (2016). I am also a graduate of the University of the Witwatersrand in South Africa and hold a B.Sc. (Hons) Geology (1992).
3. I am a geologist who has worked in the minerals industry for 24 years with specific involvement in mine production and Mineral Resource estimation, mainly for gold. I have worked as a geological consultant for eight years in a technical and advisory capacity for clients covering development and mine production for a number of different mineral commodities.
4. I am a registered Professional Scientist with the South African Council for Scientific Professions (SACNASP Reg. No. 400361/12), a Member of the Southern African Institute of Mining and Metallurgy and a Fellow of the Geological Society of South Africa.
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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have visited the Sadiola mine from May 19 to May 21, 2015.
7. I am responsible for Sections 1.4 to 1.6, 1.8, 1.9, 7 to 12, 14, 25.1 and in collaboration with the other QPs to my corresponding sub-sections 1.1, 1.4, 1.5, 1.6, 1.8 and 1.9 of Section 1 and Section 26 of the Technical Report.
8. I am independent of the issuer as defined in Section 1.5 of NI 43-101;
9. I have no prior involvement with the property that is subject of the technical report;
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and NI 43-101F1.
11. At the effective date of the technical report, to the best of the my knowledge, information and belief, the sections contained in the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 15th day of March, 2016

**(Signed and Sealed) " Mark Burnett "**

Mark Burnett, *MSc (MRM), B.Sc.(Hons); PGDTE; CBM; CAG; GCG; Pr. Sci. Nat; FGSSA: MSAIMM*  
SNOWDEN

---

# Certificate of Qualified Person

---

DANIEL VALLIÈRES

I, Daniel Vallières, Ing., as an author of this report entitled "IAMGOLD Sadiola Sulphide Project (SSP) 2015 NI 43-101 Report" prepared for IAMGOLD and dated and effective March 15<sup>th</sup>, 2016, do hereby certify that;

1. I am Director Mining engineering with IAMGOLD Corporation, 1111, St. Charles Street West, Longueuil, QC, Canada, J4K 5G4;
2. I am a graduate of Laval University, Quebec; in 1991 with a BSc. Mining Engineer;
3. I am registered as a mining engineer in the Province of Quebec (OIQ #107203). I have worked as a mining engineer for a total of twenty five years since my graduation. My relevant experience for the purpose of the Technical Report is the financial evaluation.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Sadiola last on August 28<sup>th</sup>, 2015.
6. I am responsible for Sections 1.17, 19, 22 of the Technical Report.
7. I have been working for Cambior Inc. from 1992 to 2006 and Breakwater Resources from 2000 to 2002; I am a full-time employee of IAMGOLD Corporation, Canada, since 2006, and I own shares of IAMGOLD Corporation;
8. I am not independent of IAMGOLD Corporation as set out in Section 1.5 of National Instrument 43-101 -as per NI 43-101 s.8.1(2)(f) and I did receive from my employer participation incentive securities ("options") and company shares in 2009 - 2015 ;
9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and NI 43-101F1.
10. At the effective date of the technical report, to the best of the my knowledge, information and belief, the sections contained in the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated 15th day of March, 2016

**(Signed and Sealed) " Daniel Vallières "**

Daniel Vallières, Ing. (OIQ # 107203)  
Director, Mining engineering  
IAMGOLD Corporation

# Certificate of Qualified Person

JÉRÔME GIRARD

I, Jérôme Girard, ing., P. Eng., as an author of this report entitled "IAMGOLD Sadiola Sulphide Project (SSP) 2015 NI 43-101 Report" prepared for IAMGOLD and dated and effective March 15<sup>th</sup>, 2016, do hereby certify that:

1. I am the Manager of Metallurgy with IAMGOLD Corporation, 1111, St. Charles Street West, Longueuil, QC, Canada, J4K 5G4;
2. I am a graduate of Laval University, Quebec; in 1995 with a BSc. Eng. Materials and Metallurgy;
3. I am registered as a Metallurgical Engineer in the Province of Quebec (OIQ #116471) and Ontario (PEO # 100160489). I have worked as a metallurgical engineer in mineral processing for a total of 20 years since my graduation. My relevant experience for the purpose of the Technical Report is:
  - a. MASA-A / Tailings Storage Facility, Tailings Treatment Plant and Plant Detailed Engineering for the Sadiola Sulfide Project
  - b. Update of subsequent documents and engineering related to aforementioned report
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Sadiola last on August 28<sup>th</sup>, 2015.
6. I am responsible for Sections 1.7, 1.12, 1.13.3, 1.13.4, 5.6.3, 13, 17, 18.8 and 21.6.4 of the Technical Report.
7. As a Manager, Metallurgy since 2012, I am a full-time employee of IAMGOLD Corporation, Canada and I own shares of IAMGOLD Corporation.
8. I am not independent of IAMGOLD Corporation as set out in Section 1.5 of NI 43-101, as per NI 43-101 s.8.1(2)(f), as I am a full-time employee.
9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and NI 43-101F1.
10. At the effective date of the technical report, to the best of the my knowledge, information and belief, the sections contained in the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 15th day of March, 2016

**(Signed and Sealed) " Jérôme Girard "**

Jérôme Girard, Ing., P. Eng (OIQ # 116471) (PEO # 100160489)  
Manager, Metallurgy  
IAMGOLD Corporation

# Certificate of Qualified Person

LUC-BERNARD DENONCOURT

I, Luc-Bernard Denoncourt, ing., as an author of this report entitled "IAMGOLD Sadiola Sulphide Project (SSP) 2015 NI 43-101 Report" prepared for IAMGOLD and dated and effective March 15<sup>th</sup>, 2016, do hereby certify that;

1. I am a Project Director with IAMGOLD Corporation, 1111, St. Charles Street West, Longueuil, QC, Canada, J4K 5G4;
2. I am a graduate of Laval University, Quebec City; in 2002 in Mining Engineering.
3. I am registered as a Mining Engineer in the Province of Quebec (OIQ #129874). I have worked as a mining engineer and project management for a total of fourteen years since my graduation.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Sadiola Mine Site on August 28<sup>th</sup>, 2015.
6. I am responsible for Chapters 1.1 to 1.2.3, 1.3, 1.14, 1.16.1, 1.16.2, 1.16.3, 1.18, 1.19, 2, 3, 4, 5.2 to 5.5, 6, 21.1 to 21.5, 23, 24, 25.2, 26, 27 and 28 of the Technical Report.
7. I have been working for IAMGOLD since 2015 as a Project Director. I am a full-time employee of IAMGOLD Corporation, Canada and I own shares of IAMGOLD Corporation;
8. I am not independent of IAMGOLD Corporation as set out in Section 1.5 of National Instrument 43-101 -as per NI 43-101 s.8.1(2)(f) and I did receive from my employer participation incentive securities ("options") and company shares in 2015 ;
9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and NI 43-101F1.
10. At the effective date of the technical report, to the best of the my knowledge, information and belief, the sections contained in the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated 15th day of March, 2016

**(Signed and Sealed) " Luc-B Denoncourt "**

Luc-B Denoncourt, Ing. (OIQ # 129874)  
Project Director  
IAMGOLD Corporation

# Certificate of Qualified Person

LOUIS-PIERRE GIGNAC

I, Louis-Pierre Gignac, as an author of this report entitled "IAMGOLD Sadiola Sulphide Project (SSP) 2015 NI 43-101 Report" prepared for IAMGOLD and dated and effective March 15<sup>th</sup>, 2016, do hereby certify that;

1. I am a mining engineer with the firm of G Mining Services Inc. with an office at 7900 W, Taschereau Blvd., Building D, Suite 200, Brossard, QC, Canada J4X 1C2;
2. I am a graduate of the École Polytechnique de Montréal and of the McGill University with a Master in Applied Science in 2002 and a Bachelor Degree in Mining Engineering in 1999. I have practiced my profession continuously since 2002. I have over 10 years experience in exploration and consulting. Prior to joining G Mining Services Inc., I worked for Cambior Inc. and Iamgold Corporation as a Financial Analyst and Mine Engineer. My experience includes providing expertise for the open-pit aspect of various mining Projects and the financial modeling and economic evaluation. Areas of expertise are open-pit and financial modeling;
3. I am a Professional Engineer registered with the Ordre des ingénieurs du Québec (Licence: 132995) and CIM member and holder of a CFA charter;
4. I visited the project area on August 27 to August 31 2015;
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
6. I am independent of the issuer as defined in Section 1.5 of NI 43-101;
7. I am responsible for the preparation of Sections 1.10, 1.11, 1.15.3, 5.6.4, 5.6.5, 15, 16, 21.6 (except for 21.6.4) and 21.7 of the Technical Report, which are also referred to in portions of Sections 14 to 21, 24 and 25;
8. I have no prior involvement with the property that is subject of the technical report;
9. At the effective date of the technical report, to the best of the my knowledge, information and belief, the sections contained in the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading;
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and NI 43-101F1; and
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 15th day of March, 2016

**(Signed and Sealed) " Louis-Pierre Gignac "**

Louis-Pierre Gignac, Eng. (OIQ 132995)  
Co President  
G Mining Services Inc.