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IN COLLABORATION WITH

SGS DRA MET-CHEM WSP

Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania

Submitted to:

ALGOLD RESOURCES LTD

Algold Resources Ltd.

Effective Date:
March 9, 2018

Issue Date:
July 4, 2018

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Respectfully submitted to:
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Date and Signature

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1. EXECUTIVE SUMMARY

1.1 Introduction

Algold (ALG - TSX.V) is a Canadian junior company focusing on exploration and development of gold deposits in West Africa.

TIREX SA, a public limited company under Mauritanian law regularly incorporated and registered in the Nouakchott trade register on April 27, 2016 under the numbers 91408/GU/12417 (analytic register) and 1994 (chronological register), is a subsidiary of Kanosak Barbados, which is a subsidiary of the Algold Resources Company Ltd. ("Algold"), a Canadian company listed on the Toronto Stock Exchange in Canada.

Algold, on behalf of TIREX SA, commissioned Ausenco Engineering Canada (Ausenco), along with Met-Chem/DRA, WSP Canada Inc (WSP), and SGS Geostat to prepare a Preliminary Economic Assessment (PEA) of the Tijirit Project in Mauritania.

The Tijirit Project includes the 1117B2 exploration permit and the 306 km^2 2480C2 mining concession granted to TIREX by the Mauritanian government on June 12, 2017 for a period of 30 years. This concession gives TIREX SA exploration and exploitation rights for the gold and related substances covered by this permit. The PEA and the resources estimate only refer to the mining concession.

The PEA is also based on resources estimation statement already published in a previous technical report prepared by SGS Geostat and titled: Tijirit Project NI-43-101 Technical Report with Resources Estimate Update dated April 10, 2018 and already available on SEDAR.

A list of the Qualified Persons (QP) under NI 43-101 which collaborated to this report is presented in Section 2.0. Algold Resources (Algold) may use this Technical Report to satisfy disclosure and filing requirements of Canadian securities regulators. The effective date of the Technical Report is March 9, 2018.

1.2 The Project

Algold wishes to exploit a gold deposit in the area of Tijirit located in the Wilaya of Inchiri and that of Dakhlet-Nouâdhibou in Mauritania. The Tijirit Mining Project consists of setting up mining infrastructure required for the development of an open-pit mine and the construction and operation of a plant for the treatment of the mineralized material with an average daily throughput capacity from 2,976 (Phase 1) to 4,500 (Phase 2) metric tonnes per day, to produce gold bullion. The mineralised material will be processed by a conventional gravity and cyanide leaching process.

The Project consists of the construction of a process plant and the establishment of related infrastructure such as mine services buildings and infrastructure, fuel tanks, and a tailings facility and the establishment of a camp for workers. The Project also includes the development of wells for industrial water supply for process.

1.3 Responsibility Matrix

This report is a synthesis of several technical studies undertaken by TIREX SA / Algold on the Mining Permit covering some 306 square kilometers. Various contributors have been mandated to carry out these studies so that it conforms to generally accepted mining industry practices.

- Resource estimation: SGS-Geostat
- Metallurgical Testing: SGS Lakefield
- Mining Operations, Mining Planning, Capex and Mining Opex: DRA/Met-Chem
- Development of Recovery Schemes, Infrastructure, Tailings Stockpile, Opex, Financial Analysis: Ausenco.

1.4 Property Description and Location

The Tijirit project is located in northwestern Mauritania, 260 km NNE of the capital Nouakchott and 200 km ESE (direct) of the major city of Nouâdhibou, located on the Atlantic coast. The project is comprised of the Tijirit mining concession 2480C2, covering 306 km², and of the adjacent exploration permit 1117B2, covering 460 km², in the administrative districts of Inchiri and Dakhlet Nouâdhibou Districts.

The world-class Tasiast deposit is located approximately 40 km north of Tijirit camp. The mineralized zones are oriented north to Tasiast and NNE to Tijirit. The mineralization is separated by the dune of the Azéfal and could be separated by regional faults or large folds. Tijirit mineralization data and their locations near Tasiast make it an area of high mineral potential.

Initially, the Tijirit property consisted of permits 447B2 and 1117B2. The mining property was originally owned by Shield Mining from 2007 to 2010 who sold the rights to Gryphon Minerals who in turn sold the rights to Algold (see Section 4).

The project is free of royalties to past owners. Since the modification of the Mining Code in 2014, a royalty (linked to the gold price) which varies from 4.0 to 6.5 % is payable to the RIM (Islamic Republic of Mauritania). An additional NSR royalty of 1.5 % has be granted to Osisko Royalties as of February 1, 2018.

1.5 History

The 447B2 exploration permit covered 1000 km². It was first granted to Shield Mining Mauritania SA under Decree 2007-200 by RIM (November 20, 2007), for a period of three years in the Guelb Enich - Inchiri area. It was renewed for the first time by decree 2010-256 (November 24, 2010). The permit was transferred to Gryphon Minerals Mauritania SA by decree 2013-876 (May 23, 2013). The permit was renewed a second time by Gryphon Minerals by decree 2014-036 (April 20, 2014). The 447B2 license expired after a period of three years in 2017. On June 12, 2017 Algold obtained from the Government of Mauritania a separate mining concession (2480C2) for a 306 km² portion of the 447B2 permit. The concession is valid for a period of 30 years.

Adjacent to the east of the mining lease, the 1117B2 exploration permit covers an area of 460 km². The permit was granted to Shield Mining Mauritania SA by RIM by Decree 2010-278 (December 13, 2010) for a period of three years in the Guelb Enich - Inchiri area. It was transferred to Gryphon Minerals Mauritania

SA by Decree 2013-877 (May 23, 2013). The permit was renewed for a period of three years by Decree 2014-121 (August 2014) for a period of three years. The license expires after a period of 9 years in 2020.

Since the discovery of gold anomalies in Tijirit soils between 1994 and 1996, extensive exploration by soil sampling, trenching, and core drilling completed by Shield Mining and Gryphon Minerals revealed gold mineralization. The gold is concentrated in surface areas of 1.5 x 4 km in 4 large separate areas, Sophie I and II, Sophie III, Lily, Eleonore and 3 less important areas located in the extensions.

Mineral Resource estimates were carried out by SGS in 2016, 2017 and 2018 with Technical reports filed on SEDAR.

1.6 Accessibility

The Tijirit property is accessible by a paved road 150 km from Nouakchott (south) and by a 130 km road north-east along the Akchar dune bar. Nouâdhibou is closer as the crow flies (200 km), but further away by road. Year-round access by all-terrain vehicle (4x4) is possible. The camp built on mining concession 2480C2 can accommodate more than one hundred (100) people and has offices and dormitories. It is equipped with two main-200 kVA electric generators and one 80 kVA. Fresh water is currently brought to site by means of a tanker truck from the Tijirit water drill hole located 80 km to the southwest of camp. The camp is also equipped for local communication, satellite phones and VSAT station for internet. Sample storage, a sample preparation laboratory operated by SGS and working facilities are also available on site.

1.7 Geological Context and Mineralization

The Tijirit gold mineralization is believed to be directly related to a north-north-east oriented deformation event where hydrothermal fluids were routed along these sulphides and silicification conduits, particularly:

- In contact between meta-igneous and metabasic rocks;
- In quartz-carbonate veins and stockworks in contact with metasediments and iron formation;
- In shear contacts within metasedimentary or metabasic units or;
- In large syntectonic quartz veins within sheared metabasic units.

Similar mineralization is found by extension of large prospect zones, but also along sheared contacts between large syntectonic plutons and the metasedimentary units to the east.

1.8 Exploration Work and Drilling

Since acquiring its operating license in 2007, Shield Mining (Shield) has conducted extensive exploration including a regional and detailed soil sampling study, trenching and some reverse circulation drilling. These led to the discovery of five prospective zones before Gryphon Minerals acquired them and proceeded with additional work totalling 37,703 m of RC drilling, 3,814 m of core drilling as well as a detailed study using magnetic and radiometric airborne surveys. The RC drilling program alone was able to detect more than one hundred mineralized intersections with significant gold values (greater than 2 m at 0.30 g / t Au).

Between April and June 2016, Algold discovered a network of quartz veins in trenches and pits in the Eleonore area that could possibly represent a large stockwork within a major shear zone that extends over 10 km towards north northeast. Algold has drilled to increase the resource base primarily in the Eleonore area. Since the beginning of May 2016, the Company has completed more than 50,000 metres of RC and core drilling.

From the beginning, a total of 92 Diamond Drill Holes (DDH) and 629 RC Drilling were completed and analysed on the Tijirit property for respective lengths of 10,706 and 80,337 metres.

1.9 Sample Preparation, Analyses, and Security

Algold developed a Standard Operating Procedures Manual (SOP) based on the CIM Best Practice Guidelines to provide a comprehensive guide to quality awareness, implementation and monitoring of control measures (QA/QC) during reverse circulation (RC) drilling. The procedures and techniques outlined in this protocol are considered to be in compliance with NI 43-101 standards, corroborated by SGS Canada and by Mr. André Ciesielski, D. Sc. Geo. of Algold.

The QA/QC results were reviewed by SGS Canada - and the results are deemed satisfactory to allow for the inclusion of this data in the resource estimate. SGS Canada, given the successful verification of the data and given that most items of the QA/QC are satisfactory, believes that the sample preparation, security and analytical procedures are adequate to support the estimation of resources presented in this technical report.

1.10 Data Verification

SGS Canada did the following to ascertain that the database supporting the estimation of resources is sound and reliable:

- Verification of the highest assays of the Algold 2017 data against analytical certificates;
- Site visit (August 14 to 18, 2017);
- Independent sampling;
- Multiple database verifications;
- Verification of bias for RC holes and trenches.

1.11 Metallurgical Testwork

To date, two series of metallurgical tests have been completed on the Tijirit project by Algold. The first series was completed on 4 composites in fall 2016. The second series on 16 composites started September 2017 for variability testing and was completed in February 2018.

A first series of tests were conducted at SGS-Lakefield in Canada in 2016 on composite samples of HQ drill core collected in the Eleonore, Sophie I, Sophie II and Lily zones. The composite from the Eleonore Zone was characterized by quartz veins associated with metasediments, whereas Sophie I's composite consisted of quartz veins within a banded iron formation (BIF). The composite of Sophie II consisted of a disintegrated ribbed iron formation (BIF) and that of Lily, of metasediments. Each of the composites weighed between 70 kg and 130 kg.

Following the metallurgical tests of 2016, additional tests to confirm the first results were carried out during 2017, again at SGS Lakefield facilities. Preliminary variability tests were carried out on the different zones.

From the samples taken, 16 composites were assembled for the variability tests. Subsequently, all the composites of the same zone were assembled to serve as feed for the CIL tests. The samples came from drill cores collected from Eleonore North, Central and South zones, Sophie II, and Lily.

Both series of CIL tests show that the Tijirit material responds well to the overall gravity and cyanidation process. In 2017 variability testwork performed, gold recovery using this flowsheet ranged from 91.2 to 99.4 % for an average of about 95 %. CIL and gravity testwork on the Master Composites reported gold recovery ranging from 95.1 % to 98.1 % and averaged about 96%.

1.12 Mineral Resource Estimation

The database contains 718 drillholes and 265 trenches with 76,297 assay results.

The SGS Genesis software was used for the modelling and estimation. Table 1-1 shows the base case resource with a cut-off grade of 0.4 g/t inside pits and 1.7 g/t below pits except for Eleonore at a global COG of 1.5 g/t.

A modelling cut-off grade of 0.3 g/t Au and minimum thickness of two metres along hole and a minimum accumulation of 1.2 m.g/t were used to delineate mineralised volumes. The 2,340 two-metre composites were capped at grades varying between 3.5 g/t Au and 45 g/t Au based on local extreme grades. Only 14 composites were capped. The gold loss is approximately 8 % for the resource. The density used for the estimation of the resource is 2.00 t/m^3 for saprolite and 2.7 t/m^3 for fresh rock in the Lily zone, 2.8 t/m^3 in the Sophie III zone, 2.85 t/m^3 in the Sophie II zone, 3.0 t/m^3 in the Sophie I zone and 2.86 t/m^3 in the Eleonore zone.

Table 1-1: Base Case Resources

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Eleonore	Indicated	4.08	719,000	94,250
Eleonore	Inferred	4.07	3,016,000	394,690

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Sophie/Lily	Measured	0.98	376,000	11,900
Sophie/Lily	Indicated	0.93	2,122,000	63,300
Total Sophie/Lily	Measured + Indicated	0.94	2,498,000	75,200
Sophie/Lily	Inferred	1.06	7,476,000	254,100

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Total Sophie/Lily/Eleonore	Measured	0.98	376,000	11,900
Total Sophie/Lily/Eleonore	Indicated	1.72	2,841,000	157,550
Total Sophie/Lily/Eleonore	Measured + Indicated	1.64	3,217,000	169,450
Total Sophie/Lily/Eleonore	Inferred	1.92	10,492,000	648,790

1. Effective date for Eleonore and Sophie/Lily resources is January 19, 2018.
2. The independent QP for this resources estimate is Yann Camus, P. Eng., SGS Canada Inc.
3. The mineral resources are presented at a 0.4 g/t Au cut-off grade in pits and 1.7 g/t Au cut-off grade under the pits, except Eleonore at a global cut-off 1.5 g/t Au.
4. The resources are presented without dilution.
5. Whittle pits have been utilized based on a gold value of US$1,500/oz.
6. Mineral resources that are not mineral reserves do not have demonstrated economic viability. This disclosure does not include economic analysis of the mineral resources.
7. Totals may not add up due to rounding.
8. No economic evaluation of the resources has been produced.
9. This Resource estimate has been prepared in accordance with CIM definition (2014).
10. Density used is between 2.0 and 3.0 depending on rock type and alteration based on measurements.
11. Capping varies from 3.5 g/t Au (Lily) to 45 g/t Au (Eleonore) depending on extreme local grade.

1.13 Mining Methods

The mining methods and In-pit Mineral Resource estimate for the Tijirit deposit were prepared by DRA/Met-Chem. All work related to Pit Optimization for the PEA was performed using Whittle software. MineSight was used for Pit design and Mine Planning. Both Whittle and MineSight are commercially available software.

The mining method selected for the Project is a conventional truck and shovel, drill and blast quarry operation. The mineralised material and waste rock will be mined at 5 m high benches, drilled, blasted and loaded into rigid quarry haul trucks with hydraulic excavators.

The mine plan was established annually for the first four (4) years of production at a rate of 2,976 t/d, followed by a four (4) year period at 4,500 t/d. The annual production target will be 1,086,240 tonnes for Phase 1 and 1,642,500 tonnes for Phase 2. (with 138,805 tonnes in Year 8).

Seventeen (17) pits were designed for the Project resulting in just over 7 years of production between Phase 1 and Phase 2.

Table 1-2 represents all mineralized content by pit for the Project.

Table 1-2: Mineralized Content per Pit (*)[1]

Zone		Waste (diluted)	Tonnes (diluted)	Grade (g/t)	Gold contained (oz)
Eleonore	E1	3,436,956	92,536	3.16	9,389
	E2	12,103,885	572,527	5.33	98,132
	E3	2,053,961	78,504	4.86	12,278
	E4	7,871,838	715,359	4.04	92,967
	E5	19,316,801	805,820	2.97	76,839
	E6	4,278,097	213,942	2.21	15,220
	E7	22,020,215	815,311	3.04	79,679
	E8	1,366,511	52,393	5.13	8,641
	E9	2,122,812	154,691	1.53	7,625
	E10	629,596	64,760	1.59	3,300
Lily	LILY	14,553,500	3,204,867	0.92	94,831
Sophie	S1A	2,948,156	395,152	2.06	26,118
	S1B	2,065,921	233,893	1.87	14,062
	S2	16,116,200	1,578,786	1.29	65,281
	S3A	223,399	41,928	0.77	1,039
	S3B	127,613	7,306	3.33	781
	S3C	679,599	111,971	1.13	4,078
TOTAL		**111,915,060**	**9,139,745**	**2.08**	**610,260**

The open pit designs include 9.14 M tonnes of measured, indicated and inferred mineral resources. In order to mine these resources, 111.92 M tonnes of waste material will need to be mined and stockpiled in waste dumps on the property.

1.14 Recovery Methods

The processing plant will have an average daily capacity of 2,976 t/d for the first 4 years of operation and will increase to 4,500 t/d thereafter to produce gold bullion. The treatment process that will be used will be a conventional gravity and cyanide leaching process. Gold production will range from more than 100 k ounces per year initially to around 50 k ounces per year enduring the final years of the mine life.

[1] (*) Cautionary statement NI 43-101: The PEA was prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"). Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

In the present study, the target grind size of the Eleonore deposit (Phase 1) is 105 µm with a plant recovery during Phase 1 of 96.0 % Au. For Phase 2 (Sophie and Lily) a grind size of 75 µm and a recovery of 93.6 % Au have been applied.

For now, the alternative of filtered dry stacked tailings has been chosen vs the use of a conventional tailings pond. The chosen process for tailings processing consists in pumping the thickened slurry to a filtration circuit for washing and filtering for water removal where the solids will then be transported to the tailings stockpile by truck. A bulldozer will be used to spread the tailings material similar to a waste rock dump. The base of the tailings stockpile area will not be covered with a geomembrane. Only preliminary tests were conducted on acid generation potential and they are not conclusive about the acid generation potential from the tailings. More tests will have to be conducted to evaluate the actual acid generation potential for each deposit.

The tailings stockpile area will be built in cells to facilitate its rehabilitation. Run-off water from the tailings stockpile area will be recycled for reuse in the process plant.

1.15 Project Infrastructure

New infrastructure will have to be built during the development of the Tijirit Mining Project, the main ones are:

- A new base camp;
- Water supply: Fresh Water will be brought by trucks and a 70 km pipeline will provide brackish water from well fields to the processing plant;
- Power generation (power line, substation);
- Processing plant;
- Development of access roads and haul roads;
- Development of the tailings stockpile area;
- Development of waste dumps;
- Construction of service infrastructure, including maintenance workshops and warehouses.

1.16 Manpower and Training

It is estimated that about 300 people (with peaks of up to 400-500 people) will be required during construction of the mine and that more than 400 people will be employed by Algold or various contractors when operating.

One of the main challenges of the Tijirit Mining Project will be finding qualified personnel within a relatively small pool of skilled workers. To counter this, Algold has planned, as soon as the development phases of the project begin, to develop a detailed training plan. This training plan will be carried out in collaboration with the Mauritanian technical and professional training centers and will train staff to be able to occupy various positions within the production, maintenance and management teams. It is also expected that during the years of operations, Algold will recruit trainers for the mine and processing plant who will provide ongoing training.

1.17 Environmental Studies, Permitting and Social and Community Impacts

In 2016, Algold commissioned AECOM to prepare the Tijirit Environmental and Social Impact Assessment (ESIA) study. As part of the public consultations, AECOM met with stakeholders to obtain more information and begin discussions with stakeholders. AECOM also held meetings with the employees of Algold as well as with the Wilayas of Inchiri and Dakhlet-Nouâdhibou of the districts of Chami Moughataa, Chami and Tmeimichat.

Algold submitted the ESIA study to the Mauritanian government at the end of 2016. In early 2017, the Minister of Environment and Sustainable Development of Mauritania issued a favourable opinion on the admissibility of the Environmental and Social Impact Assessment study.

1.18 Capital and Operating Costs

The total Capital costs (Capex) for the Project are estimated at $145.5M for the Pre-Production Period for Phase 1 and $16.3M to increase capacity before Phase 2. These include direct costs (mine, process, infrastructure, residuals, E, I & T, etc.), indirect costs (EPCM, construction, etc.), the tailings and other costs (contingency, etc.).

Capital costs (Initial Capex and Sustaining Costs) for mining operations, including mining equipment initial purchase and replacement, are valued at $45.1M over the life of the mine, this is including $6.0M for the pre-stripping of the mine.

The operating costs (Opex) of the mining operations are estimated with an accuracy of ±30 %. They are based on costs from mining suppliers or data from similar projects or operating under similar conditions. Operating costs for mining operations are $18.77 USD / t milled.

The operating costs (Opex) of the plant and the general costs are estimated with an accuracy of ±30 %. They are based on internal data, supplier costs or data from similar projects. The operating costs of the plant including energy costs are $13.84 USD / t milled. To that, G&A costs of $5.01 USD per t milled should be added.

1.19 Economic Analysis

The economic/financial assessment of the Tijirit Project of Algold Resources Ltd. is based on Q2-2018 price projections and cost estimates in U.S. currency. No provision was made for the effects of inflation. The evaluation was carried out on a 100 %-equity basis. Current tax regulations in Mauritania were applied to assess the project's state royalty liabilities. However, as it is unsure at this time whether the state will impose outright the mining code regulations associated with corporate income taxes, these were ignored for the purpose of this assessment.

The financial indicators under base case conditions are:

Table 1-3: Financial Model Indicators

Base Case Financial Results	Unit	Value
Pre-State Royalty (PSR) NPV @ 8 %	M USD	94.9
After-State Royalty (ASR) NPV @ 8 %	M USD	69.0
PSR IRR	%	28.4
ASR IRR	%	23.5
PSR Payback Period	years	1.6
ASR Payback Period	years	1.8

Figure 1-1 illustrates the after-state royalty cash flow and cumulative cash flow profiles of the Project for base case conditions.

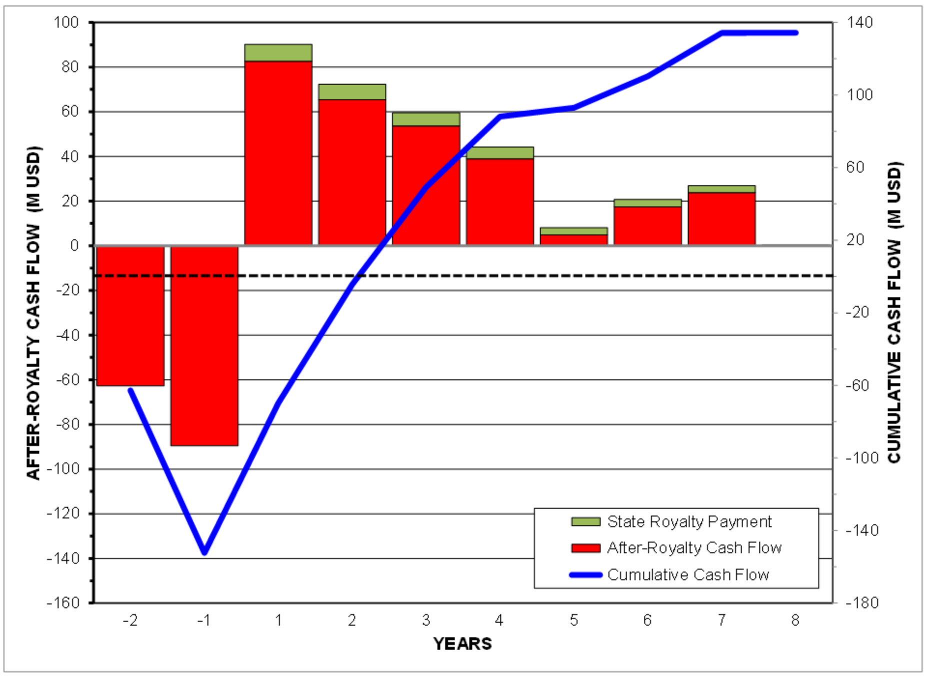


Figure 1-1: After-state Royalty Cash Flow and Cumulative Cash Flow Profiles

A sensitivity analysis reveals that the Project's after-state royalty viability is moderately vulnerable to variations in capital and operating costs within the margins of error associated with PEA-level estimates. However, the Project's viability remains more vulnerable to the larger uncertainty in future gold market prices.

1.20 Adjacent Properties

According to the 2017 RIM Exploration and Exploration License database, there are three exploration licenses adjacent to the Tijirit property. Two of the licenses belong to a local Mauritanian owner and one to Kinross through Tasiast Mauritania. The properties are located in the granitic-volcanic complexes of the southwestern Reguibat Archean Ridge and are partly covered by sand from the dune ridges of Azéfal to the northwest and Akchar southeast of Tijirit.

1.21 Interpretation and Conclusions

The Tijirit project is comprised of the Tijirit mining concession 2480C2, covering 306 km^2, and of the adjacent exploration permit 1117B2, covering 460 km^2. The project is located in western Mauritania, in the southwestern Precambrian Reguibat shield and shows mostly Archean granite greenstone terrains. The project shows sheared and folded NNE-trending Archean metasedimentary and intermediate to ultramafic volcanic rocks in contact with porphyry, syn-tectonic granitoids and basement quartzo-feldspathic gneiss. The project was acquired by Shield Mining in 2007 after the discovery of gold mineralization and transferred to Gryphon Minerals in 2010 and acquired by Algold in March 2016.

A fair amount of exploration works was carried out by past owners including a large soil survey, an airborne magnetic and radiometric survey, trenches, rock samples, auger drilling and 37,703 m of RC and 3,814 m of core drilling. Large gold soil anomalies were revealed in five prospect areas and a limited IP survey was done on the Sophie prospect.

Since the acquisition of the project, Algold has completed over 50,000 metres of core drilling and reverse circulation drilling in the Tijirit project. Algold has also prepared an environmental and social impact study that was submitted to the Mauritanian government in 2016. SGS Lakefield was commissioned to implement two metallurgical test programs in 2016 and 2017. In February 2017, the Minister Environment and Sustainable Development of Mauritania has agreed to the environmental feasibility of the Tijirit gold mining project.

To date, Algold has been able to substantially increase resources on the project, especially on Eleonore and to identify new promising areas (Salma, Eleonore East, Nour, Southeast). Efforts over the next few months will be required to better define these new areas to hopefully grow the overall resources and to increase drilling density on already known areas in order to transform the majority of existing inferred resources into indicated or measured resources.

Mining

The current Tijirit project results in an annual mining production of 1.1 Mt of ROM material per year at 3.3 g/t of gold during Phase 1 and approximately 1.6 Mt of ROM material per year at 1.1 g/t of gold during Phase 2. The total In-Pit Resources, spread between the Eleonore, Sophie and Lily zones, include 9.14 Mt of Mineralized Material grading 2.08 g/t of gold. The ROM material and Waste rock will be mined at 5 m high benches drilled, blasted and loaded into rigid quarry haul trucks with hydraulic excavators.

Mineral Processing

To date, two (2) metallurgical testwork campaigns have been performed on the Tijirit gold project. Both campaigns demonstrated that all deposits within the Tijirit property responded well to a traditional grinding – gravity concentration – cyanidation process. The process plant design criteria have been selected according to the metallurgical testwork results. The resulting process plant has been designed for a throughput of 1.09 Mt/y in Phase 1, with an expansion to 1.64 Mt/y in Phase 2. Gold recovery is expected to be 96.0% for Phase 1 and 93.6% for Phase 2, generating between 50,0000 and 120,000 oz of doré per year. Additional testwork should be performed in subsequent project phases to refine reagents and air consumption as well as thickening and filtration parameters.

Algold has also started a research program for water to supply the Project as well as a program of geotechnical studies to better define the parameters related to the stability of the slopes of the pits of the future mining operation.

There are no uncertainties or risks related to the information inspected by the authors other than standard risks in the mining industry like price drops and such.

Many studies and additional work will have to be carried out to validate and optimize the scenarios presented in this Report. The main steps to be completed before the start of production will be:

- Updating the resource estimation;
- Fieldwork;
- Geological, metallurgical, geotechnical and hydrogeological studies;
- Analysis of variants;
- Pre-Feasibility and Feasibility Studies;
- Updating the ESIA;
- Grant and fulfilment of an EPCM mandate;
- Basic Engineering (FEED) and early works;
- Construction;
- Pre-commissioning, Commissioning, Start-up, and Ramp up to full Production.

In conclusion, the work done to date has demonstrated the potential of the Tijirit Project and the preliminary economic analysis foresees positive outcome.

1.22 Recommendations

Given the positive economic results presented in this Report, Ausenco recommends that the Project be advanced to the next stage with the following recommendations.

1.22.1 Resources

The following development plan is recommended for the Project:

- Complete infill drilling to cover current resources. The aim is to increase the quantity of indicated resources (and possibly measured);
- Focus on in-pit resources first as they are closer to surface, probably easier to extract, and they require shorter drillholes so much less costly drilling;
- Explore new recent discoveries that are not at the resource development stage including mapping, structural study, additional soil survey, trenching, additional geophysics, hydrological and geotechnical studies, and preliminary drilling;

It is assumed that between 20,000 to 50,000 metres will be required for the next drilling phase.

1.22.2 Mining

The following items are recommended for consideration in the future studies:

- Consider the use of potential drill and blast of mining contractor in order to lower the initial capital requirement
- Perform a more detailed dilution study, especially for the Eleonore mineralized zone
- The Eleonore zone mineralization being open at depth, an underground operation could be considered if confirmed by further drilling

1.22.3 Geotechnical

The following recommendations are provided for consideration for future studies:

- It is recommended that a 3D geological and lithological model, including identification of pit size and regional structural features such as contact, faults and shears, be completed prior to future geotechnical studies.
- It is recommended that the assumptions for the structural regime and orientation of structures be verified with future geotechnical data collection. This could include line mapping of trenching to validate the orientations of foliation and joint discontinuities, as well as to assess any additional structural discontinuities (faults/shears). Due to the lack of this data, there is uncertainty in the 3D geotechnical model and associated pit design domains.
- It is recommended that rock strength data collection be completed for direct shear, compressive strength and triaxial strength tests to add to the geotechnical dataset. Rock strength testing should also collect data associated with density of various lithologies. Rock strength, cohesion and friction angles for all rock types were assumed for this study based on empirical studies and field descriptions.
- For future data collection purposes, detailed descriptions of all discontinuity conditions (veins, foliation, fractures and faults/shears) including spacing of defects, thickness of alteration should be detailed.
- Overburden soils logging and collection including in situ densities, sieve tests, and depth should be completed.
- Collection of fracture frequency and spacing is required to provide a better understanding of the quantity and variability of discontinuities.
- Location and size of any waste dumps associated with the planned pits should be included in future stability assessments to determine the effect of surface loading in proximity to the pits.
- Continued collection of hydrogeological data, including recording lack of water presence in drillholes to fully characterize expected hydrogeological conditions should be completed.

Stability of the final bench faces can be improved using controlled blasting techniques, scaling of loose rock from the bench faces, and removal of loose on benches after blasting. This design has an operational constraint which includes controlled blasting considerations.

Extensive monitoring of pit slopes and ongoing commitment to data collection throughout the life of the Project is required to ensure design appropriateness and to validate assumed design parameters.

A more detailed geological understanding of the location and continuity of weak or altered zones, including weathered zones within the pit are required to fully assess the pit stability. Further drilling, as well as Televiewer (geocamera) studies or oriented core collection and geological interpretation are required.

1.22.4 Processing Plant and Process

The development of the Tijirit process plant at this phase of development has led to several recommendations for project improvements in subsequent phases. A comparative study related to the grinding circuit was carried out by Ausenco. This study considered three different grinding circuits, for the two phases of the project:

- A conventional circuit that was selected for this phase of the project and which includes a semi-autogenous mill with the addition of a ball mill for the second production stage.
- An alternate circuit that included a ball mill with the addition of a high pressure grinding rolls press to increase mill capacity in the second production phase.
- A modified alternative circuit that included a smaller SAG mill than the reciprocating circuit for Phase 1, and the addition of a high pressure grinding rolls and a second ball mill for Phase 2.

The conclusion of this study was to maintain the conventional circuit at this stage of the Project but to continue to explore alternative scenarios in the next steps of the project aiming for greater cost accuracy. In fact, the net present values of the three scenarios proposed are very similar at this study level, slightly higher in the case of the alternative scenario and the modified alternative scenario. Given the accuracy of the current study, it is not possible to conclude that one scenario is more economically advantageous than the other.

Ausenco believes that using a high-pressure grinding rolls could have several benefits for the project, including:

- A possible reduction in plant water consumption in Phase 2, HPGRs operating almost entirely dry.
- Few modifications to Phase 1 facilities would be required to proceed to Phase 2 as opposed to the conventional scenario. This would reduce the transition time between the two phases which could be important for a short-term project like this one.

In order to better evaluate this technology, it will be necessary to test this equipment to determine the capacity and specific energy requirements for this equipment processing Tijirit ore.

The metallurgical tests performed at SGS, and the development of the process by Ausenco, have also led to several other process recommendations:

- The addition of a flotation circuit on the tailings discharge of the gravity circuit. Preliminary tests for this purpose have already been carried out on Eleonore material and have demonstrated recoveries above 90%. More testing of all project zones should be undertaken to better understand the potential of this process for the project.
- Additional metallurgical tests at the CIL circuit in order to optimize the residence time of this process, determine the optimal oxygen and reagent consumptions of this circuit.
- The preparation of a comparative study on the use of thickeners in the process. For the moment, a tailings thickener has been included in the process. A thickener could also be used upstream of the CIL circuit (Pre-Leach Thickener) which could reduce the size of the CIL tanks.

- Conduct sedimentation and filtration tests to optimise sizing of the process thickeners and tailings disc filters. These facilities have been sized on the basis of similar projects.
- Samples for future test programs should be based on the major lithological units of each deposit as defined in the resource and block models. A review of the geology and mineralization of each lithological unit should be conducted to determine the range of variability within each unit to select possible variability samples.

1.22.5 Tailings Stockpile and Waste Dumps

The following are recommendations for the management strategies to be reviewed and the investigations to complete for the tailings storage facility:

- Review the location of the tailings site in light of dominant winds and consider relocating it to reduce exposure of mine personnel to fugitive dust.
- Conduct geotechnical and borrow pit investigation at the future tailings and waste rock sites to determine suitability of foundation ground and locate potential construction materials
- Determine maximum design flood for runoff collection and channel sizing.
- Proctor testing on filtered tailings to obtain optimum bulk density and maximum density at filtered moisture contents.
- Review fugitive dust management and crust/hard-pan forming behaviour of filtered tailings
- ARD screening: Comprehensive static geochemical tests (NAG, some sequential NAG, modified Sobek ABA,) on tailings and waste rock material using a suitable number of representative samples selected by an environmental engineer or geologist.
- Revisit the requirements for membranes and seepage drainage below the waste rock and tailings piles following review of geochemical results.

1.22.6 Proposed Work Program Budget

The expected cost to complete the above-mentioned exploration, drilling and sites costs, and metallurgical tests, is between $5 and $8.3MUS; a high-level budget recommendation is outlined in Table 1-4.

Table 1-4: Recommended Budget

Description	$US
Drilling Costs (additional infill program): Total of between 20,000 and 50,000 m of RC and DDH drilling with assays and logging (75 % RC, 25 % DDH). This drilling should be enough to cover current resources with infill drilling. This will help for future technical/economical studies.	2,200,000 to 5,500,000
Site Costs: Including camp costs, salaries and transportation costs	1,000,000
Metallurgical Tests: Heap Leach amenability test Gravimetric and cyanidation tests Comminution tests Environmental Testwork	500,000
Pre-Feasibility Study (PFS) including: Resources Update Process engineering Mine engineering	500,000
Other Suggested Budgets: Mapping, structural study, additional soil survey, trenching, additional geophysics, hydrological and geotechnical study	800,000
TOTAL (varies depending on the infill drilling required to delineate enough resources to an indicated or measured resources)	5,000,000 to 8,300,000

2. INTRODUCTION AND TERMS OF REFERENCE

2.1 General

Algold (ALG - TSX.V) is a Canadian junior company focusing on exploration and development of gold deposits in West Africa.

TIREX SA, a public limited company under Mauritanian law regularly incorporated and registered in the Nouakchott trade register on April 27, 2016 under the numbers 91408 / GU / 12417 (analytic register) and 1994 (chronological register) is a subsidiary of the Algold Resources Company Ltd. ("Algold"), a Canadian company listed on the Toronto Stock Exchange in Canada.

Algold, on behalf of TIREX SA, commissioned Ausenco Engineering Canada (Ausenco), along with DRA /Met-Chem, WSP Canada Inc (WSP) and SGS Geostat to prepare a Preliminary Economic Assessment (PEA) of the Tijirit Project in Mauritania.

The Tijirit Project includes the 1117B2 exploration permit and the 306 km^2 2480C2 mining concession granted to TIREX by the Mauritanian government on June 12, 2017 for a period of 30 years. This concession gives TIREX SA exploration and exploitation rights for the gold and related substances covered by this permit. The PEA and the resources estimate only refer to the mining concession 2480C2.

2.2 Terms of Reference

The PEA report was prepared according to National Instrument 43-101 guidelines for mineral deposit disclosure and describes historic works, mineralization types and mineral potential of the project. Recommendations are presented for further exploration works.

Information for this Technical Report has been gathered from several geologists, government reports, and independent technical reports. The PEA is also based on resources estimation statement already published in a previous technical report prepared for Algold by SGS Geostat that was titled: Tijirit Project NI-43-101 Technical Report with Resource Estimate Update dated April 10, 2018 and already available on SEDAR.

Algold Resources may use this Technical Report to satisfy disclosure and filing requirements of Canadian securities regulators. The effective date of the PEA Report is March 9, 2018.

2.3 Scope of the Study

This Report entitled: Preliminary Economic Assessment of the Tijirit Project is a synthesis of several Technical Studies undertaken by TIREX SA / Algold on the Mining Permit covering some 300 square kilometres. Various contributors have been mandated to carry out this study so that it conforms to generally accepted mining industry practices

- Resource estimation: SGS-Geostat;
- Metallurgical Testing: SGS Lakefield;
- Mining Operations, Mining Planning, Capex and Mining Opex: DRA/Met-Chem;
- Development of Recovery Schemes, Infrastructure, Tailings Stockpile, Opex, Financial Analysis: Ausenco.

2.4 Qualified Persons

The responsibilities for the preparation of the different sections of this Report are shown in Table 2-1.

Table 2-1: Responsibility Matrix by Contributing Organization

Section	Title of Section	Qualified Persons
1	Executive Summary	Thomas Zwirz and related QPs
2	Introduction	Thomas Zwirz (Ausenco)
3	Reliance on Other Experts	Thomas Zwirz
4	Property Description and Location	Yann Camus (SGS Geostat)
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Yann Camus
6	History	Yann Camus
7	Geological Setting and Mineralization	Yann Camus
8	Deposit Types	Yann Camus
9	Exploration	Yann Camus
10	Drilling	Yann Camus
11	Sample Preparation, Analysis and Security	Yann Camus
12	Data Verification	Yann Camus
13	Mineral Processing and Metallurgical Testing	Robert Raponi (Ausenco)
14	Mineral Resources Estimates	Yann Camus
15	Mineral Reserve Estimates	N/A
16	Mining Methods	Patrick Perez (DRA/Met-Chem) / Jean-Sébastien Houle (WSP -16.2.2 only)
17	Recovery Methods	Robert Raponi
18	Project Infrastructure	Thomas Zwirz David Sims (Ausenco – 18.5)
19	Market Studies and Contracts	Thomas Zwirz
20	Environmental Studies, Permitting and Social or Community Impact	Thomas Zwirz
21	Capital and Operating Costs	Capex - Martin St. Amour (DRA/Met-Chem) Opex Mining (21.2.3) Patrick Perez Opex for other areas except Mining – (21.2.4) R. Raponi / T. Zwirz
22	Economic Analysis	Thomas Zwirz
23	Adjacent Properties	Yann Camus
24	Other Relevant Data and Information	N/A
25	Interpretation and Conclusions	Thomas Zwirz and related QPs
26	Recommendations	Thomas Zwirz and related QPs

Section	Title of Section	Qualified Persons
27	References	Thomas Zwirz and related QPs

The entire project team has reporting responsibility for Section 1 (Executive Summary) and Sections 25 and 26 (Interpretations and Conclusions, Recommendations).

2.5 Site Visits

Yann Camus, P. Eng. (SGS Canada–Geostat) travelled to the Tijirit project on August 14 to 18, 2017, visited trench and drill sites, looked and photographed core mineralized intersections, inspected drill core and reject facilities and warehouse.

Elie Acad. P. Eng. (Ausenco) travelled to Tijirit on October 14, 15, and 16, 2017 to visit the trenches, potential camp sites, to assess the locations for suitability for the process plant and infrastructure to support the mining and mineral processing. Information gathered during the site visit was incorporated into the PEA. This information was shared with current Ausenco QP Thomas Zwirz

Thomas Zwirz, P. Eng. (Ausenco) responsible for leading Ausenco report chapters did not visit the site.

Robert Raponi, P. Eng. (Ausenco) qualified person for Section 13 Mineral Processing and metallurgical Testing and Section 17 Recovery Methods did not visit the site.

Mr. Jean-Sébastien Houle, a mining engineer with WSP, visited the Tijirit property from November 21 to 24, 2017. During the visit, Mr. Houle compared the structural and geotechnical logs of selected boreholes for each of the proposed pits. He consulted with the on-site geological team assess methods for geotechnical description and data collection with industry standards and best practices.

Mr. Houle also visited the artisanal mining and trenching sites in the areas of the mineralized deposits and potential future pits: north of Eleonore (Salma), Eleonore, Sophie, and Lily, and south-west of Lily. He was accompanied by Mr. Dia Ibrahim, a geologist with Algold. This visit allowed for further assessment of the bedrock weathering profile according to the various lithologies encountered in several sectors of the property.

2.6 Currency, Units, Abbreviations, and Definitions

Units of measurement used in this report conform to the SI (metric) system. All currency in this report is US dollars ($US) unless otherwise noted. The coordinate system used in this report is the WGS84 ellipsoid, UTM zone 28, Northern Hemisphere. Abbreviations used in this Report are in Section 28.

2.7 Disclaimer

This report was prepared as a NI 43-101 Technical Report for Algold Resources Ltd. (Algold). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Ausenco Engineering Canada (Ausenco) services, based on: 1) information available at the time of preparation, 2) data supplied by outside sources, 3) assumptions, conditions, and qualifications set forth in this report. This document and its contents are for the private information and benefit only of Algold, for whom it was prepared and for the particular purpose which Algold described to Ausenco. The contract permits Algold to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. The contents of this document are not to be reused in whole or in part, by or for the benefit of others without prior adaptation by, and the prior specific written permission of, Ausenco.

Particular financial and other projections, analysis and conclusions set out in this document, to the extent they are based on assumptions or concern future events and circumstances over which Ausenco has no control are by their nature uncertain and are to be treated accordingly. Ausenco makes no warranty regarding any of these projections, analysis and conclusions. Ausenco, its affiliates and subsidiaries and their respective officers, directors, employees and agents assume no responsibility for reliance on this document or on any of its contents by any party other than Algold.

The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

3. RELIANCE ON OTHER EXPERTS

Messrs. Yann Camus, P. Eng., Thomas Zwirz, P. Eng, Martin Saint-Amour, P. Eng, or Patrick Perez, P. Eng are not qualified to comment on issues related to legal agreements, royalties, permitting, and environmental matters. The authors have relied upon the representations and documentations supplied by the Company management. The authors have reviewed the mining titles, their status, the legal agreement and technical data supplied by Algold, and any public sources of relevant technical information.

The authors also rely on the expertise of Mr. David Vilder, M. Env., CSR Advisor for Algold, for the information contained in Section 20 of this Technical Report.

The economic analysis is based on a gold price of $1,250 US per ounce which reflect Algold outlook of the future market for gold.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Tijirit gold project is located in northwestern Mauritania, 260 km NNE of the capital Nouakchott and 200 km ESE (direct) of the major city of Nouâdhibou, located on the coast. The project comprises the Tijirit mining concession 2480C2, covering 306 km^2, and the adjacent exploration permit 1117B2, covering 460 km^2, in the administrative districts of Inchiri and Dakhlet Nouâdhibou Districts, Figure 4-1 and Figure 5-1. Both properties are governed by two different Mining Conventions (“Convention Minière”). All resources are on the mining licence 2480C2.

The two permits of the project are limited by the following UTM coordinates (UTM zone 28N, datum WGS84), Table 4-1.

Table 4-1: Tijirit Project Coordinates in UTM

Id	East	North
2480C2		
1	472,000	2,242,000
2	489,000	2,242,000
3	489,000	2,260,000
4	472,000	2,260,000
1117B2		
1	491,000	2,270,000
2	500,000	2,270,000
3	500,000	2,225,000
4	480,000	2,225,000
5	480,000	2,230,000
6	491,000	2,230,000
UTM zone 28N, datum : WGS84		

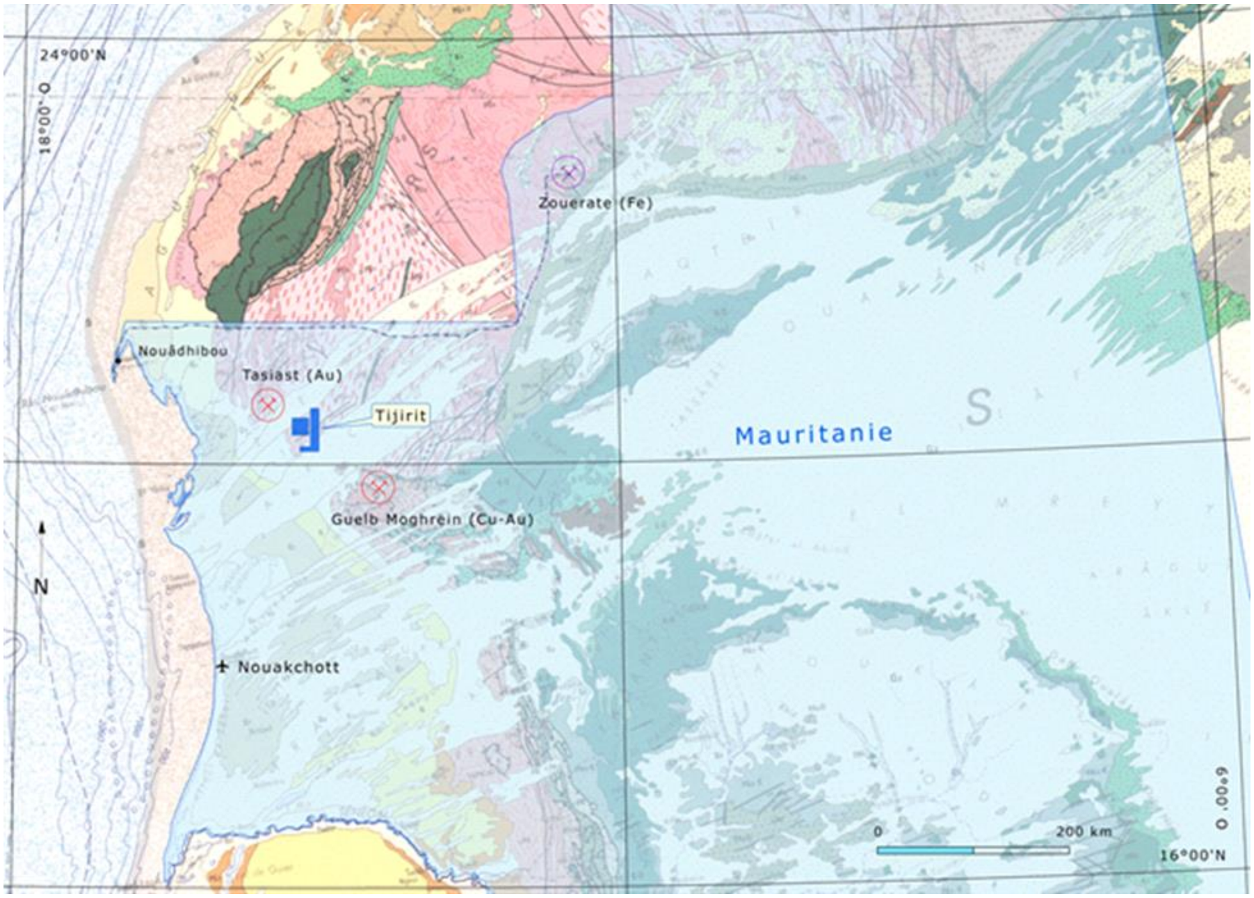


Figure 4-1: Geological Map of West Africa (1:5,000,000) with Location of Tijirit Properties in relation to the Capital Nouakchott and Other Major Mines in RIM. (after UNESCO Geological Map 1988)

4.2 Exploration Rights

Tirex holds the rights to the Tijirit project which consists of an exploration license (1117B2) and a mining concession (2480C2) as detailed in Item 4.1. The Tijirit-East exploration permit 1117B2 is adjacent to the east to the mining concession and covers 460 km^2. The exploration permit expires after a 6-year period in 2020. Since June 12, 2017, a portion of 306 km^2 of the original exploration permit 447B2 has been converted into a mining concession valid for 30 years.

4.3 Ownership, Royalties and Agreements

Algold has exercised the option (Press Release of March 11, 2016) to acquire the properties from Gryphon Minerals. The Tijirit project was one of three properties acquired by Algold in Mauritania from Gryphon. The details of the acquisition can be found in press releases from Algold dated October 28, 2015, February 12 and March 11, 2016.

4.3.1 Ownership

Tirex is 100 % owner of the Project.

4.3.2 Royalties

The Project is free of royalties to past owners.

Since the modification of the Mining Code in 2014, a royalty (linked to the gold price) which varies from 4.0 to 6.5 % is payable to the RIM (Islamic Replublic of Mauritania), according to the following gold price levels: An additional NSR royalty of 1.5 % has be granted to Osisko Royalties as of February 1, 2018.

- below 1,000 USD per ounce: 4.0 %;
- between 1,000 and 1,200 USD per ounce: 4.5 %;
- between 1,200 and 1,400 USD per ounce: 5.0 %;
- between 1,400 and 1,600 USD per ounce: 5.5 %;
- between 1,600 and 1,800 USD per ounce: 6.0 %;
- above 1,800 USD per ounce: 6.5 %.

4.4 Permits and Environmental Liabilities

To the knowledge of the author, the Tijirit project is only subjected to the environmental guidelines of the mining code of Mauritania.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Physiography

The topography of the Tijirit project area consists mainly of flat, barren plains which are primarily covered by regolith and locally by sand dunes, or eroded paleo-lateritic profiles. It is located between the Azefâl and Akchâr main dune bars oriented to the northeast. The average elevation is approximately 130 masl to 150 masl.

The area has no permanent watercourse but is crossed by numerous, intermittent watercourses, known as "wadis", which flow for only a few days per year during rare high rain falls.

The Project is located in the arid Saharan zone, the predominant ecological area in Mauritania, where flora is very scarce and is mainly colonized by Aghaya (Zygophyllum album), together with Aghar (Maerua crassifolia, atil) and Drinn (Aristida pungens, sbot). Acacias are also present along many of the wadis. There are no forests in the area.

Small rodents (such as hares, hamsters and gerbils) are the most common mammals in the area, while jackals, fennec fox and zorille fox have been observed in the area.

5.2 Accessibility

The Tijirit project is accessed from Nouakchott on a paved highway for 150 km to the north and via a dirt track for 130 km to the northeast. Nouâdhibou is closer, 200 km direct, but much farther by road.

The main ports of entry for goods and consumables are either Nouakchott or Nouâdhibou. Materials are transported by road to the various sites.

Access within Mauritania is provided by a road network, of approximately 3,000 km of paved highways and approximately 8,000 km of unpaved highways as well as numerous desert tracks.

A paved 470 km long, two-lane highway runs between Nouakchott and Nouâdhibou, 425 km to the north as shown in Figure 5-1.

A 717-km long railway located along the border between Mauritania and Western Sahara is owned and operated by Société Nationale Industrielle et Minière de Mauritanie (SNIM). It is primarily used to haul iron ore from SNIM's iron ore mine at Zouérate to the port of Nouâdhibou.

Access to the major urban centers of Mauritania is also possible via air. Nouakchott is accessible via international flights operated by numerous West and North African carriers; Air France also provides a direct connection to Paris.

Figure 5-1: Political Map of Mauritania with Administrative Regions

5.3 Climate

Mauritania’s climate is classified as arid desert (under the Köppen climate scale), with average annual high temperature above 45°C between May and August. Minimum temperatures may go below 10°C in December and January. Sandstorms frequently occur from January to March causing sand build up and dune formation. Sandstorms do vary in intensity and visibility can be reduced to several metres.

A rainy season, usually between July and September, does exist; however, the amount of rainfall and length of season varies spatially and temporally in the various regions of the country. Annual rainfall varies from a few millimetres in the desert regions to highs of 450 mm in the south along the Senegal River. During the last 30 years, the country has recorded two periods of drought, namely 1984-85 and 1991-92.

Mauritania is located along the northwestern coast of Africa and is bordered by the Atlantic Ocean to the west. The country's land mass covers the western portion of the Sahara Desert. Mauritania's land mass consists mainly of flat and barren desert landscape surfaces that are cross cut by three large NE-SW trending longitudinal dune fields. In the central part of the country, near Adrar and Tagant, several hills and mountains rise up to 915 metres above sea level (masl). In the desert regions, vegetation is sparse, consisting of various species of thorny trees (acacia, etc.) and grasses.

5.4 Local Resources and Infrastructure

The Tijirit project is located in a remote area deprived of resources and infrastructure. The nearest major center is located in Nouakchott, a distance of 300 km to the southwest by road. All supplies to the exploration works have to be transported to site.

A camp has been built on the mining concession 2480C2 that can fully accommodate around 100 persons, with offices and dormitories. It is equipped with two (2) main 200 kva and one (1) 80 kva electrical power generators. Fresh water is obtained from the Tijirit spring 80 km to the southwest and transported to site by tank truck. The camp is also equipped for local communication, satellite phones and VSAT station for internet. Samples storage, a sample preparation laboratory operated by SGS, and working facilities are also available at site.

Current third-party land use in the area consists of occasional nomadic camel and sheep farmers. There are no villages or agricultural farms activity within or around the project area.

As of April 2016, a fair number of artisanal miners have started to congregate on the Eleonore zone of the Project. This activity was considered illegal. As of May 2016, following modifications to the mining code, the government issued many artisanal miner's licences. However, these licences were all approved outside of the Project area, thus all artisanal mining activity on the Project is illegal. As a matter of good social corporate responsibility, Algold decided to not interfere and simply monitor the artisanal work for the time being.

5.5 Surface Rights

To the knowledge of the authors, the Project is not subjected to any surface rights. To the knowledge of the authors, there are no significant factors or risks that may affect the access to the properties, the mining titles and the ability to perform exploration works on the properties, including Tijirit.

6. HISTORY

6.1 Prior Ownership of the Property and Ownership Changes

Initially, the Project consisting of permit 447B2 and 1117B2 originally belonged to Shield Mining from 2007 to 2010 who sold the rights to Gryphon who in turn sold the rights to Algold Resources (see Section 4).

Originally, the Tijirit exploration permit 447B2 covered 1000 km^2. It was first granted to Shield Mining Mauritania SA under RIM decree 2007-200 of November 20^t, 2007 for three-year period in the Guelb Enich - Inchiri area. It was renewed first time under decree 2010-256 of November 24, 2010. The permit was transferred to Gryphon Minerals Mauritania SA under decree 2013-876 of May 23, 2013. The permit was renewed second time to Gryphon under decree 2014-036 of April 20, 2014. Permit 447B2 expired after a three-year period in 2017. On June 12, 2017, Algold obtained from the Government of Mauritania a separate mining concession (2480C2) for a 306 km^2 portion of the 447B2 concession. The permit is valid for a period of 30 years.

The Tijirit-East exploration permit (1117B2) is adjacent to the east to permit 447B2 and covers 460 km^2. The permit was first granted to Shield Mining Mauritania SA under RIM decree 2010-278 on December 13, 2010 for three-year period in the Guelb-Enich - Inchiri area. It was transferred to Gryphon Minerals Mauritania SA under decree 2013-877 on May 23, 2013. The permit was first renewed for a three-year period under decree 2014-121 on August 2014 for three-year period. The permit expires after a 6-year period in 2020.

6.2 Regional Geological Studies

The Project area is located in the southwestern part of the Reguibat shield, a large north-trending Archean and Proterozoic ridge that occupy most of northern Mauritania, Figure 6-1. In the second half of the 20th century, a number of scholars and institution geologists carried out various studies and mapping resulting in the publication of scientific papers and maps of northern Mauritania (BRGM, 1968 and Choubert et Faure-Muret, 1971). Later doctoral dissertations on the metamorphic evolution of the southern Reguibat craton were produced and geological and metallogenic regional studies published (Potrel, 1994; Fabre, 2005, El Hadji, 2002).

Between 1962 and 1993, BRGM (Bureau de Recherches Géologiques et Minières) and later SNIM (Société Nationale Industrielle et Minière) carried out various exploration programs in northwestern Mauritania.

- Mission Pegmatite: 1960-1962 - The mission targeted Be and Li located in the Tijirit pegmatite intrusive rocks, located north of the Tasiast mine.
- Mission nickel sulfuré: 1972 - The mission focused on ultramafic units located in the Tasiast and Tijirit areas.
- Mission Fer: 1973-1975 - SNIM targeted the Lebzénia Hills and Aouéouat greenstone belt.

BRGM and OMRG carried out prospecting and geological mapping in NW Mauritania Tasiast-Tijirit domain from 1992 to 1996 and produced the first 1:200 000 scale compilation of the Chami sheet. Later a wider geological compilation project lead by the BGS and funded by the World Bank produced a series of 1:200,000, 1:500,000 scale maps and reports on the geology and the metallogeny of the southern

Mauritania including the SW Reguibat Shield and the Mauritanides (BGS 2004a and b) as shown on Figure 6-1.

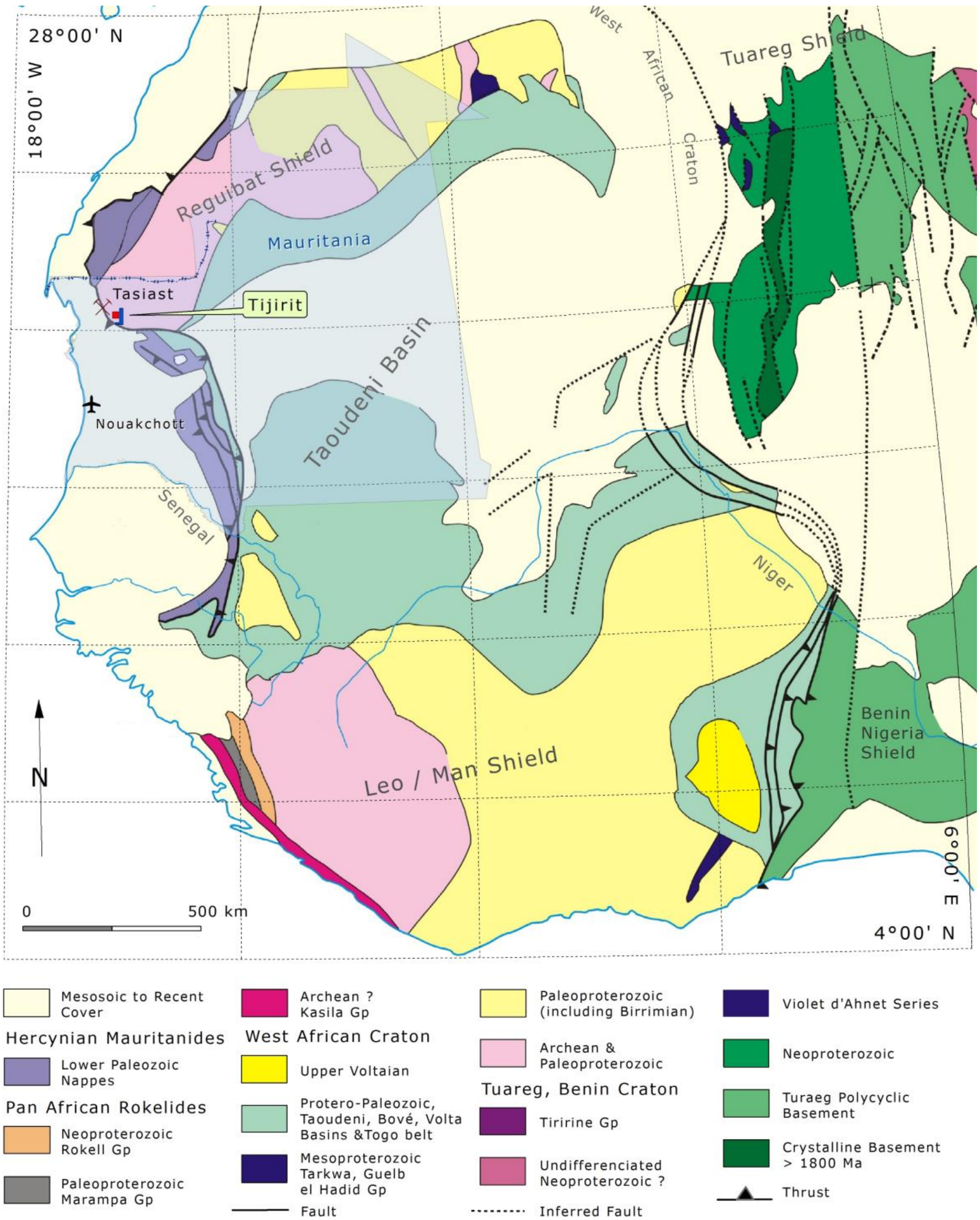


Figure 6-1: Geology Sketch of West Africa showing the Location of the Tijirit Project Southwest of the Reguibat Shield (After BGS 2004a)

6.3 Regional Geochemistry

Regional regolith, outcrop and soil sampling was conducted during the 1993-1996 BRGM-OMRG program and by the PRISM compilation program in the 2000’s on the Chami map sheet. The samples were taken every kilometre along E-W lines separated by 4 or 5 km. A total of 1187 samples were taken on the Chami map sheet. Results have been mapped in the BGS (2004b) report on mineral potential of northern Mauritania.

6.4 Regional Geophysics

In Mauritania, regional airborne magnetic and radiometric surveys were funded by United Nations and by PRISM in the years 2000. Fugro and Sanders carried out the surveys at 500 m line spacing during PRISM I. The data was merged with older UN data to produce a coherent database. The data excludes most of the Taoudeni Basin. A gravity database also exists for Mauritania that excludes the Taoudeni Basin.

Regional 1:500 000 and 1:200 000 scale aeromagnetic and radiometric maps are available for NW Mauritania. Geophysical data accompany the 1:200 000 scale geological compilations carried out by BGS (20004a and b). Figure 6-2 shows the Chami sheet (#2015 in the 1:200,000 map grid of Mauritania) thorium signal, the major geological contacts and the Tijirit project area. The Tasiast mine units are trending to the NNW and show various volcanic and iron-rich sedimentary sequences, marked by a strong magnetic signal. The Project units are trending to the NNE and composed of mafic and ultramafic volcanics, metasediments, iron formations and meta-igneous rocks that give a higher Th signal on Figure 6-2.

Volcanic and sedimentary rocks are contrasting with much higher Th concentration granitoids (in red). It also contrasts with lower signal in quartz-feldspathic gneisses to the east and west of the map. The Tasiast mine is located across a major NNW-trending metasedimentary zone.

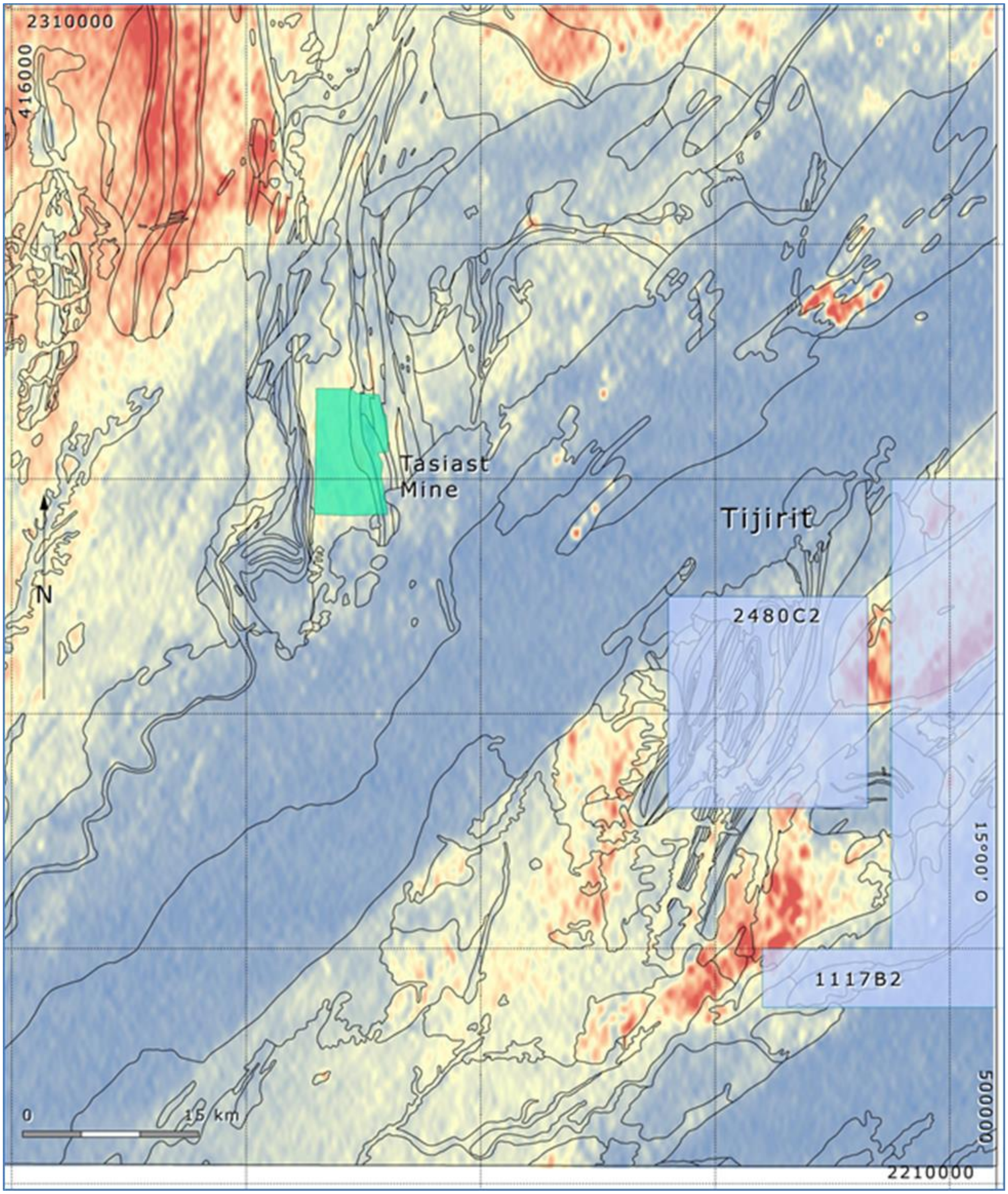


Figure 6-2: Radiometric Thorium Signal over the 1:200,000 Chami Map Sheet (#2015)

6.5 Previous Estimates

A previous estimate was carried out by SGS in 2017 with a technical report entitled "Tijirit Property NI 43-101 Technical Report with Resource Estimate Update, Tijirit, Mauritania" dated June 15, 2017. The previous estimate had two effective dates: March 17, 2017 for Eleonore zone and November 4, 2016 for Sophie and Lily zones.

Table 6-1 shows the previous base case estimates with a cut-off grade of 0.4 g/t inside pits and 1.4 g/t below pits except for Eleonore at a global COG of 1.5 g/t.

Table 6-1: Previous Estimate of June 2017 (Base Case)

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces
Sophie/Lily	Measured	1.03	73,000	2,420
Sophie/Lily	Indicated	1.04	1,226,000	41,010
Total	**M+Ind**	**1.04**	**1,299,000**	**43,430**
Sophie/Lily	**Inferred**	**1.37**	**5,528,000**	**244,210**

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces
Eleonore	**Inferred**	**4.18**	**2,665,000**	**357,920**

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces
Total Eleonore, Sophie/Lily	**Inferred**	**2.29**	**8,193,000**	**602,130**

1. Effective dates for the resource is March 17, 2017 for Eleonore and November 4, 2016 for Sophie and Lily zones
2. The Independant QP for this Resources Statement is Yann Camus, ing at SGS Canada Inc
3. The mineral resources are presented at a 0.4 g/t Au COG in pits - 1.4 g/t Au COG under the pits except Eleonore at a global COG of 1.5 g/t Au
4. The resources are presented without dilution
5. Whittle pits have been used using a gold value of 1500 $/oz
6. Mineral resources that are not mineral reserves have not demonstrated economic viability
7. Total may not correspond due to rounding
8. No economic evaluation of the resources have been produce
9. This resources estimates have been prepared according to CIM definition (2014)
10. Density used are between 2.0 and 3.0 depending on rock type and alteration based on measurements
11. Capping varies from 3.5 g/t to 45 g/t Au depending on extreme local grade

6.6 Past Production

A fair number of artisanal miners have started to congregate on the Project Apart from these activities, there is no known past production on this Project.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project area lies in the southwestern portion of the Reguibat craton mostly underlain by Precambrian crystalline rocks as shown in Figure 7-1. The craton forms a northeast-trending ridge limited to the NW by the Tindouf basin and to the SE by the Taoudeni basin. The southwestern half of Reguibat is underlain by rocks of Mesoarchean age (~2.9 to ~3.1 Ga), whereas the northeastern half are of Paleoproterozoic ages (~2.5 Ga).

The basement is mostly composed of large granitic intrusions and moderate to high-grade metamorphic rocks including gneisses, amphibolites, volcanogenic schists and gneisses and metasediments. Metamorphic grade in the Reguibat generally increases from mid-greenschist in the southwest through to granulite grade in the northeast.

Numerous north-trending elongate volcano-sedimentary (v-s) belts occur over a broad area of the southwestern Reguibat, each marking major shear and crustal discontinuities. These belts are composed of metavolcanic, meta-plutonic and metasedimentary rocks ranging from ultramafic to felsic composition and likely represent the roots of volcanic arcs and inverted elongated sedimentary basins, (Figure 7-1).

Most of the v-s belts are trending north and in contact with granitic to gabbroic intrusive rocks and gneissic suites. In the east, some belts show arcuate shapes possibly related to regional folding or small scale doming tectonic. The v-s belts consist of ultramafic to intermediate and felsic volcanic and volcanogenic sedimentary sequences with variably preserved iron formations and iron and quartz-rich metasediments as shown in Table 7-1.

Within the belts, rock successions have undergone mid greenschist to lower amphibolite grade metamorphism and multiple deformation events. Swarms of non-foliated mafic (basaltic) dikes dominantly striking NNE-SSW and E-W crosscut all other rocks in the district including undeformed pegmatite units, and are interpreted to be Proterozoic, possibly younger in age.

The major north-south structural fabric in the belts is clearly evident in both regional satellite images and geological maps for long strike distances. Steep foliations and localized isoclinals folds with north-south axial surface orientations are ubiquitous across the district and formed through a combination of strong early E-W shortening accompanied by sinistral shear as depicted in Figure 7-2 (BGS, 2004a). Structural investigations by Key et al. (2008) recognized tightening of folds, sheared folds, strike-slip and low- angle reverse faults that overprinted the north-south foliation. This event is interpreted to have occurred during later NW-SE compression.

More recent sub-tropical conditions in the Tertiary formed laterite profile over the eroded basement with thin ferricrete remnants. Elsewhere, the surface alteration has been eroded and re-deposited as reg and gravel. In the area, depth of oxidation ranges from 5 to 10 m or more. On the property, the oxidation is limited to few metres at surface (Bolster, 2011).

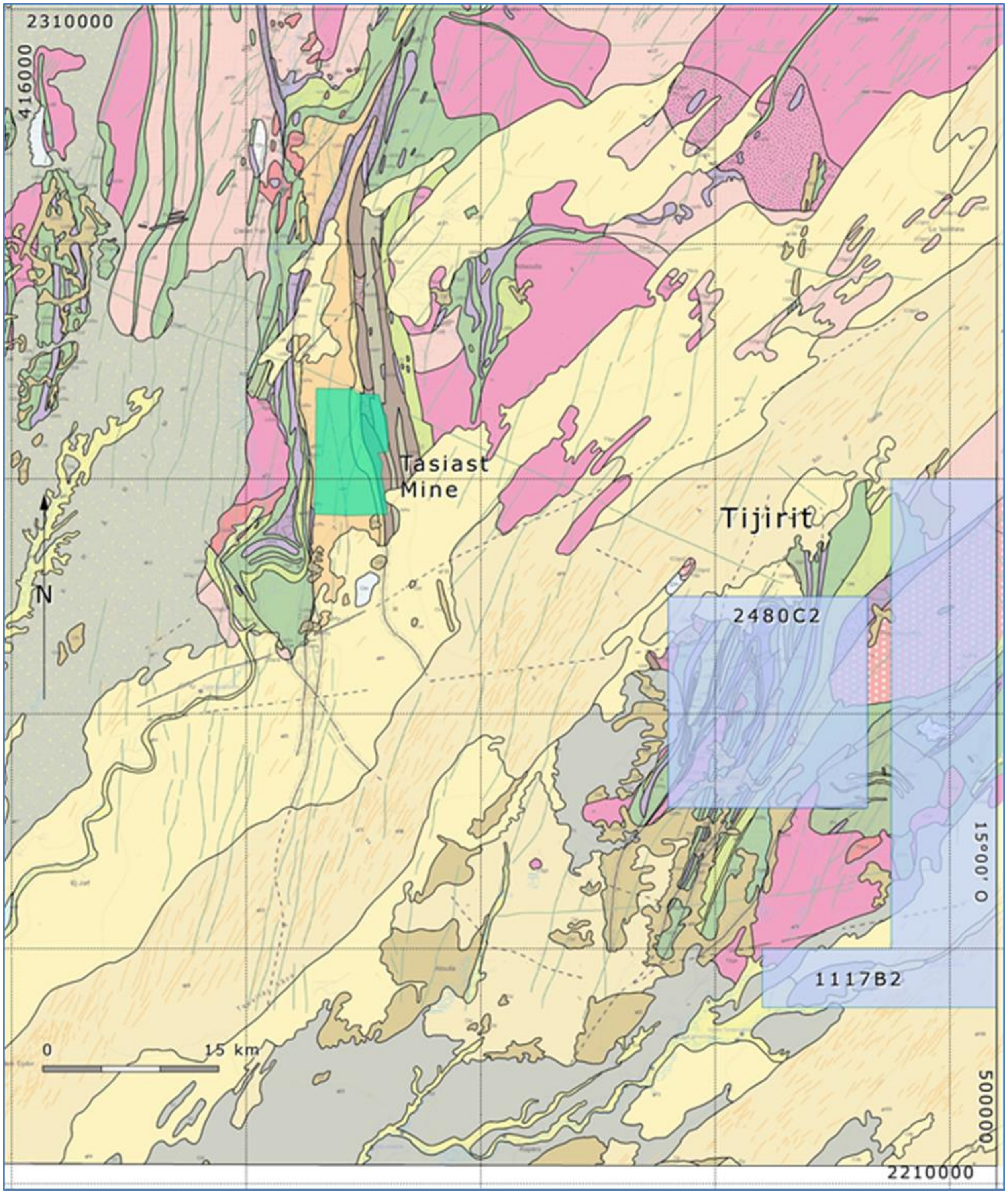


Figure 7-1: Geology of the 1:200,000 Chami Map Sheet Showing the Location of the Tijirit Project

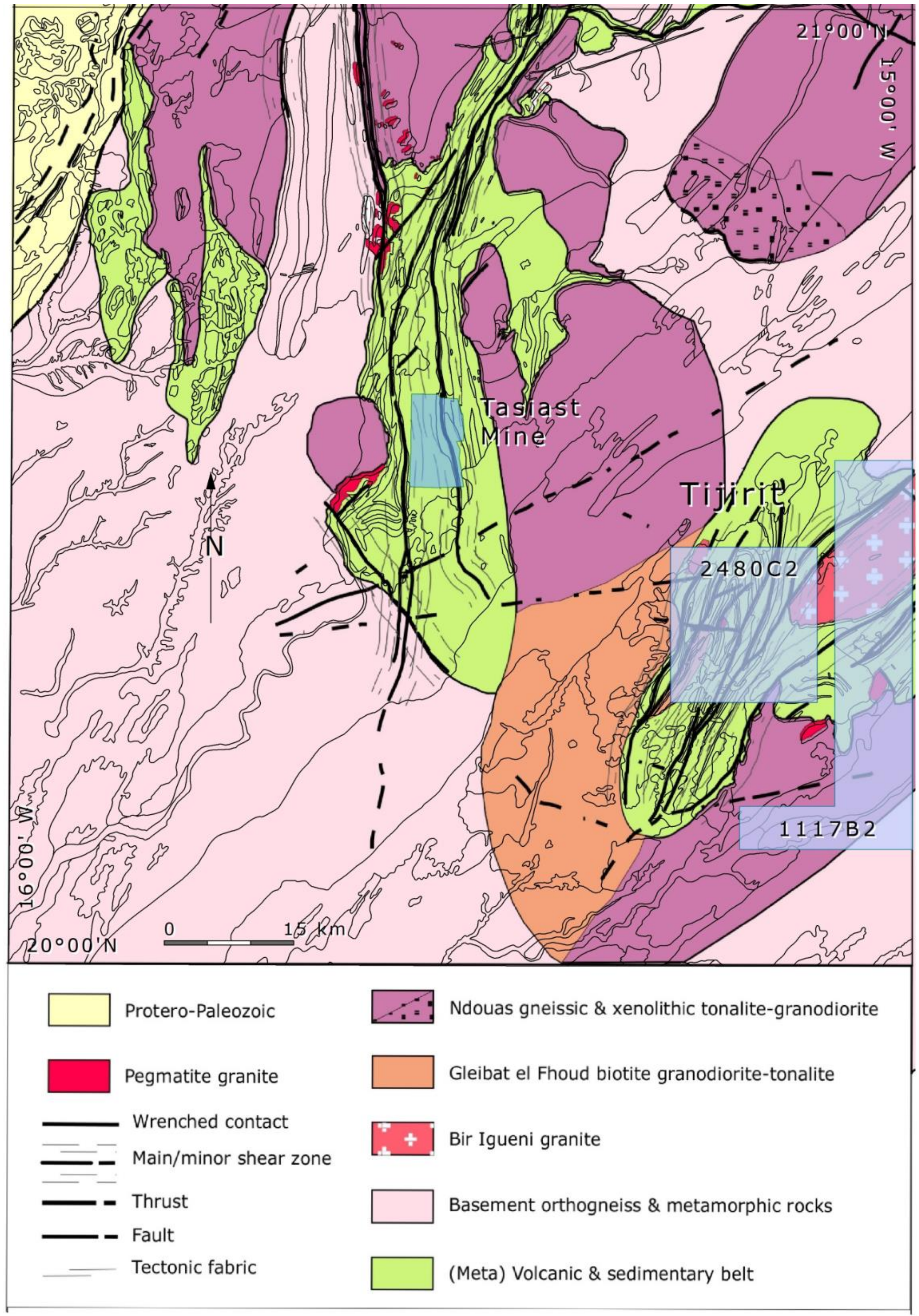


Figure 7-2: Major Shear Zones and Deformation Affecting the Tasiast-Tijirit Aouéouat Volcanic and Sedimentary Assemblages (BGS 2004a)

Table 7-1: Interpreted Stratigraphy of the Aouéouat v-s Belt (after Maurin et al. 1996)

Recent Cenozoic Phanerozoic	Fluvial gravel, sand, clay, silt, latosols, duricrust, sand dunes etc. Formation of lateritic soils and saprolite (older??) Gabbro – dolerite dyke swarms (120-65 Ma?)
Unconformity	~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~
Middle to Lower Proterozoic (?)	Unmetamorphosed quartzites and mica-schists (with intercalated amphibolites and orthogneiss)
Unconformity	~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~
Archean	
Late intrusive rocks	Calc-alkaline granite-granodiorite, pegmatites, granitoids (~2.7 Ga) Gabbro – Diorite, dolerite dikes (age?)
Intrusive contact	---
Volcano-Sedimentary pile (2.9 Ga)	Greywackes (epiclastics), mafic volcanics (basalts?) Banded iron/magnetite formations inter-layered with alternating garnetiferous schists, micaschists and tholeiites ultramafic, mafic and intermediate or felsic rocks
Basement (>2.9 Ga)	Granite gneiss, orthogneiss, migmatite complexes

7.2 Mineral Potential

Since the discovery of gold soil anomalies on the Project in the 1994-1996 period, major exploration works carried out by Shield Mining and Gryphon Minerals showed gold mineralization outlined by soil anomalies, trenches, and RC and diamond drilling. Most of gold soil anomalies and trench and drill intersections were discovered along or on either side of contact zones between metasedimentary rocks, serpentinites and porphyritic granitoids. It is distributed in zones measuring up to 1.5 x 4 km in four main areas namely Sophie I & II, Sophie III, Lily and Eleonore and 3 minor zones located more or less on strike.

The Tasiast world-class deposit is located approximately 40 km NW Tijirit area, the mineralization zones are not on strike but rather sub parallel as depicted in Figure 7-2. The Tasiast and Tijirit mineralized zones are separated by the Azefal dune bar and may be separated by regional faulting and large folding. The actual data of gold mineralization of the Project added to its location in a relatively close environment capable of producing a world-class multi-million-ounce deposit (Tasiast) makes it a high potential area.

7.3 Property Geology

The Project is in the southwestern Reguibat shield in granite greenstone terrain of the Aouéouat Belt. The project is located on Sebkhet Nich volcano-sedimentary (v-s) local belt, Figure 6-1. The sequences located in the centre west of the project are mainly composed of NNE-trending and regionally folded ultramafic, mafic to intermediate and felsic volcanics interstratified with various metasedimentary units and banded iron or iron-rich formations.

The various NNE-trending contacts are locally tectonized and show faults and shear. To the west, the v-s sequence is in contact with syn-tectonic tonalitic granitoids and main porphyritic quartz-feldspar granitoid. The main belt is intruded by syn-tectonic tonalitic granitoids and minor late-tectonic intruding granodiorite.

The centre of the project is underlain mainly by basalts and associated gabbro sills and some ultramafic rocks further to the east. The v-s sequence is bordered by quartz-feldspathic orthogneisses to the south, in contact with porphyritic granitoids to the west and intruded by somewhat more felsic granitoids to the northeast, Figure 7-1, Figure 7-2. The Project shows a fair coverage of regolith, sand and minor laterite ferricrete remnants. A study on the nature of the various regolith and local laterite cover using remote sensing was carried out by Bolster (2011).

7.4 Mineralization

On Tijirit, gold mineralization is related to deformation zones affecting the iron formations, various metasedimentary sequences and intermediate and mafic metavolcanic and meta-igneous rocks. Gold is locally visible, mostly as native medium to small grain located in quartz ± carbonate veins and silicified zones within highly schistozed metabasites and locally associated with sulphides and metasediments or with quartz veins and altered zones near meta-igneous rocks.

The Tijirit mineralization can be classified according to the following:

- Sophie I, II - gold is related to quartz-carbonate-sulfide veining located in altered, sheared and folded zone within or near contacts between BIF (iron formation), metabasic and metasedimentary sequences.
- Sophie III - gold is located in the north limb of a regional fold in NE-trending sheared contact between metavolcanics and metasediments.
- Lily - gold appear as fine grains located in widespread quartz-carbonate-sulfide veinlets associated to various zones, showing disseminated pyrrhotite and pyrite, alteration and shear and transposition deformation mostly in metasediments, meta-igneous and metabasic NE-trending sequences.
- Eleonore[2] - medium to fine grained gold is seen in various size quartz-carbonate-sulfide veining associated with shear and alteration zones within or in contact zones between meta-igneous, metasediments and metabasic rocks.
- Salma - gold is related to late-tectonic quartz veins locally showing sulfides and host-rock alteration. Gold is concentrated in quartz veins trending north, dipping west and located either at the contact between the metabasite sequences and the granite, 250 or 600 m east of the contact in identical veins or within the metabasite sequence, 1 km west of the contact with similar type of quartz veins. The Salma structure can be followed over a 10-km strike.

At present, different types of gold mineralization have been identified on the Project. It appears that gold mineralization is not always related to the iron formations but as in Tasiast, the presence of iron-rich rocks in the vicinity or lithological density contrasts could have play a structural and/or mechanical role in the precipitation of gold. It is carried through fluid vector circulation resulting in potassic alteration and quartz veins or stockworks or pervasive silica and/or carbonate and sulphide-rich precipitation.

[2] The name of this zone has been modified to Eleanor in various text, and thus for clarity, the reference has been made to refer this zone as Eleonore.

Figure 7-3 shows the major metavolcanic and sedimentary successions of the Project with interpreted faults and shears separating the various local magmatic and v-s domains. The map outlines the NNE-trending regional tight folds with parallel traces and shears or faults.

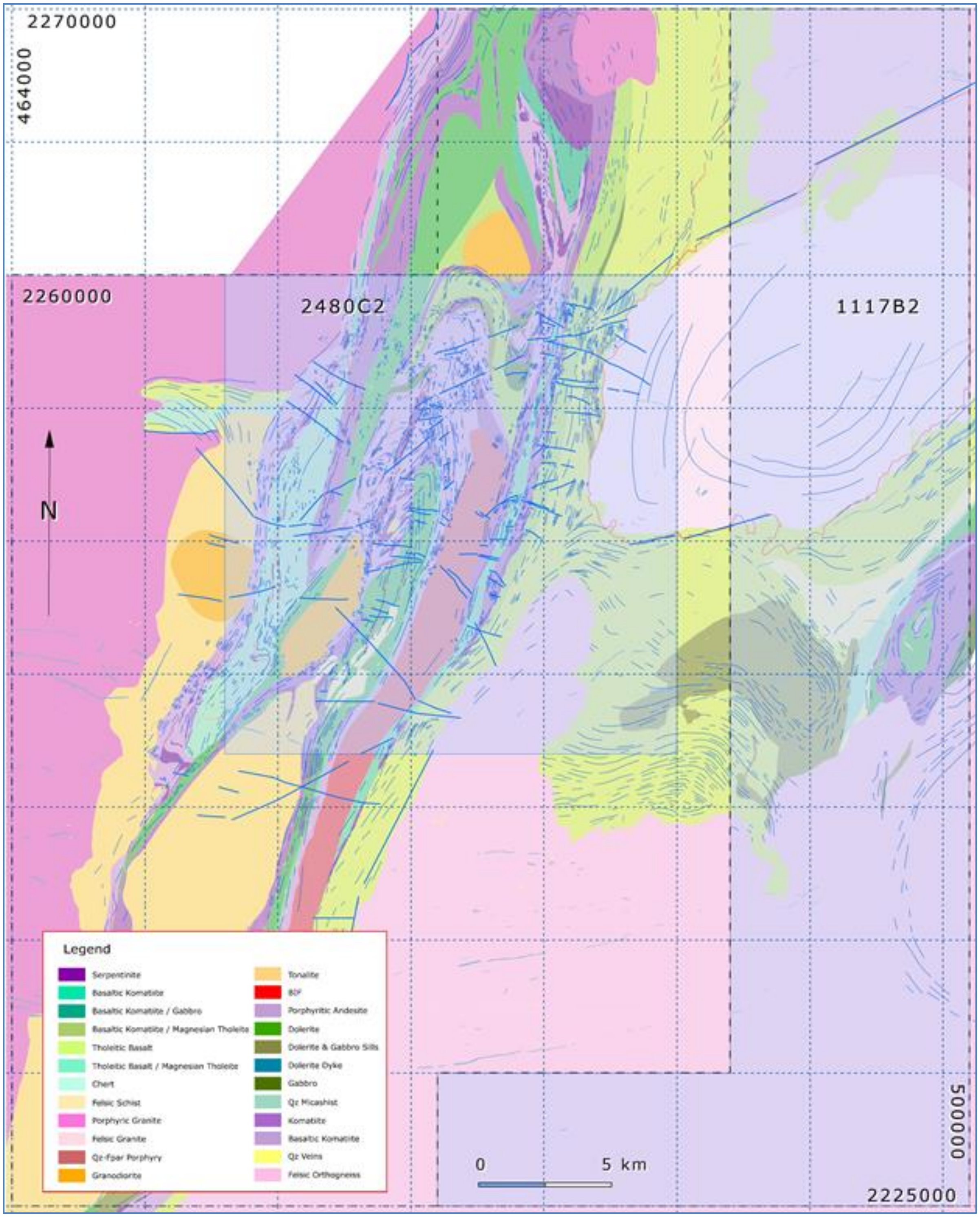


Figure 7-3: Geological Map of the Tijirit Project with Interpreted Structures. After Gryphon Geological Map (2015)

8. DEPOSIT TYPES

The Project is located approximately 40 km southeast of the Tasiast gold deposit. The Tijirit mineralized zones are trending to the NNE and are rather sub-parallel to the Tasiast north-trending deposit as depicted in Figure 9-1. The style of gold mineralization varies on the Project; however, two major types have been identified as unique: Eleonore-type mineralization and BIFs type. For the purpose of this section and given the information on the mineralization provided above, one assumes that Tijirit and Tasiast show similarities in the type of gold mineralization. One also takes in account differences in the lithologies and the tectonic contexts but with iron formations as common denominator.

Tasiast and Tijirit can be classified in part as shear-hosted and / or non-stratiform iron-formation (BIF) hosted deposit in the sense of Kerswill (1993) classification with some differences, since the mineralization are related primarily to deformation whether sulphides are present or not and the iron formation contains a limited amount of primary sulphides outside deformation zones. The Eleonore zone is comprised of a predominantly westerly-dipping metasediment and volcanic sequence with upper greenschist to lower amphibolite metamorphic assemblages.

Gold is hosted in quartz veins striking parallel to local foliation and the regional trends. It is believed to be emplaced relatively early in the tectonic history with veins often displaying a sheeted texture distinguishable from later crosscutting quartz veins. Several families of east-west and north-east trending brittle faults are believed to crosscut the foliation and offset geological units and mineralization. Initial results suggest that the area is extremely rich with coarse, nugget-type gold.

The host volcano-sedimentary Chami sequences are older than 2,933 Ma, according to BGS (2004) and may be as old as 3.2 Ga, which makes it much older than any known Archean mesothermal gold deposit, although the age of the deformation and the sulphide and gold precipitation is not known and may be much younger. The Salma granite (dated at Bir Igueni) is 2,933±16 Ma - mineralization in Salma post dates this intrusion.

The Tasiast deposit shows similarity with Archean greenstone gold deposits classified as mesothermal where deformed mineralized zones, either quartz or quartz-carbonate vein, stockworks or silicification are hosted in low to medium metamorphic grade terrains in in variously deformed supracrustal rocks. Deposits are characterized by high Au/Ag ratios and a carbonate host rock alteration. Low grade mineralization can be extended but high-grade lodes are much more reduced in dimension.

As for Tasiast, gold mineralization is hosted in Archean iron-rich sedimentary sequences and felsic sedimentary and/or volcanic rocks that have been deformed and metamorphosed at greenschist and lower amphibolite grade. Mineralization are both structurally and lithologically controlled and show mesothermal characteristics. The mineralized lithological sequences can be defined as follow:

- Epiclastic sediments and greywackes;
- Oxide-silicate and magnetite quartzite (grunerite-cummingtonite);
- Garnetiferous chlorite schist (metasediments);
- Biotite chlorite schist (garnet);
- Dacite, fine to medium grained overlain by an aphanitic pyrite/pyrrhotite-rich interval at the upper contact interpreted as a chert;

- Late cross-cutting mafic (gabbro) dykes.

It contains gold in various shapes and habits as:

- Isolated grains in quartz veins, commonly near lithological contacts;
- Isolated grains near fragments of host rock;
- Closely associated with pyrrhotite: generally, against or at the periphery and occasionally as inclusion;
- Associated with pyrite;
- In fractures within garnets crystals;
- Seldom associated with arsenopyrite;
- Rare occurrence within carbonate veins.

As another example in the area, gold is documented at surface on the Chami sheet 50 km NW of the Tasiast deposit. The iron-rich v-s Hadeibt Agheyâne belt has surface gold showings that were identified by Xstrata during their iron exploration program's in the recent past. It is trending NNW parallel to the Tasiast mineralized sequences and show similar orientation and intensity of the magnetic signature.

9. EXPLORATION

Between April and June 2016, Algold uncovered a series of quartz veins in trenches and pits on the Eleonore zone. These findings could possibly represent a large stockwork enclosed within a major shear zone, striking for more than 10 km in a north-northeast direction. Based on these new findings as well as on previous work, Algold strongly believes that this area hosts a high-grade gold deposit. This is a completely new high-grade gold occurrence in Mauritania.

All past exploration work carried out on the project is detailed below.

From 1993 to 2007

Based on geological mapping and previous results, exploration programs funded by Fonds de Développement Européen (EDF Project) took place between 1993 and 1996 and was contracted by BRGM and Office Mauritanien de Recherches Géologiques (OMRG). The program focused on the Chami and Ahmeyim 1:200,000 scale sheets and consisted in reconnaissance geological mapping and sampling. It identified significant gold anomalies in the Tasiast area.

Although the Tasiast deposit is located about 40 km to the northwest of the Project, much knowledge can be granted by studying the research history that lead to the discovery and subsequently the exploitation of the Tasiast gold mine.

In the mid 1990’s, La Source Développement SA, a French Mining and Exploration company was granted the right to the Tasiast and Tasiast Sud licenses. From 1996 to early 2000, La Source (and later on Normandy-La Source, NLSD) completed at regional and target scale soil geochemistry, geological mapping, airborne and ground geophysics, trenching and mapping, at a number of locations within the licenses, but focusing on the Tasiast deposit which was discovered in late 1997. NLSD completed a major exploration program within the Tasiast Permit Area, including some 32,000 m of RC drilling and 3,300 m of diamond drilling. An extensive amount of work was completed during the program.

A soil and stream sampling program in 1993-1994 by BRGM in the Tasiast-Tijirit area lead to the discovery of major anomalies on the actual Tijirit project between 91 and 1000 ppb Au.

In 2000, following a 94 to 96 European Development Fund regional soil survey a JV between NLSD and BRGM lead to a surface geology, soils and rock sampling program in "The Tijirit" area on the Ahmeyim map sheet located 100 km NE of the Algold’s Tijirit project. It is mainly composed of granitoids and volcano-sedimentary assemblages.

In the early 2000's an important geological mapping and compilation program was carried out in western Mauritania and funded by the World Bank and Projet de Renforcement Institutionel du Secteur Minier (PRISM). The work was contracted by the British Geological Survey (BGS) and lead to the production of twelve 1: 200 000 map sheet and related geological and metallogenic studies (BGS 2004). In 2005 and 2006 JICA (Japan International Cooperation Agency) produced various synthetic studies on northern Mauritania gold potential including Tasiast and Tijirit. In 2007, following PRISM II program, the USGS produced a variety of short studies on metallogeny of precious and base metals and geology, geophysics and remote sensing of various locations in Mauritania.

Since 2007

In 2008, Shield Mining Mauritania (SMM) followed up on the 1993-1994 BRGM gold anomaly discovery on Tijirit and carried out a more detailed sampling and a geological survey. The analysis of 3130 samples and geological surface studies lead to the discovery of major anomalous zones confirming the BRGM gold anomalies. Four (4) different zones showed 135, 45, 68 and 100 soil samples >60 ppb Au. In 2009, Shield carried out trenches, small soil sampling grids and rock sampling on the anomalous zones already outlined by previous works.

It was followed by a 1:2000 scale geological mapping, an RC drilling campaign and a major regional soil sampling program. In 2010, a supplementary RC drill campaign and some soil sampling were carried out. In 2011, a diamond drill program, trenches, soil sampling and detail geological surveys were carried out on the Lily, Sophie, Eleonore and Nancy prospects and a total of 14325 soil samples were taken on the 447B2 permit.

In 2012, 17189 soil samples were taken on the 447B2 and 3932 on the 1117B2 permits. Two trenches adding to 446 m were sampled on the Sophie III prospect and 75 RC holes were drilled and samples adding to 11910 m on the Lily, Eleonore and Sophie III prospects of the 447B2 permit. An extended auger sampling program was also completed.

Both the 447B2 and 1117B2 permit were transferred to Gryphon Minerals Mauritania (GMM) in May 2013. In the same year, GMM carried out semi regional and detail mapping on both 447 and 1117 permits, local sampling and RC drill result compilation on the 447 permit.

Table 9-1 describes the data produced during semi regional and detail exploration geochemistry, geology and geophysical work carried out on permits 447B2 and 1117B2 since 2008 either done by Algold, Shield Mining or Gryphon Minerals. Information is taken from unpublished annual reports.

Table 9-1: Tijirit Exploration Work Statistics since 2007

Property	Soils	Trenches		Rocks	Auger	PP	Mag & Rad
		nbr	m			km	km^2
Tijirit	38,107			1,447	1,300	16.2	1,917.6
Eleonore	1,037	56	5,183				
Lily		16	6,594			4.8	
Nancy	31	8	650				
Sophie		112	6,594				
Sophie I						2.8	
Sophie II		13	768			5.4	
Sophie III						3.2	
Tijirit South		5	2,810				
Total Tijirit		210	17,007				

9.1 Geochemistry

Soils

The Tijirit soil database contains 37,039 samples analyzed for gold and 7267 analyzed by XRF, including Ag, Cu, Pb, Zn, etc. In 2017, a new sampling campaign was carried out comprising 1068 soil samples. At the time of writing this report, the results of the analysis received are partial because the majority of the samples have not yet been analyzed. Therefore, the statistics below remain unchanged since the previous report. The data shows an Au mean of 15 ppb, a median of 8 ppb and a standard deviation of 37 ppb. None of the elements analyzed by XRF show a correlation with Au. A summary of the soil population shows:

90th percentile…………… 30 ppb Au;

95th………………………. 50 ppb Au;

98th………………………. 91 ppb Au.

A threshold of 51 ppb Au was chosen to show the anomalous areas on Tijirit and in normal centered distribution corresponds to the mean (15 ppb) + 1σ (standard deviation), in Figure 9-1. It reveals the presence of soil anomalies related to NNE-trending shearing separating or crosscutting various Archean metasedimentary, mafic, ultramafic and meta-igneous units (see geology section below). A compilation of the soils shows distinct concentration of anomalous values in four main area namely Sophie I, II & III, Lily, Nancy and Eleonore, in Figure 9-2. It also shows NNE and SSW extensions of many anomalies along deformation zones and many smaller or less concentrated anomalous areas of interest along more or less sheared meta-igneous and volcanic or sedimentary contacts (Bolster, 2011).

Trenches

In the past, a total of 197 trenches have been excavated for a total of close to 17 km on 5 anomalous areas following the soil results. The trenches are normally 1 to 2 m deep and have been sampled and analyzed every 2 metres. The sixth area in the south Tijirit is much less anomalous than Sophie, Lily and Eleonore areas. Figure 9-3 shows 25 results above 1 g/t Au (1 ppm) mostly concentrated on soil anomalies concentrated at lithological contacts.

In 2017, Algold completed 13 new trenches in the Sophie II/III area for a total of 768 m and 78 channel samples for a total of 170.47 m. The trenches are approximately 2 m deep and have been sampled at intervals ranging from 1 to 2 m. The channel samples were carried out at various depths below surface in artisanal workings.

Rock Samples

<u>2009 - 2012</u> – There were 989 surface rock samples collected all over both 447 and 1117 permits with a higher concentration on the soil anomaly areas of Sophie, Lily and Eleonore, as shown in Figure 9-4. Grades vary from 1.7 to 38.9 g/t Au. Of the 16 samples showing Au > 1 ppb Au, 11 richest samples are located in Eleonore along deformational zones and lithological contacts. There are 10 samples that are quartz veins and one show silicification of a Fe-rich carbonate rock.

<u>2016 - 2017</u> – Algold analyzed 704 samples collected mainly from the mineralized zones and the southern and northern extensions of Eleonore East. At the time of writing this report, the results of 2 samples were

pending. As shown in Figure 9-4, 225 samples show Au > 1 g/t Au varying from 1.04 to 102.5 g/t Au. There are 38 samples that show grades above 30 g/t Au mainly concentrated in the Salma and Eleonore East zones and further to the SSW suggesting the presence of a new high-grade gold corridor trending to the NNE and parallel to the main tectonic trends at the contact with a syn-tectonic granitoid intrusion, 2.5 km east of the Eleonore mineralized zone.

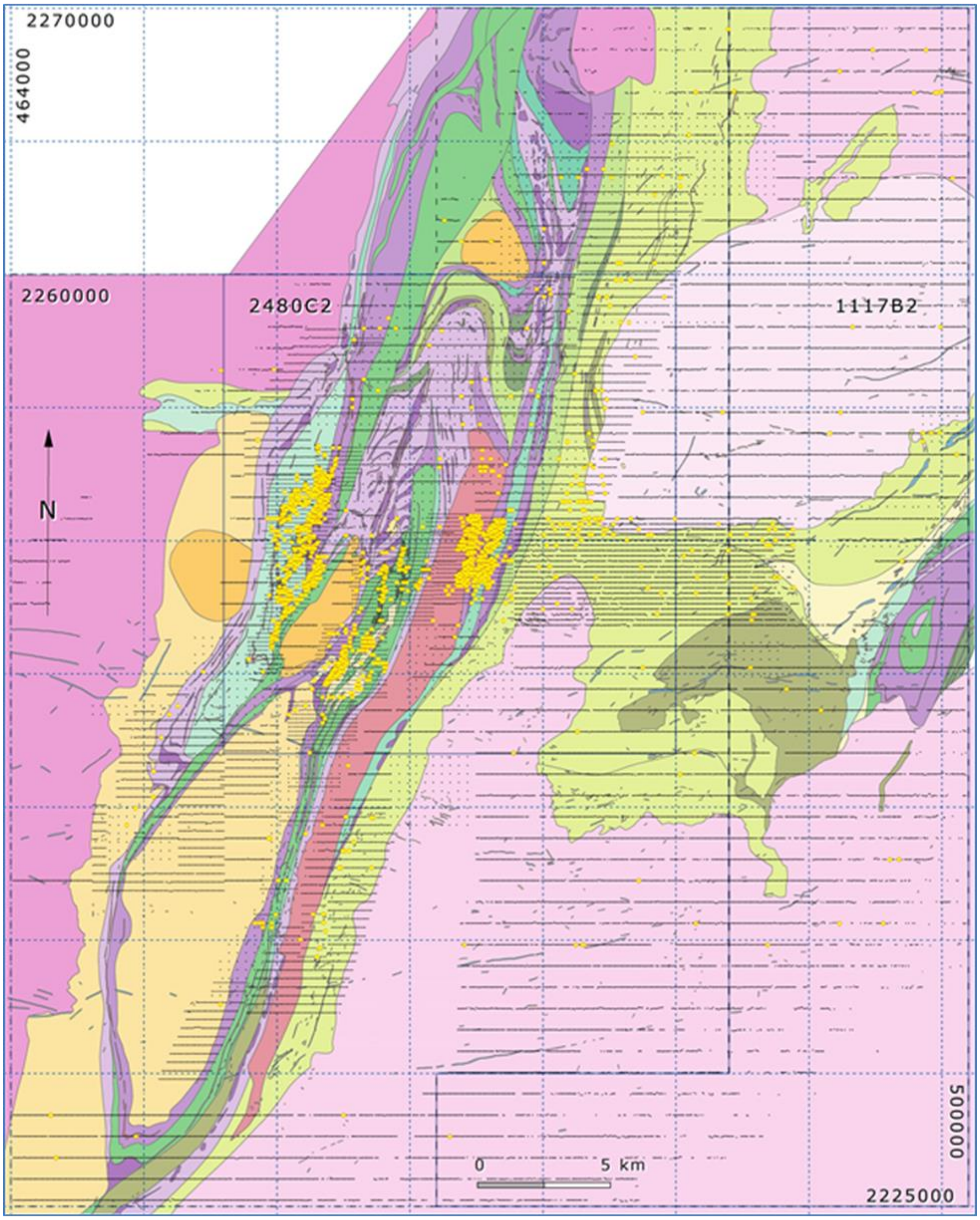


Figure 9-1: Distribution of 37,039 Soil Analyses and the >51 ppb Au Threshold (Big Yellow Symbols) Underlain by Tijirit Geology

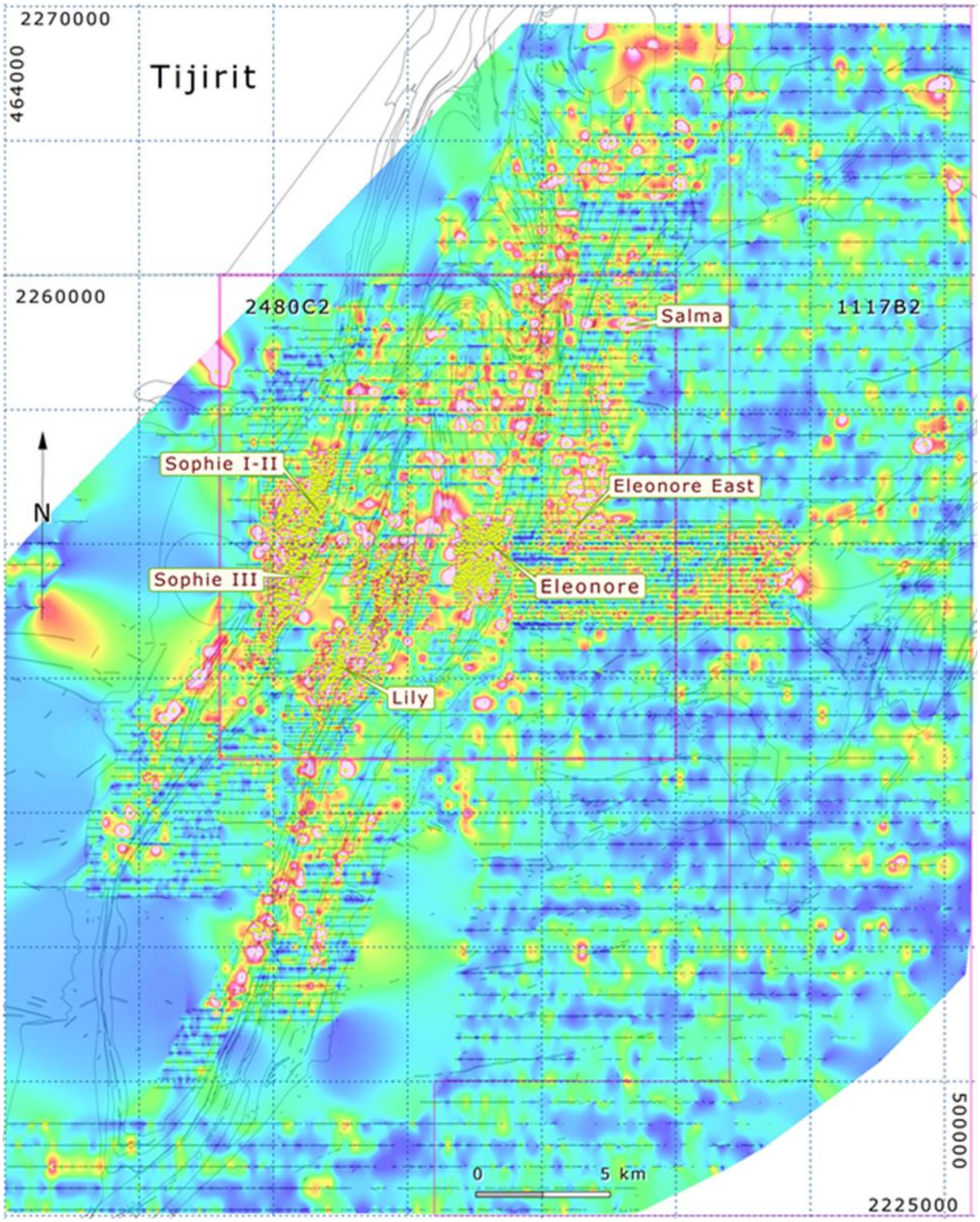


Figure 9-2: Distribution of Soil Analyses >51 ppb Au Threshold (Blue Symbols) Underlain by Total Soil Interpolation Map

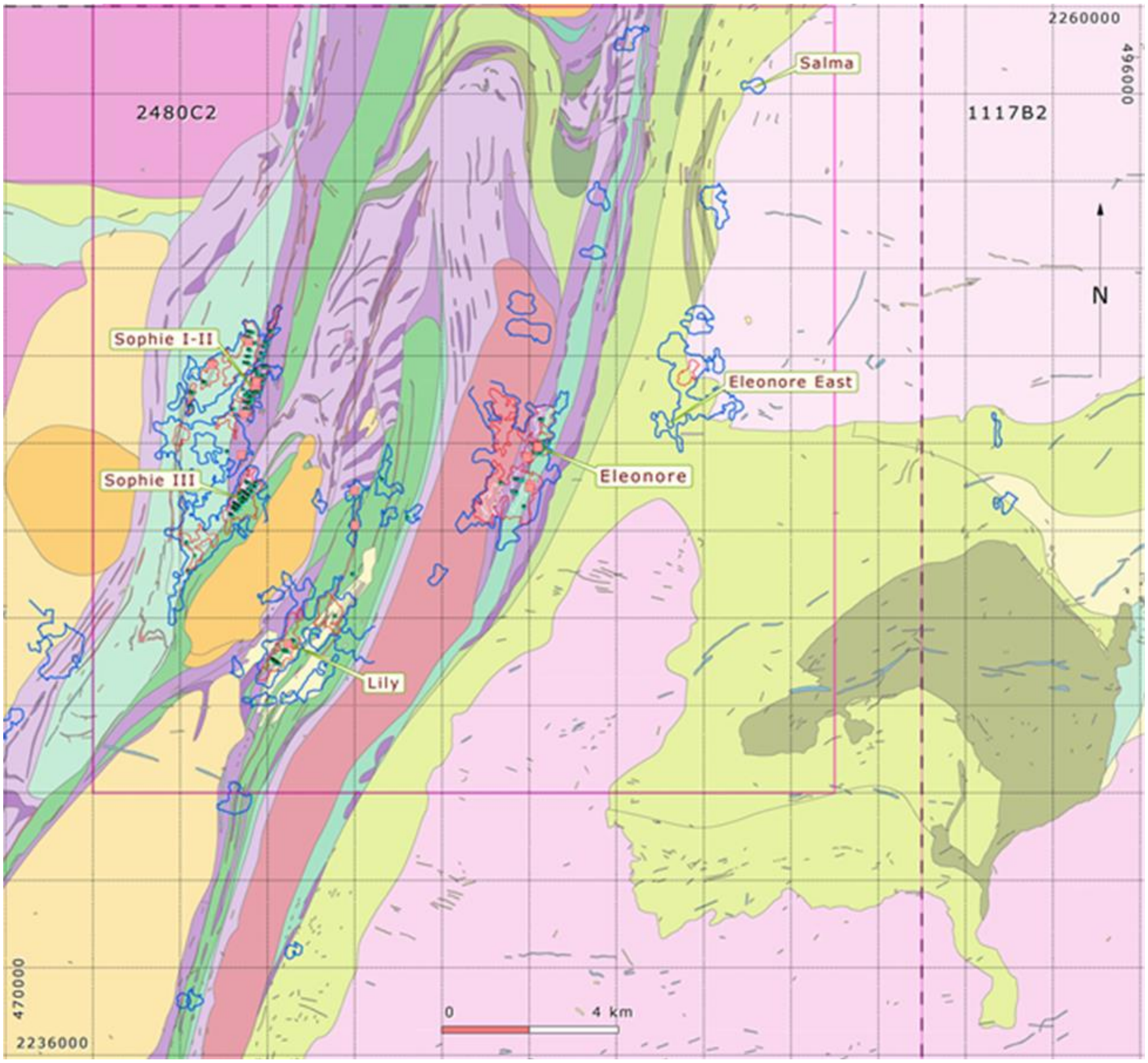


Figure 9-3: Tijirit Close-Up Distribution of Trenches, Trench Samples and Results above 1 g/t Au (Pink Symbols)

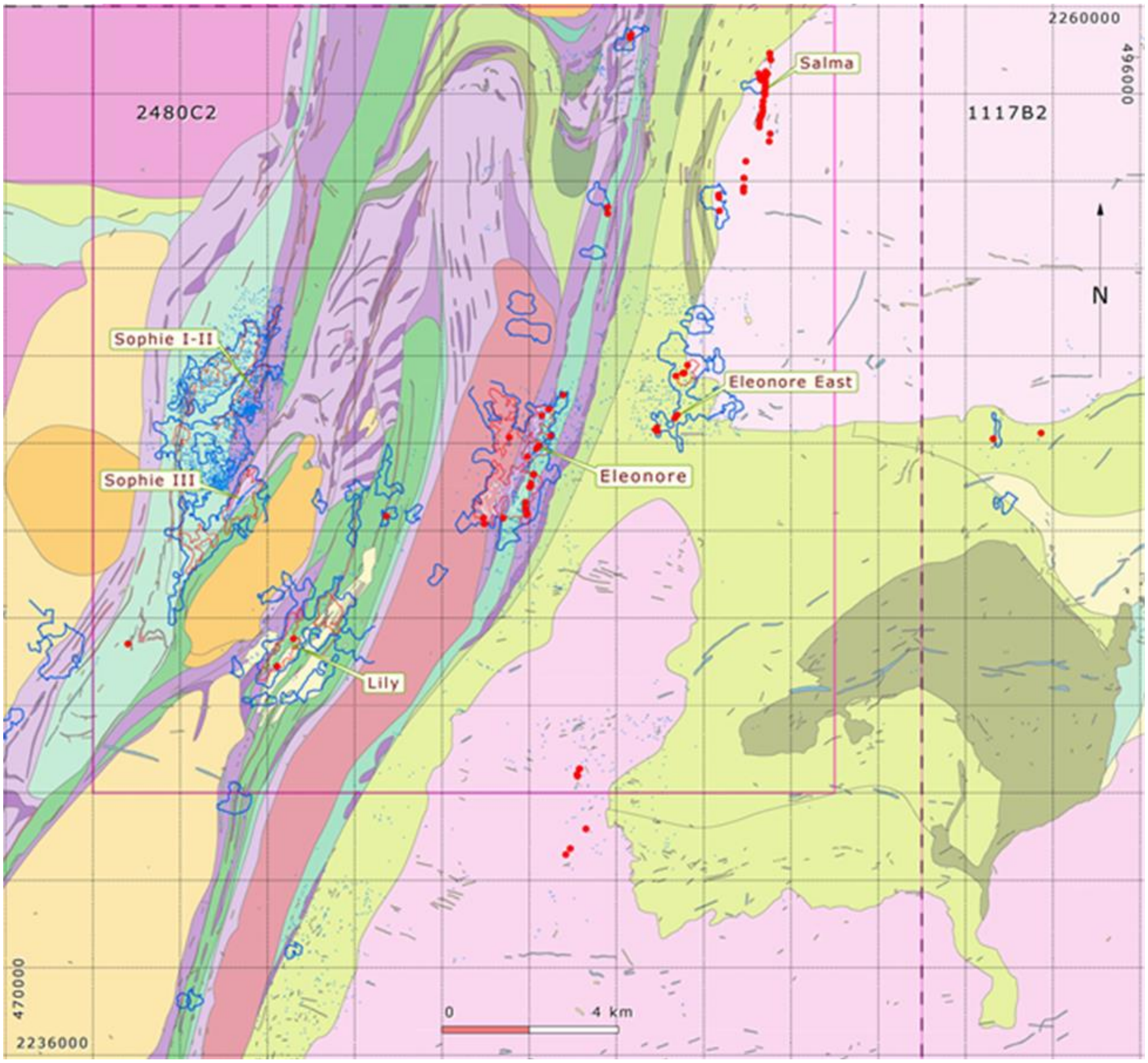


Figure 9-4: Tijirit Soil Anomaly Contours for 30, 50 and 100 ppb Au and the Distribution of Surface Samples (blue) and Samples Above 1 g/t Au as Red Symbols. Some samples (7) in the Salma Zone are Above 30 g/t Au

Auger Drill Holes

A 1300 drill auger holes program was carried out in 2012 in the center and southwestern Tijirit. 284 holes were done on the Eleonore zone and the rest in the southern 447B2 permit at a mean sampling depth of 6 m. The results show only one analysis at 281 ppb Au in the southwestern part of Figure 9-5. No results match the anomalous Eleonore soil values.

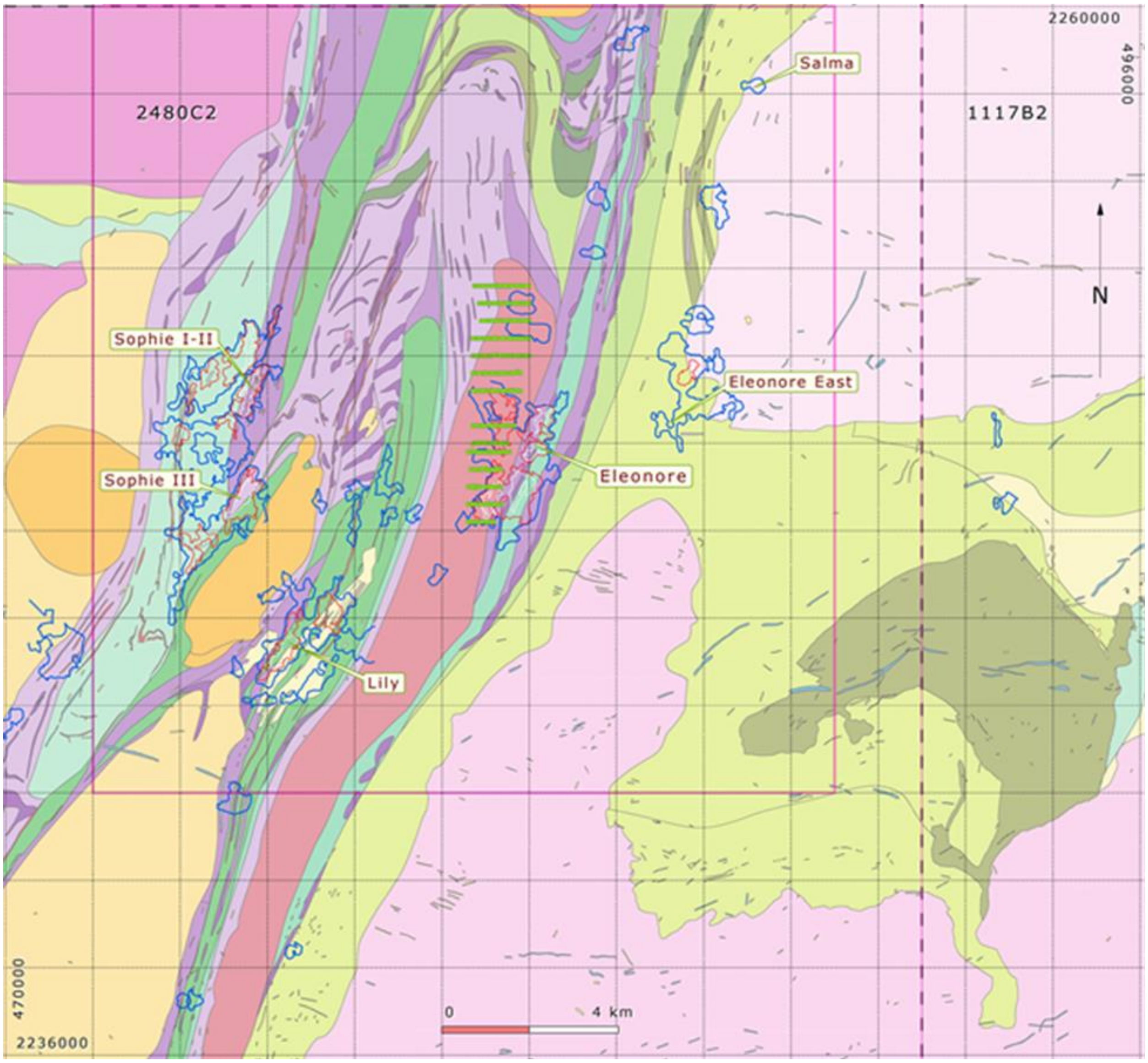


Figure 9-5: Tijirit Distribution of Auger Drill hole with a Mean Depth of 6 m

9.2 Satellite Imagery

Ikonos and 50 cm WorldView Images

One-metre resolution Ikonos satellite images were acquired on the project covering 120 km^2 and centered on the main mineralized prospects, Sophie, Eleonore, etc. It includes various visible and infrared band images. A WorldView mosaic was also acquired at 50 cm resolution and covers the main prospect area on permit 447B2. Bolster (2011) carried out a detail study using WorldView images. Figure 9-6 shows a structural interpretation by Davies (2012) using the 50-cm mosaic and the detail of the tight folding and shearing of the 447B2 permit involving Archean iron-rich metasediments, metabasites, meta-igneous and ultramafic rocks and the location of the main Tijirit gold prospects.

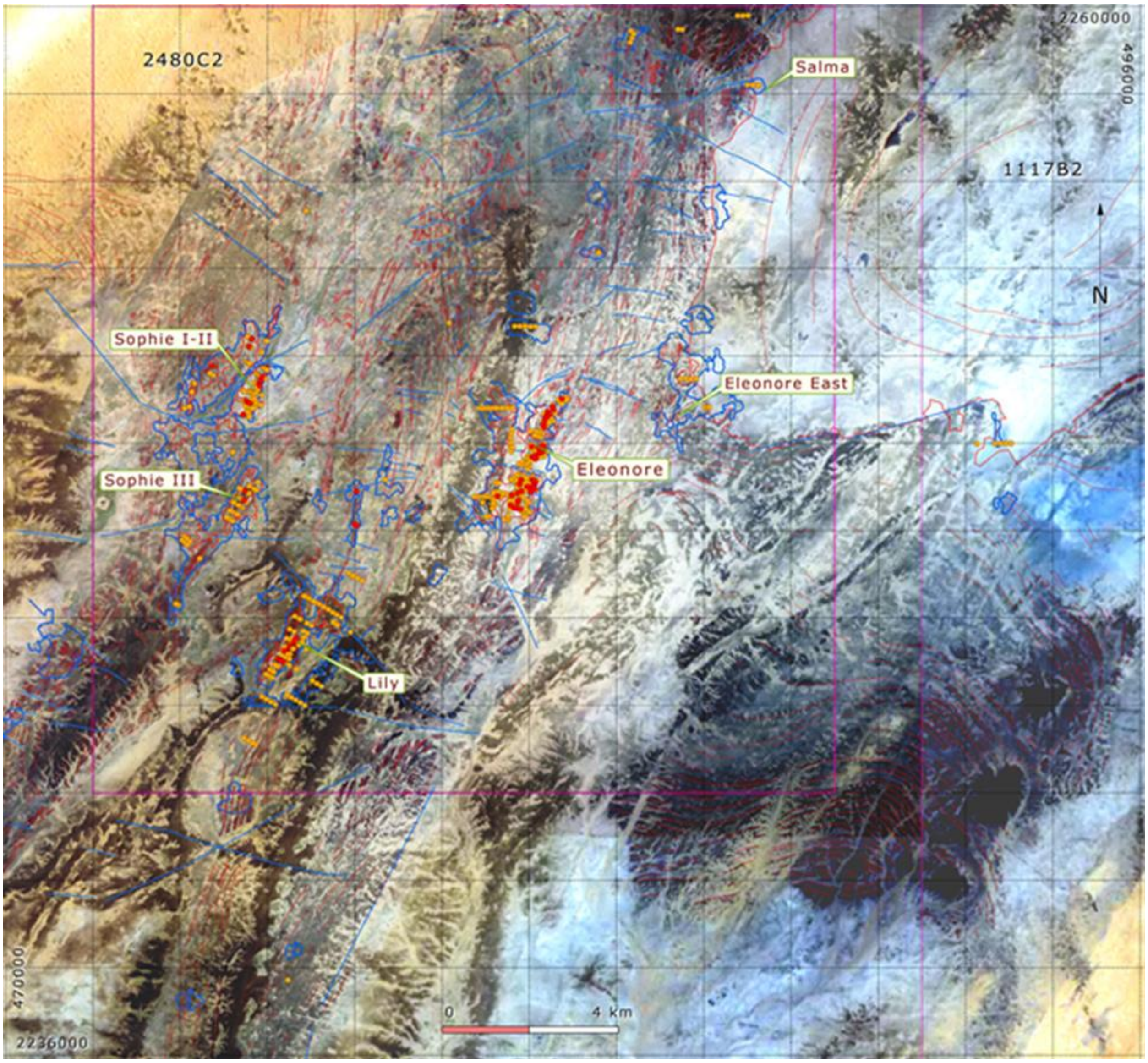


Figure 9-6: Tijirit 50 cm Hi-Res Satellite Image with Structural Interpretation, 51 ppb Soil Anomalies, RC and Core Drilling Distribution. After Davies (2012)

9.3 Geophysics

Airborne Magnetic & Radiometric Survey

An airborne magnetic and radiometric survey was commissioned on the Project by Gryphon Minerals. The survey was flown with 200 m line spacing oriented E-W and 1,000 m tie line spacing, oriented N-S. Various maps including magnetic vertical derivative and analytical signal and U-Th-K where produced by BIG-Consulting.

Figure 9-7 shows structural interpretation and the distribution of the various prospects with soil 51 ppb anomalies, RC and core drill holes. It shows that each of the main prospects are located on specific portion of the NNE-trending deformation pattern. It is revealed by magnetic highs due to magnetite presence in iron formation associated with some of the mineralization. For its part, the thorium radiometric signal gives information on the compositional variation of meta-igneous rocks involved with the mineralization, Figure 9-8. It shows compositional variations within the tonalite unit of the Tijirit South area and a marked signal of the quartz porphyry elongated unit involved in the deformation and mineralization. It also shows similarities between the felsic granite and the orthogneisses mapped as separate units. See detailed geological map in the next section.

Induced Polarization

In 2008, Sagax Afrique S.A. carried out 16.2 km of dipole-dipole (dp-dp) induced polarisation lines in four distinct locations on the Tijirit project, namely Sophie I & II, III, Lily and Eleonore. Figure 9-9 shows the distribution of the dp-dp lines on the various prospects.

- Eleonore - Only one small chargeability target was established by Shield Mining on the northernmost line and most of the IP data is said to be unusable (SGS, 2008).
- Lily - Two NNE-trending elongated and one small chargeability targets were established west of the main soil anomaly. A low current penetration was documented in the eastern third of the survey.
- Sophie III - An elongated NE-trending chargeability anomaly has been detected and a target was established parallel to the main deformation and coincides with the soil anomaly. Two minor targets were also proposed just to the east, but one considered that the 45° angle between the survey and the schistosity makes a line correlation more hazardous.
- Sophie I & II - Sophie shows high resistive zone to the west coincident with the soil anomaly and quartz-rich metasediment unit. On Sophie II, only a small high resistive portion to the SW corresponds to the contact zone with intermediate to mafic volcanics to the east. Most of Sophie I and II show NNE-trending low resistivity areas coincident with volcanic rock. Sophie II shows a high chargeability zone coincident with the soil anomaly and the contact zone between metasediments and metavolcanic rocks. Both high chargeability and low resistivity are considered disseminated sulphide IP signature and are regarded as priority targets, Figure 9-9.

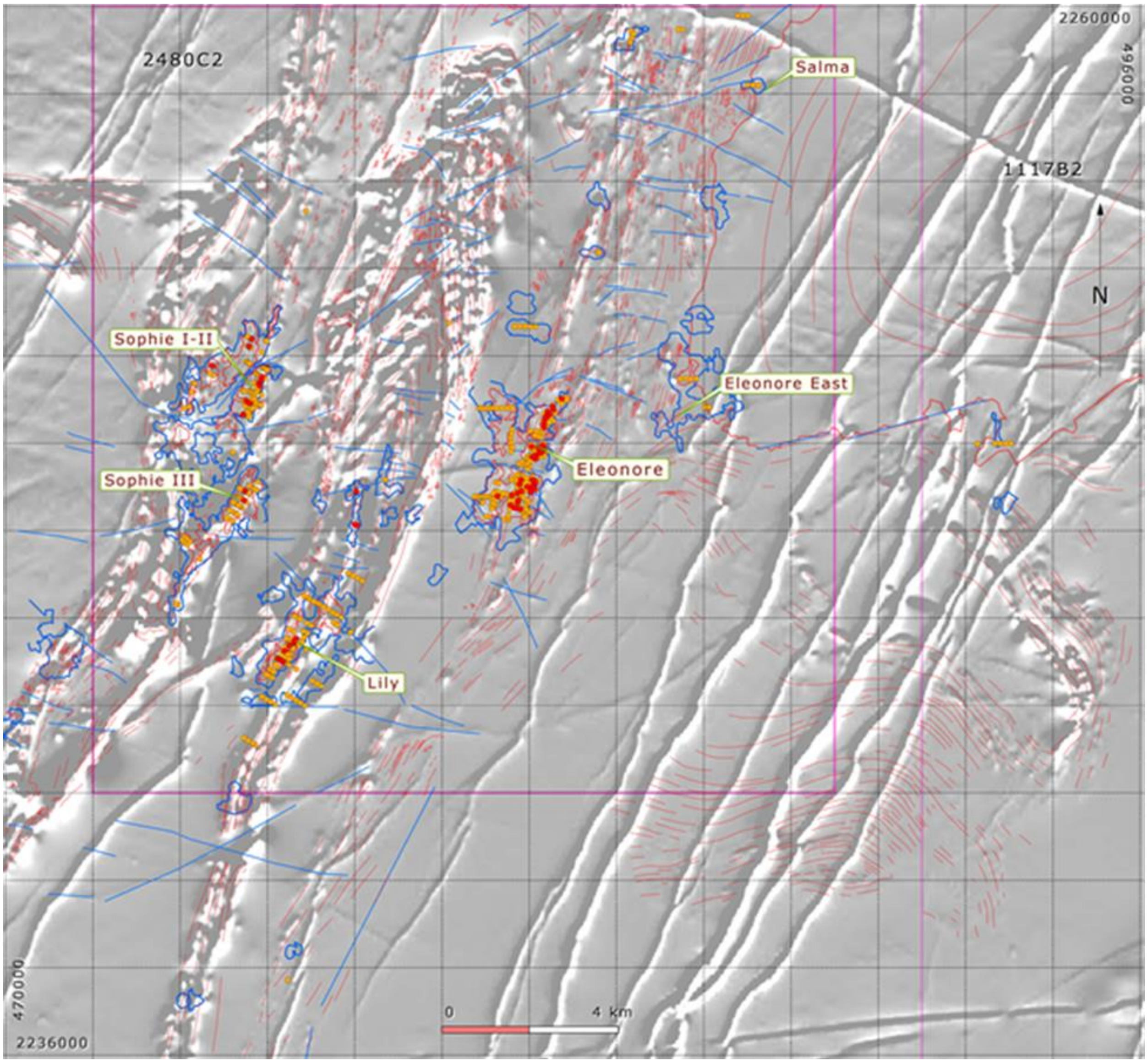


Figure 9-7: Magnetic Vertical Derivative Signal with Structural Interpretation, 51 ppb Soil Contours (Yellow), Drill Sites and Target Areas

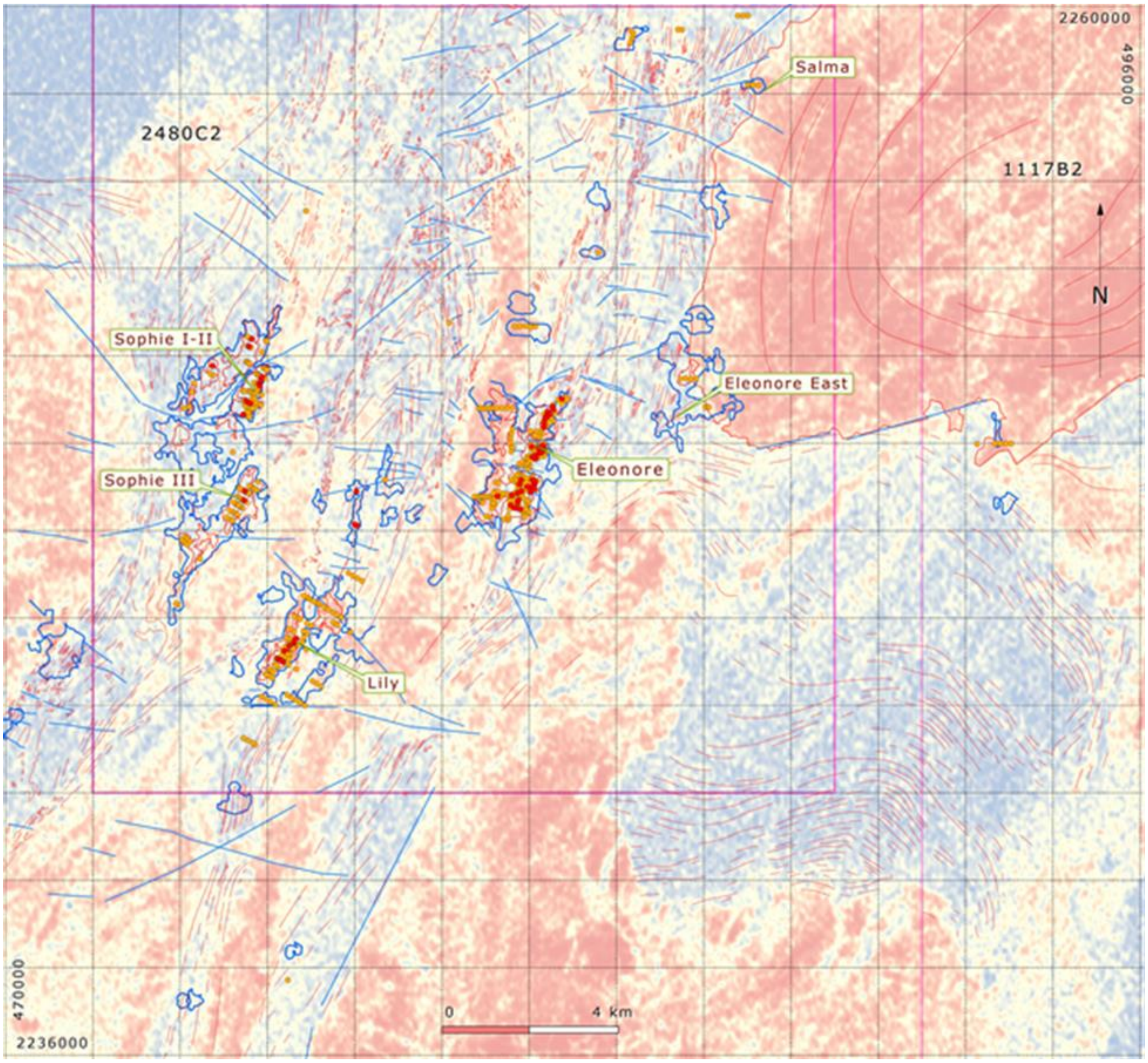


Figure 9-8: Thorium Radiometric Signal with Structural Interpretation, Drill Sites and Target Areas

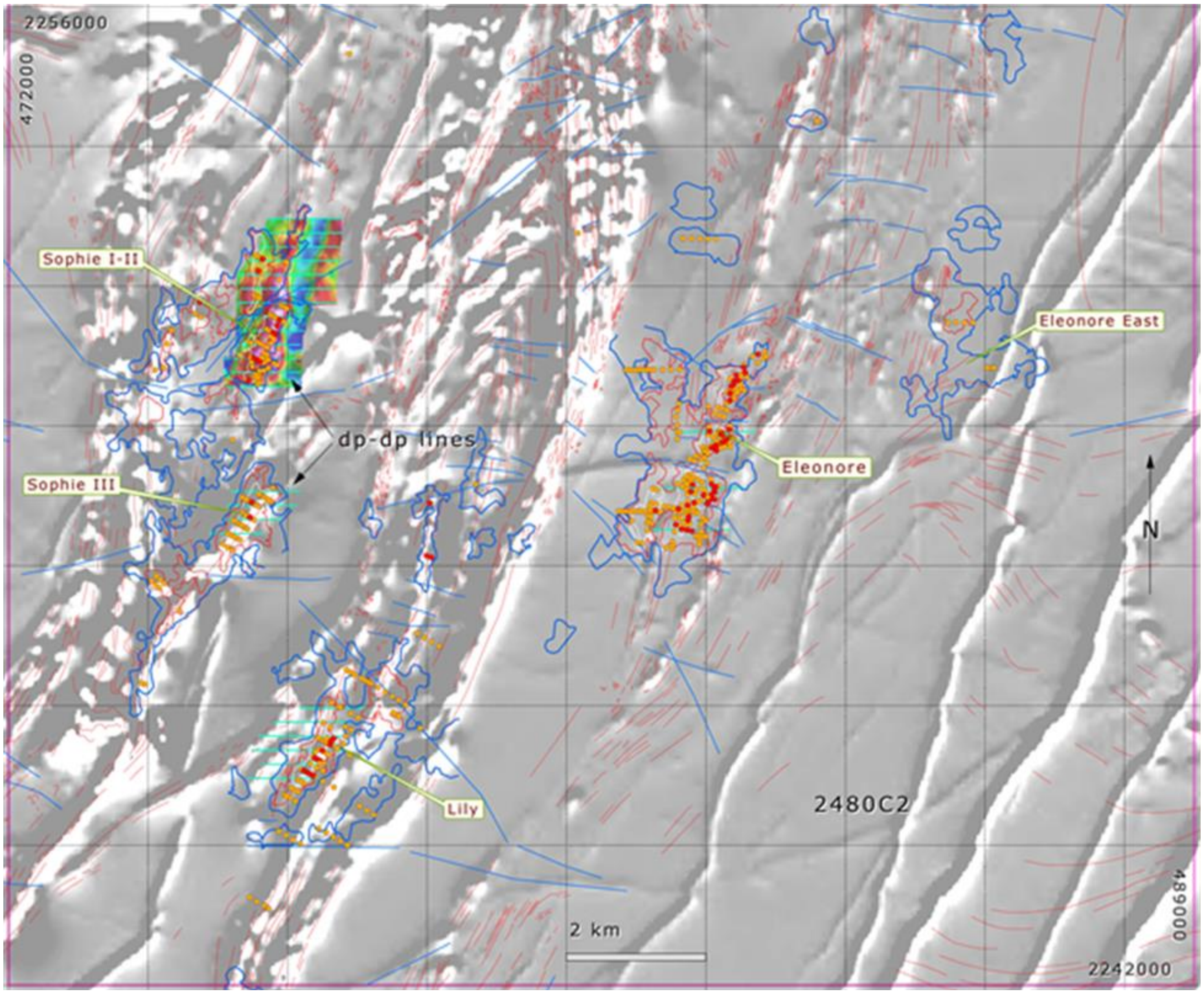


Figure 9-9: Prospect IP dp-dp Lines showing 51 ppb Anomaly Contours (yellow) and RC and Core Drill Sites

9.4 Exploration of Salma and Eleonore East

The Salma-Eleonore East mineralized zone extends for 8,5 km north to south and is located 3 km east of the main Eleonore mineralized zone on the 2480C2 Tijirit mining permit, Figure 9-10 and Figure 9-11.

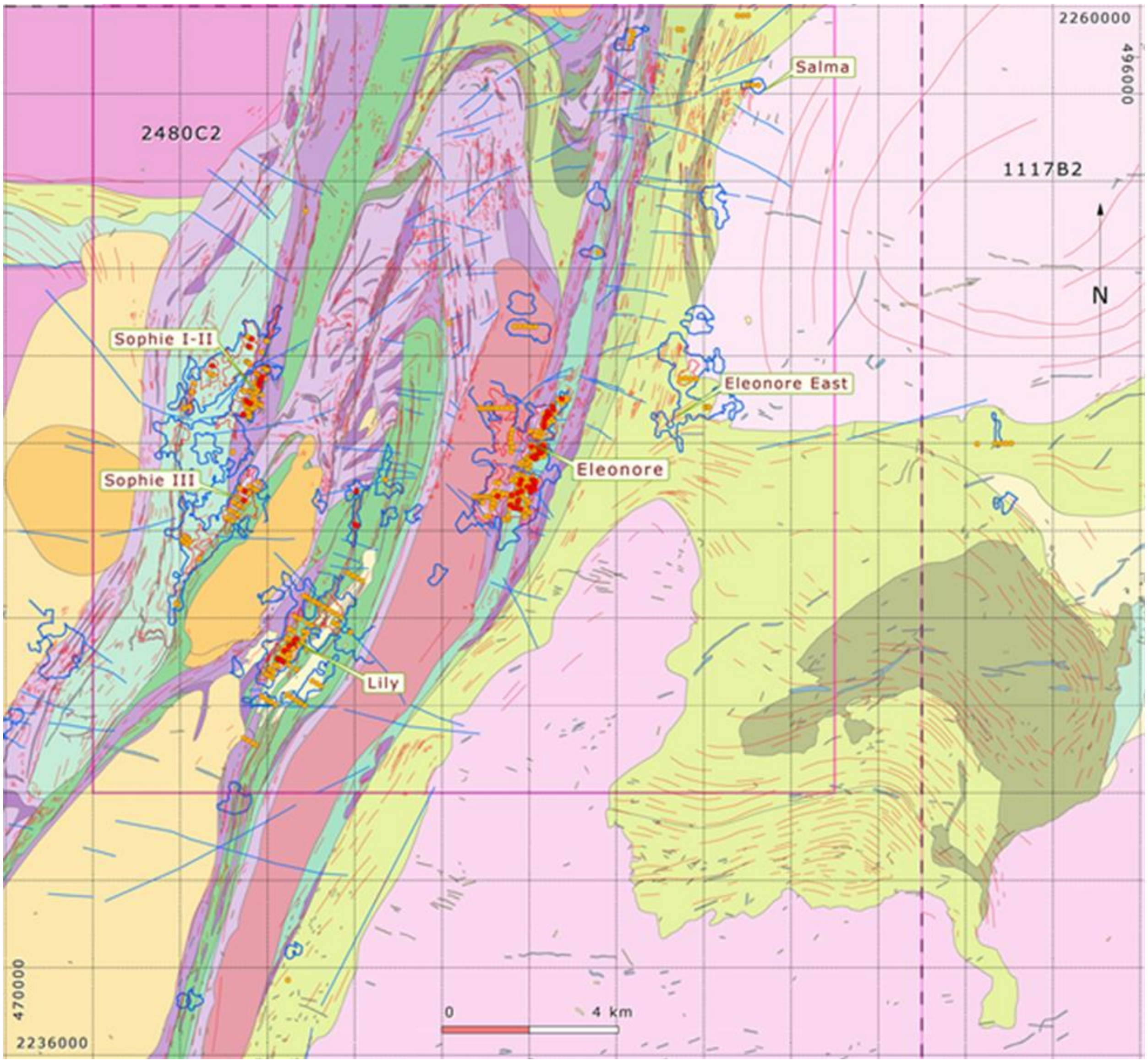


Figure 9-10: Location of the Salma-Eleonore-East area, 3 km east of the main mineralized Eleonore zone. Geology after Davies (2012).

The area was previously sampled for soil and surface rock geochemistry. The area was also mapped at regional scale and locally drilled following soil anomalies or mineralized quartz veins. Mineralized intersections have been described in RC holes of the Tijirit-East area, Figure 9-11.

The area is being currently worked by Algold where geological mapping, surface and channel sampling and drilling has been conducted during Q1 – 2018. No resources estimation has been done for this area.

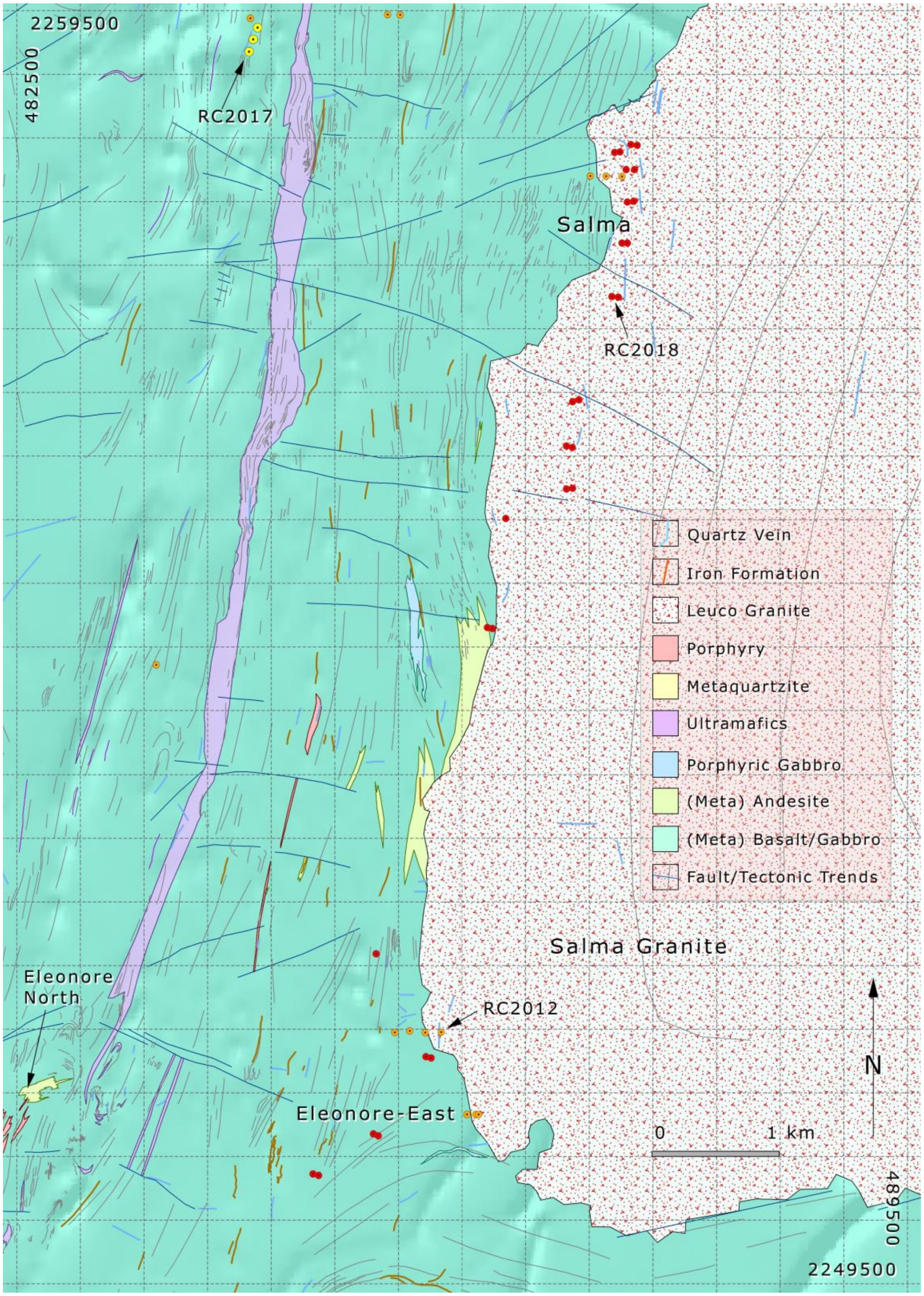


Figure 9-11: Geology of the Salma zone showing the limit of the eastern granite in contact with underlying metabasite sequences (green-blue)

The metabasite sequences show conformable intercalated serpentinites (purple), porphyritic gabbro (pale blue), meta-andesite concentrated along the contact (green) and minor porphyry, meta-arenite, BIF and felsic dykes. The area shows mostly NNE-trending schistosity and folds. The contact zone with the granite is highly deformed and shows conformable sheared quartz veins. 30 and 50 ppb soil contours in green and red, surface chip and channel samples > 1 g/t Au as red dots, previous RC drill holes as orange dots, Algold RC drilling in Nour as yellow symbols, RC drilling in the Salma contact zone as open symbols.

The Salma zone is oriented north-south and located at the limit between the Eastern Granite and the main mafic volcanic assemblages that underlie most of the area, Figure 9-11. The metabasalt sequences show conformable metre to hectometre size intercalation of serpentinite, meta-andesite concentrated along the western contact, porphyritic metagabbro and minor bands of granite porphyry, metaquartzite or meta-arenite, iron formation (BIF) and felsic dykes. Locally, granite porphyry is associated with meta-arenites and iron formation. The whole area shows conformable late tectonic and post-tectonic quartz veins locally oriented east or northeast.

The area is affected by the main north-northeast-trending schistosity. It is locally trending east in the southern portion of the map and locally shows various scale folding, the contact with the Eastern Granite is highly deformed, mostly dipping to the west and shows kaolin-rich white and limonite-rich brown oxidation corresponding to granite and basalt alteration.

As in other mineralized zones on the Tijirit project, gold is related to late-tectonic quartz veins locally showing sulfides and host-rock alteration. In the Salma zone, gold is concentrated in quartz veins trending north, dipping west and located either at the contact between the metabasite sequences and the granite, 250 or 600 m east of the contact in identical veins or within the metabasite sequence, 1 km west of the contact with similar type of quartz veins.

RC drilling by the previous operator in 2012 showed the following mineralized intersections:

- 1 km east of the Nour zone, northwest of the map area, 12RC078;
- at the granite contact in the Salma zone, 12RC141-142;
- at the granite contact and 250 m to the east in the Eleonore-East zone, 12RC134-137; and
- at the granite contact, 750 m southeast of the Eleonore-East zone, 12RC166-167, Table 9-2, Figure 9-11.

Table 9-2: 2012 RC Drilling Mineralized Intersections in the Salma Map Area

HoleId	East	North	Zone	from	to	m	Au g/t
12TRC078	485513	2259467	Nour	0	1	1	0.62
12TRC134	485808	2251476	Eleonore-East	41	45	4	2.07
12TRC135	485674	2251473	Eleonore-East	54	55	1	0.77
12TRC136	485557	2251485	Eleonore-East	47	52	5	0.70
12TRC136	485548	2251485	Eleonore-East	55	71	16	0.74
12TRC137	485436	2251474	Eleonore-East	52	54	2	3.16
12TRC141	487149	2258198	Salma	165	166	1	2.92

HoleId	East	North	Zone	from	to	m	Au g/t
12TRC142	487021	2258199	Salma	168	169	1	11.75
12TRC166	486091	2250829	Southeast	81	85	4	1.52
12TRC167	486117	2250830	Southeast	14	15	1	16.95

The 10 drillholes from Gryphon presented in Table 9-2 were never used in any resource estimation. The QAQC data has been verified in the maiden report dated August 2016, but the quantity of QAQC data specifically for this area is low. The results of the QAQC in the maiden report were satisfactory, and as the overall Gryphon database there is no reason to believe that there is a specific problem. New drilling and future work will validate that the data from these 10 drillholes are reliable.

Surface chip samples revealed mineralized quartz in the Nour zone, along the Salma granite contact and within the metabasite sequence. Following positive results, Algold proceeded with channel sampling of all accessible artisanal diggings along the granite contact, Figure 9-11. Gold assaying revealed 34 samples above 1 g/t Au over 186 samples with a maximum of 42 g/t.

Following positive surface sampling in the Nour zone, Algold proceeded with three RC holes in the area but failed to intersect significant mineralization.

Following positive channel sampling in the Salma contact zone, Algold carried out 28 RC drill holes in February 2018 mostly trending east with EOH between 60 and 120 m. As of January 19, 2018, results were not available and no resources have been estimate for this area.

9.5 Dispersion of Gold in the Eleonore laterites - Implications for Exploration

Understanding the alteration profile and the gold dispersion of the Eleonore zone are essential for the exploration follow-up especially for the western portion where a thick ferricrete is observed.

The mineralized Eleonore zone in the Tijirit permit is located in northwest Mauritania in a desert area where rain does not reach 100 mm / year. Now the alteration profile shows a marked variation in the various climates affecting the area and without being able to date it, the climate has been and for a long time in a humid tropical zone.

The profile shows a transition toward a more desert zone at the end of the laterite and ferricrete process and the beginning of the surface and saprolite erosion. In Tijirit, the absence of mottle clay zone underneath the ferricrete suggests that an erosion period may have preceded the last phase of ferricrete formation.

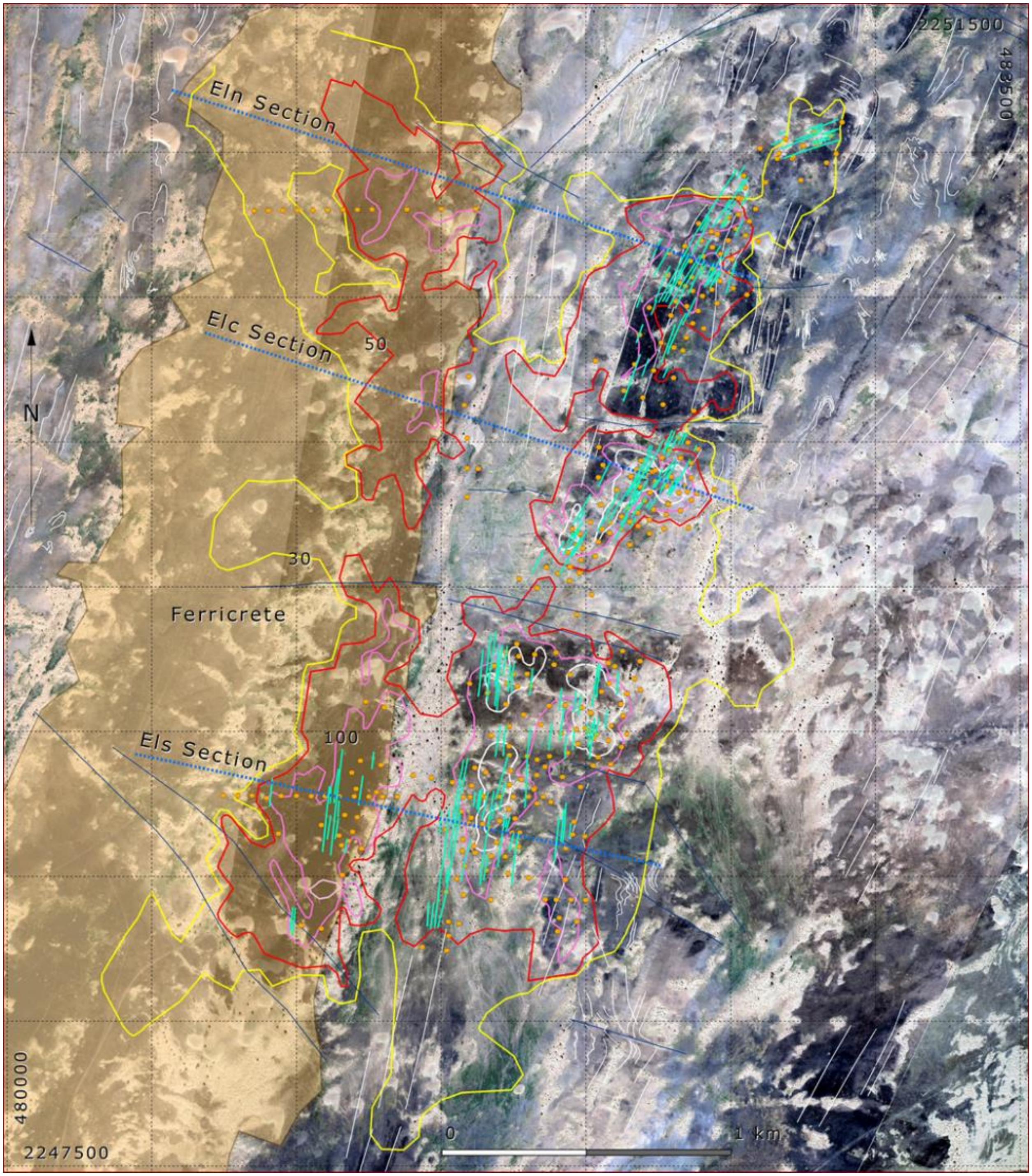


Figure 9-12: Distribution of the ferricrete on the Eleonore zone, with soil anomalies (30, 50, 100 & 200 ppb Au), drill holes (yellow symbols), the projection of mineralized zones (green lines), the faults (blue lines) and the structural trends (gray lines).

It is fair to say that the establishment of the laterite profile, the understanding of gold dispersion and the application of a genetic model should lead the exploration works.

Mineralized Zone Modelling

It was noted above that the soils of Eleonore east zone are residual with local abundant clay. Soils are believed to be in situ with the exception of transported sand and gravel in identified wadi (streams). It shows more or less concentric gold anomalies between 30 and 200 ppb Au and more with a notable continuity. According to the drilling, it corresponds to major underlying mineralization. It should be noted that the limits of the mineralized zones are schematic, and the host rock may contain various mineralization zones outside the proposed limits.

On the west side of Eleonore, the pisolithic ferricrete is considered in-situ and also contains more or less concentric and continuous gold anomalies.

Previous Works

The Eleonore west zone was already worked by previous operators, using soil geochemistry described above and auger and reverse circulation (RC) drilling.

- Auger - Auger drill sections were carried out on all the Eleonore western zone at 50 x 400 m grid. The data show that the analyzed samples were taken at various depths and that no laterite profile was established. No results could compare to the soil geochemistry.
- RC drilling - The RC drill section located immediately south of the Eln section on Figure 9-12 did not reveal notable gold intersection on Eleonore west.

The drill section located on section Els shown in Figure 9-12 revealed a mineralized zone sketched on Figure 9-13. It is located 500 m west of the main Els zone, underneath the 100-ppb anomaly. Furthermore, the drilling revealed a narrow-mineralized zone 45 m across located immediately east of the main zone that caused the 30 and 50 ppb overlying soil anomalies.

Projected Mineralized Zones

The application of the dispersion standard model in soils or ferricrete with collapse of the saprolite and in situ dispersion of gold-bearing quartz veins at surface shows the presence of mineralized zones at depth more or less centered on the 50 and 100 ppb Au anomalies. Thus, 3 sections were modelled on the Eleonore zone, Figure 9-12, with 5 mineralized zones detected by drilling, Figure 9-13, Figure 9-14 and Figure 9-15. Following the same pattern, 2 zones are projected on the west portion of sections Eln and Elc, centered underneath the 50 ppb anomalies found on the ferricrete of Eleonore west zone, Figure 9-13 and Figure 9-14.

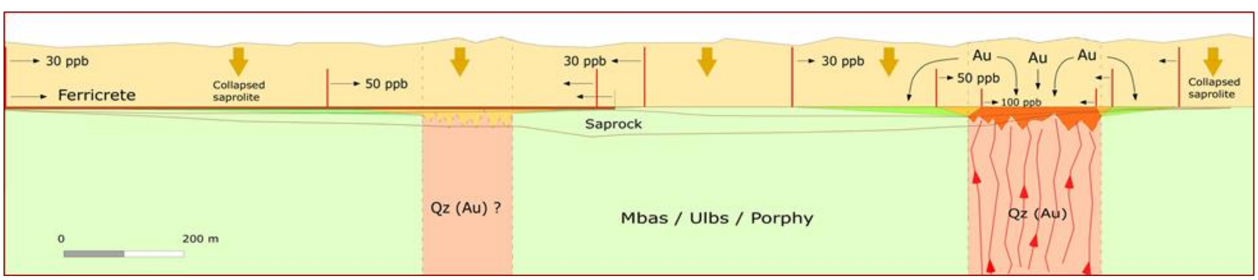


Figure 9-13: Sketched mineralized zone detected by drilling in the eastern portion of the Eln section and projected zone on the west side underneath the 50 ppb anomaly of Figure 9-12.

The soil anomalies are created by the collapse of the saprolite, the dismantlement of the quartz veins and the concentration of gold.

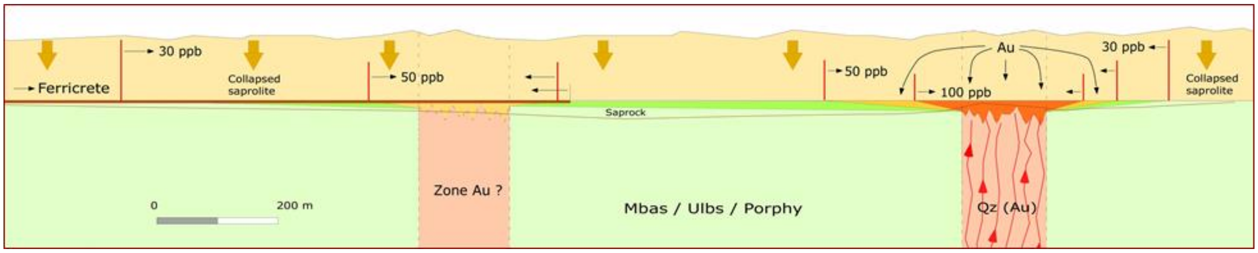


Figure 9-14: Sketched mineralized zone detected by drilling in the eastern portion of the Elc section and projected zone on the west side underneath the 50 ppb anomaly of Figure 9-12.

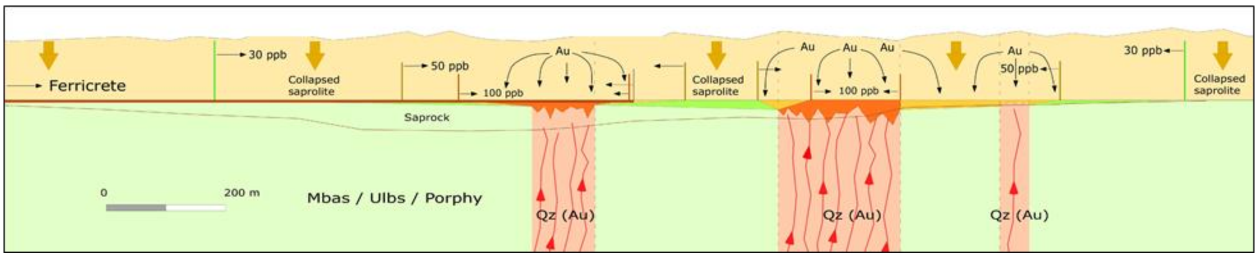


Figure 9-15: Sketched mineralized zones detected by drilling on the eastern and western portion of the Els section underneath the 100 and 50 ppb anomalies of Figure 9-12.

Conclusion

The lateritic profile of the Eleonore zone was characterized and compare with the standard profile in West Africa. One has observed the absence of the mottle clay zone underneath the ferricrete on the west Eleonore zone and the absence of both the mottle clay zone and the ferricrete on the east Eleonore zone.

The soils are residual and considered in situ on the east Eleonore zone with local presence of clay of possible lacustrine origin. The dominant ferricrete on the west Eleonore zone is also considered in situ.

The Freyssinet (1993) model was applied to the Eleonore soil anomalies and mineralization where gold is related to the collapse of the upper part of the saprolite, the dispersion of quartz veins and the concentration of heavy minerals including gold more or less centered above the mineralized zones.

Three sections were modelled, Eln, Elc and Els with 5 zones detected by drilling and located as predicted, centered underneath the 50 and 100 ppb gold anomalies. Following the same pattern, two zones have been modelled centered underneath the anomalies located in the ferricrete of west Eleonore.

It is concluded that the establishment of the lateritic profile, the origin of the soils and the understanding of the gold dispersion pattern at surface is critical to for the choice of a model that best represents the gold mineralization. This information should lead the exploration works at Eleonore and on the other mineralized zones of Tijirit.

10. DRILLING

Algold initiated a 10,000 metres RC drilling program in early May 2016, followed by another 10,000 metres from September to December, In February 2017, a 30,000 metre drilling program was initiated that ran until the end of August 2017. The last two programs were more focused on expending the resources at Tijirit's *Eleonore Zone*.

Past operators Shield Mining and Gryphon Minerals have completed more than 35,000 metres of RC drilling and 3,500 metres of core drilling on the project.

Core and RC Drill Holes

A total of 92 diamond drill holes (DDH) and 629 reverse circulation (RC) holes have been completed and analyzed on the Tijirit project for respectively 10,706 and 80,337 metres of drilling. These DDH and RC holes were carried out on the project both by Algold Resources in 2016 and 2017 and past operators Shield Mining and Gryphon Minerals from 2009 to 2012. Annex II – List of Reverse Circulation Holes on the Project. A total of 10 RC hole from Gryphon (12TRC120 to 12TRC125, 12TRC152, 12TRC153, 12TRC157, 12TRC158) and 5 trenches from Gryphon (12TT093 to 12TT097) are located outside of the mining concession 2480C2 and the exploration permit 1117B2.

Algold drilled a total of 32 complete diamond drillholes and 37 diamond extensions. Note that one drill hole is not accounted in the resource estimation because it does not contain assays that allow for it use, but they are listed in the in Annex I. A total of 4 RC holes from Algold (T16RC082 to T16RC085) are located outside of the mining concession 2480C2 and the exploration permit 1117B2.

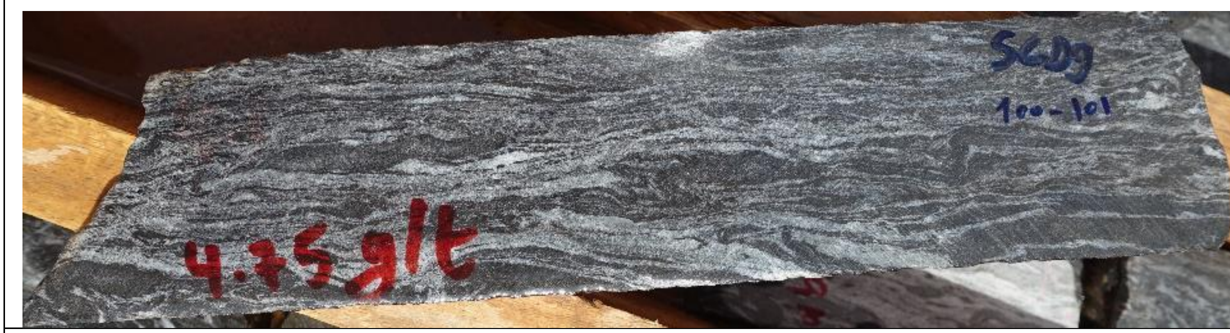
Drill holes are distributed throughout the main anomalous zones. These have been defined by soil geochemistry, surface rock chip samples and trenches and are mainly concentrated on Sophie I & II, Sophie III, Lily and Eleonore prospective zones, as shown in Figure 9-6. More RC holes have been carried out as exploration on various lithological and deformational contacts NNE and east of Eleonore in metasediments and meta-igneous rocks and SSW of Lily in various sheared contacts.

As example, the following shows HQ core mineralized intersections. (Figure 10-1)

- T17DD0001 in quartz veins in altered epidote-albite-chlorite shear zone in intermediate volcanics or metasediment from Eleonore south;
- LCD6 in silicified porphyry from Lily;
- SCD9 in a quartz-carbonate veined iron-rich metasediments from Sophie I & II.

In Eleonore South, T17DD0001 sample A035959 shows 2.97 g/t Au from quartz veins in altered epidote-albite-chlorite shear zone in intermediate volcanics or metasediment. In Lily, LCD6 contains a fair amount of sulphides, predominantly pyrite, but gold is primarily related to the silica replacement. In Sophie. SCD9 iron-formation also shows some sulphides but fine-grained gold is believed to be related to the deformational and fluid event that allowed quartz and carbonate veining. Iron-formation in Figure 10-1 shows heavy folding. The core angle is not representative of the entire hole.

T17DD0001 core in quartz veins in altered epidote-albite-chlorite shear zone in intermediate volcanics or metasediment from Eleonore South.



SCD9 core in mineralized quartz-carbonate veined iron-formation from Sophie I & II.

LCD6 core in sulphide-rich mineralized silicified porphyry from Lily.

Figure 10-1: HQ Core Mineralized Intervals

Various RC holes confirmed mineralized intersections in the main anomalous soil areas, but also in Eleonore East and Salma Zone, see Figure 10-2 and Figure 10-3. Outside the main prospects, mineralization is also related to NNE-trending sheared deformation zones or to variously oriented sheared contact between meta-igneous and metasedimentary rocks. Annex III – Drillholes Intersections Above 0.27 g/t Au (see in Annex III) is a compilation of the RC and DDH intersections above 0.27 g/t Au. It shows grade values between 0.3 and 76.5 g/t Au and a median value of 0.6 g/t Au. Intersection lengths vary between 0.25 and 38 m with a median value of 2 m. Most of the intersections do coincide with major soil anomalies in Sophie I, II & III, Lily and Eleonore, see Figure 9-1, Figure 9-2, Figure 10-3 and Figure 9-6.

Survey of the Drilling

Drysdale and Associates surveyed from February 19 and 23, 2017. Ground control points were established in the main prospects and WGS 84 UTM coordinates generated by post processing. The 2016 Algold drill holes were surveyed by RTK GPS and several historic drill holes surveyed to confirm accuracy.

The 2017 Algold Drilling Survey was conducted by Consulting Training Group (CTG). The device was calibrated by ground control points positioned by the Gryphon company and by the validation of the coordinates of some polls done by RTK GPS in February 2017.The concrete DH collar monuments were surveyed at the casing (X, Y) and elevation (Z) calculated from the actual ground level.

All 2016 drilling except 4 holes; T16RC023, TC16RC41, T16RC078, T16RC079, where the collar was destroyed, were surveyed to better than 5 cm accuracy in the XYZ.

There was excellent comparison in the X and Y between the Algold survey and the previous survey that generated the historic drill hole collar coordinates. However, the elevation (z) could not be reconciled based on the post processing data generated in the February 2017 survey and the historic survey. With no comparable geoid found to calculate the historic DH collar elevations, it was decided that the new survey be shifted to use the existing elevation “mine height”. A +8.75 m shift was applied to the EGM2008 Geoidal height to bring it in line with the previous survey. It is assumed that there was an error when entering the location for the original survey ground control point hence the shift in elevation.

Verification between the coordinates of the holes in the database and the surveyed holes was done by the author of the current report. The fieldwork included the verification of collar locations by GPS readings with the WAAS correction. All drillholes can be found in the field and most are well identified with a proper monument. Table 12-1 shows the list of 18 collars with position comparisons that match very well each time.

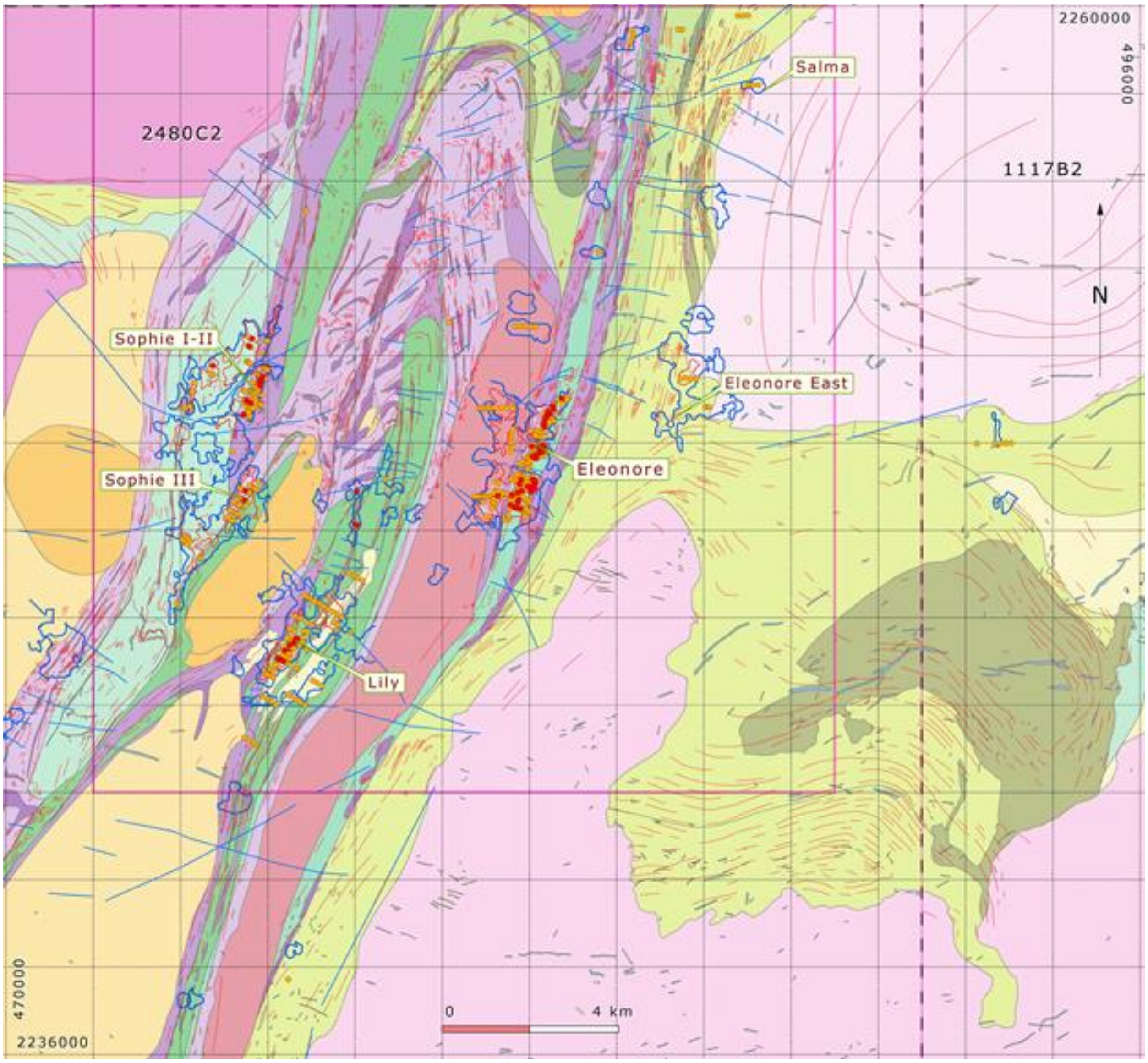


Figure 10-2: Tijirit Distribution of RC (orange) and Core Drill (red) Holes

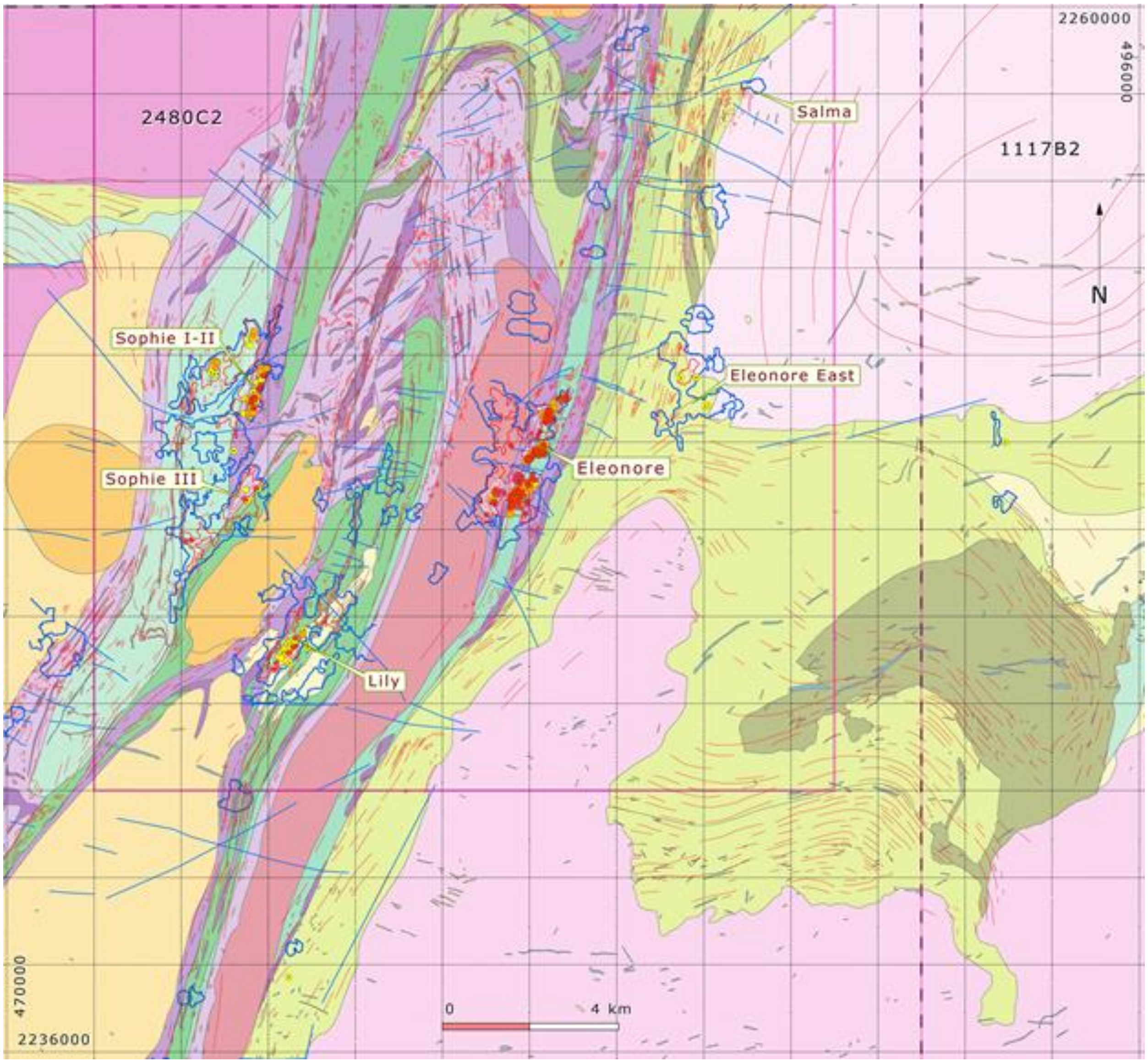


Figure 10-3: Tijirit Mineralized Drillholes > 0.9 g/t Au (yellow: 2009-2012 brown: 2016 red: 2017)

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

A standard operating procedure (SOP) document has been established by Algold, based on CIM best practice guidelines, and provides guidance and details on quality assurance, quality control (QA/QC) implementation and monitoring during Reverse Circulation (RC) drilling. The procedures and techniques outlined within this protocol are considered to be compliant with NI 43-101 regulations. This has been confirmed by SGS Canada and by Mr. André Ciesielski, D.Sc., Geo., from Algold.

Data provided by previous owners, Shield Mining and Gryphon, are described briefly in the current report and in details in the previous 2016 technical report. The data available suggests that QA/QC for the Shield Mining work (2009-2010, RC holes and trenches) included field duplicates, umpire laboratory duplicates and laboratory checks (same laboratory). Shield Mining apparently did not have standards and blanks inserted in sample batches.

In contrast, there is the full range of QA/QC data for Gryphon work including standards, blanks, field duplicates and pulp duplicates. There is no indication in the QA/QC data if an umpire laboratory was used.

11.1 Procedures Used by Algold

The following is the description of the procedures as described in the standard operating procedure (SOP) document that has been established by Algold and provides guidance and details on quality assurance, quality control (QA/QC) implementation and monitoring during Reverse Circulation (RC) drilling.

In 2016, Algold used ALS Minerals Laboratory (ALS) in Nouakchott, Mauritania and ALS OMAC, Ireland as the primary laboratory for preparation and assaying samples generated on its project. ALS is an accredited laboratory in accordance with ISO 17025 with respect to its 50g fire assay analytical technique (Method Code Au-AA26) for gold samples. Since the beginning of 2017, the primary assay laboratory has been SGS Bamako.

In October 2016, Algold detected some anomalies identified from review of the QA/QC program. Concerns were raised to ALS and in order to guarantee the integrity of the overall process, all batches were verified and corrected data provided to ensure that all 2016 data is valid.

Since the beginning of 2017, analytical work for drill core and reverse circulation chips, geochemical samples and rock chip samples has been carried out at the independent SGS Laboratories Ltd. in Bamako, Mali. The 50 g fire assay with ASS finish analytical services are accredited by SANAS and are carried out with a quality assurance protocol in line with ISO 17025:2005. Prior to 2017, drill samples were prepared in the independent ALS Laboratory in Nouakchott, Mauritania and analysed at ALS Laboratories Ltd. in Loughrea, Co. Galway, Ireland, an ISO 17025 (2005) Certified Laboratory.

- All drilling was conducted by reverse circulation drilling with sampling conducted by riffle splitting to an average of 3 kg for dispatch to the assay laboratory;
- All sampling is conducted on a 1 m basis in mineralized areas (zones of interests, domains);
- Composite sampling is conducted on a 2 m basis in presumed barren geological domains
- Recovered sample weight is recorded at time of sample recovery on a 1 m basis. Data is used to verify recoveries and sample quality. Drilling is terminated if wet samples or poor recovery encountered;

- All drill chips logged on site for geology, alteration and mineralization for incorporation into geological models. A representative sample of the chips on a 1 m basis is retained on site;
- Sample preparation is conducted in a SGS on-site mobile sample preparation laboratory.
- Assaying is conducted at SGS Laboratory Ltd. In Bamako, Mali, since 2017 and previously prepared in the independent ALS Laboratory in Nouakchott, Mauritania and analysed at ALS Laboratories Ltd. in Loughrea, Co. Galway, Ireland;
- In both 2016 and 2017, samples are stored at the Algold field camp and put into sealed bags until delivered by a geologist on behalf of Algold to the respective laboratory where samples are prepared and analyzed. Since September 2017, samples are prepared at a samples preparation laboratory, located within the limit of the Tijirit camp and operated by SGS;
- Algold's samples are logged in the tracking system, weighed, dried and finely crushed to better than 70 %, passing a 2 mm (Tyler 9 mesh, US Std. No.10) screen. A split of 1,000 g is taken and pulverized to better than 85 %, passing a 75-micron (Tyler 200 mesh) screen, and a 50-gram split is analyzed by fire assay with an AA finish. Selected samples may be re-analyzed using a 1 kg cyanide leach (Bottle Roll) using "LeachWELL" or a 1 kg screen fire assay method. These results automatically supersede the original 50 g fire assay result.
- As part of Algold's Quality Assurance and Quality Control (QA/QC) procedures; blanks, duplicates and certified reference material (standards) are routinely inserted within the sample stream to monitor laboratory performance during the preparation and analysis:
- A CRM sample (50-100g vacuum sealed sachet) is inserted into the sample stream after every 20 samples;
- Coarse blank samples are inserted into the sample stream at the beginning of every 20 samples and at selected positions within and immediately following known mineralization, to monitor potential contamination during sample preparation. Algold has used locally sourced granite, and barren quartz material provided by the sample preparation laboratories;
- Field duplicates are generated from a selected 1 m sample at a frequency of 1 in 20 samples. Ideally duplicates are selected from possible mineralized samples;
- Pulp duplicates are carried out routinely by the assay laboratory;
- Between 4-5 % of original assay pulps from the primary laboratory is submitted to a second accredited laboratory (SGS Lakefield, Canada) for umpire analysis to monitor bias and reproducibility of results. Whilst samples are randomly selected, it is ensured that the full spectrum of grades and geology be re-analyzed. CRMs are also added in sequence at a frequency of 1 in 20 along with the pulp duplicates;
- In summary, QC samples make up of a total of 20-25 % of the samples submitted and analyzed;
- If some samples come back barren where mineralization was anticipated, the “B” sample of the 1 m split is re-analysed using the screen fire assay method. A new sample ID is given (from a difference sequence than the regular samples).

For drill core, the same QA/QC procedures are used as per the above description. Sampling is carried out honouring differences in alteration and lithological contacts with core split in half using a saw. Field duplicates are taken as quarter core splits of the same interval. Procedures for DDH are likely the same as for RC drilling.

11.1.1 Core Sample Quality and Sample Representativity

The author noted that the recovery was good to very good (95-100 %) in fresh rock, and generally better than 70 % in saprolite. For RC drilling, recovery is generally excellent, the top 4 metres of the hole experience some sample loss.

11.1.2 Standards Statistics

A total of 10 certified standards were used by Algold. SGS Geostat reviewed 1736 standards in the QAQC database at the effective date of the report since the previous technical report dated June 2017. The 1736 assays belong to 10 certified standards. The basic statistics of the standards are in Table 11-1. There were no evidences of wrongly identified standards (see Figure 11-1). The ratio of warnings is of 1.5 % and the ratio of failures is of 0.23 %. The detailed performance graphs are presented in Figure 11-2 to Figure 11-6.

The right graph of Figure 11-2 shows the results of the Oreas-65a standards, which recorded 3 warnings and 4 failures. Despite the 15 % rule of SGS Bamako, Algold should take action against the failures. A standard that has less gold than expected may be more acceptable because of partial digestion, but a higher grade than the expected value is problematic. Concerning the 4 failures, the batches that included 3 of these failures were not used in the resource estimate, and for one of the four failures, Algold contacted the laboratory who repeated the assays of 10 % of the samples and Algold followed up.

The right graph of Figure 11-4 shows results for the Oreas-15d standards that lists 6 warnings. The results are 3 % lower than the expected value. The digestion method is Aqua Regia and cyanide extraction, and it is a partial digestion so lower values are expected. SGS Bamako's rule is to re-analyse a sample batch if the gold is 15 % below the expected value.

The left graph of Figure 11-4 shows results for the Oreas-224 standards that lists 5 warnings. These warnings are still within acceptable limits according to the author and the laboratory.

The right graph in Figure 11-6 shows the results of the Oreas-228 standards that have recorded 9 warnings. Algold contacted the laboratory who reported that there was a problem with the micropipette used for dilution. As the problem seemed to concern only standards with high grades and that other standards behaved in an acceptable way, no changes were made. We can also notice that the warnings took place at the beginning of the period of use of this standard, and that then the values stabilized.

The results of the Oreas-200, AMIS0482, and Oreas-206 standards in Table 11-1 show that there was one warning for each of these standards. These warnings are still within acceptable limits according to the author and the laboratory.

These results are acceptable and confirm that the database is reliable.

Table 11-1: Basic Statistics on the Standards

Standards Id	Count	Expected Value	Standard Deviation	Warning Count	Fails Count
Oreas-200	313	0.34	0.012	1	0
Oreas-65a	209	0.52	0.017	3	4
AMIS0221	73	1.14	0.040	0	0
AMIS0482	241	1.46	0.085	1	0
Oreas-15d	138	1.56	0.042	6	0
Oreas-224	27	2.15	0.053	5	0
Oreas-206	321	2.20	0.081	1	0
Oreas-67a	163	2.24	0.096	0	0
Oreas-228	78	8.73	0.293	9	0
Oreas-208	173	9.25	0.438	0	0
Total	**1736**			**26**	**4**

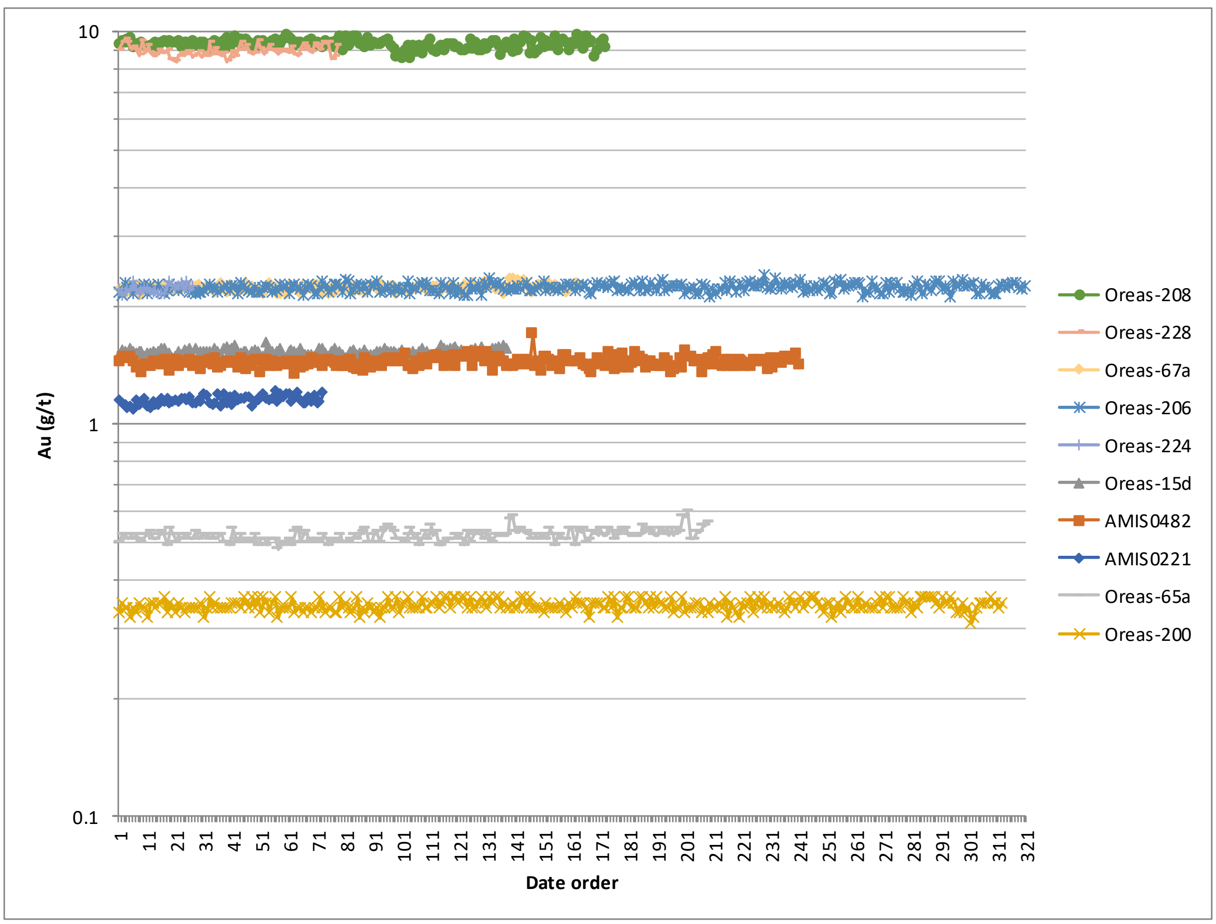
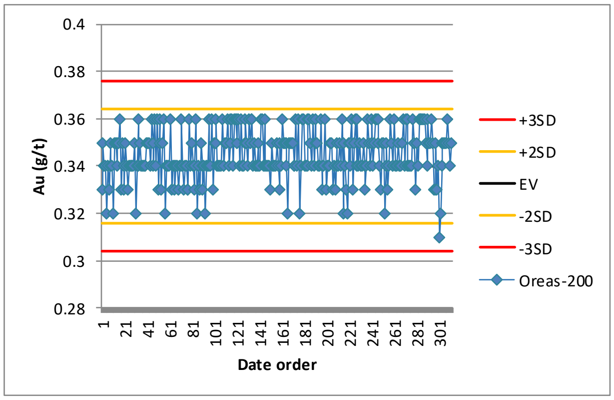


Figure 11-1: Individual Assays for the 10 Standards Studied by SGS



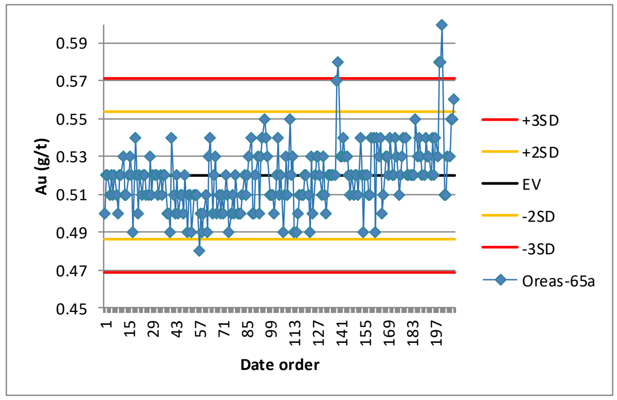
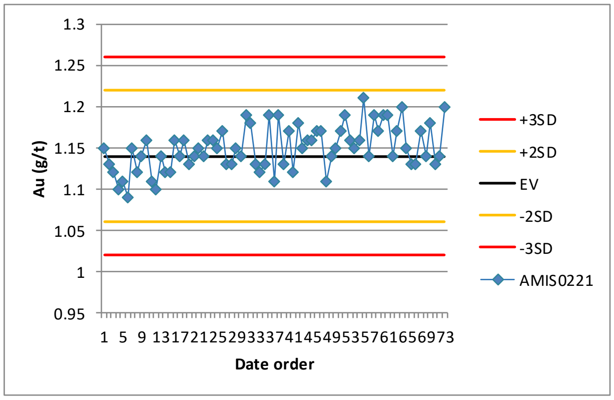


Figure 11-2: Performance for the QA/QC Standards Oreas-200 (Left) and Oreas-65a (Right)



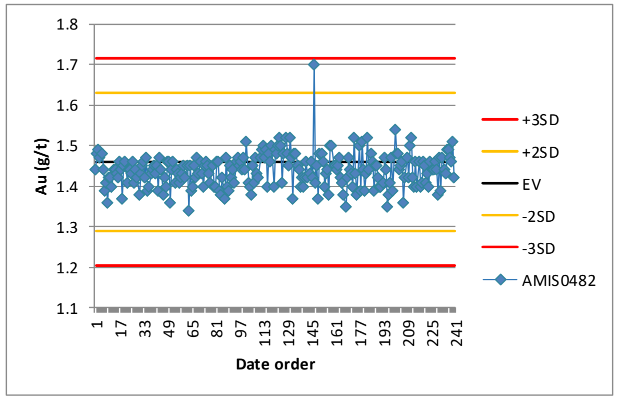
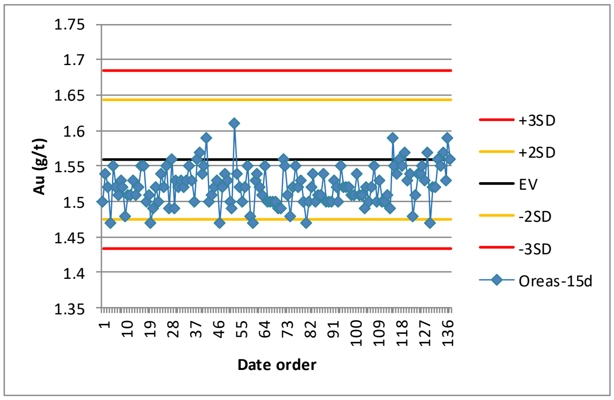


Figure 11-3: Performance for the QA/QC Standards AMIS0221 (Left) and AMIS0482 (Right)



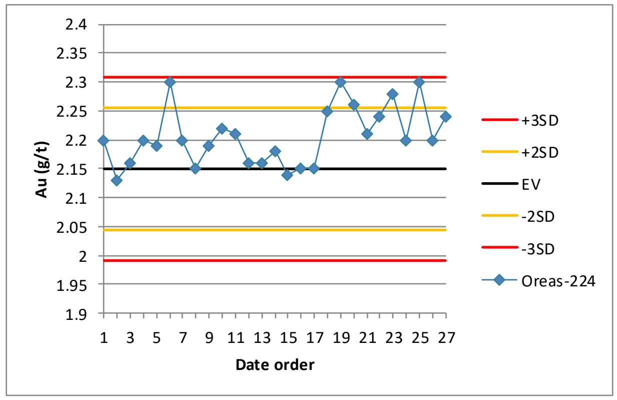


Figure 11-4: Performance for the QA/QC Standards Oreas-15d (Left) and Oreas-224 (Right)

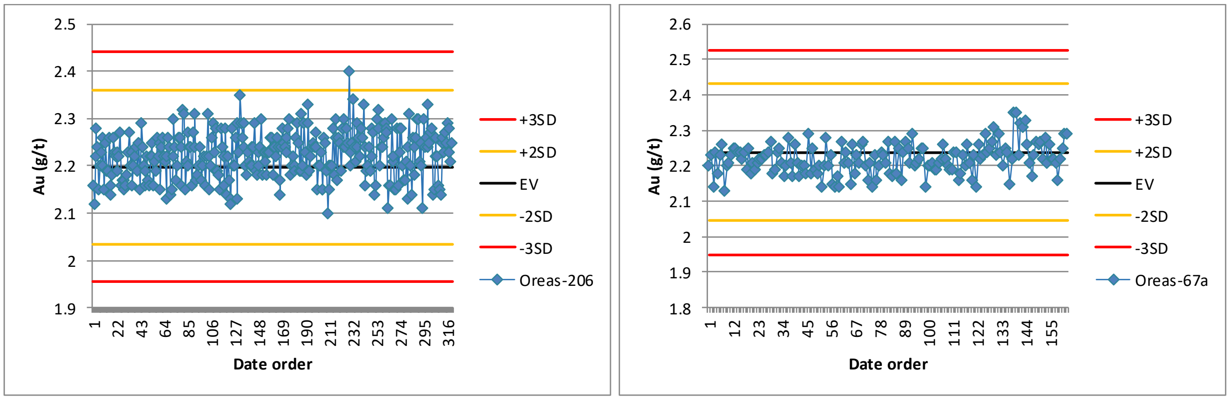


Figure 11-5: Performance for the QA/QC Standards Oreas-206 (Left) and Oreas-67a (Right)

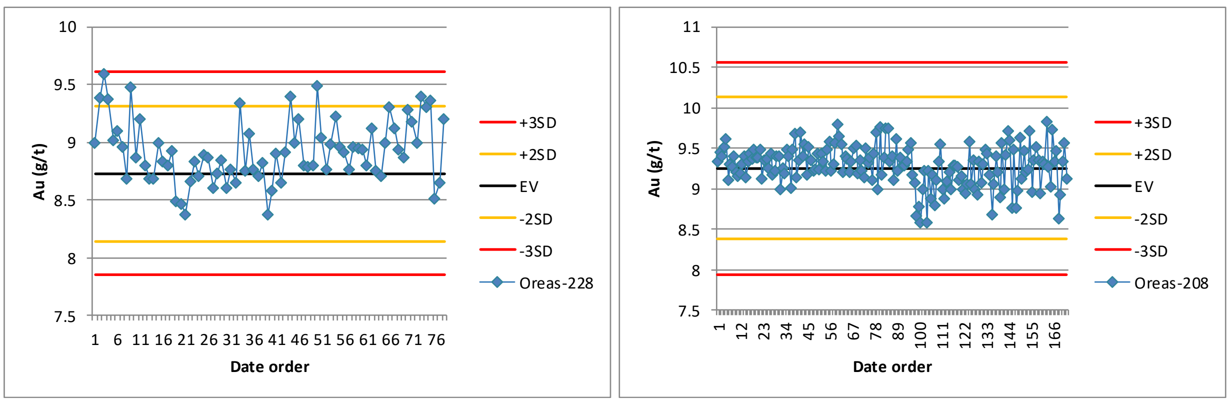


Figure 11-6: Performance for the QA/QC Standards Oreas-228 (Left) and Oreas-208 (Right)

11.1.3 Blanks Statistics

A total of 1,015 blanks were in the QA/QC tables of Algold at the effective date of the report since the previous technical report dated June 2017. The blanks were dated from April 2, 2017 to December 28, 2017. If we set the warning threshold at 5 times the detection limit (0.01 g/t) and the failure threshold at 10 times the detection limit, we get no warning and one failure. This failure was included in the following graph presented in Figure 11-7.

For the control samples that exceed the failure threshold at ten (10) times the detection limit, Algold did a follow up with the laboratory, but it was not possible to do a re-assay. Algold traced the error back to site with how the blank was prepared (crushed on site by Algold using contaminated crushing pot) being the problem. There were no significantly mineralised samples 77 samples after and 68 samples prior, so no re-assaying was taken. As the lab's internal QC was correct and all other external QC also, Algold didn't re-assay the pulp. The umpire lab samples from this batch confirmed the original assays.

The database that SGS used to estimate the resources of the Project includes the results of the certificate of analysis associated with this control sample. These results are acceptable and confirm that the database is reliable.

Figure 11-7: Performance for the QA/QC Blanks

11.1.4 Field Duplicates (Mixed RC and DDH, Prepared by Algold on Site)

A total of 1,025 field duplicates were prepared by Algold at the effective date of the report since the previous technical report dated June 2017. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 215 pairs with an average grade above 0.05 g/t. The pairs resulting graphs (scatterplot and QQ plot) are shown in Figure 11-8. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and no bias can be identified. The average grade of the 215 pairs is of 1.36 g/t, the average variance for the pairs is of 9.65 therefore the coefficient of variation is of 228 %. There are 2 high grades in the original assays that could not be reproduced by the duplicates. These 2 results are not statistically significant. Therefore, these results are acceptable and confirm that the database is reliable.

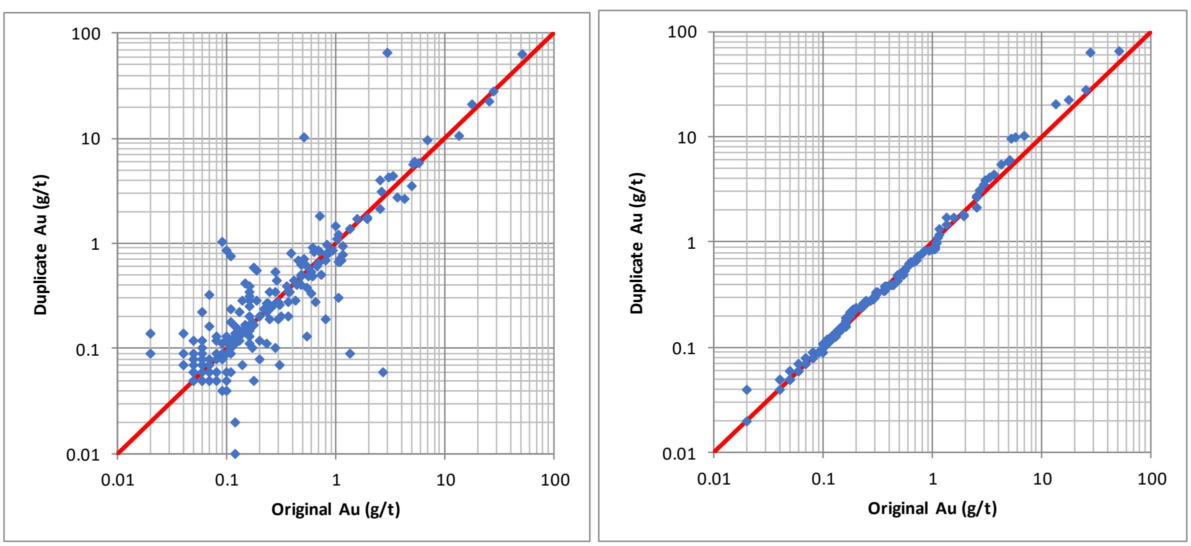


Figure 11-8: Scatterplot of the Algold Field Duplicates (Left) – QQ plot (Right)

11.1.5 Pulp Duplicates (Randomly Selected by Algold)

Since the June 2017 technical report, no pulp duplicates were prepared by Algold. SGS did the verification of bias for the 2017 report, and the results of the pulp duplicates were acceptable and confirmed that the database was reliable. Please refer to the technical report dated June 2017 for detailed information on pulp duplicates.

11.1.6 Umpire Duplicates on the Algold RC Drillholes

A total of 937 umpire duplicates sent to SGS Lakefield were prepared on the Algold RC drillholes at the effective date of the report since the previous technical report dated June 2017. Algold randomly selected pulps and sent them to SGS. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 415 pairs with an average grade above 0.05 g/t. The pairs resulting graphs (scatterplot and QQ plot) are shown in Figure 11-9.

SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and no bias can be identified. The average grade of the 415 pairs is of 1.46 g/t, the average variance for the pairs is of 5.34 therefore the coefficient of variation is of 158 %. By removing the pair with the largest variance, the statistical test results remain conservative. Thus, the average grade of the 414 pairs is of 1.39 g/t, the average variance for the pairs is of 0.47 therefore the coefficient of variation is of 49 %. The QQ plot reveals that the low values are slightly lower for the umpire lab and that the highest values are slightly higher for the umpire lab. These results are acceptable and confirm that the database is reliable.

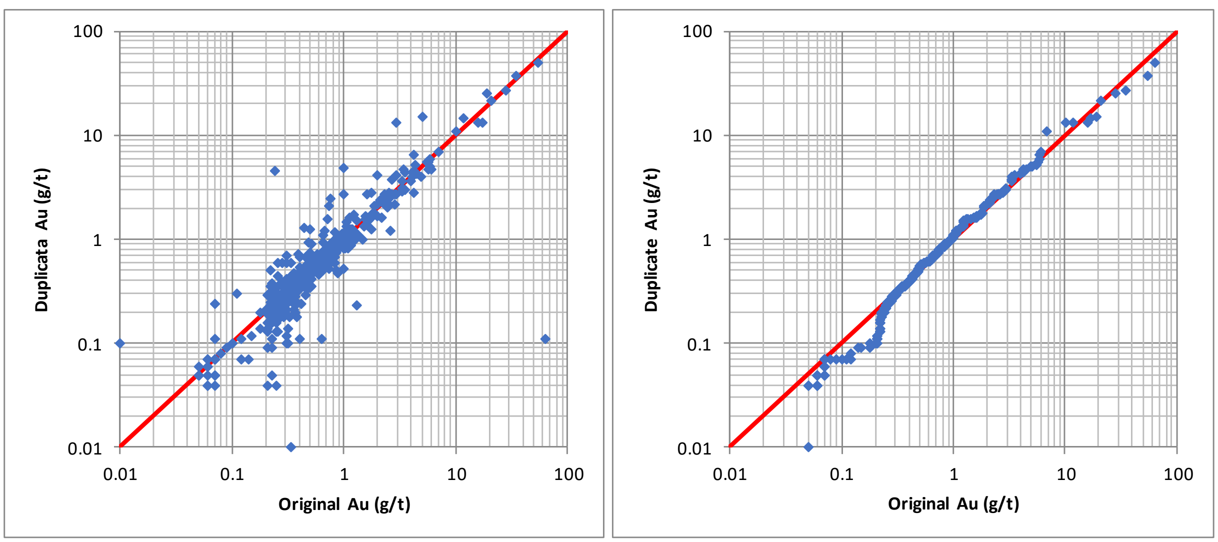


Figure 11-9: Scatterplot of Umpire Duplicates on the Algold RC Drillholes (Left) – QQ plot (Right)

11.1.7 Laboratory Duplicates on Algold RC Drillholes (Chosen by the Lab)

A total of 1807 duplicates were prepared on Algold 2017 RC drillholes up to the effective date of the report since the previous technical report dated June 2017. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 1132 pairs with an average grade above 0.05 g/t. CK1 and/or CK2 assays were performed by the laboratory and were used together. No variability of the data was found.

The pairs resulting graphs (scatterplot and QQ plot) are shown in Figure 11-10. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and no bias can be identified. The average grade of the 1132 pairs is of 1.27 g/t, the average variance for the pairs is of 0.55 therefore the coefficient of variation is of 58 %. These results are acceptable and confirm that the database is reliable.

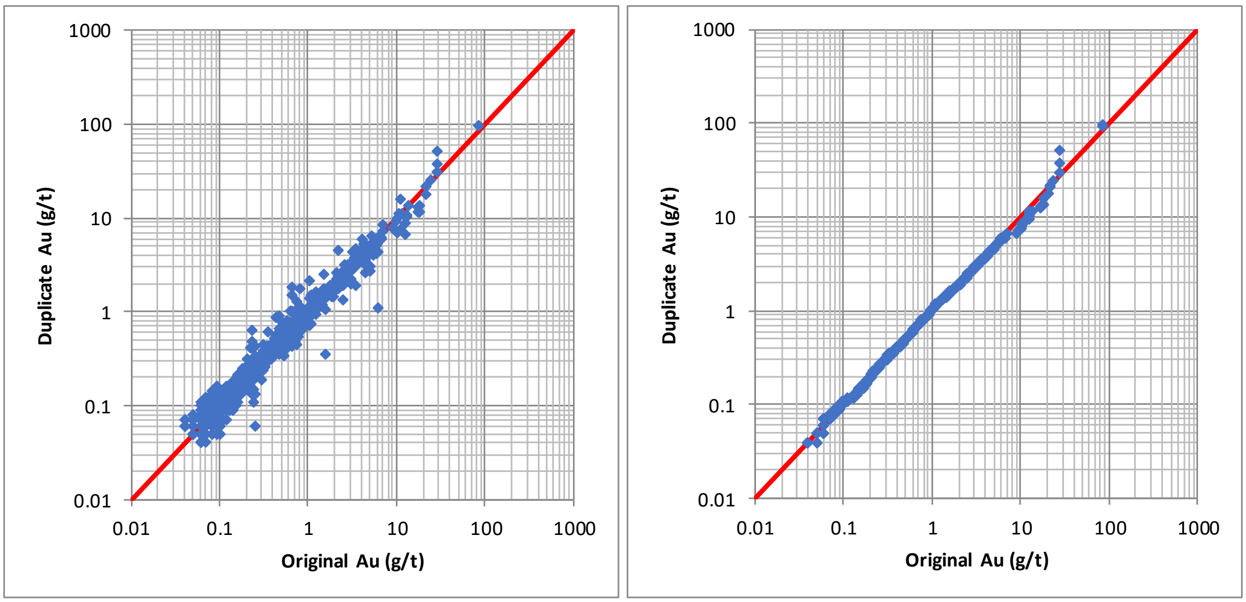


Figure 11-10: Scatterplot of the Algold RC Drillholes Duplicates (Left) – QQ plot (Right)

11.1.8 Laboratory Duplicates on Algold DDH (Chosen by the Lab)

A total of 758 duplicates were prepared on Algold 2016-2017 DDH up to the effective date of the report since the previous technical report dated June 2017. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 466 pairs with an average grade above 0.05 g/t. CK1 and/or CK2 assays were performed by the laboratory and were used together. No variability of the data was found.

The pairs resulting graphs (scatterplot and QQ plot) are shown in Figure 11-11. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and no bias can be identified. The average grade of the 466 pairs is of 1.98 g/t, the average variance for the pairs is of 0.24 therefore the coefficient of variation is of 25 %. These results are very good and confirm that the database is reliable.

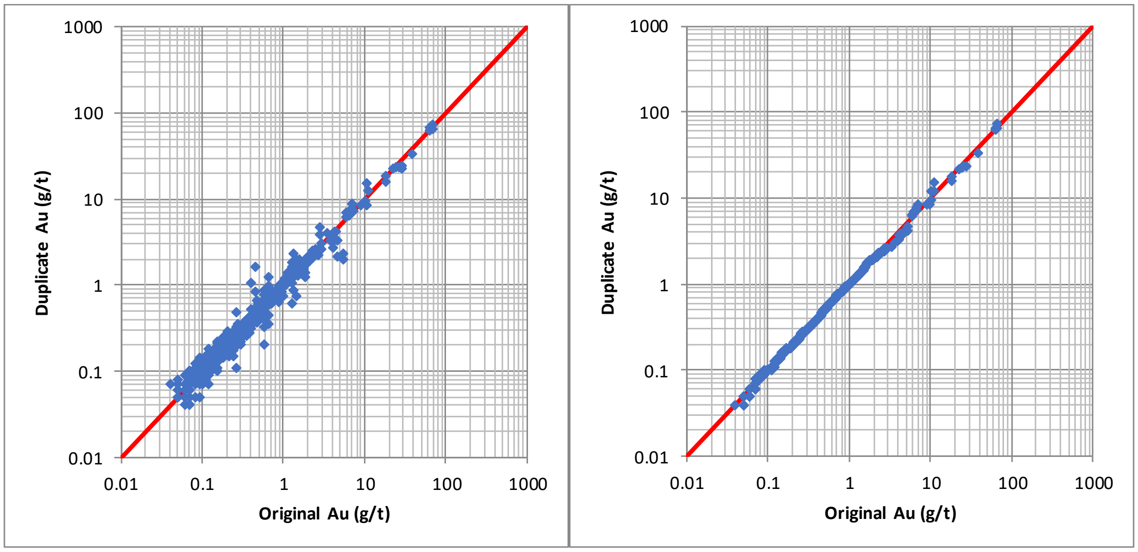


Figure 11-11: Scatterplot of the Shield RC Drillholes Duplicates (Left) – QQ plot (Right)

11.1.9 Cyanide Bottle Roll Tests

A total of 478 cyanide bottle roll tests were prepared on Algold 2016-2017 RC drillholes and DDH up to the effective date of the report. The pairs resulting graphs (scatterplot and QQ plot) are shown in Figure 11-12. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test. A bias was detected by the student t-test and logarithmic student t-test. SGS Geostat observed a bias on high values and noted that high values are higher for cyanide bottle roll tests.

The average grade of the 477 pairs is of 1.80 g/t, the average variance for the pairs is of 1.43 therefore the coefficient of variation is of 66 %. These results are good and confirm that the database is reliable.

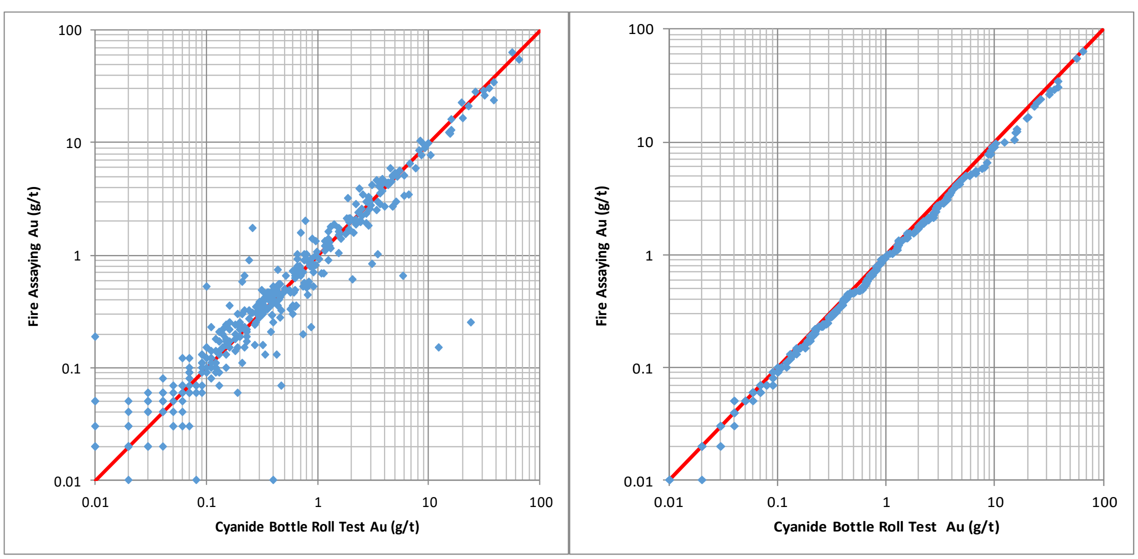


Figure 11-12: Scatterplot of the Cyanide Bottle Roll Tests on RC Drillholes and DDH (Left) – QQ plot (Right)

11.2 Procedures Used by Gryphon

The following is the description of the procedures as described by Mr. Ahmed Mohamed Lemin, Database Manager of Gryphon:

- All drilling was conducted by reverse circulation drilling with sampling conducted by riffle splitting to 3 kg for dispatch to the assay laboratory;
- All sampling conducted on a 1 m basis;
- Moisture content and recovered sample weight were recorded at time of sample recovery on a 1 m basis. Data used to verify recoveries and sample quality. Drilling terminated if wet samples or poor recovery encountered;
- No sample recovery or quality issues were encountered during the drill program likely to impact on the quality of data derived;
- All drill chips logged on site for geology, alteration and mineralization for incorporation into geological models. A representative sample of the chips on a 1 m basis retained on site;
- All RC chips are photographed for digital storage;
- Assaying and sample preparation conducted at SGS Laboratory in Kayes, Mali;
- 3 kg samples as received from Gryphon Minerals are dried and crushed to 6 mm before being quartered using a Rocklabs splitter;
- 1 quarter is then pulverised by ring mill to 70-75 microns and 200g recovered as the master pulp for 50g fire assay;
- All assaying conducted by fire assay with an AAS finish on a 50 g charge;
- Blind standards, blanks and field duplicates inserted at a rate of 5 % in the field and results analysed in the Gryphon Minerals database system. Acceptable accuracy and precision have been established for all samples reported.

11.3 Procedures Used by Shield Mining

No information has been found on Shield Mining’s exact procedures. The data was validated by the author and some QAQC (field duplicates, umpire laboratory duplicates and laboratory checks) is available and shown in the next sub-sections.

11.4 Procedures Used by Gryphon for the Diamond Drillholes

While no details are available, the core is well identified, with some minor depth errors noted in the core box blocks. The same QAQC procedures and laboratory procedures as the Gryphon RC holes were used.

11.4.1 Core Sample Quality and Sample Representativity

The author noted that the recovery was good to very good in mineralized zones observed in holes ECD2, LCD6, SCD1, SCD7 and SCD9.

11.4.2 Standards Statistics

A total of 17 standards (certified (13) and uncertified (4) materials) were used by Gryphon. SGS reviewed 690 of the 872 standards available in the QA/QC database. The 690 assays belong to 9 standards (6 certified and 3 uncertified). There were evidences of wrongly identified standards. SGS did many changes to get the results. While the number of warnings and failures were very much reduced, the author believed the revised results are closer to reality. The final ratio of warnings is of 4 % and the ratio of failures is of 1.7 %. These results are acceptable and confirm that the database is reliable.

According to the 2016 technical report, it was recommended that Algold staff going forward take preventative measures to eradicate the problem of mislabelling of standards.

11.4.3 Blanks Statistics

A total of 948 blanks were sent to the laboratory by Gryphon. If the warning threshold is set at 5 times the detection limit (0.01 g/t) and the failure threshold at 10 times the detection limit, the results shown 1 warning and 1 failure. These results are acceptable and confirm that the database is reliable.

11.4.4 Field Duplicates (Gryphon)

A total of 655 field duplicates were prepared by Gryphon. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 62 pairs with an average grade above 0.05 g/t. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and no bias can be identified. The average grade of the 62 samples is of 0.51 g/t, the average variance for the pairs is of 0.66 therefore the coefficient of variation is of 159 %. These results are acceptable and confirm that the database is reliable.

11.4.5 Pulp Duplicates (Gryphon)

A total of 422 pulp duplicates were prepared by Gryphon. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 57 pairs with an average grade above 0.05 g/t. The QQ plot reveals a strong bias. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and a strong bias can be identified by all methods.

The average grade of the 57 samples is of 0.26 g/t, the average variance for the pairs is of 0.18 therefore the coefficient of variation is of 168 % (too high for pulp duplicates). The average for the original assays is of 0.36 g/t and the average for the duplicate assays is of 0.16 g/t, a relative difference of -56 %. The observed bias is unacceptable but should have a sensible explanation. At the time of the writing of the previous 2016 technical report, SGS can only observe but cannot provide an explanation.

11.4.6 Field Duplicates in Trenches

A total of 450 field duplicates were prepared for trenches. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 55 pairs with an average grade above 0.05 g/t. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and no bias can be identified. The average grade of the 55 samples is of 0.21 g/t, the average variance for the pairs is of 0.25 therefore the coefficient of variation is of 239 %. These results are acceptable and confirm that the database is reliable.

11.4.7 Umpire Duplicates on the Shield Trenches

A total of 160 umpire duplicates (probably sent to an umpire laboratory) were prepared on the Shield trenches. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 23 pairs with an average grade above 0.05 g/t. The QQ plot reveals a strong bias.

SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and a strong bias can be identified by all methods. Both t-tests just reach the conclusive threshold of "95 % chance of a bias". This is quite understandable with only 23 pairs to work with. The average grade of the 23 samples is of 0.13 g/t, the average variance for the pairs is of 0.01 therefore the coefficient of variation is of 91 % (reasonable for gold duplicates). The average for the original assays is of 0.16 g/t and the average for the duplicate assays is of 0.09 g/t, a relative difference of -41 %. The observed bias is unacceptable but should have a sensible explanation. At the time of the writing of the 2016 technical report, SGS can only observe but cannot provide an explanation.

11.4.8 Duplicates on Shield RC Drillholes

A total of 435 duplicates were prepared on Shield RC drillholes. Most of the pairs (original and duplicate analysis) reveal a barren material. To avoid the problems caused by material close to the detection limit, SGS retained the 120 pairs with an average grade above 0.05 g/t. SGS did the verification of bias by statistical sign test, student t-test and logarithmic student t-test and no bias can be identified. The average grade of the 120 samples is of 0.71 g/t, the average variance for the pairs is of 0.07 therefore the coefficient of variation is of 38 %. Such a low coefficient of variation should be due to these duplicates being some pulp duplicates. These results are very good and confirm that the database is reliable.

11.5 Conclusion

The QA/QC was reviewed by SGS Geostat and the results are satisfactory. SGS Geostat, given the successful verification of the data and given that most items of the QA/QC are satisfactory, believes that the sample preparation, security and analytical procedures are adequate to support the estimation of resources presented in this Report.

SGS Geostat noted that QA/QC warnings and failures are well attended by Algold employees. There are good standard operating procedures that focus on the resolution of any QA/QC problems found in the results from the laboratory. Algold follows well the current standard operating procedures. Most QAQC warnings and failures found by SGS were "false warnings" because Algold had remedied the situation with the laboratory.

SGS Geostat recommends that Algold removes QA/QC warning and failure data after solving a problem in the data. In this way, the QA/QC warning and failure data present in the database will better represent the quality of the data in the database.

12. DATA VERIFICATION

SGS Geostat performed the following to ascertain that the database supporting the estimation of resources is sound and reliable:

- Verification of the highest assays of the Algold 2017 data against analytical certificates;
- Site visit;
- Independent sampling;
- Multiple others database verifications;
- Verification of bias for RC holes and trenches.

SGS Geostat has prepared public resource technical reports for Algold in August 2016 and June 2017. For both reports, SGS Geostat performed some verification, concluding to adequate data for resource estimation.

12.1 Verification of the Algold 2017 Data

SGS Geostat did a verification of available analytical certificates to ascertain drill database conformity. A total of 200 of the highest assays belong to Algold 2017 new data, which represents 2.7 % of the new assays, have been verified against analytical certificates.

SGS Geostat found exact match between the verified certificates and 200 assays from the database coming from Algold drillholes, previous owners' data has been verified for the preparation of the previous technical report.

12.2 Site Visit

Mr. Yann Camus visited the Tijirit project from the August 14 to 18 of 2017 (5 days).

At the Tijirit camp, Mr. Camus was with Mr. Alastair Gallaugher, C.Geo., BSc. Geology, Algold's on-site geologist responsible for the 2017 drilling campaign. SGS Geostat also met with several Algold geologists, including Messrs. Mohameden Oued Khadim Awen, Kane, and Malum, Mohamed Ghoulam, geographic assistant, Ahmed Brahim M'Body, warehouse manager and responsible for density measurements, and Mohamed Keine, responsible of Algold logistics.

Many subjects were discussed including, but not limited to:

- Structural geology;
- Known mineralized structures and available data;
- Preparation of the 2017 drilling campaign, Phase IV;
- Procedures put in place for drilling, logging, sampling, QA/QC, etc.;
- Potential new targets;
- Comments on the most recent mineralized envelopes by SGS;
- Availability of material for independent sampling by SGS.

During Mr. Camus' visit at the Tijirit project site, the following actions were taken:

- Visit of the diamond drill in operation and the core shack where the core is transported and treated by Algold geologists and drill observation in action, core recovery, core orientation identification, handling and storage in wooden boxes, identification of footage on wooden blocks;
- Visit of the reverse circulation drill in operation and observation of the procedure followed for the preparation of 1 m samples and sometimes 2 m samples when the geologist does not identify any trace of gold mineralization;
- Visit of the storage site for the reverse circulation drill samples at Tijirit camp;
- Visit of a trench;
- Visit of the Algold core shack in Nouakchott with Mr. Mohamed Keine;
- Revision of the protocol used for on-site density measurement with measurement of weight in the air and weight in water with Mr. Ahmed Brahim M'Body;
- Revision of the core sampling protocol;
- Review of Quality Assurance and Quality Control Protocols (QAQC) with Mr. Alastair Gallagher;
- Visit of illegal miners;
- Independent sampling of 45 witness RC samples from 4 holes (see details in the *Verification Re-Assays - Independent Sampling by SGS section*);
- Visit and field observation of the following outcrops:
  - Eleonore (West, South, Center, and North parts);
  - Lily;
  - Sophie I and II.
- The Sophie III zone has been seen without dwelling because it is less important in terms of resources;
- The fieldwork included some structural measurements and verification of collar locations by GPS readings with the WAAS correction. Revision of the procedure used by Algold for determining coordinates using hand-held GPS (Garmin Legend Etrex HCx model). All drillholes can be found in the field and most are well identified with a proper monument. Table 12-1 shows the list of 18 collars with position comparisons that match very well each time.

Figure 12-1: Typical Drillhole Monument (Left) – Messrs. Gallaugher, Ciesielski, and Camus somewhere between Sophie I and Sophie II (Right)

Table 12-1: List of Independently Measured Collar Locations and Validation

Verification Date	Hole Name	Operator	Mineralization Bodies Encountered	SGS Geostat- GPS		Database		Distance (m)
				Easting	Northing	Easting	Northing	
15-Aug-17	T17RC128	Algold	Lily C, Y	476,565	2,245,530	476,563	2,245,530	1.7
	T17RC174	Algold	Lily D, G, M, R	476,154	2,244,769	476,152	2,244,768	2.2
	T16RC112	Algold	Lily, no significant Au	476,776	2,245,522	476,775	2,245,521	1.8
	T16RC142	Algold	Lily, no significant Au	476,867	2,245,546	476,865	2,245,542	4.7
	T16RC113	Algold	Lily B	476,862	2,245,758	476,863	2,245,758	0.9
16-Aug-17	T16RC143	Algold	Sophie 3M	475,472	2,249,684	475,472	2,249,683	1.3
	T16RC020	Algold	Sophie 2I	475,653	2,250,674	475,653	2,250,672	2.3
	T16DD006m	Algold	Sophie 2I	475,603	2,250,784	475,602	2,250,785	1.6
	SRD19	Algold	Sophie 2G, 2H	475,607	2,250,916	475,607	2,250,918	1.8
	T16RC122	Algold	Sophie 2B	475,810	2,251,458	475,809	2,251,460	1.7
	T16RC065	Algold	Sophie 2B, 2M, 2N	475,828	2,251,619	475,824	2,251,622	4.9
	T17DD005	Algold	Eleonore North 2	482,751	2,251,022	482,751	2,251,024	1.5
	T16DD010	Algold	Eleonore North 10	482,527	2,250,785	482,527	2,250,785	0.5
	T16RC099	Algold	Eleonore North 25	482,404	2,250,692	482,403	2,250,691	1.6
	T17DD006	Algold	Eleonore Centre, no significant Au	482,241	2,249,902	482,240	2,249,902	1.1
	T17RC025	Algold	Eleonore West 1, 2	481,211	2,248,804	481,206	2,248,802	5.3
	T17RC154	Algold	Eleonore South-South 1, 5, 20, 21, 23	481,557	2,248,688	481,554	2,248,683	5.7
	T16RC147	Algold	Eleonore South-South 16, 17	481,579	2,248,436	481,575	2,248,436	3.6

12.3 Witness Samples Found at the Tijirit Camp Site

The witness samples that are present at the Tijirit Camp Site are mainly of 4 types:

- Half cores of the diamond drill holes that are well preserved in core boxes;
- RC trays containing coarse material representative of each metre (See Figure 12-2);
- Witness samples split at the drill from the RC drilling (about 2 kg each);
- Pulps returned from the laboratory.

While the first 2 first types of witness samples are fairly well organised, the last 2 types of witness samples, at the time of the site visit, were not well organised; some labels had been erased or sometimes bags are pierced rendering the sample unusable. Some double bagging procedures and improved bag identification should be explored by Algold.

Figure 12-2: Core Boxes from Diamond Drillholes (Left) and Coarse Chip Trays from RC Drillholes (Right)

Figure 12-3: Witness Samples from the RC

The half cores are well identified, and Mr. Camus reviewed the core for drillholes SRD70 and T17RD165. Some visible gold was observed inside hole T17RD165 at about 98 m depth (corresponds to mineralized volume Sophie 3D). Figure 12-4 presents the visible gold as photographed by Mr. Camus.

Figure 12-4: Visible Gold (Inside Red Circles) as Photographed by the Author – Hole T17RD165 (Sophie 3)

12.4 Verification Re-Assays – Independent Sampling by SGS

For the validation of the data used for this resource estimation, SGS Geostat looked for identifiable witness samples whilst on site in Tijirit. A total of 45 witness samples were selected to be re-sampled, all of which are from Algold's 2016-2017 reverse-circulation (RC) drilling program. They are witness samples split at the drill and are of about 2 kg each and are from 4 holes: T16RC021 (13 from Sophie II), T16RC052 (8 from Lily), T17RC045 (12 from Eleonore South) and T17RC050 (12 from Eleonore Center). The author supervised the collection of the witness samples and the re-bagging of the 45 samples (See Figure 12-5 for pictures of the bags before final packing for the transfer to SGS Bamako laboratory in Mali). A total of 5 QAQC samples were added. The author himself escorted the samples from the Tijirit camp to the truck leaving Nouakchott for the SGS laboratory in Bamako – Mali for preparation of the samples and then the samples have been analysed at SGS laboratory in Ouagadougou – Burkina.

The list of the re-sampled intervals can be found in Table 12-2. Since some of the samples in the database are 2 m in length, while the independent samples that were selected and analyzed are all 1 m in length, an average of the independent sample results corresponding to the same lengths has been done. From 45 intervals of 1 m collected for independent analysis, SGS Geostat compiled 30 intervals to compare with the original samples from the database.

In addition to independently verifying the gold grades in the database, SGS aimed to verify the potential benefit of using the bottled cyanidation method, which is known to be the best method for revealing gold for some coarse gold projects. Results on the 30 samples showed a bottled cyanidation content 17 % lower than the average of the fire assay contents. Given the poor results on 30 samples, fire assay could be encouraged because it is less expensive. However, QA/QC results on 478 samples show that rich samples are generally enhanced by bottled cyanidation. On average, there is a gain in the use of cyanidation in bottles. The QAQC analysis is more reliable given the number of samples and therefore bottled cyanidation is a recommended method for the Project.

In 2016, SGS Geostat wanted to justify the possible advantage of using the Metallic Screen method. Given the poor results on 45 samples, Fire Assaying was encouraged because it is less expensive.

The comparison of the Database Au grade versus the SGS Bamako independent laboratory Au grade showed that the database is reliable. SGS Geostat tried to identify a bias using 3 statistical tests: the sign test, the student-t test and the logarithmic student-t test. None of them identified a bias. The results of the comparison are shown in Table 12-2 and Figure 12-6.

Figure 12-5: Pictures of the 45 Independently Re-Assayed Samples

Table 12-2: List of 45 Independently Re-Assayed Samples – Database Au (g/t) vs Independent SGS Bamako Au (g/t)

Sample Number	Hole Name	From (m)	To (m)	Algold Database Au (g/t)	SGS Bamako Ind. Sampling Au (g/t)	Difference
C36626	T17RC045	52	54	0.005	0.005	0 %
C36628	T17RC045	54	56	65.2	8.365	-87 %
C36630	T17RC045	56	58	15.3	11.19	-27 %
C36633	T17RC045	58	60	1.9	0.93	-51 %
C36635	T17RC045	60	62	2.08	1.68	-19 %
C36637	T17RC045	62	64	0.21	0.21	0 %
C36639	T16RC052	94	96	0.29	0.19	-34 %
C36641	T16RC052	96	97	0.37	0.54	31 %
C36643	T16RC052	97	98	5.11	4.89	-4 %
C36644	T16RC052	98	100	1.55	1.36	-12 %
C36646	T16RC052	100	102	0.48	0.425	-11 %
C36648	T17RC050	22	24	0.05	0.045	-10 %
C36650	T17RC050	24	26	38.2	53.06	28 %
C36653	T17RC050	26	28	32.1	34.84	8 %
C36655	T17RC050	28	30	4.69	21.945	79 %
C36657	T17RC050	30	32	0.75	0.845	11 %
C36659	T17RC050	32	34	0.06	0.045	-25 %
C36661	T16RC021	66	67	0.12	0.11	-8 %
C36662	T16RC021	67	68	5.05	4.7	-7 %
C36664	T16RC021	68	69	0.69	1.14	39 %
C36665	T16RC021	69	70	0.25	0.28	11 %
C36666	T16RC021	70	71	0.07	0.13	46 %
C36667	T16RC021	71	72	0.32	0.24	-25 %
C36668	T16RC021	72	73	0.45	0.4	-11 %
C36669	T16RC021	74	75	3.46	2.52	-27 %
C36670	T16RC021	75	76	3.18	1.85	-42 %
C36671	T16RC021	76	77	1.78	1.3	-27 %
C36673	T16RC021	77	78	1.4	1.26	-10 %
C36674	T16RC021	78	79	0.53	0.1	-81 %
C36675	T16RC021	79	80	0.07	0.47	85 %
AVERAGE				**6.19**	**5.17**	**-17 %**

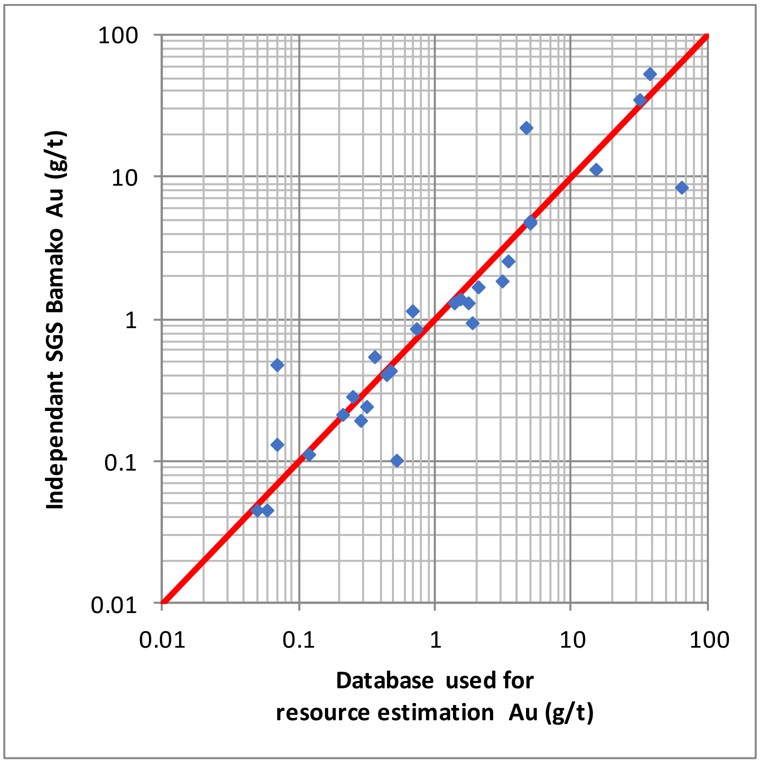


Figure 12-6: Scatterplot of the 30 Independently Re-Assayed Samples – Independent SGS Bamako Au (g/t) vs Database Au (g/t)

12.5 Database Verification

Standard verifications were carried out: extreme values, data going beyond hole depth, check of gaps in the information, search of collars inconsistencies. Only minor details needed any changes and all was fine for the resource modeling and estimation.

But since the database included different types of data, it was important to verify if a bias could be found. It is well recognised in the industry that diamond drillholes (DDH) are very reliable. Then reverse circulation (RC) drillholes have to be watched and also trenches.

In the 2017 technical report, the DDH by Gryphon were compared to some RC drillholes by Shield. The trenches by Shield and Gryphon were compared to DDH by Gryphon. Since all verifications are satisfying, it is also accepted that both work by Gryphon and Shield are sound and reliable for the estimation of resources. Please refer to the technical report dated June 2017 for detailed information of this verification.

12.6 Conclusion

The verification of the 2017 Algold database is satisfactory for the preparation of the resource estimation. The site visit allowed multiple verifications including the identification of visible gold in one drillhole. Everything in reality corresponded well to the information on paper. The independent sampling of 45 witness samples confirmed the database information. No bias was identified.

The standard database verifications run by SGS Geostat indicates a sound database, reliable for the estimation of resources. From the bias study between RC and DDH, and between trenches and DDH, SGS Geostat concludes that the DDH, RC and trenches can be used together for the estimation of resources.

In the previous technical report, the DDH by Gryphon were compared to some RC drillholes by Shield. The trenches by Shield and Gryphon were compared to DDH by Gryphon. Since all verifications are satisfying, it is also accepted that both work by Gryphon and Shield are sound and reliable for the estimation of resources.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

To date, two series of metallurgical tests have been completed on the Project by Algold. The first series performed in the fall of 2016 included 4 composites. The second series started in September 2017 on 5 composites.

13.1 Mineral Processing Testwork 2016

Preliminary metallurgical testing was carried out in Q4-2016 at SGS located in Lakefield, Canada, on four composites using HQ core samples collected from the Eleonore, Sophie I and Sophie II, and Lily zones. The Eleonore composite was characterized by quartz veins associated with metasediments. The Sophie I composite was made up of quartz veins within banded iron formations ("BIF"). The Sophie II composite was made up of weathered BIF and the Lily composite was made up of metasediments. Each composite weighted between 70 kg and 130 kg.

Utilizing fire assay analysis, the head grade for each of the composites were as follows in Table 13-1.

Table 13-1: Head Grades

Zone	Composite No	Rock Type	Samples Weight (kg)	Head Grade (g/t Au)
Eleonore	1	Quartz Vein with Metasediments	130.10	22.8
Lily	2	Metasediments	131.5	0.67
Sophie II	3	Weathered BIF	70.55	1.53
Sophie I	4	Qtz Veins and BIF	90.20	5.36

The testwork program focused on the amenability of the samples to a heap leaching, gravity separation and cyanidation. A comminution study and environmental (acid rock drainage) testwork were also completed. Full HQ core samples were received for the test program and they were prepared into 4 main composites (Comp 1 to Comp 4). Two additional composites (Comp 5 and Comp 6) were also created but were only submitted for gold head assay.

A summary of the head assays is presented in Table 13-2.

Table 13-2: Head Assay Summary

Element	Unit	Comp 1 Eleonore	Comp 2 Lily	Comp 3 Sophie 2	Comp 4 Sophie 1	Comp 5 Eleonore	Comp 6 Eleonore
Au Cut A	g/t	18.3	0.59	1.55	5.16	2.78	<0.02
Au Cut B	g/t	27.2	0.74	1.51	5.56	0.42	<0.02
Au Avg.	g/t	22.8	0.67	1.53	5.36	1.60	<0.02
Au Calc.	**g/t**	***19.4 ****	***1.22***	***1.98***	***5.29 *****	…	…
Ag	g/t	15.1	1.0	0.6	0.9	…	…
S_T	%	2.12	1.14	0.17	3.76	…	…
$S^=$	%	1.93	1.04	0.17	3.64	…	…

Au Calc. = calculated average head grade from testwork
** Calculated (weighted average) from G-1, G-5 and CN-17R*
*** Calculated (weighted average) from G-4 and G-6R*

The direct gold head grades (duplicate 50 g fire assay aliquots) ranged from 0.67 g/t to 22.8 g/t for Comp 1 to 4. The silver and sulphur head grades ranged from 0.6 g/t to 15.1 g/t and 0.17% to 3.76%, respectively. The gold head grades for Comp 5 and 6 were 1.60 g/t and <0.02 g/t, respectively. The calculated gold head grades from the metallurgical testwork ranged from 1.22 g/t to 19.4 g/t. In the case of Comp 1, the gold head grade of 19.4 g/t is significantly higher than the expected average plant feed grade of 3.5 g/t for the Eleonore deposit according to the mine plan. The gold head grades of Composites 2 to 4 are more in line with the expected average plant feed grade for these deposits which is 1.15 g/t according to the mine plan.

The comminution program consisted of the SMC test, the Bond ball mill grindability test, and the UCS test performed on four composite samples as well as the Bond abrasion test on two of the four samples. The comminution program consisted of the SMC test, the Bond ball mill grindability test, and the UCS test performed on four composite samples as well as the Bond abrasion test on two of the four samples. Table 13-3 summarises all the individual grindability results.

Table 13-3: Grindability Summary

Sample Name	Relative Density	JK Parameters A x b	JK Parameters t_a	JK Parameters SCSE	BWI (kWh/t)	AI (g)	UCS[1] (MPa)
Comp 1	2.85	45.8	0.41	9.6	14.6	0.534	94.0
Comp 2	2.75	30.4	0.28	11.4	15.4	-	55.5
Comp 3	2.65	65.5	0.64	8.0	12.0	0.391	29.1
Comp 4	3.10	35.4	0.29	11.5	12.7	-	102.9

[1] UCS values represent average of 3 to 8 specimens for each sample

Comp 1, Comp 2, and Comp 4 were generally classified as medium to hard in terms of the SMC, BWI and UCS tests while Comp 3 was mostly classified as soft in terms of the aforementioned tests. The two samples submitted for the abrasion test (Comp 1, Comp 3) fell in the moderately to abrasive range of abrasiveness.

The objective of the preliminary metallurgical test program was to explore processing alternatives. The metallurgical test program consisted of:

- Heap leach amenability tests with coarse ore bottle roll tests on three of the four composites.
- Gravity separation and cyanidation testwork was completed on all samples.
- A single gravity tailings flotation test using Comp 1.

The Comp 1 gravity gold recoveries were 58% and 65% from two tests. The gravity gold recoveries for Comps 2-4 ranged from 31% to 71%. The gravity concentration results indicate that a gravity concentration should be considered for the overall process flowsheet.

The Comp 1 gravity tailings sample responded well to rougher flotation. Gold and sulphur recoveries were 92.3% and 97.0%, respectively, at a mass pull of 5.7%. The overall gravity and flotation circuit recovery was 97.3%. An insufficient quantity of flotation tests were performed to consider this unit operation in the plant flowsheet. No flotation tests were performed on Composites 2 to 4.

Heap Leach amenability tests were completed using Comps 1, 2 and 3 at three crush sizes of -19 mm, -12.5 mm and -6.35 mm. The final 28-day gold extractions ranged from 12.3% to 38.9% for Comp 1. Gold extractions were much higher for Comp 2 and Comp 3 and ranged from 58.9% to 82.5% and 61.9% to 70.1%, respectively. The results showed that, while gold extraction kinetics had slowed somewhat over the final week of the tests, leaching was likely nowhere near complete. Given the high gold head grades, this was not surprising. Finer crushing of the mineralised material would likely yield higher gold extractions and faster leach kinetics but, it is difficult to determine what the optimal crush size is with the data available. No definitive conclusion can be drawn from these results, but it is unlikely that heap leaching is the metallurgically optimal treatment process for these ores. Therefore, heap leaching was not included in the plant flowsheet for this study.

In total, 12 gravity tailing cyanidation tests were completed to investigate the response of the samples to cyanidation using standard leach conditions. The relationship between grind size and leach extraction was explored for Comp 1 and Comp 4. The impact of pre-aeration and lead nitrate on cyanide consumption and leaching kinetics was also determined for Comp 1. Table 13-4 summarises the results obtained.

Table 13-4: 2016 Testwork Gravity Tailings Cyanidation Results

Comp	Feed	CN Test No.	Feed Size P_{80}, µm	Reagent Cons. kg/t of CN Feed		% Au Extraction / Recovery							Au Residue, g/t			Au Head, g/t		
				NaCN	CaO	CN (Unit) 8 h	24 h	30 h	48 h	Grav	CN	**Grav +CN**	Cut A	Cut B	Avg.	CN	Grav +CN	Direct
1	G-1	10	144	0.33	1.17	70	89	88	89.7	57.9	37.8	**95.7**	0.85	0.83	0.84	8.16	20.0	22.8
		11	100	0.75	1.11	55	83	86	91.0	57.9	38.3	**96.2**	0.68	0.84	0.76	8.43		
		12	79	1.09	1.17	51	82	87	93.5	57.9	39.4	**97.3**	0.52	0.59	0.56	8.58		
		13	49	1.86	1.20	33	74	81	89.8	57.9	37.8	**95.7**	0.78	0.94	0.86	8.44		
	G-5	18	78	1.23	1.04	59	85	89	93.0	65.1	32.5	**97.6**	0.44	0.45	0.45	6.38	19.0	22.8
		19	72	0.55	1.14	73	94	94	93.7	65.1	32.7	**97.8**	0.42	0.45	0.44	6.95		
		20	75	1.33	1.06	90	94	95	93.5	65.1	32.6	**97.7**	0.41	0.37	0.39	5.97		
2	G-2	14	77	0.78	2.49	66	83	86	90.0	71.4	25.7	**97.1**	0.04	0.03	0.04	0.35	1.22	0.67
3	G-3	15	73	0.08	3.25	82	96	96	95.5	33.3	63.7	**97.0**	0.06	0.06	0.06	1.32	1.98	1.53
4	G-4	16	74	0.84	2.88	61	84	87	87.8	43.6	49.5	**93.1**	0.36	0.35	0.36	2.90	5.15	5.36
	G-6	21	160	1.13	1.11	50	74	77	81.7	30.8	56.5	**87.3**	0.71	0.66	0.69	3.75	5.36	
		22	90	1.50	1.13	51	77	82	88.2	30.8	61.0	**91.8**	0.42	0.45	0.44	3.67		

The gravity tailings samples responded well to cyanidation and overall (gravity + cyanidation) gold recoveries ranged from 87.3% to 97.8% and averaged approximately 95%. The overall gold recoveries for Comps 1, 2, and 3 were all 96% or higher. The Comp 4 average overall gold recovery was 90.8%. The results for Comp 4 showed a strong correlation between grind and recovery.

The Comp 1 (high grade sample) leach extraction was somewhat insensitive to grind size. Gold extractions for the five grind size evaluation tests ranged from 90% to 94%. Overall gold recoveries were 96% to 97%. The test results indicate that pre-aeration and lead nitrate improved the leach kinetics and gold extraction was essentially flat after 24 hours. Cyanide (NaCN) consumption decreased to 0.55 g/t (CN-19) from 1.23 g/t (CN-18) with the addition of a 2-hour pre-aeration stage. Additional tests are warranted to better define the relationship.

The whole ore cyanidation test results confirmed that the gravity recoverable gold in the Comp 1 sample was cyanide leachable. Gold extractions were 96% to 97% (two tests), similar to the overall (gravity + cyanidation) recoveries. With the exception of the Eleonore heap leach tests, the alternatives deliver good results. However, considering the good gold recovery by gravity-leach for all zones and, low reagent consumptions, this alternative was retained for the study.

13.2 Mineral Processing Testwork 2017

Following preliminary metallurgical tests conducted at SGS-Lakefield in Canada during the 4th quarter of 2016 on composite samples of HQ drill core. Some additional tests to confirm the initial results were carried out in 2017, also at SGS Lakefield. Preliminary variability tests were carried out on samples from the different zones to confirm the gold recovery, reagent consumptions and leach retention time required. Based on the results, CIL confirmation tests have been done on a composite from each zone based on the optimized conditions. From HQ samples taken, 16 composites were assembled for variability testing. Thereafter, all composites in the same area were assembled to serve as feed to CIL tests.

The samples were from drill cores collected from Eleonore North, Central and South, Sophie II, and Lily areas. Table 13-5 and Table 13-6 show the rock types and head grades from fire assay.

Table 13-5: 2017 Testwork Head Grades

Zone	Composite	Rock Type	Head Grade (g/t Au)
Sophie II	S-1	Shear zone	4.15
Sophie II	S-2	Amphibolite	1.66
Sophie II	S-3	Amphibolite and Quartz veins	0.49
Sophie II	S-4	Banded Iron Formation	0.43
Lily	S-5	Metasediments	1.22
Lily	S-6	Metasediments	1.16
Lily	S-7	Metasediments	0.50
Lily	S-8	Metasediments	0.88
Lily	S-9	Metasediments	0.95
Eleonore-N	S-10	Shear zone and quartz veins	0.82
Eleonore-N	S-11	Shear zone and quartz veins	4.36
Eleonore-S	S-12	Metagabbro	0.12
Eleonore-S	S-13	Metagabbro	0.70
Eleonore-C	S-14	Shear zone and quartz veins	8.84
Eleonore-C	S-15	Shear zone and quartz veins	11.5
Eleonore-C	S-16	Porphyre and quartz veins	0.18

Table 13-6: 2017 Testwork Head Grades of Samples vs. Composites

	Sample ID	Au, g/t	Ag, g/t	Var Sample	Au gr/t	Au gr/t	Bulk comp Au gr/t
	959	6,73	< 0.5	S1	4,15		
	962	0,25	< 0.5	S2	1,66		
	963	0,45	< 0.5				
	964	0,74	< 0.5	S3	0,49		
MS-SOPHIE	965	1,10	< 0.5			1,47	
	966	0,95	< 0.5				
	967	0,39	< 0.5				
	968	0,02	< 0.5				
	960	0,13	< 0.5	S4	0,43		
	961	1,12	< 0.5				
	951	0,48	< 0.5	S5	1,22		
	952	1,72	0,6				
	955	0,66	< 0.5	S6	1,16		
	953	1,96	< 0.5				
	954	0,63	7,4				
MS-LILY	957	0,49	< 0.5	S7	0,50		
	956	0,90	< 0.5			0,88	
	958	2,80	0,6	S8	0,88		
	969	0,69	< 0.5	S9	0,95		
	970	1,19	< 0.5				
	971	1,20	< 0.5				0,96
	972	0,78	< 0.5				
	973	1,77	< 0.5				
	974	0,74	< 0.5	S10	0,82		
MS-ELEO-N	975	1,02	3,3	S11	4,36	1,98	
	976	3,36	1,7				
	979	samples missing					
	980	samples missing					
	981	samples missing		S12	0,12		
MS-ELEO-S	982	samples missing					
	983	samples missing				0,54	
	984	2,30	< 0.5				
	977	0,24	< 0.5	S13	0,70		
	978	0,24	< 0.5				
	986	5,08	2,4	S14	8,84		
	987	0,21	1,0	S15	11,5		
	988	0,10	< 0.5				
MS-ELEO-C	989	18,4	2,8			4,49	
	990	samples missing					
	991	samples missing		S16	0,18		
	992	samples missing					

The gold head grades for the Eleonore samples varied between 0.54 and 4.49 g/t, which is significantly closer to the expected plant feed grade for the Eleonore deposit of 3.52 g/t than the samples from the 2016 program. The average head grade for the Sophie and Lily samples (pondered by weight) is 1.22 g/t which is reasonably close to the expected plant feed grade for the Lily and Sophie deposits of 1.15 g/t.

The objective of the program was to confirm the results obtained from the 2016 program using the optimized conditions identified in the first program. Additional flotation tests were carried out to validate this option for a possible comparison study. It also included some grinding tests.

13.2.1 Grindability Test

The comminution program was performed on four composite samples. The program consisted of a Bond ball mill work index tests (BWi) and Bond abrasion (Ai) tests. Table 13-7 shows the results of each of the comminution tests. The Sophie sample would be considered moderately hard with the remaining three samples would be considered hard.

Table 13-7: Testwork Grindability Summary

Samples	BWI (kWh/t)	Ai
Sophie II	13.3	0.358
Lily	17.1	0.768
Eleonore-C	17.5	0.558
Eleonore-S	18.7	0.514

13.2.2 Variability Tests

In 2017, gravity and cyanidation tests were performed using parameters obtained in the optimized tests of 2016. Table 13-8 summarises the parameters used. The series of variability tests (16 tests) were performed under the same conditions to verify the repeatability of the results, and to determine the required retention time for the CIL process.

Table 13-8: Gravimetric and Cyanidation Test Parameters for Variability Testing

Parameters	Value
Grind Size	75 microns
Pulp Density	40 % solids w/w
Pre-aeration period	2 hours
Oxygen Concentration	5 to 8 mg/L
Cyanide Concentration	0.5 g/L
pH	10.5 to 11
Retention Time	48 h for Eleonore-S/C/N 30 h for Sophie and Lily
Carbon Concentration	15 g/L

Table 13-9: Variability Test Results

Sample		Feed	CN Test No.	Feed Size P_{80}, µm	Reagent Addition kg/t of CN Feed		Reagent Cons. kg/t of CN Feed				% Au Extraction / Recovery CN (Unit)							Au Residue, g/t			Au Head, g/t		
					NaCN	CaO	NaCN (24)	NaCN (30)	NaCN (48)	CaO	6 h	24 h	30 h	48 h	Grav	CN	**Grav + CN**	Cut A	Cut B	Avg.	CN	Grav + CN	Direct
S1		G-4	1	75	1.05	0.77	0.30		0.46	0.77	64.0	87.9	87.2	88.2	27.4	64.0	**91.4**	0.29	0.31	0.30	2.55	3.50	4.15
S2	SOPHIE	G-5	2	64	0.99	1.08	0.21	0.23	0.40	1.08	65.4	95.3	95.2	94.4	33.0	63.2	**96.2**	0.03	0.02	0.03	0.44	0.65	1.66
S3		G-6	3	64	1.06	0.77	0.23	0.29	0.35	0.75	73.9	91.3	91.2	90.2	56.8	39.0	**95.8**	0.03	<0.02	0.03	0.25	0.57	0.49
S4		G-7	4	88	1.21	0.87	0.42	0.42	0.68	0.85	62.4	84.6	84.5	83.5	46.7	44.5	**91.2**	0.06	0.07	0.07	0.39	0.72	0.43
Zone Average				**73**	**1.08**	**0.87**	**0.29**	**0.31**	**0.47**	**0.86**	**66.4**	**89.8**	**89.53**	**89.075**	**41**	**52.7**	**93.7**	**0.10**	**0.11**	**0.10**	**0.91**	**1.36**	**1.68**
S5		G-8	5	75	1.01	0.75	0.24	0.26	0.42	0.74	71.3	83.9	83.7	82.9	49.5	41.9	**91.4**	0.07	0.08	0.08	0.44	0.86	1.22
S6		G-9	6	75	1.03	0.85	0.19	0.27	0.43	0.85	76.6	90.9	90.7	89.9	31.3	61.8	**93.1**	0.07	0.07	0.07	0.69	1.00	1.16
S7	LiLy	G-10	7	61	1.02	0.81	0.22	0.27	0.42	0.80	67.2	86.5	89.6	88.8	32.6	59.9	**92.5**	0.06	0.04	0.05	0.45	0.66	0.50
S8		G-11	8	82	1.02	1.02	0.06	0.06	0.08	0.99	53.8	89.0	90.7	93.7	22.1	73.0	**95.1**	0.06	0.08	0.07	1.12	1.43	0.88
S9		G-12	9	78	0.96	0.80	0.17	0.20	0.31	0.80	61.2	90.4	90.3	89.8	40.6	53.3	**93.9**	0.07	0.06	0.07	0.64	1.07	0.95
Zone Average				**74**	**1.01**	**0.85**	**0.18**	**0.25**	**0.33**	**0.84**	**66.0**	**88.1**	**89.0**	**89.0**	**35.2**	**58.0**	**93.2**	**0.07**	**0.07**	**0.07**	**0.67**	**1.00**	**0.94**
S10	ELEO_N	G-13	10	68	1.00	0.78	0.20	0.24	0.35	0.78	57.0	93.1	87.2	91.4	61.9	34.8	**96.7**	0.02	0.03	0.03	0.29	0.77	0.82
S11		G-14	11	69	1.05	0.84	0.26	0.29	0.44	0.84	51.5	89.6	90.0	93.2	30.1	65.1	**95.2**	0.17	0.18	0.18	2.57	3.66	4.36
Zone Average				**69**	**1.03**	**0.81**	**0.23**	**0.27**	**0.40**	**0.81**	**54.3**	**91.4**	**88.6**	**92.3**	**46.0**	**50.0**	**96.0**	**0.10**	**0.11**	**0.10**	**1.43**	**2.22**	**2.59**
S12	ELE-S	G-15	12	70	0.93	0.82	0.18	0.18	0.88	0.81	29.0	77.8	80.8	96.8	20.2	77.2	**97.4**	0.03	0.03	0.03	0.93	1.16	0.12
S13		G-16	13	69	1.01	0.84	0.21	0.25	0.38	0.84	50.4	86.1	90.4	94.0	53	44.2	**97.2**	0.02	0.02	0.02	0.33	0.70	0.70
Zone Average				**70**	**0.97**	**0.83**	**0.20**	**0.22**	**0.63**	**0.83**	**39.7**	**82.0**	**85.6**	**95.4**	**36.6**	**60.7**	**97.3**	**0.03**	**0.03**	**0.03**	**0.63**	**0.93**	**0.41**
S14	ELEO-C	G-17	14	173	1.05	0.69	0.24	0.27	0.43	0.67	51.1	57.7	81.7	84.1	13.3	72.9	**86.2**	0.73	1.00	0.87	5.42	6.25	8.84
S14		G-17R	14R	59	1.11	1.40	0.25		0.44	0.98	85.0	89.0	84.1	93.9	33.4	62.5	**95.9**	0.35	0.37	0.36	5.86	8.78	8.84
S15		G-18	15	68	1.07	0.74	0.27	0.29	0.44	0.73	68.5	93.6	91.7	99.0	44.4	55.0	**99.4**	0.06	0.07	0.07	6.23	11.2	11.5
S16		G-19	16	82	0.93	0.63	0.19	0.19	0.34	0.58	84.6	93.1	92.9	92.0	85.6	13.2	**98.8**	0.03	0.02	0.03	0.31	2.16	0.18
Zone Average				**69.7**	**1.04**	**0.92**	**0.24**	**0.24**	**0.41**	**0.76**	**79.4**	**91.9**	**89.6**	**95.0**	**54.5**	**43.6**	**98.1 ***	**0.29**	**0.37**	**0.33**	**4.46**	**7.10**	**7.34**
Total Average				**67 ***	**1.0**	**0.9**	**0.23**	**0.23**	**0.43**	**0.82**	**63.9**	**88.9**	**88.8**	**91.4**	**41.8**	**53.3**	**95.1**	**0.09**	**0.09**	**0.09**	**1.47**	**2.43**	**2.37**

*: excluding results from CN- 14 which was redone (14R)

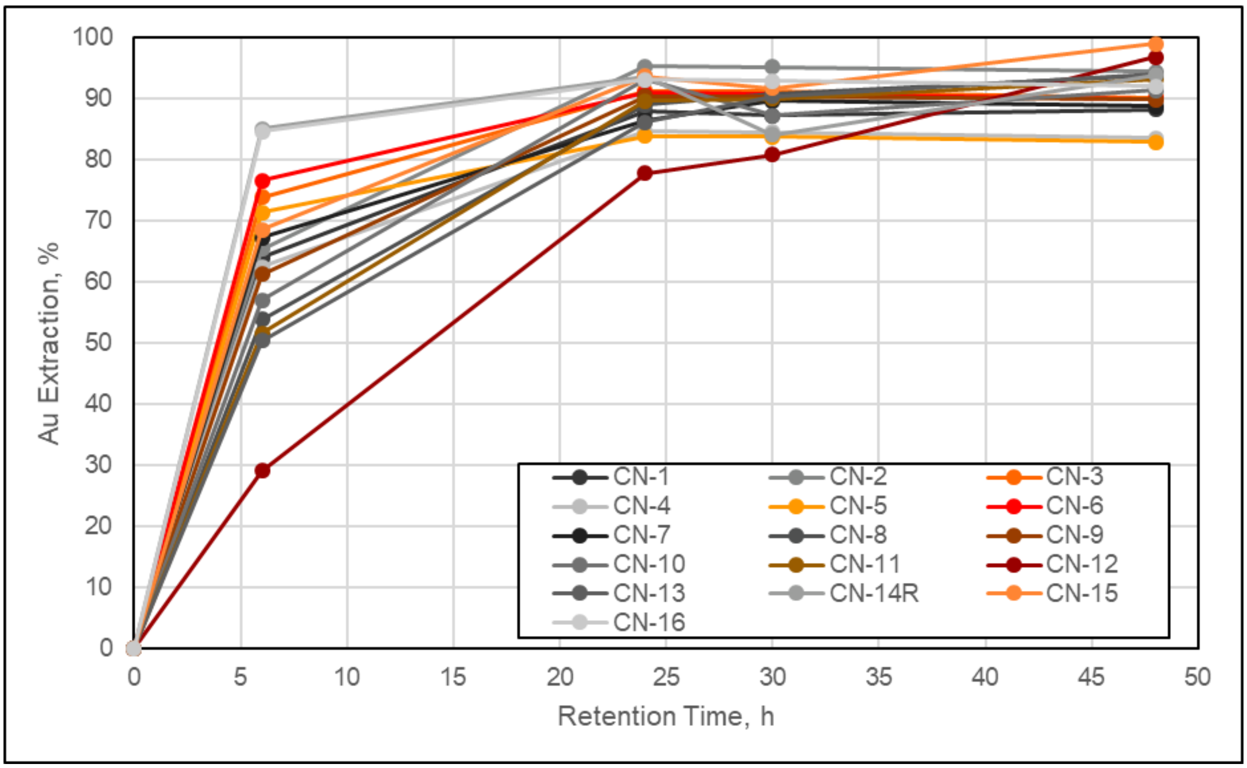


Figure 13-1: Leach Kinetics

The results of the variability tests confirmed good responses to the selected process (Table 13-9). The results confirmed that 48 hours of leach retention time for the Eleonore zone and 30 hours for the Sophie and Lily zones (Figure 13-1). These retention times were retained to conduct CIL tests on the composites from each group of deposits. Table 13-10 summarises the results achieved.

Table 13-10: CIL Results

Composites	P_{80}, µm	Cons. Reagents		% Au Extraction/Recovery				
		kg/t of CN Feed		CN (Unit)		Grav	CN	Grav + CN
		NaCN	CaO	30 h	48 h			
MS-SOPHIE	68	0.36	*0.81*	88.0	…	64.0	31.7	**95.7**
MS-LILY	78	0.34	1.29	89.8	…	51.5	43.6	**95.1**
MS-ELEO-S	71	0.59	1.24	…	92.1	52.7	43.6	**96.3**
MS-ELEO-C	78	0.50	0.80	…	97.5	23.7	74.4	**98.1**
MS-ELEO-N	70	0.65	0.90	…	93.2	31.9	63.5	**95.4**

The results confirm the previous high gravity gold recovery results. Overall recoveries obtained in the first variability tests were similar to the 2016 results and the positive response to the CIL process. The results also validate the leach retention time requirements for the different zones. The Sophie and Lily composites obtained a gravity circuit recovery of 64% and 51.5%, respectively. The Eleonore South, Central and North Composites obtained a gravity circuit recovery ranging from 23.7% to 52.7%. These results once again demonstrate that gravity concentration should be included in the process flowsheet.

The tailings samples obtained from the gravity testwork responded well to cyanidation process and to the overall recovery process (gravity separation + cyanidation). Gold recovery rates ranged from 95.1% to 98.1% for an average of 96%. This is an improvement of one percent compared to the 2016 testwork. These results validated the selection of the process flow diagrams for the project at this point.

13.2.3 Gold Deportment Study

A gold deportment study was performed on a global composite sample. The majority (96%) of the gold particles found occur as native gold (less than 20 % silver) with a silver content averaging 10.0% Ag. A few electrum particles were also found. Most of the gold is liberated and should be available for gold dissolution which is in line with the leach testwork results. The results are shown in Table 13-11, Table 13-12, and in Figure 13-2.

Table 13-11 – Chemical Composition of Gold Particles

Sample ID	Mineral ID	n	Chemical Composition (SEM-EDS, wt%)				
			Fe	Ag	Au	Pb	Te
Algold Composite	Gold	155	0,8	10,6	88,6	0,0	0,0
	Electrum	4	3,1	27,1	69,8	0,0	0,0
	Kustelite	1	0,8	64,0	35,2	0,0	0,0
	Petzite	1	0,0	41,5	27,0	0,0	31,5
	Au-Ag-Pb	1	2,6	15,0	54,6	27,8	0,0

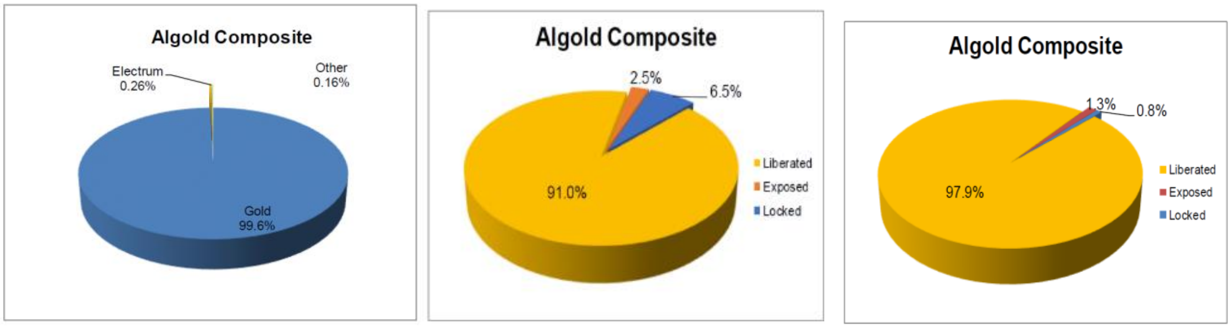


Figure 13-2 – Gold Mineral Type, Abundance and Microscopic Distribution

Table 13-12 – Overall Microscopies of Gold Distribution

Algold Composite	Association	No. of Observed Grains	Size Range (µm)	Average Size (µm)
SP Tip 1	Liberated	71	0.6 - 174.5	24.6
	Exposed	8	2.0 - 18.1	6.09
	Locked	38	0.6 - 14.2	3.47
	Subtotal	117		16.5
SP Tip 2	Liberated	45	0.6 - 414.3	44.4
	Exposed	19	0.6 - 64.8	6.57
	Locked	37	0.6 - 13.2	3.77
	Subtotal	101		22.4
SP Mag	Liberated	0	0.0 - 0.0	-
	Exposed	2	2.6 - 3.6	3.11
	Locked	2	1.5 - 7.5	4.48
	Subtotal	4		3.80
SP Sul 1	Liberated	5	0.8 - 35.5	16.8
	Exposed	3	1.0 - 12.9	5.01
	Locked	20	0.6 - 12.5	3.98
	Subtotal	28		6.37
SP Sul 2	Liberated	2	61.5 - 63.8	62.6
	Exposed	11	1.3 - 15.2	6.30
	Locked	16	0.6 - 8.5	2.72
	Subtotal	29		8.21
SP Mid	Liberated	1	1.0 - 1.0	0.98
	Exposed	0	0.0 - 0.0	-
	Locked	1	1.1 - 1.1	1.13
	Subtotal	2		1.05
SP Tail	Liberated	5	2.5 - 6.5	4.83
	Exposed	5	0.6 - 3.8	1.95
	Locked	3	2.3 - 5.6	3.66
	Subtotal	13		3.45
HLS Float	Liberated	2	3.0 - 4.6	3.80
	Exposed	0	0.0 - 0.0	-
	Locked	1	32.2 - 32.2	32.2
	Subtotal	3		13.3
TOTAL	Liberated	131	0.6 - 414.3	30.4
	Exposed	48	0.6 - 64.8	5.71
	Locked	118	0.6 - 32.2	3.79
	Total	**297**		**15.8**

Algold Composite	Association	No. of Observed Grains	Total Observed Surface Area (um2)	% Observed Surface Area Fraction	Overall Gold Distribution
SP Tip 1 /Tip 2	Liberated	116	362,221	98.3	62.8
	Exposed	27	4,113	1.12	0.71
	Locked	76	2,244	0.61	0.39
	Subtotal	*219*	*368,578*	*100*	***63.9***
SP Mag	Liberated	0	0.00	0.00	0.00
	Exposed	2	15.5	25.4	0.20
	Locked	2	45.6	74.6	0.59
	Subtotal	*4*	*61.1*	*100*	***0.78***
SP Sul 1	Liberated	5	1,720	74.0	2.79
	Exposed	3	133	5.72	0.22
	Locked	20	470	20.2	0.76
	Subtotal	*28*	*2,322*	*100*	***3.77***
SP Sul 2	Liberated	2	6,166	89.9	5.77
	Exposed	11	514	7.49	0.48
	Locked	16	176	2.57	0.16
	Subtotal	*29*	*6,855*	*100*	***6.41***
SP Mid	Liberated	1	0.75	42.9	1.55
	Exposed	0	0.00	0.00	0.00
	Locked	1	1.00	57.1	2.07
	Subtotal	*2*	*1.75*	*100*	***3.61***
SP Tail	Liberated	5	101	64.4	7.04
	Exposed	5	19.3	12.3	1.35
	Locked	3	36.3	23.2	2.54
	Subtotal	*13*	*156*	*100*	***10.9***
HLS Float	Liberated	2	23.8	100.00	10.5
	Exposed	0	0.00	0.00	0.00
	Locked	0	0	0.0	0.0
	Subtotal	*2*	*24*	*100*	***10.5***
TOTAL	**Liberated**	**131**	**370,232**	**97.9**	**91.0**
	Exposed	**48**	**4,794**	**1.3**	**2.50**
	Locked	**118**	**2,972**	**0.8**	**6.55**
	Total	***297***	***377,998***	***100***	***100***

13.3 Design Criteria Selection

The process design criteria were selected on the basis of all the testwork performed. Plant operation is divided into two distinct phases, Phase 1 which will exploit the Eleonore deposits, and Phase 2, which will exploit the Lily and Sophie deposits.

For the purpose of grinding mill selection, a Bond ball mill work index of 18.1 kWh/t was selected. This value represents the average of the tests performed on the Eleonore material during the 2017 testwork campaign. Since the mill will operate exclusively on Eleonore material for the first few years of operation, the mills must be sized with sufficient capacity to handle this material. The Sophie and Lily deposit properties, which will be mined after the Eleonore deposit, were not used for mill sizing as both the 2016 and 2017 test work campaigns have shown that Eleonore is the harder material. Similarly, an abrasion index of 0.534 was selected for the process design criteria.

Gravity recovery for both phases was selected using the average results of the variability test work. A gravity circuit recovery of 44.8% was selected for Phase 1, and gravity circuit recovery of 38.6% was selected for Phase 2.

Initially, a target cyanidation feed grind size of 75 microns was selected for both phases. At this grind size, a CIL retention time of 48 hours for Phase 1 and 30 hours for Phase 2 was selected (throughput increases during Phase 2). These retention times include 2 hours of pre-aeration. Overall plant recoveries (including gravity circuit) were selected using the calculated average of the variability testing performed. Phase 1 recovery was established at 97.1% and Phase 2 recovery was established at 93.6%. Subsequent plant development identified the advantages of selecting a coarser grind size for Phase 1 operations on both OPEX and CAPEX costs. The 2016 testwork demonstrated that the Eleonore deposit CIL circuit parameters were not very sensitive to grind size as shown in Table 13-4. To improve project economics, the Phase 1 grind size selection was changed to 105 microns. This change does not impact the recovery of the gravity circuit, but the overall circuit recovery was decreased to 96% as a result of the lower recovery in the cyanidation circuit, using the 2016 results. At this study level and considering the preliminary results it appears that this change is relevant considering the positive impact on the project economics.

Additional variability testwork is recommended for Eleonore as well on Sophie and Lily at this grind size for future phases of this project.

13.4 Conclusion

The metallurgical testing completed to date demonstrates that the samples from the major deposits within this Tijirit project are free milling, with a significant gravity recoverable gold component. There are indications of some cyanide consuming mineralization as shown by the beneficial effects of pre-aeration on leach kinetics and cyanide consumption. The metallurgical testwork completed to date is appropriate for a PEA level study, utilizing composite samples of the major deposits.

14. MINERAL RESOURCE ESTIMATES

Resource estimates on the Tijirit deposit were estimated with an effective date of January 19, 2018 for the Eleonore, Sophie I-II-III and Lily zones. These numbers have been publicly disclosed February 26, 2018 in the press release: “Algold Increases Gold Resources at Tijirit”. This report explains additional details about this updated resource estimation.

14.1 Drill Hole Database

Algold provided SGS Geostat with the electronic version of the drilling campaign data. The data was imported into a Geobase format emphasizing on the collar identifications, deviations, lithologies and assay results (see Table 14-1).

Table 14-1: Summary of Database Entries Used for the Estimates

Field	Number of entries	Length (m)
Collars [Drill Holes (RC and DDH) and sampled trenches]	983	112,343
Deviations	6,975	
Lithologies	113,544	
Assays	76,297	102,567

A total of 627 RC drill holes and 91 Diamond-drill holes are included in the database, along with 265 Trenches. (see Table 14-2). Drilling used for the resource estimate totals 112,343 m. All holes were surveyed using a Reflex downhole orientation instrument and appear to be sampled consistently every 50 m or less down the hole. Drill holes and trenches are surveyed using the UTM projection, WGS84 datum 28 northern hemisphere.

Note that 4 drill holes (1 trench, 2 RC and 1 DDH) are not accounted for because they do not contain assays that allow for their use in the resource estimation.

Table 14-2: Summary of Database Entries by Hole Type

Hole Type	Number of Drill Holes	Sum of Length (m)	Number of Assays	Sum of Assayed Length (m)
DDH	91	14,704	10,276	10,706
RC	627	80,549	58,675	80,337
Trenches	265	17,090	7,346	11,524
Total	**983**	**112,343**	**76,297**	**102,567**

The database contains 76,297 assay results for gold. A total of 10,276 assays are from diamond drill rigs and represent 10,706 metres, and 58,675 assays are from RC drill holes representing 80,337 drilled metres, and 7,346 assays come from Trenches and represent 11,524 trenched metres (see Table 14-2).

Assays were made into mineralized intervals (MI). A modelling cut-off grade of 0.3 g/t Au, a minimum thickness of 2 metres along hole and a minimum accumulation of 1.2 m.g/t were used to delineate mineralized volumes. There are 689 MIs. The total length for the MIs is of 4,491.7 m. The shortest MIs created are of 2 m and there are 71 of them. The longest MI created is of 94 m and is located in the Lily mineralized zone.

14.2 Mineralized Volumes (Envelopes)

Some mineralized volumes were modeled over the MIs. The process involved the creation of closed polygons on section views. The sections are not always on a regular grid as the drilling is not always on a standard azimuth and the drill hole spacing is not perfectly even. There are more than 900 polygons that were interpreted on about 300 sections. There are currently 220 mineralized volumes (envelopes) in the Tijirit project. There are 134 of them in Eleonore, 28 in Lily and 58 in Sophie I, II and III. The volumes under topography are listed in Table 14-3 along with the number of holes that pierce these volumes.

Table 14-3: List of the Mineralized Volumes and the Count of Mineralized Intervals

Name	# of Mis (Holes Count)	Vol (m^3)	Name	# of Mis (Holes Count)	Vol (m^3)	Name	# of Mis (Holes Count)	Vol (m^3)	Name	# of Mis (Holes Count)	Vol (m^3)
Eleonore_A	27	207,675	Eleonore_Ouest_6	1	16,205	Eleonore_ss_32	1	6,624	Sophie1_A_5	1	8,268
Eleonore_B	3	4,525	Eleonore_Ouest_7	1	16,880	Eleonore_ss_33	1	6,003	Sophie1_A_6	1	7,387
Eleonore_C	3	36,967	Eleonore_Ouest_8	1	20,402	Eleonore_ss_34	2	22,750	Sophie1_B	2	52,215
Eleonore_D	6	30,534	Eleonore_Ouest_9	1	19,713	Eleonore_ss_35	1	9,446	Sophie1_C	7	65,958
Eleonore_E	6	35,607	Eleonore_P	1	9,569	Eleonore_ss_36	3	31,509	Sophie1_D	3	20,425
Eleonore_F	6	30,278	Eleonore_Q	2	20,715	Eleonore_ss_37	2	48,560	Sophie1_E	5	34,435
Eleonore_G	2	23,229	Eleonore_R	1	15,767	Eleonore_ss_38	2	29,218	Sophie1_F	3	47,964
Eleonore_H	6	54,671	Eleonore_S	4	35,722	Eleonore_ss_39	1	11,834	Sophie1_G	1	4,092
Eleonore_I	1	8,358	Eleonore_s_1	5	57,543	Eleonore_ss_40	1	8,554	Sophie1_H	1	6,225
Eleonore_J	1	8,757	Eleonore_s_10	1	7,728	Eleonore_ss_41	2	23,192	Sophie2_A	4	53,495
Eleonore_K	5	32,675	Eleonore_s_11	1	7,903	Eleonore_ss_42	1	15,902	Sophie2_A_1	1	2,258
Eleonore_L	1	15,639	Eleonore_s_12	1	22,087	Eleonore_ss_43	1	11,698	Sophie2_A_2	1	3,602
Eleonore_M	3	25,144	Eleonore_s_13	1	6,744	Eleonore_ss_44	1	14,269	Sophie2_A_3	1	4,058
Eleonore_N	2	7,675	Eleonore_s_14	2	53,747	Eleonore_ss_45	1	7,986	Sophie2_B	25	375,236
Eleonore_Nord_01	14	160,609	Eleonore_s_15	2	31,529	Eleonore_ss_46	1	18,176	Sophie2_B_1	1	6,803
Eleonore_Nord_02	9	124,131	Eleonore_s_16	1	11,634	Eleonore_ss_47	2	70,034	Sophie2_C	4	69,972
Eleonore_Nord_03	6	95,005	Eleonore_s_2	7	68,512	Eleonore_ss_48	1	15,322	Sophie2_D	2	27,932
Eleonore_Nord_04	6	54,114	Eleonore_s_3	4	45,476	Eleonore_ss_49	1	12,238	Sophie2_E_1	4	10,069
Eleonore_Nord_05	5	33,200	Eleonore_s_4	2	34,153	Eleonore_ss_50	1	14,342	Sophie2_E_2	3	12,674
Eleonore_Nord_06	3	25,786	Eleonore_s_5	10	75,119	Eleonore_T	1	15,275	Sophie2_F	4	28,013
Eleonore_Nord_07	4	32,921	Eleonore_s_6	1	15,663	Eleonore_U	1	59,970	Sophie2_G	9	115,506
Eleonore_Nord_08	2	24,119	Eleonore_s_7	1	7,298	Eleonore_V	1	5,444	Sophie2_H	13	195,165
Eleonore_Nord_09	2	35,199	Eleonore_s_8	1	8,659	Eleonore_W	1	10,274	Sophie2_H_2	1	5,937
Eleonore_Nord_10	3	18,561	Eleonore_s_9	5	10,750	Eleonore_X	1	6,246	Sophie2_I	5	137,762
Eleonore_Nord_11	1	20,954	Eleonore_ss_01	12	89,202	Lily_1	1	22,171	Sophie2_J	2	975
Eleonore_Nord_12	1	13,007	Eleonore_ss_02	6	42,586	Lily_2	1	10,965	Sophie2_K	5	42,962
Eleonore_Nord_13	2	59,537	Eleonore_ss_03	1	24,578	Lily_3	1	16,546	Sophie2_L	1	6,560
Eleonore_Nord_14	1	19,512	Eleonore_ss_04	19	194,070	Lily_A	3	144,254	Sophie2_M	1	26,312
Eleonore_Nord_15	4	25,090	Eleonore_ss_05	4	43,510	Lily_B	4	362,352	Sophie2_N	3	62,948
Eleonore_Nord_16	3	26,661	Eleonore_ss_06	3	19,737	Lily_C	10	1,739,469	Sophie2_O	1	12,287
Eleonore_Nord_17	1	10,593	Eleonore_ss_07	3	24,458	Lily_D	29	1,429,983	Sophie2_P	1	7,706
Eleonore_Nord_18	1	5,326	Eleonore_ss_08	1	14,517	Lily_E	3	11,309	Sophie2_P_2	1	7,706
Eleonore_Nord_19	2	11,486	Eleonore_ss_09	1	7,426	Lily_F	1	5,458	Sophie2_Q	1	9,042
Eleonore_Nord_20	1	8,398	Eleonore_ss_10	2	18,164	Lily_G	18	1,462,870	Sophie2_R	1	9,738
Eleonore_Nord_21	2	11,411	Eleonore_ss_11	5	88,770	Lily_H	3	113,249	Sophie2_S	1	16,024
Eleonore_Nord_22	1	10,738	Eleonore_ss_12	4	51,765	Lily_I	1	5,138	Sophie2_T	2	44,861
Eleonore_Nord_23	1	6,832	Eleonore_ss_13	2	14,202	Lily_J	5	68,483	Sophie3_A	23	2,632,966
Eleonore_Nord_24	1	6,036	Eleonore_ss_14	1	4,047	Lily_K	4	23,106	Sophie3_B	3	12,690
Eleonore_Nord_25	4	28,582	Eleonore_ss_15	1	40,147	Lily_L	10	440,170	Sophie3_C	3	23,663
Eleonore_Nord_26	1	7,603	Eleonore_ss_16	4	103,583	Lily_M	2	29,515	Sophie3_C1	1	19,196
Eleonore_Nord_27	2	8,728	Eleonore_ss_17	6	98,657	Lily_N	5	201,003	Sophie3_C2	1	3,549
Eleonore_Nord_28	1	16,543	Eleonore_ss_18	1	55,931	Lily_O	1	36,644	Sophie3_D	8	390,988
Eleonore_Nord_29	1	4,092	Eleonore_ss_19	2	7,715	Lily_P	1	7,669	Sophie3_D_2	1	21,242
Eleonore_Nord_30	1	22,186	Eleonore_ss_20	1	5,095	Lily_Q	1	18,513	Sophie3_E	1	8,667
Eleonore_Nord_31	1	9,454	Eleonore_ss_21	1	4,267	Lily_R	4	94,940	Sophie3_F	1	7,594
Eleonore_Nord_32	1	17,203	Eleonore_ss_22	1	3,303	Lily_S	1	25,158	Sophie3_G	1	5,023
Eleonore_Nord_33	1	5,216	Eleonore_ss_23	1	2,806	Lily_T	1	30,294	Sophie3_H	1	4,723
Eleonore_Nord_34	1	11,083	Eleonore_ss_24	1	16,728	Lily_U	1	19,464	Sophie3_I	1	3,962
Eleonore_O	2	35,180	Eleonore_ss_25	1	17,264	Lily_V	1	11,062	Sophie3_J	1	11,883
Eleonore_Ouest_1	8	123,164	Eleonore_ss_26	1	19,990	Lily_X	2	55,709	Sophie3_K	2	37,988
Eleonore_Ouest_11	2	36,632	Eleonore_ss_27	9	102,620	Lily_Y	2	15,384	Sophie3_L	2	33,020
Eleonore_Ouest_2	5	51,616	Eleonore_ss_28	5	41,013	Lily_Z	2	35,443	Sophie3_M	4	73,404
Eleonore_Ouest_3	2	42,384	Eleonore_ss_29	1	6,115	Sophie1_A	5	140,886	Sophie3_N	1	13,280
Eleonore_Ouest_4	2	39,316	Eleonore_ss_30	7	132,187	Sophie1_A_1	4	67,645	Sophie3_West_A	2	38,537
Eleonore_Ouest_5	1	17,250	Eleonore_ss_31	2	27,573	Sophie1_A_4	1	6,117	Sophie3_West_B	1	20,491
									TOTAL	689	15,810,010

14.3 Composite Data

The assays inside MIs have a total length of 4,491.5 m. There are 2,911 assays at 1 m or less in length (totaling 2,709.36 m in length), 68 assays of 1.5 m length (totaling 102.05 m in length), 827 assays of 2 m length (totaling 1,654 m in length) and a single assay of 3 m length. Composites have been created inside mineralized intervals (MI). The “round closest” setting have been used to eliminate the problem of remainders. A 2-m long MI produced a single composite while a 3-m or 4-m produced 2 composites and a 5 m or 6 m long MI produced 3 composites and so on. The resulting 2,340 composites have a length between 1.37 m and 2.65 m. There are 0.6 % of the composites (14) that are under 1.5 m and three composites over 2.5 m. The other composites have lengths between 1.5 m and 2.5 m, which is very acceptable.

There are 2,340 composites in total. The composites were loaded into Genesis, with Au and Au_Cap and extracted by Tags, one for each of the 220 zones. Table 14-4 and Table 14-5 show the composite statistics by zone for Au and Au cap respectively

Table 14-4: Statistics on the Composites (Au) for Each Major Zone

Statistics Composites Au (g/t)						
	Eleonore	**Lily**	**Sophie I**	**Sophie II**	**Sophie III**	**Total**
Count	799	722	75	397	347	2,340
Min	0.005	0.005	0.005	0.005	0.005	0.005
Max	150.69	20.90	17.70	24.90	5.02	150.69
Mean	2.38	0.64	1.60	1.12	0.50	1.32
Median	0.74	0.41	0.81	0.63	0.37	0.52
Stdev	7.74	1.10	2.51	1.96	0.58	4.73

Table 14-5: Statistics on the Composites (Au Cap) for Each Major Zone

Statistics Composites Au Cap (g/t)						
	Eleonore	**Lily**	**Sophie I**	**Sophie II**	**Sophie III**	**Total**
Count	799	722	75	397	347	2,340
Min	0.005	0.005	0.005	0.005	0.005	0.005
Max	45.00	6.00	10.00	10.00	3.50	45.00
Mean	2.15	0.60	1.50	1.06	0.49	1.22
Median	0.74	0.41	0.81	0.63	0.37	0.52
Stdev	4.91	0.66	1.93	1.40	0.51	3.06

14.4 Capping

The capping study has found very nuggety gold and therefore while few composites are capped, the impact on gold content can be significant in some of the zones. Eleonore: 4 composites capped (9 % of the gold lost), Lily_G: 1 composite capped (22 % of the gold lost), Lily other than Lily_G: 2 composites capped (3 % of the gold lost), Sophie I-II: 4 composites capped (6 % of the gold lost), Sophie III: 3 composites capped (2 % of the gold lost). These gold losses are indicative and only based on composite statistics. Table 14-6 shows the details of the capping.

Table 14-6: Capping and Gold Loss Based on Composites

Zone	Capping Zone	Count	Max Au (g/t)	Au Capping Grade (g/t)	Capped Count	Gold Loss	Mean Au (g/t)	Mean Au Cap (g/t)
Eleonore	**All**	799	150.69	45	4	-9 %	2.38	2.15
Lily	**All but G**	609	11.26	6	2	-3 %	0.63	0.61
	G	113	20.90	4	1	-22 %	0.69	0.54
Sophie I-II	**All**	472	24.90	10	4	-6 %	1.20	1.13
Sophie III	**All**	347	5.02	3.5	3	-2 %	0.50	0.49
Total		2,340	150.69	Variable	14	-8 %	1.32	1.22

14.5 Density

The density used for the estimation of the resource is 2.00 t/m³ for saprolite and 2.7 t/m³ for fresh rock in the Lily zone, 2.8 t/m³ in the Sophie III zone, 2.85 t/m³ in the Sophie II zone, 3.0 t/m³ in the Sophie I zone and 2.86 t/m³ in the Eleonore zone.

In 2016-17, there were 2,570 density measurements made by Algold. About 2,464 are from fresh rock, the others are from oxidized areas with a lower density than normal. SGS Geostat has decided to use a density of 2.00 t/m³ for all oxidized material as it is a reasonable value and there is no representative data for this material at this stage of the project. Indeed, it is very difficult to measure the density of a material that does not have good recovery during diamond drilling.

In 2016, SGS Geostat used density measurements available from the work of the previous owners of the project. It was calculated that the method used (water displacement) had a significant error to it (± 0.2 t/m³) and that new data was required. SGS received 560 density measurements done on the 2016 diamond drillholes. On December 11, 2016, SGS received 560 density measurements from the 2016 diamond drill. The method used was the weight in air (W_A) versus the weight in water (W_W). The density (D) is then calculated with the formula: $D = W_A / (W_A - W_W)$. The weight measurements have a precision of 1 g, the minimum sample weight was 388 g and the mean weight was 1,849 g. SGS estimates the precision of the method to be at least of ± 0.1 t/m³ but in general around ± 0.01 t/m³. The method employed in 2016 for the assessment of the density is adequate. The saprolite data was separated from the fresher rock material. This saprolite has more variable density data and it was decided to use 2.00 for the density of the saprolite material in the model. It is a reasonable value. The densities statistics for older data and 2016-2017 data for the fresher rock are shown in Table 14-7. A total of 3 outliers were removed leaving a population of 557 densities to work with.

Table 14-7: 2016-2017 Density Data Compared to Resource Densities

Zone		Resource Density	2016-2017 Resource Density			Comment on the density used for resources
			N of Data	N of Holes	Mean	
Eleonore	North	2.86	325	9	3.01	Conservative (5 %)
	Centre		398	10	3.00	Conservative (5 %)
	South		177	4	2.97	Maybe conservative
	South-South		378	8	3.01	Conservative (5 %)
	West		43	1	2.76	Maybe optimistic (3 %)
Sophie	I	3.00	36	1	3.12	Maybe conservative
	II	2.85	151	5	3.01	Conservative (5 %)
	III	2.80	68	1	3.02	Maybe conservative
Lily		2.70	No data for 2016-2017			

14.5.1 Modeling of the Saprolite Depth for Density Attribution

A saprolite surface was generated using X, Y, Z points from the lithological table. Then the blocks were tagged as saprolite and an appropriate density was applied.

For resource estimation, SGS Geostat received a geology database on October 22, 2017 with oxidation information for a total of 720 holes including 317 from 2010, 2011, and 2012 by Shield and Gryphon and a total of 403 by Algold (176 in 2016 and 227 in 2017). The file named "TIJ_ALG_ALL_Geology_Compiled_22102017.xlsx" contains the geological intervals with the weathering, primary lithology and secondary lithology information. These depths were used to update the saprolite / fresher rock interface surface model. This new surface was used for the attribution of densities in the model. These data were used to update the saprolite / fresher rock interface surface model. The new surface was used for the attribution of densities in the model. Blocks between the topography and the saprolite / fresher rock interface surface are attributed a saprolite density while material below the saprolite / fresher rock interface surface are attributed a fresh rock density. Refer to the Density paragraph for the detailed densities used in the resource model.

14.6 Interpretation of the Mineralization

Each of the 220 mineralized envelopes was considered as a hard boundary mineralization; composites for estimation were limited to inside each mineralized zone for block estimation, removing influence of values of adjacent mineralized structures.

14.7 Resource Block Modeling

The project consists of 6 block models oriented in 4 different grids having the parameters shown in Table 14-8. Block sizes vary between 2 m × 5 m × 2.5 m and 5 m × 5 m × 5 m. Except Lily, block models are turned to conform to the general orientations of the zones. The coordinates of the origin correspond to the center of the first block. Full blocks were used that means that if the center of the block falls inside the volume, it is counted at 100 %, if the center of the block falls outside the volume, it is counted at 0 %. Each of the 220 mineralized volumes was tagged in the block model and estimated separately. Discretization of 2 × 2 × 2 (8 sub-blocks) was used to better estimate the distance between composites and blocks.

Table 14-8: Block Models Parameters

Block Model Zone	Rotation	Blocks Size (m)			Block Grid Origin (Block Centre)			Block Grid Number		
		X	Y	Z	X	Y	Z	X	Y	Z
Eleonore	30	2	5	2.5	480,200	2,248,400	-200	926	801	161
Lily	0	5	5	5	475,600	2,244,400	-250	341	341	91
Sophie I-II	15	2	5	2.5	473,800	2,250,600	-200	1,201	561	161
Sophie III	30	2	5	5	473,500	2,247,600	-200	850	700	81

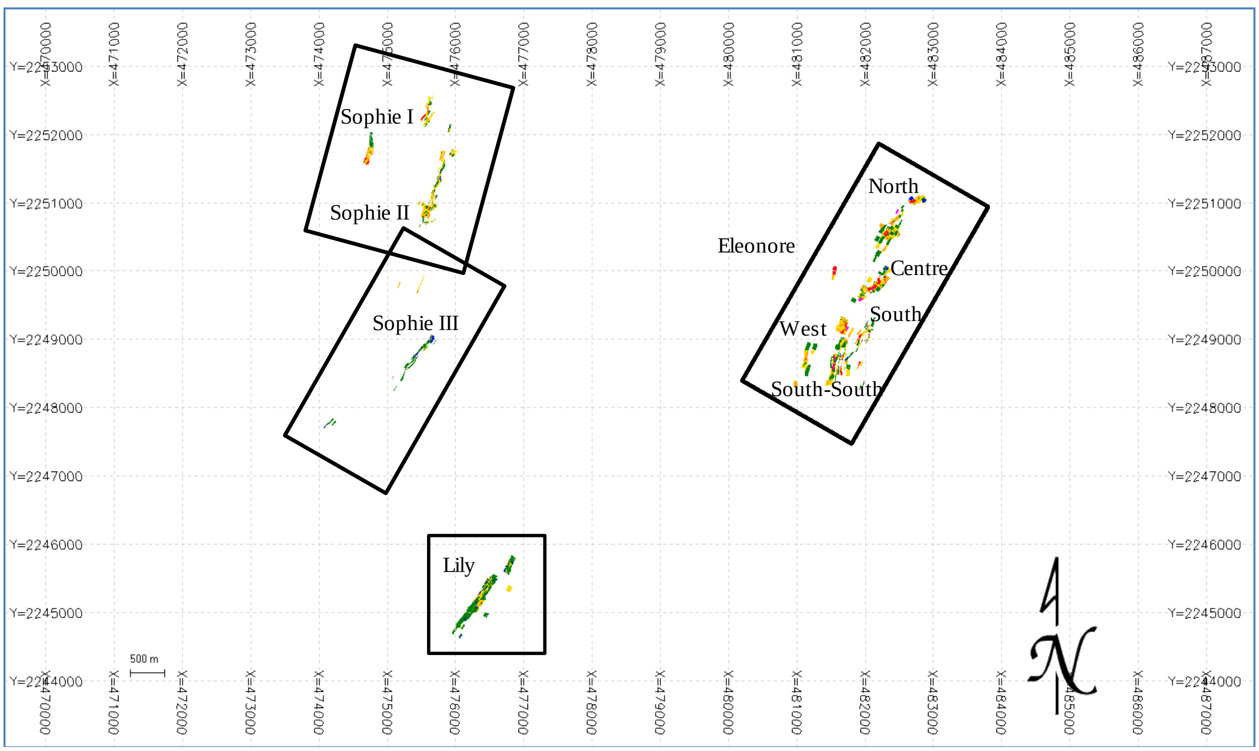


Figure 14-1: Map of the 4 Grids Used for Block Models

14.7.1 Variography

The variography has been revised on January 4, 2018. A block model has been estimated by kriging. The global results are less than 5 % different from the Inverse Square Distance model. The kriged model was not retained as the final model for this report. Like in June 2016, the short range variogram is acceptable. Actually, SGS Geostat calculated a correlogram. It is presented in Figure 14-2. For the first time, the long range variogram was quite successful thanks to some heavy manipulation of the data to maximize the number of pairs along the strike and dip of the mineralized zones. In order to prepare the long range variogram presented in Figure 14-3, the composites were unfolded using projections along the X axis. The Figure 14-4 presents the short and long range variogram. The Table 14-9 shows the summary of an integrated variogram. Note that the proper orientation was customized for each individual mineralized volume to be worked on. So, to summarise the results, one could say that the effective range perpendicular to the mineralized structures is of about 4.8 m while it is of 75 m along the structures. The author believes that there are different ranges for the different mineralized zones (Eleonore vs Lily for example) but more data will be needed to assess this assumption.

For June 2016 estimation, the smoothing of the final estimate was calibrated against the natural variance of blocks of 6 m x 6 m x 6 m. The result is that the final estimate has smoother variance then the natural 6 m x 6 m x 6 m blocks. The predicted variance of these blocks is done using the prediction of the internal variance of such a block. Having a final estimate smoother than estimated variance in reality is ideal because the final block model is therefore judged as on the conservative side when a cut-off grade is applied.

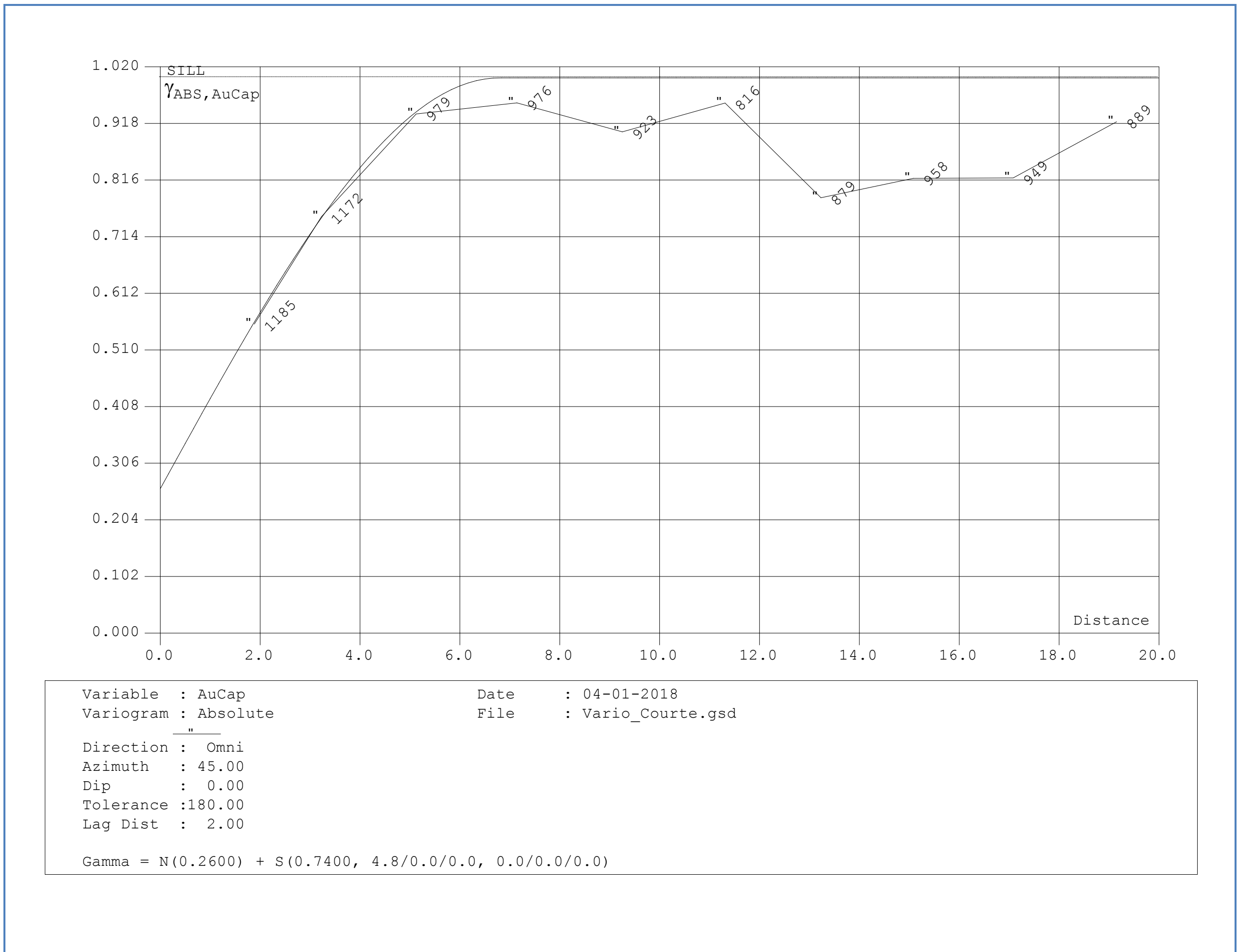
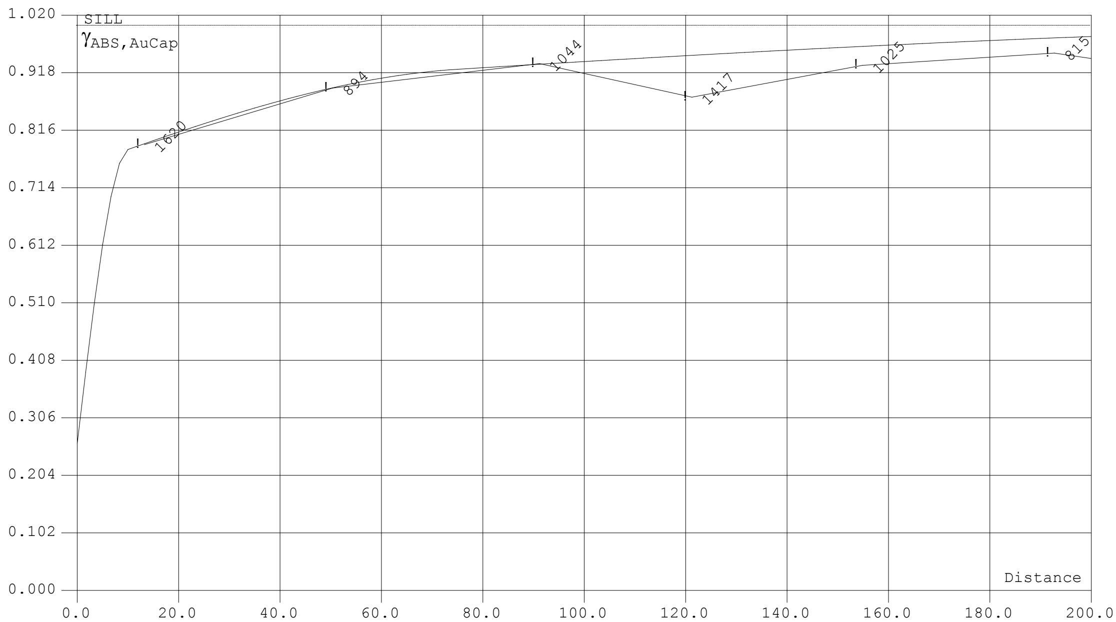


Figure 14-2: Short Range Variogram Model




```
Variable  : AuCap                      Date     : 04-01-2018
Variogram : Absolute                   File     : Vario_Longue2(yc2).gsd

Direction :  Long
Azimuth   :180.00
Dip       :-45.00
Tolerance :170.00
Lag Dist  : 35.00

Gamma = N(0.2600) + S(0.4900, 10.0/10.0/10.0, 0.0/0.0/0.0) + S(0.1300, 75.0/75.0/75.0, 0.0/0.0/0.0)
      + S(0.1200, 300.0/300.0/300.0, 0.0/0.0/0.0)
```

Figure 14-3: Long Range Variogram Model

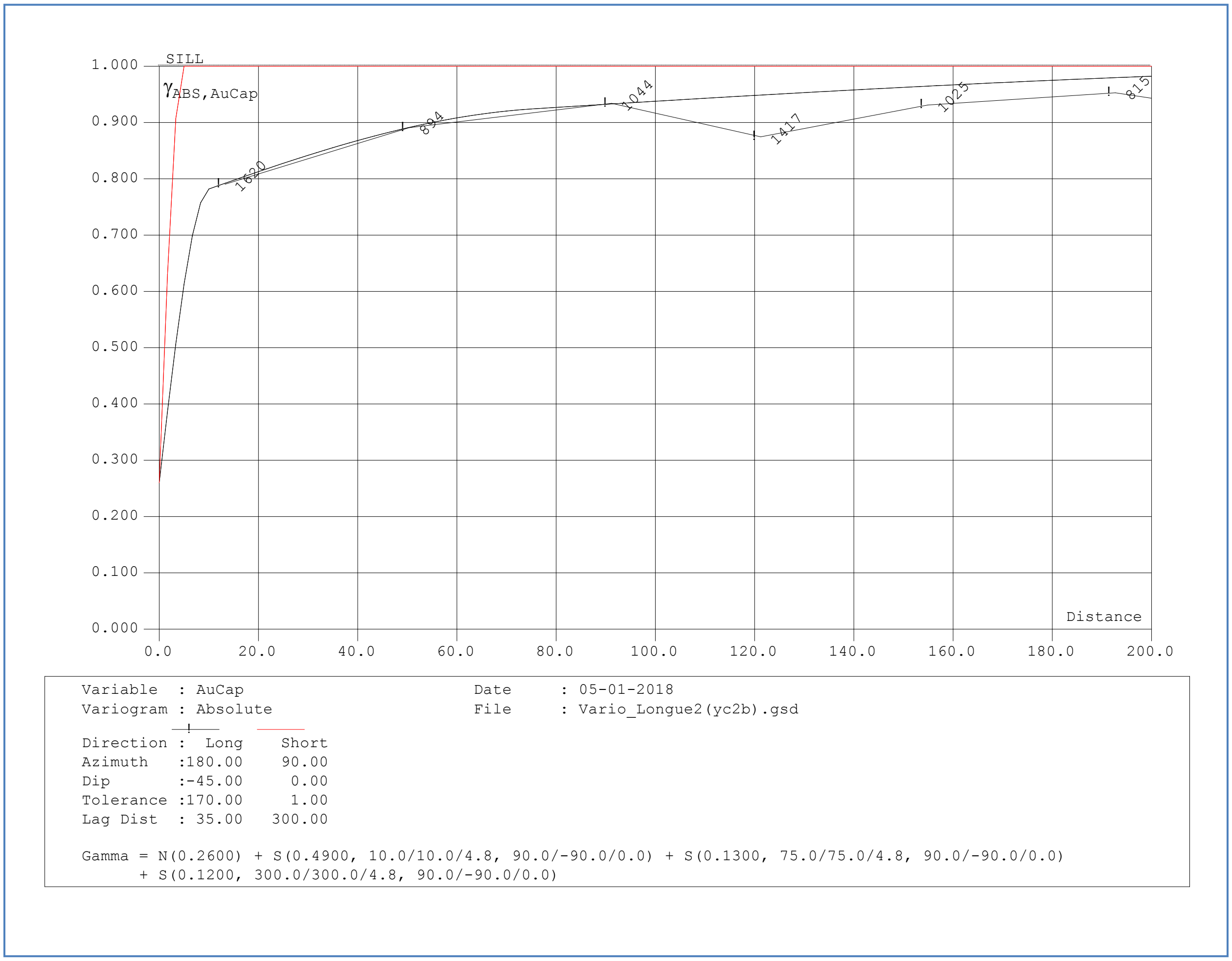


Figure 14-4: Short Range and Long Range Variogram Model

Table 14-9: Summary of the Variogram Model

Component			Ranges (m)		
Number	**Type**	**C (%)**	**Short**	**Medium**	**Long**
4	Spherical	12%	4.8	300	300
3	Spherical	13%	4.8	75	75
2	Spherical	49%	4.8	10	10
1	Nugget	26%	NA	NA	NA

14.7.2 Grade Interpolation Methodology

For the new estimation in 2018, most of the methodology has been retained from the 2017 one. To interpolate Au grade, the Inverse Square Distance (ISD) method was used, with anisotropic distances influenced by ellipsoids in the calculation and the composite selection. Block discretization was set to 2×2×2 for the estimation of block to composite distance. Blocks were created within all the mineralized envelopes, and each composite was tagged with an envelope name. A composite set was created for each envelope

(220 composite sets), containing both the Au and Au_Cap values. Two estimation passes were used with a small ellipsoid for the first pass and larger ellipsoid for the second pass. The small ellipsoids have radiuses of 75 m x 75 m x 25 m, and the large ellipsoids have radiuses of 150 m x 150 m x 50 m.

A variable orientation for the ellipsoids has been used to ensure that the orientation of the composite search is always in line with the estimated mineralized volume extension.

The first pass of the estimation used a minimum of 4 and a maximum of 7 composites, with the additional limit of 2 composites per drillhole. The second pass of the estimation used a minimum of 2 and a maximum of 7 composites, with the additional limit of 2 composites per drillhole except for 35 volumes with a minimum of one. There are exceptions for 27 volumes in Eleonore, 8 volumes in Sophie I-II-III and no volume in Lily: the minimum number of composites used in the second pass was lowered to 1 for these 35 volumes.

All blocks inside the mineralized volumes (envelopes) were estimated.

14.8 Classification

14.8.1 Definitions

Definitions are from the Canadian institute of Mining, Metallurgy and Petroleum (CIM):

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently been defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralisation may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Mineralisation or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

14.8.2 Classification Method

The categories Measured and Indicated have been given to blocks by drawing outlines manually on longitudinal views. Everything outside these limits have been attributed Inferred category. Also, only Sophie and Lily have been attributed Measured, Indicated and Inferred resources, Eleonore has been attributed with only Indicated and Inferred resources. The extents of the outlines are based on the distance between drill holes Mineralized Intersections (MI). The full thicknesses of the mineralized structures have the same category.

There must be at least 3 MIs in close neighbourhood to get some indicated resource. There must also be at least 2 MIs from RC or diamond drill holes in close neighbourhood to get some indicated resource. Drilling every 40 metres (Eleonore and Sophie I), 45 metres (Sophie II-III) and 50 metres (Lily) was classified as indicated.

There must be at least 4 MIs in close proximity to get some measured resource. There must also be at least 3 MIs from RC or diamond drill holes in close neighbourhood to get some measured resource. Drilling every 30 metres (all except Lily) and 35 metres (Lily) was classified as measured.

Inferred resources have been modeled using RC holes, diamond drill holes and trenches. The distance between Mineralized Intersections can be of up to 200 m where the geology map is more straightforward and when the structures appear to be straight and have grades continuously above the modeling cut-off. While the distance between Mineralized Intersections is of 200 m, the extensions are kept on the conservative side with extensions of 45 m both laterally and at depth. For the LilyG structure, extensions of up to 80 m have been used at depth, as this zone have the thickest mineralized intervals (around 25 m) and simple geometry. In Eleonore, the distance between Mineralized Intersections can be of up to 100 m. Eleonore extensions of 45 m both laterally and at depth have been used. The half distance to the next hole have been used when a drill hole cut short a mineralized volume by showing grades below the modeling grade. Eleonore has a maximum interpolation up to 100 metres. Other zones have interpolation up to 200 metres. All zones have extrapolation limited to 45 metres.

14.9 Optimized Open Pits

To limit resources to quantities with reasonable potential for profitable extraction, resources are limited by optimized pits. The new assumptions are a gold price of US $ 1,500/oz., mining recovery of 95 %, mining dilution of 5 % (for Sophie/Lily) and 10 % (for Eleonore), processing recovery of 96 % (for Sophie/Lily) and 97 % (for Eleonore), a processing cost of $12.02/t, a G&A cost of $3.69/t and an open pit mining cost of $1.41/t. For underground mining costs, the current assumption is $40/t. Based on these assumptions; the economically viable cut-off grades are 0.4 g/t Au in open pits and 1.5 g/t Au under the pits. Accordingly, Algold decided to retain COGs of 0.4 g/t Au in open pits and 1.7 g/t Au for Sophie/Lily and a global COG of 1.5 g/t Au for Eleonore.

Whittle software was used to create pit shells based on the resource model and the topographic surface. The open pits presented here to limit the resources are shells without ramps. The gold price, mining and processing costs and slope angle used for this optimization are shown in Table 14-10. Capped Au was used for the optimizations and is reported in resource tables.

The current resources evaluation with an effective date of January 19, 2018 considers the 5 % RIM royalty in calculating COG.

Table 14-10: Assumptions Used for the Optimization of the Open Pits and Economical COG Calculations of Open Pit Resources

Block Model Zone Name	Gold Price	Pit Angle	Processing		Mining			Royalties	COG
	($/oz)	(SAP/ROC)	Cost (%/t)	Recovery	Cost (%/t)	Recovery	Dilution	(%)	(g/t)
Sophie I-II-III	$ 1,500	40 / 50	$ 15.71	96%	$ 1.41	95%	5%	5%	0.4
Lily	$ 1,500	40 / 52	$ 15.71	96%	$ 1.41	95%	5%	5%	0.4
Eleonore (North, Centre, West)	$ 1,500	40 / 52	$ 15.71	97%	$ 1.41	95%	10%	5%	0.4
Eleonore (South, South-South)	$ 1,500	40 / 52	$ 15.71	97%	$ 1.41	95%	10%	5%	0.4

Note: Any COG above 0.4 g/t has the potential to be economic

Table 14-11: Assumptions Used for the Economical COG Calculation of Underground Resources

Block Model Zone Name	Gold Price	Processing		Underground Mining			Royalties	COG
	($/oz)	Cost (%/t)	Recovery	Cost (%/t)	Recovery	Dilution	(%)	(g/t)
All zones	$ 1,500	$ 15.71	97%	$ 40.00	95%	10%	5%	1.45

Note: Any COG above 1.45 g/t has the potential to be economic

Figure 14-5 to Figure 14-8 illustrate the block model used to estimate the base case mineral resource. Note the topographic surface in green, the Saprolite limit in blue and the pit shell in dark gray.

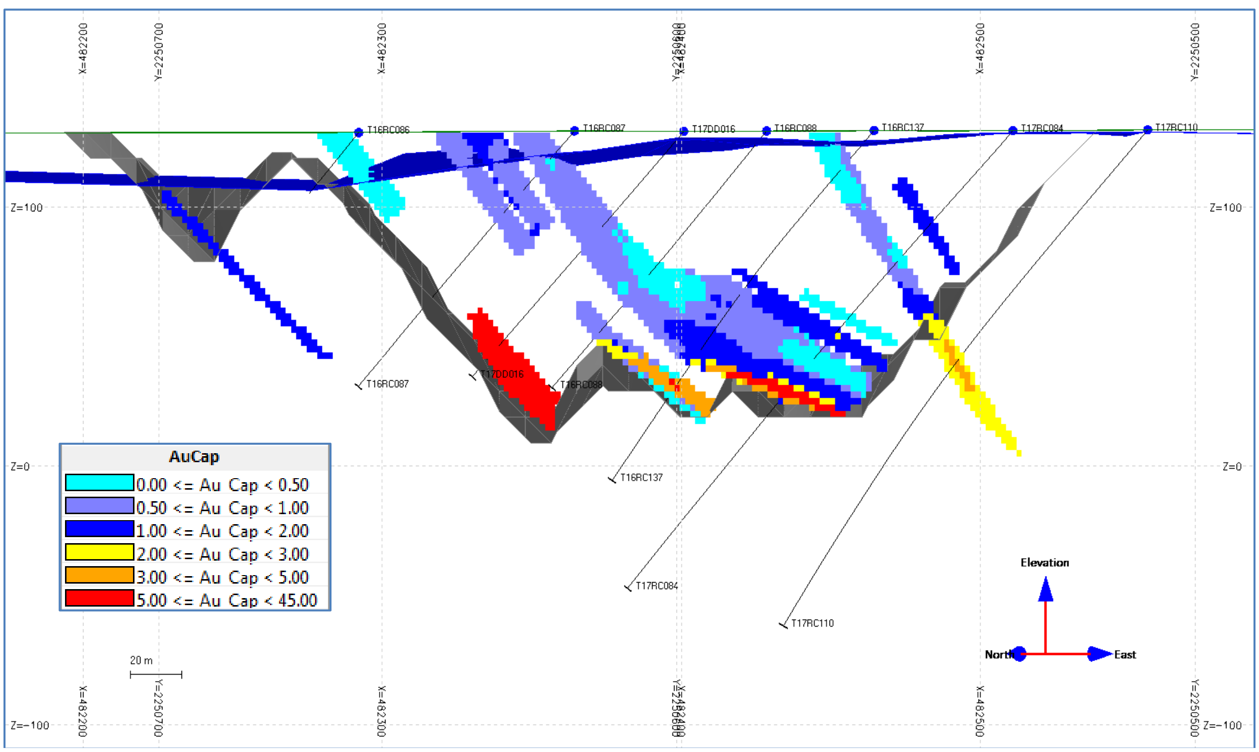


Figure 14-5: Section S9840 through Eleonore Pits with Blocks Visible from Zones Eleonore_Nord_12, 8, 16, 24, 19, 7, 1, 4, 26, 5, and 18 from Left to Right

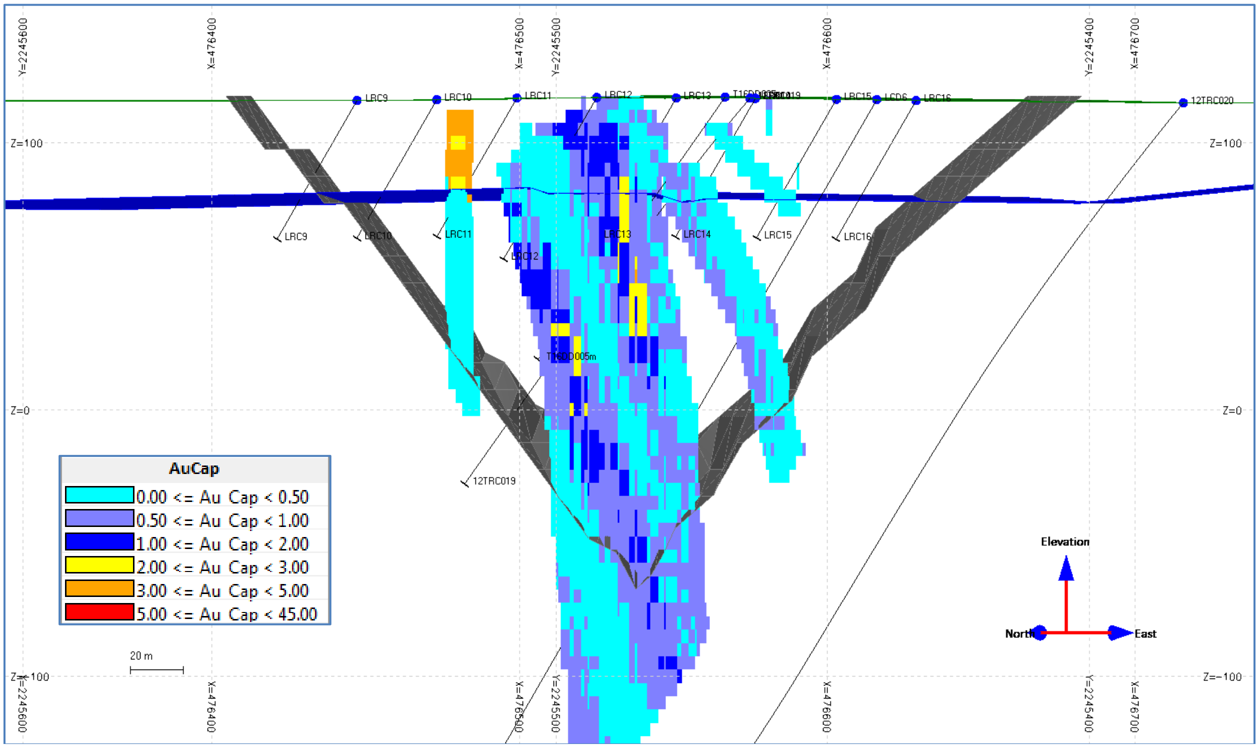


Figure 14-6: Section S2480 Through Lily Pits with Blocks Visible from Zones Lily_N, C, D and E from Left to Right

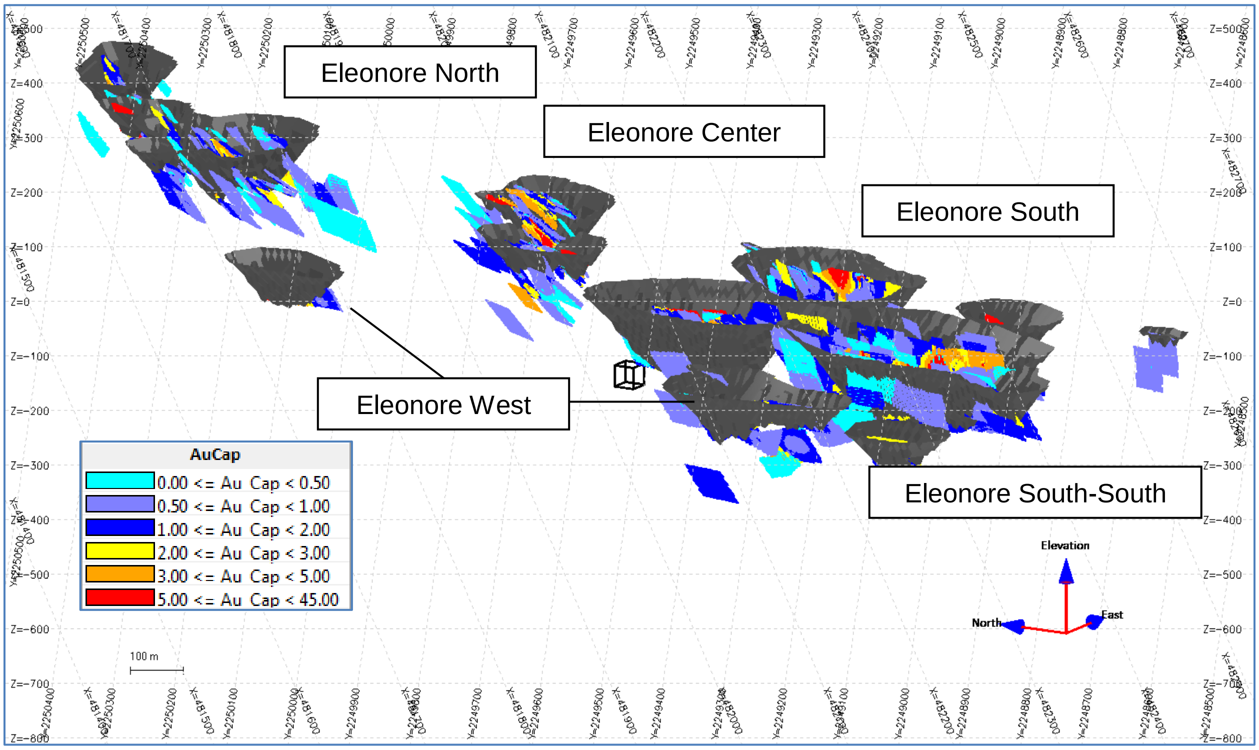


Figure 14-7: Eleonore Pits with Resource Block Models

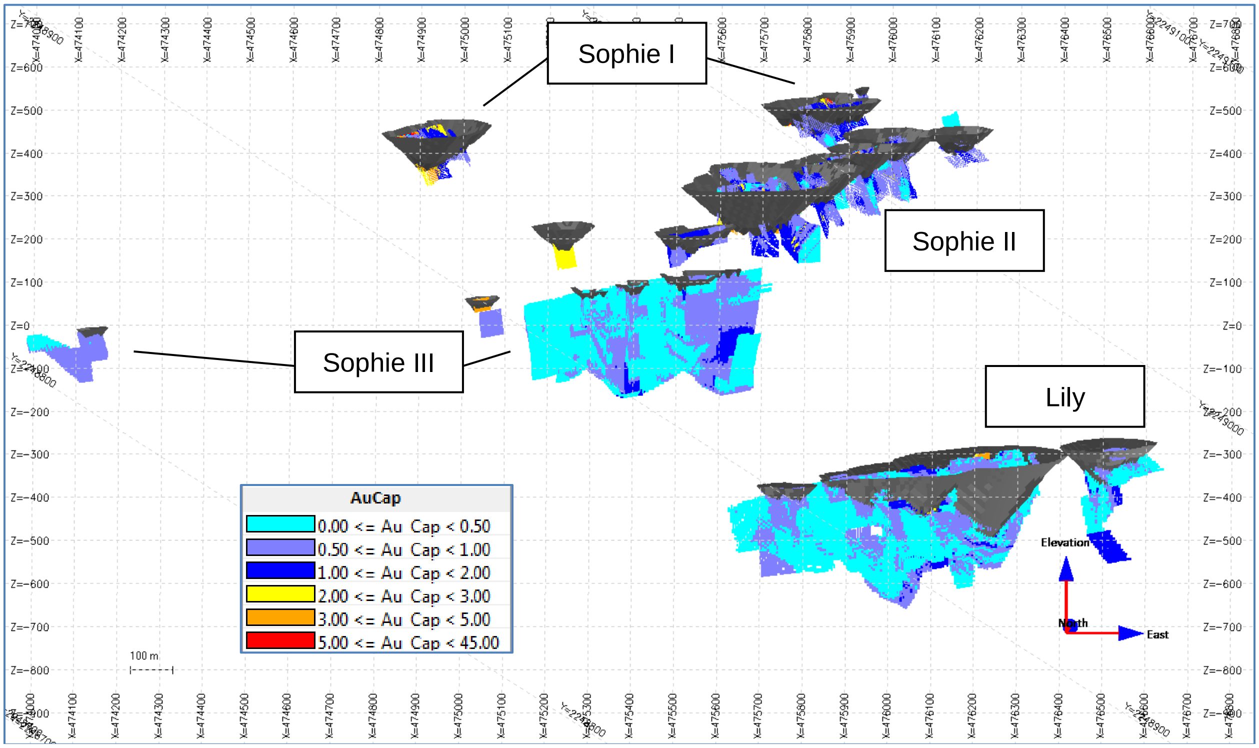


Figure 14-8: Sophie and Lily Pits with Resource Block Models

14.10 Mineral Resource Estimates (Base Case)

Resource estimation tables were created using Genesis and MS Access software to add up block data. Capped Au is reported in resource tables. The base case is presented in Table 14-12 has a COG of 0.4 g/t inside pits and 1.7 g/t below pits except for Eleonore at a global COG of 1.5 g/t.

Totals may not add up due to rounding. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The resource, especially the inferred category, could be materially affected by the mineralization interpretation. To the knowledge of the author, there are no other factors such as environmental, permitting, legal, title, taxation, socio-economical, marketing or political factors that could materially affect these estimates.

Table 14-12: Base Case Resources

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Eleonore	Indicated	4.08	719,000	94,250
Eleonore	Inferred	4.07	3,016,000	394,690

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Sophie/Lily	Measured	0.98	376,000	11,900
Sophie/Lily	Indicated	0.93	2,122,000	63,300
Total Sophie/Lily	Measured + Indicated	0.94	2,498,000	75,200
Sophie/Lily	Inferred	1.06	7,476,000	254,100

Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Total Sophie/Lily/Eleonore	Measured	0.98	376,000	11,900
Total Sophie/Lily/Eleonore	Indicated	1.72	2,841,000	157,550
Total Sophie/Lily/Eleonore	Measured + Indicated	1.64	3,217,000	169,450
Total Sophie/Lily/Eleonore	Inferred	1.92	10,492,000	648,790

1. Effective date for Eleonore and Sophie/Lily resources is January 19, 2018.
2. The independent QP for this resources estimate is Yann Camus, P.Eng., SGS Canada Inc.
3. The mineral resources are presented at a 0.4 g/t Au cut-off grade in pits and 1.7 g/t Au cut-off grade under the pits, except Eleonore at a global cut-off 1.5 g/t Au.
4. The resources are presented without dilution.
5. Whittle pits have been utilized based on a gold value of US$1,500/oz.
6. Mineral resources that are not mineral reserves do not have demonstrated economic viability. This disclosure does not include economic analysis of the mineral resources.
7. Totals may not add up due to rounding.
8. No economic evaluation of the resources has been produced.
9. This Resource estimate has been prepared in accordance with CIM definition (2014).
10. Density used is between 2.0 and 3.0 depending on rock type and alteration based on measurements.
11. Capping varies from 3.5 g/t Au (Lily) to 45 g/t Au (Eleonore) depending on extreme local grade.

14.11 Mineral Resource Estimates (Various Alternate Cases)

The following resource tables are provided to test the sensitivity of the resource to a change in the COG that can be caused by a change in the gold price for example. The COG for a particular project can also change if any of the cost and mining assumptions change. All the notes provided for the base case except note 3 also apply to the alternative cases unless noted.

The Table 14-13 for the Eleonore zone and Table 14-14 shows the detailed mineral resources for the Sophie and Lily zones at COGs of 1.5 g/t and 2.0 g/t Au without use of pit constrains. COGs above 1.45 g/t Au have reasonable prospect for economic extraction for current underground mining assumptions.

Table 14-13: Eleonore Resources at COGs of 1.5 g/t and 2.0 g/t Au

Zone	COG (g/t Au)	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Eleonore	1.5	Indicated	4.08	719,000	94,250
Eleonore	1.5	Inferred	4.07	3,016,000	394,690
Eleonore	2.0	Indicated	4.88	535,000	84,010
Eleonore	2.0	Inferred	5.12	2,089,000	343,490

Table 14-14: Sophie/Lily Resources at COGs of 1.5 g/t and 2.0 g/t Au

Zone	COG (g/t Au)	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)
Sophie/Lily	1.5	Measured	2.55	51,000	4,200
Sophie/Lily	1.5	Indicated	2.18	260,000	18,210
Sophie/Lily	1.5	Measured + Indicated	2.24	311,000	22,410
Sophie/Lily	1.5	Inferred	2.26	1,551,000	112,820
Sophie/Lily	2.0	Measured	3.03	33,000	3,180
Sophie/Lily	2.0	Indicated	2.67	125,000	10,720
Sophie/Lily	2.0	Measured + Indicated	2.74	158,000	13,890
Sophie/Lily	2.0	Inferred	2.97	682,000	65,100

14.12 Mineral Resource Estimates (For Eleonore – Alternate Cases with Open Pits)

The Eleonore pits have been used to prepare the following tables. These resource tables show the sensitivity of the resource to a change in the COG. They also show the proportion of resources contained in the optimized open pits vs under the pits. The scenarios with different COGs in pits and under pits are presented in

Table 14-16 to Table 14-18. The scenarios with equal COGs in pits and under pits are presented in Table 14-19 to Table 14-21. This exercise shows that most of the ounces are located inside optimized pits. We can estimate that between 84 % and 94 % of the ounces could possibly be accessed by open pit mining.

Table 14-15: Eleonore Zone Sensitivity Analysis - Various Cut-off Grades (within 0.4 g/t Au Whittle pit constraint)

COG (In Pit) Au (g/t)	COG (Under Pit) Au (g/t)	Class Name	Au (g/t)	Tonnage (t)	Ounces (Au)	Comments
0.4	1.7	Indicated	2.48	1,414,000	112,900	COG different in pit and under pit
0.4	1.7	Inferred	2.66	5,349,000	457,500	
0.6	1.7	Indicated	2.77	1,237,000	110,090	
0.6	1.7	Inferred	2.92	4,768,000	448,130	
1.0	1.7	Indicated	3.45	921,000	102,200	
1.0	1.7	Inferred	3.49	3,773,000	422,900	

Table 14-16: Mineral Resources for Eleonore In Pit vs Under Pit at COGs of 0.4 g/t and 1.7 g/t Au

COG (g/t Au)	In Pits/ Under Pits	Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)	% Ounces in Pits vs Under Pits
0.4	In Pits	Eleonore	Indicated	2.49	1,382,000	110,550	92 %
0.4	In Pits	Eleonore	Inferred	2.67	4,813,000	413,080	
1.7	Under Pits	Eleonore	Indicated	2.31	31,000	2,310	8 %
1.7	Under Pits	Eleonore	Inferred	2.58	536,000	44,430	

Table 14-17: Mineral Resources for Eleonore In Pit vs Under Pit at COGs of 0.6 g/t and 1.7 g/t Au

COG (g/t Au)	In Pits/ Under Pits	Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)	% Ounces in Pits vs Under Pits
0.6	In Pits	Eleonore	Indicated	2.78	1,206,000	107,780	92 %
0.6	In Pits	Eleonore	Inferred	2.97	4,232,000	403,700	
1.7	Under Pits	Eleonore	Indicated	2.31	31,000	2,310	8 %
1.7	Under Pits	Eleonore	Inferred	2.58	536,000	44,430	

Table 14-18: Mineral Resources for Eleonore In Pit vs Under Pit at COGs of 1.0 g/t and 1.7 g/t Au

COG (g/t Au)	In Pits/ Under Pits	Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)	% Ounces in Pits vs Under Pits
1.0	In Pits	Eleonore	Indicated	3.49	890,000	99,910	91 %
1.0	In Pits	Eleonore	Inferred	3.64	3,237,000	378,430	
1.7	Under Pits	Eleonore	Indicated	2.31	31,000	2,310	9 %
1.7	Under Pits	Eleonore	Inferred	2.58	536,000	44,430	

Table 14-19: Mineral Resources for Eleonore In Pit vs Under Pit at a COG of 1.5 g/t Au

COG (g/t Au)	In Pits/ Under Pits	Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)	% Ounces in Pits vs Under Pits
1.5	In Pits	Eleonore	Indicated	4.18	682,000	91,640	89 %
1.5	In Pits	Eleonore	Inferred	4.59	2,317,000	341,950	
1.5	Under Pits	Eleonore	Indicated	2.20	37,000	2,620	11 %
1.5	Under Pits	Eleonore	Inferred	2.35	699,000	52,730	

Table 14-20: Mineral Resources for Eleonore In Pit vs Under Pit at a COG of 2.0 g/t Au

COG (g/t Au)	In Pits/ Under Pits	Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)	% Ounces in Pits vs Under Pits
2.0	In Pits	Eleonore	Indicated	4.97	516,000	82,360	91 %
2.0	In Pits	Eleonore	Inferred	5.66	1,688,000	306,960	
2.0	Under Pits	Eleonore	Indicated	2.58	20,000	1,650	9 %
2.0	Under Pits	Eleonore	Inferred	2.84	401,000	36,530	

Table 14-21: Mineral Resources for Eleonore In Pit vs Under Pit at a COG of 2.5 g/t Au

COG (g/t Au)	In Pits/ Under Pits	Zone	Classification	Au (g/t)	Tonnage (t)	Ounces (Au)	% Ounces in Pits vs Under Pits
2.5	In Pits	Eleonore	Indicated	5.64	414,000	75,060	94 %
2.5	In Pits	Eleonore	Inferred	6.60	1,323,000	280,700	
2.5	Under Pits	Eleonore	Indicated	3.02	10,000	920	6 %
2.5	Under Pits	Eleonore	Inferred	3.44	195,000	21,550	

15. MINERAL RESERVE ESTIMATES

This Report is a Preliminary Economic Assessment (PEA), no Mineral Reserves have been estimated for the Tijirit Algold Project as per NI 43-101 regulations. In-pit resources are described in Section 16 of this Report.

16. MINING METHODS

The mining methods and In-pit Mineral Resource estimate for the Tijirit deposit were prepared by Richard Bonici, Eng., Mining Engineer with DRA/Met-Chem and verified by Patrick Perez, Eng., Director of Mining and Geology DRA/Met-Chem as a Qualified Person. All work related to Pit Optimization for the PEA was performed using Whittle software. MineSight was used for Pit design and Mine Planning. Both Whittle and MineSight are commercially available software.

Topographic Surface

The mine design for the Report was carried out using a topographic surface that originated from a Laser Imaging Detection and Ranging Survey (LIDAR). The topographic surface was supplied to DRA/Met-Chem as 1 m elevation contours.

Resource Block Model

The mine design for the Report is based on the 3-dimensional geological block model that was prepared by SGS-Geostat and presented in Section 14. The Tijirit property is divided into three distinct mining zones, Eleonore, Lily and Sophie. SGS-Geostat provided DRA/Met-Chem with a block model for each zone. The block size for Eleonore is 2m x 5m x 2.5m, Lily's block size is 5m x 5m x 5m and Sophie's block size is 2m x 5m x 2.5m. Each model's grade is represented with an Au grade item.

Each block in the resource model contains an Au grade attribute, a resource classification attribute (Measured, Indicated and Inferred), and a density attribute. Using the DDH collars, an overburden surface was created by SGS-Geostat and used to differentiate the non-mineralized material as either saprolite un-mineralized or waste rock.

Material Properties

The material properties for the different rock types are outlined below. These properties are important in estimating the mineral resources as well as the dump and stockpile design capacities.

<u>Density</u>

As was discussed in Section 14 of this Report, the in-situ dry density of the saprolite material is 2 t/m^3 for, and the fresh rock has an in-situ dry density of 2.85 t/m^3.

For haulage calculation, the tailings density has been assumed at 2.22 t/m^3.

<u>Swell Factor</u>

The swell factor reflects the increase in volume of material from its in-situ state to after it is blasted and loaded into the haul trucks. A swell factor of 35% was used in this Report.

<u>Moisture Content</u>

The moisture content reflects the amount of water that is present within the rock formation. It affects the estimation of haul truck requirements and must be considered during the payload calculations. The moisture content is also an important factor for the process water balance.

Since the mineral resources are estimated using the dry density, they are not affected by the moisture content value. A moisture content of 5% was used for the Report. This value is typical for similar projects in the region.

16.1 Open Pit Optimization

The pit optimization analysis uses economic criteria to determine to what extent the deposit can be mined profitably.

The pit optimization analysis was done using Whittle optimizing tool. The optimizer uses the 3D Lerch-Grossmann algorithm to determine the economic pit limits based on input of mining and processing costs and revenue per block. In order to comply with NI 43-101 guidelines regarding the Standards of Disclosure for Mineral Projects, blocks classified in the Measured and Indicated and Inferred categories are allowed to drive the pit optimizer for a PEA Study.

Table 16-1 presents the parameters that were used for the pit optimization analysis. All figures are in US Dollars. The cost and operating parameters that were used are preliminary estimates for developing the economic pit and should not be confused with the operating costs subsequently developed for the Report and presented in Section 21.

Table 16-1: Pit Optimization Parameters

Item	Units	Eleonore	Sophie & Lily
Mining Cost (Waste)	$/t (mined)	1.41	1.41
Mining Cost (Ore)	$/t (mined)	1.41	1.41
Processing Cost + G&A	$/t (milled)	23.28	15.71
Mining Dilution	%	10	5
Sales Price	$/oz (product)	1,250	1,250
Average Mining Recovery	%	95	95
Average Mill Recovery	%	97	97
Mining Rate	t/j	2,976	4,500
Royalty Rate	%	6.5 %	6.5 %
Overall Pit Slope*	degree	40.6/52.6	40.6/52.6
All Prices and operating costs are in USD * 40.6° in saprolite and 52.6°in hard rock			

Using the cost and operating parameters, a series of pit shells were generated for each of the mining zones, Eleonore, Sophie and Lily. By varying the selling price (revenue factor) from $12.06 to 80.38 /g (0.3 to 2.0 revenue factor), an optimal pit shell was chosen for each zone. For Eleonore, PIT 36 was chosen with a revenue factor of 1. The pit optimization analysis shows PIT 36 as the pit shell with the best NPV. This pit shell has a strip ratio of 17.49 to 1. Mining additional resources with an open pit beyond the limits of this pit shell does not provide an increase in NPV. Upon completion of the PEA Study, DRA/Met-Chem confirmed that the pit optimization exercise was still valid using the updated cost estimate developed in the Study.

The tonnages and grades associated with each of the pit shells are presented in Table 16-2. The Net Present Value (NPV) of each shell was calculated assuming a selling price of $1,250/oz of product, a discount rate of 10%. A variable production rate was considered given the project's two (2) phases approach. Phase 1 production will consider an annual production of 1,086,240 tonnes per year. Phase 2 increases the plant capacity up to 1,642,500 tonnes per year in Year 5 of production.

Table 16-2: Pit Optimization Results (Eleonore)

PIT	Strip Ratio	Rev Factor	Waste Tonnes	Mineralised Material Tonnes	NPV	Max. Bench	Min. Bench	Gold Contained (g)	Gold Grade (g/t)
15	15.35	0.58	23,521,527	1,438,348	200,611,800	35	23	7,953,069	5.53
16	18.34	0.6	41,043,952	2,122,090	221,313,570	35	15	9,814,394	4.62
17	18.23	0.62	42,432,684	2,206,814	222,720,716	35	15	9,990,245	4.53
18	18.1	0.64	43,871,942	2,296,688	224,089,343	35	15	10,169,125	4.43
19	18.04	0.66	44,746,354	2,350,636	224,880,930	35	15	10,274,436	4.37
20	19.49	0.68	56,024,255	2,734,497	233,176,263	35	15	11,280,572	4.12
21	19.45	0.7	58,120,628	2,842,370	234,621,846	35	14	11,498,031	4.04
22	19.23	0.72	58,954,592	2,913,929	235,199,860	35	14	11,608,112	3.98
23	19.28	0.74	61,526,216	3,033,529	236,576,125	35	14	11,849,732	3.91
24	19.19	0.76	63,909,105	3,164,703	237,697,820	35	14	12,083,622	3.82
25	19.11	0.78	65,115,208	3,238,417	238,174,305	35	14	12,204,868	3.77
26	18.94	0.8	65,959,415	3,308,094	238,553,551	35	14	12,302,473	3.72
27	18.87	0.82	67,445,158	3,395,149	239,106,232	35	14	12,442,217	3.66
28	18.59	0.84	68,465,668	3,494,959	239,480,766	35	14	12,566,310	3.6
29	18.54	0.86	71,664,044	3,667,656	240,305,384	35	14	12,841,116	3.5
30	18.25	0.88	72,908,650	3,787,511	240,615,290	35	14	12,983,580	3.43
31	18.14	0.9	74,767,966	3,906,903	240,943,130	35	14	13,149,222	3.37
32	17.98	0.92	75,394,081	3,973,252	241,039,802	35	14	13,222,298	3.33
33	17.78	0.94	75,880,018	4,039,831	241,094,604	35	14	13,288,162	3.29
34	17.57	0.96	76,271,826	4,106,468	241,124,625	35	14	13,349,133	3.25
35	17.58	0.98	77,791,367	4,186,842	241,163,235	35	14	13,462,441	3.22
36	**17.49**	**1**	**79,374,779**	**4,293,815**	**241,144,379**	**35**	**14**	**13,592,923**	**3.17**
37	17.37	1.02	80,105,492	4,361,610	241,119,550	35	14	13,663,934	3.13
38	17.35	1.04	81,918,462	4,464,708	241,010,555	35	14	13,795,962	3.09
39	17.25	1.06	83,382,243	4,568,668	240,869,768	35,	14	13,913,274	3.04
40	17.15	1.08	84,513,777	4,655,527	240,757,052	35	14	14,006,074	3.01
41	17	1.1	85,208,337	4,734,632	240,649,536	35	14	14,077,086	2.97
42	16.99	1.12	86,379,431	4,800,278	240,489,919	35	14	14,155,505	2.95
43	16.87	1.14	87,476,738	4,896,518	240,278,947	35	14	14,247,557	2.91

PIT	Strip Ratio	Rev Factor	Waste Tonnes	Mineralised Material Tonnes	NPV	Max. Bench	Min. Bench	Gold Contained (g)	Gold Grade (g/t)
44	16.75	1.16	87,965,979	4,954,461	240,185,054	35	14	14,296,189	2.89
45	16.75	1.18	90,005,647	5,071,165	239,769,386	35	14	14,427,533	2.84
46	16.69	1.2	91,695,843	5,183,536	239,367,360	35	14	14,542,671	2.81
47	16.56	1.22	92,521,505	5,270,175	239,163,846	35	14	14,614,928	2.77
48	16.95	1.24	99,651,423	5,551,830	237,514,512	35	14	14,984,641	2.7
49	16.89	1.26	100,565,146	5,622,150	237,266,134	35	14	15,048,773	2.68
50	16.9	1.28	102,255,467	5,714,018	236,836,813	35	14	15,146,239	2.65

The Eleonore zone will feed the Phase 1 to completion and will supply the plant with a higher-grade (Au) resources. Sophie and Lily will be considered in Phase 2 of production at which point an increase in production to 4,500 t/day will be considered. Figure 16-1 and Figure 16-2 present the results of the Eleonore pit optimization.

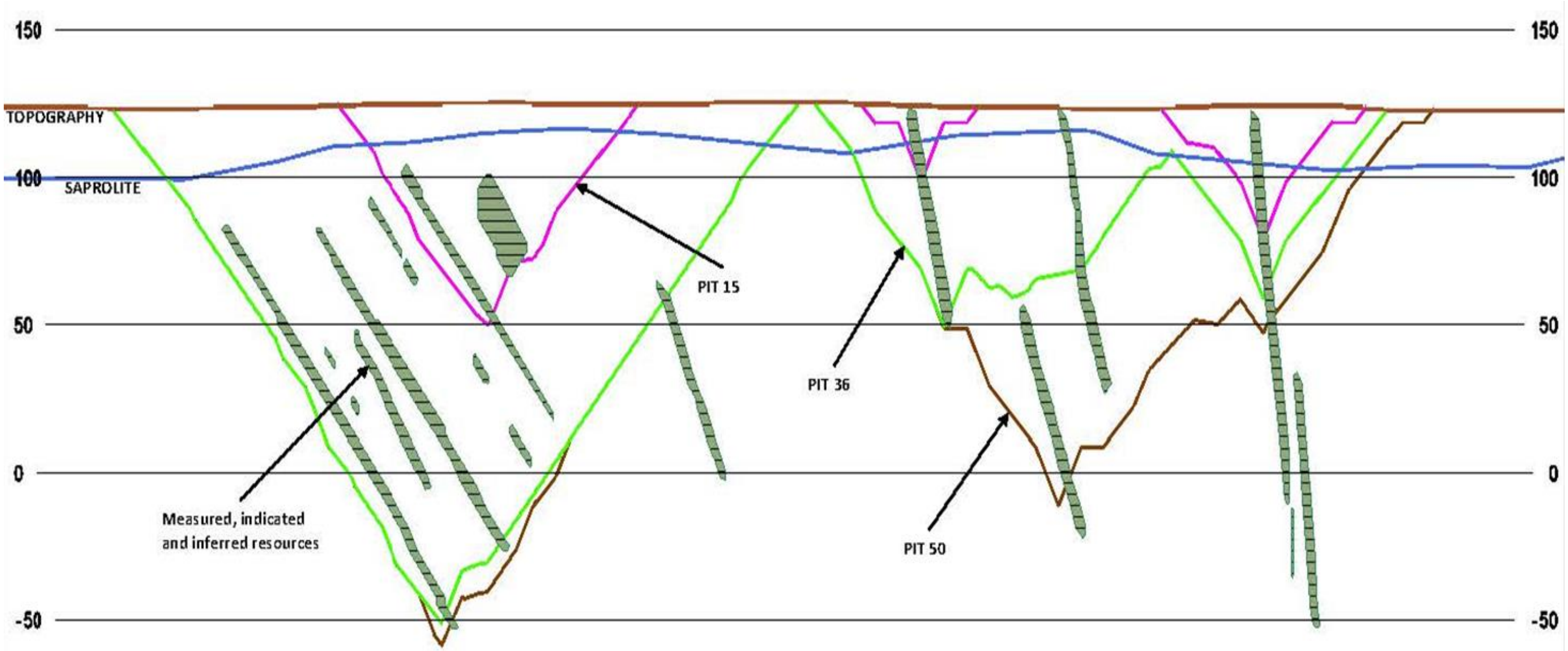


Figure 16-1: Pit Shell Graphical Representation (Eleonore South)

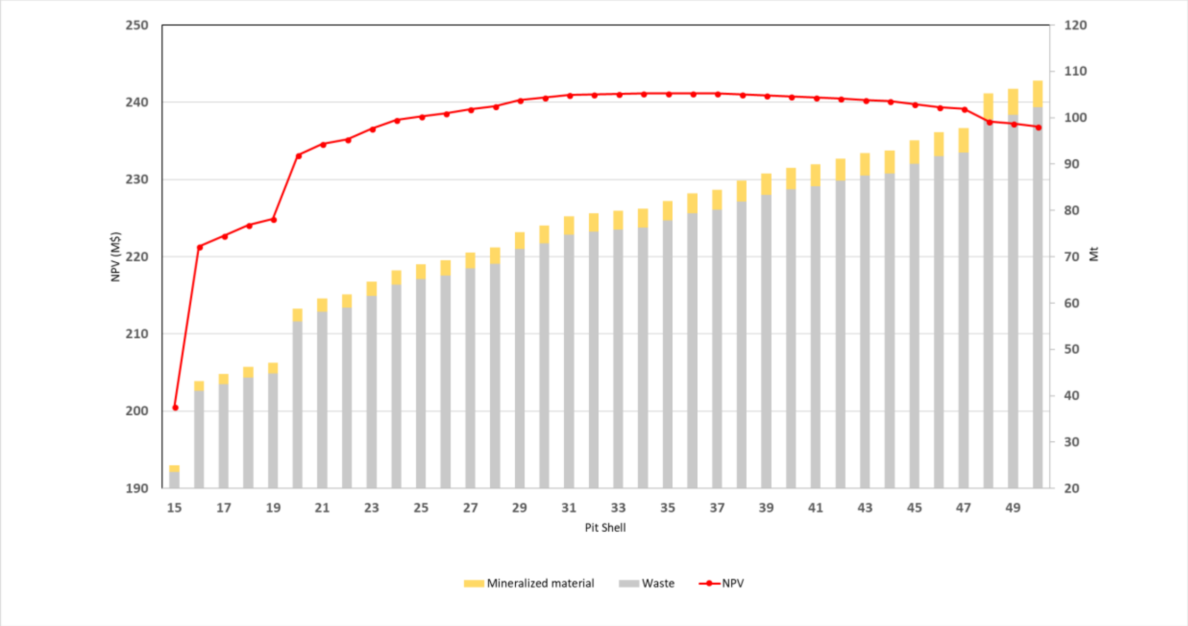


Figure 16-2: Pit Optimization /NPV Analysis – Eleonore Mining Zone

By applying the pit optimization method previously explained, multiple pit shells were generated in a similar fashion for both Sophie and Lily mining zones. Table 16-3 summarises the results of this analysis by mining zone targeting the most profitable pit for in each.

Table 16-3: NPV Analysis Results by Zone

Zone	PIT Chosen	Revenue Factor	NPV ($M)	S/R
Eleonore	36	1	241	17.49
Sophie 1&2	36	1	46	10.6
Sophie 3	34	1.02	3	5.01
Lily	35	1	28	4.1

Based on the results of the NPV analysis summarized above, a series of pit designs were created for each mining zone. The Eleonore zone located north east on the property includes 10 pit designs as shown in Figure 16-3 and Figure 16-4 (E1-E10), the Sophie zone includes 6 pits (S1A, S1B, S2, S3A, S3B and S3C) and the Lily zone located to the south includes 1 pit. Figure 16-5 presents a plan view of the Project giving the reader a sense of scale for the project

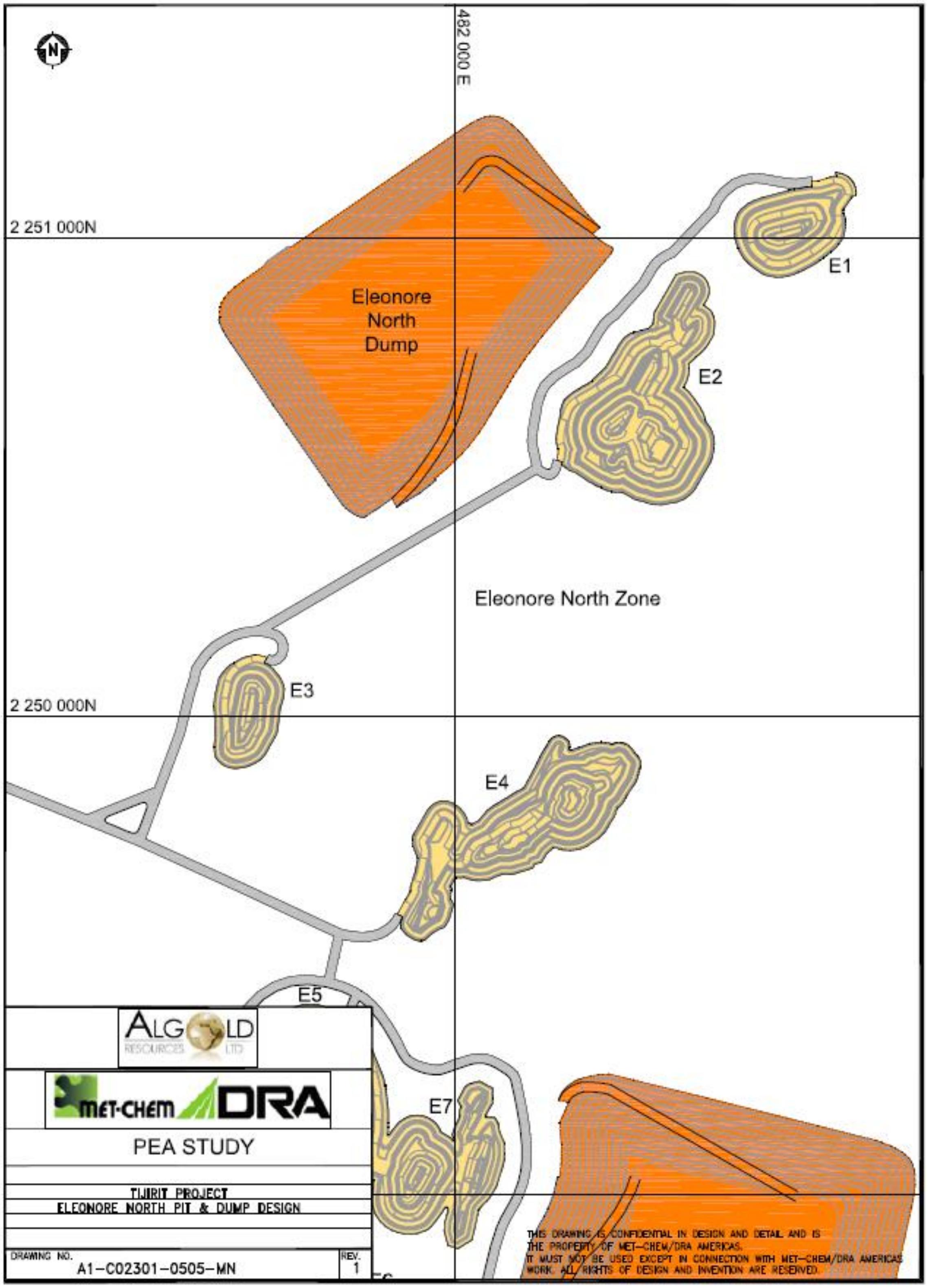


Figure 16-3: Eleonore Pits E1 to E4

Figure 16-4: Eleonore Pits E5 to E10

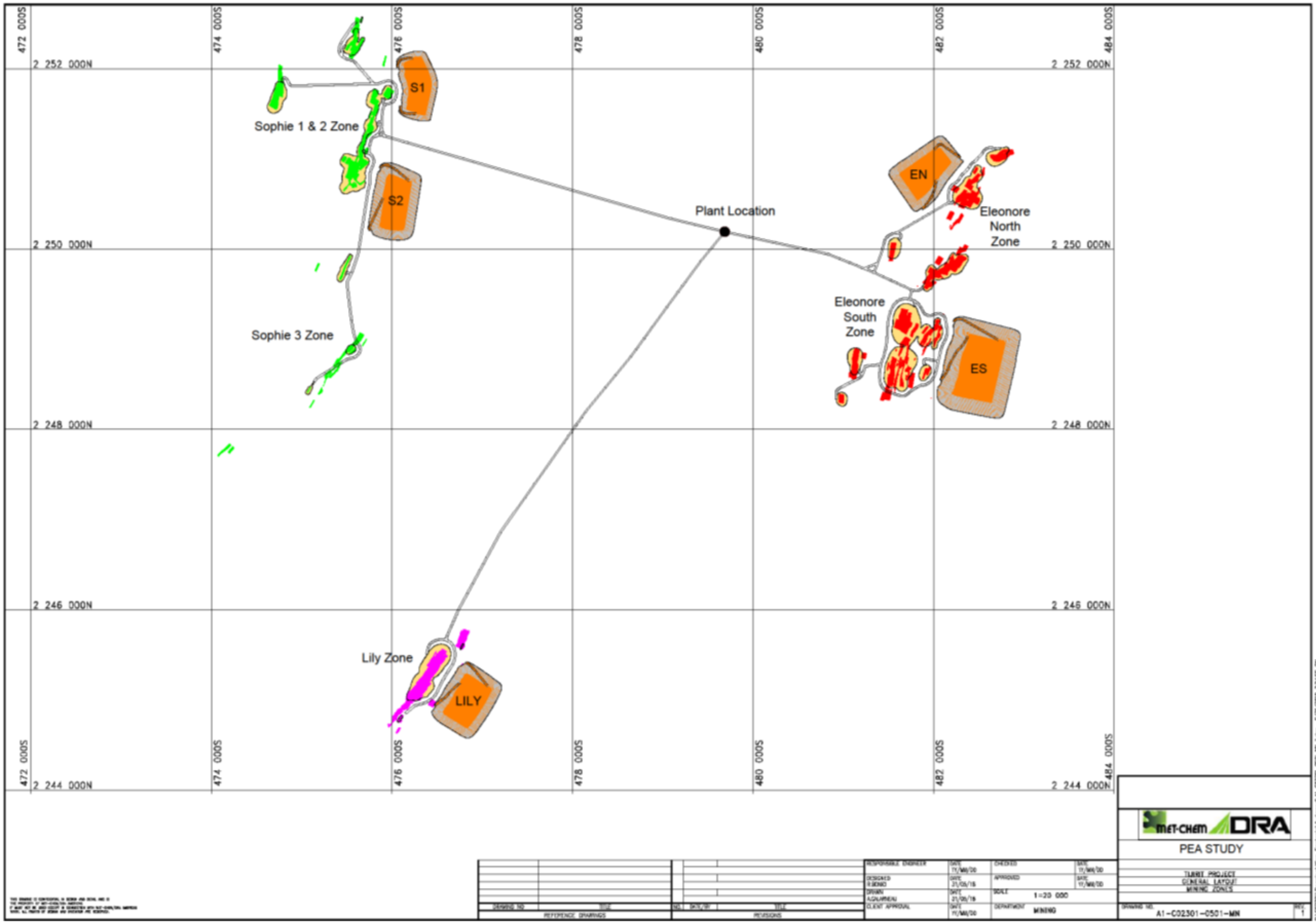


Figure 16-5: Plan View (Lily-Sophie-Eleonore)

16.2 Open Pit Design

The next step in the in-pit resource estimation process is to design an operational pit that will form basis of the production plan. This pit design uses the pit shell as a guideline and includes smoothing the pit wall, adding ramps to access the pit bottom and ensuring that the pit can be mined using the initially selected equipment. The following section provides the parameters that were used for the open pit design and presents the results.

16.2.1 Mining Methods

The mining method selected for the Project is a conventional truck and shovel, drill and blast quarry operation. The mineralised material and waste rock will be mined at 5 m high benches, drilled, blasted and loaded into rigid quarry haul trucks with hydraulic excavators.

16.2.2 Geotechnical Pit Slope Parameters

16.2.2.1 Introduction

WSP completed the PEA level geotechnical design for the Tijirit Project. The purpose of the study was to determine the geotechnical open pit design parameters derived from the characterization of the rock mass and geological data.

16.2.2.2 Work Completed

Geological and geotechnical logging data was provided by Algold for 55 core-oriented diamond drill boreholes and included identification of field rock strength, weathering, and rock quality designation. Data for the various lithologies, geological domains, and weathering profiles was compiled and analyzed for this study. An estimation of the Rock Mass Rating (RMR) parameters was assigned to each of the identified lithologies to establish the rock mass strength and deformability parameters.

Oriented core logging data was analyzed for each deposit to establish the orientation of the structural discontinuities. Oriented data was also compiled to assist in determining local and regional trends. To assist with the selection of bench face angles, kinematic stability assessments were completed for the Lily, Eleonore, and Sophie pits to determine the probability and likelihood of sliding, wedge, and toppling failures.

A limit equilibrium analysis was completed to determine pit stability for an assumed depth of 115 metres. The geological and structural model is currently being completed by Algold. The stability analysis considered various lithological types and blast damage or deformation scenarios. Pit stability analyses were completed in hydrogeological unsaturated conditions as Algold has reported that only a single drillhole on site has encountered water and at below the floor elevation of the proposed pits.

16.2.2.3 Recommended Slope Angles

A) Bench Geometry

The recommended bench geometry within the slightly weathered to fresh rock is a bench face angle of 70 degrees, a bench height of 25 m, and a bench width of 10 m. Within the highly weathered, moderately weathered rock, a bench face angle of 65 degrees is recommended with a bench height of 10 m and a bench width of 7 m.

The benches will be constructed in 5 m lifts and stacked according to the design taking into consideration the weathering profiles.

B) Inter-Ramp Angle

Inter-ramp angle (IRA) can be calculated based on the bench width (w), bench height (H), and bench face angle (BFA). The formula is as follows:

$$RA = tan^{-1}\left\{\frac{H\ (\tan\langle BFA\rangle)}{w(tan\langle BFA\rangle) +\ H}\right\}$$

Table 16-4 illustrates the calculated inter-ramp angles for a given height of 10 m, using the Modified Ritchie Bench Width criterion.

Table 16-4: Geotechnical Parameters

Domain	Face angle (°)	Berm length (m)	Bench height (m)	IRA (°)
Slightly Weathered (SW) to Fresh Rock (FR)	70	10	25	52.6
Completely Weathered (CW), Highly Weathered (HW) and Moderately Weathered (MW) Rock	65	7	10	40.6

The overall ramp angle has not been provided in this Report but will be determined upon finalization of the open pit design by others. The final design will include ramping and other infrastructure considerations to be provided by Algold.

The pit wall configuration considered for this Report is presented in Figure 16-6.

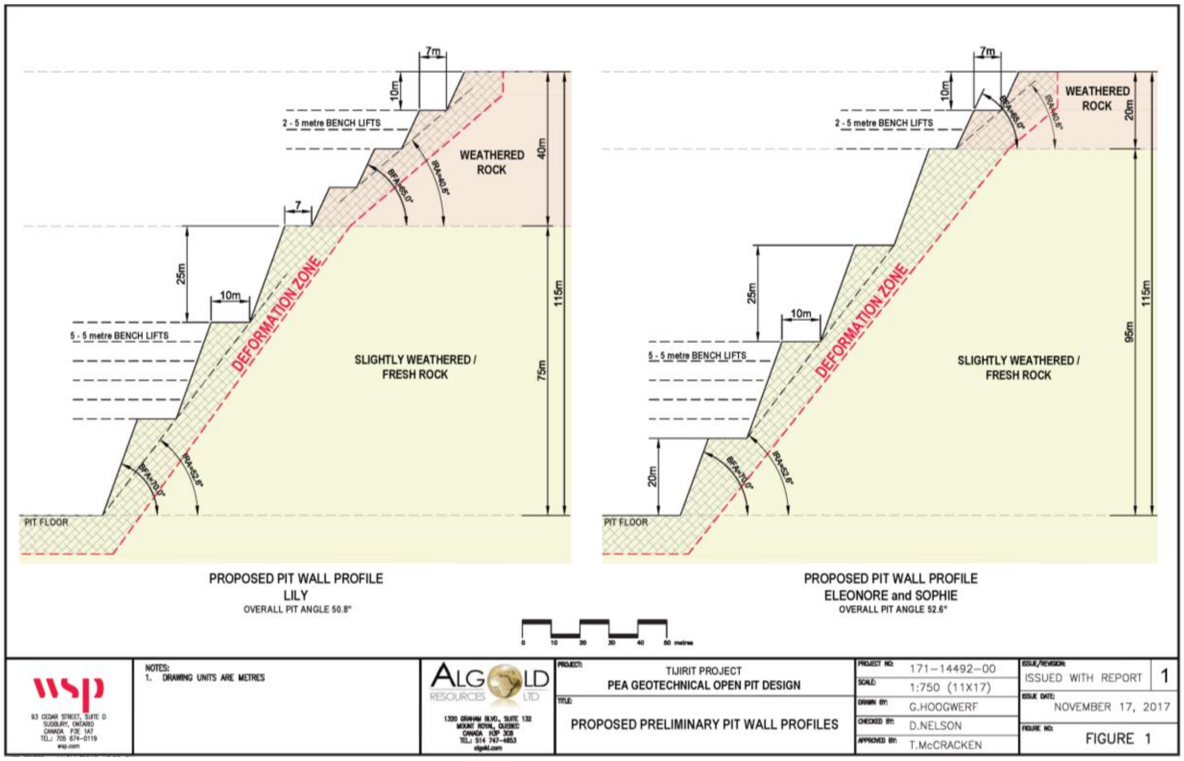


Figure 16-6: Pit Wall Configuration

16.2.2.4 Conclusions

Given the quality of the rock mass, the strength of the rock, the generalized fabric description, and the relative shallow depth of the proposed pit, the stability of the pit will be governed by the geomechanical attributes of the rock mass as well as global stability and associated rock mass strength characteristics.

Limit equilibrium analysis modelling of the global pit slope was undertaken to analyze for stability. Using the average values for rock quality rock and rock strength contributes to a factor of safety greater than 1.3 for a fully dewatered pit with good blasting conditions. It should be noted that the dataset is limited and a 3D geological and structural model has not been completed. Strength testing is required for future analysis. Further geotechnical campaigns should be completed to validate the input data parameters and to complete a more comprehensive geographically spaced dataset for additional detailed stability assessments.

16.2.3 Haul Road Design

The ramps and haul roads were designed with an overall width of 12 m. Both single line and double line traffic will take place in the pit. For double lane traffic, a width of 2.5 times the width of the largest truck was considered. The overall width of a 55-tonne quarry haul truck is 2.9 m which results in a running surface of approximately 8 m. The allowance for berms and ditches increases the overall haul road width to 12 m.

A maximum ramp grade of 12 % was considered for pit design. This grade is acceptable for a 55-tonne rigid frame quarry haul truck. Figure 16-7 presents a typical section of the in-pit ramp design.

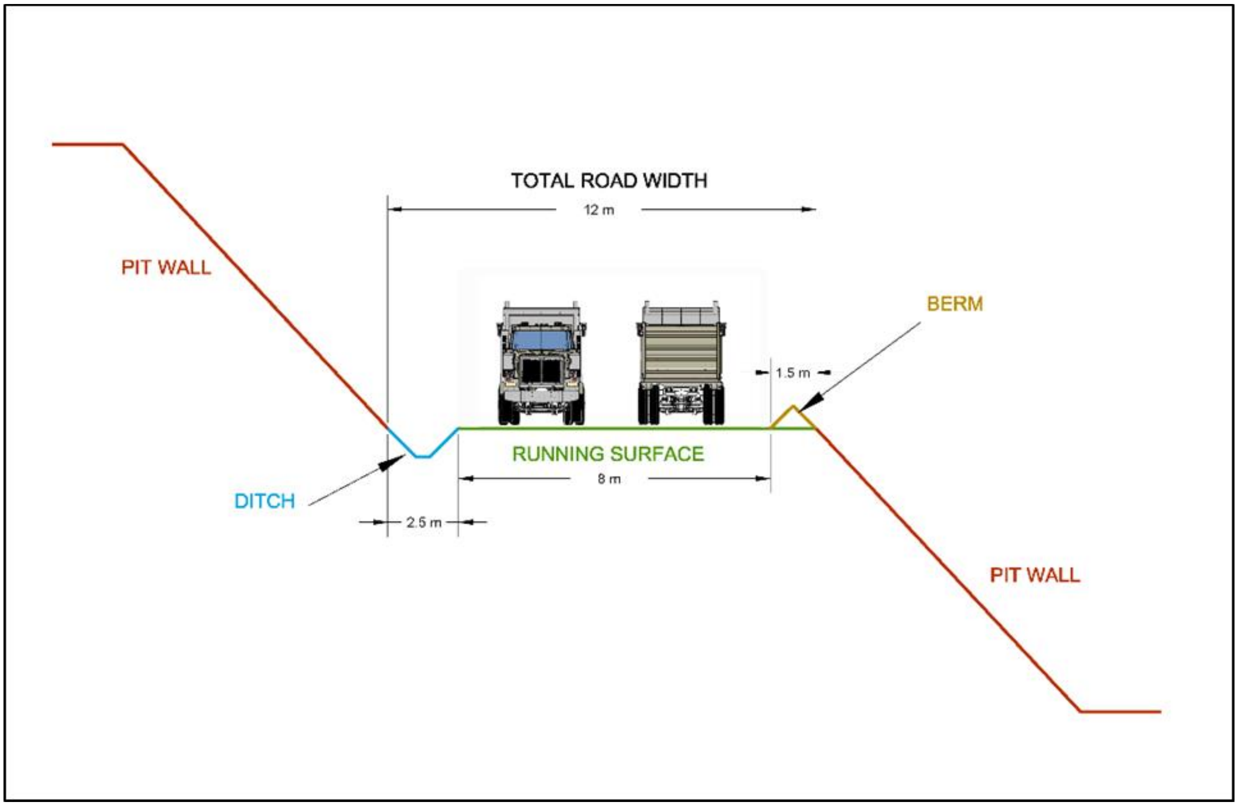


Figure 16-7:– Ramp Design

16.2.4 Mine Dilution and Mining Loss

In every mining operation, it is impossible to perfectly separate the mineralised material and waste because of the large scale of the mining equipment and the use of drilling and blasting equipment. In order to account for this, DRA/Met-Chem assumed a loss of 5% for Lily and Sophie and 10 % for Eleonore. The loss in tonnage associated with a 5%/10% in mineralised material loss was removed from the yearly production and was added to the waste material category.

16.2.5 Minimum Mining Width

A minimum mining width of 12 m was considered for the open pit design. This is based on a 9 m turning radius for a 55-tonne haul truck plus several metres on each side for safety.

16.2.6 Open Pit Design Results

Seventeen (17) pits were designed for the Project resulting in just over 7 years of production between Phase 1 and Phase 2.

Table 16-5 represents all mineralized content by pit for the Project.

Table 16-5: Mineralized Content per Pit (*)[3]

Zone		Waste (diluted)	Tonnes (diluted)	Grade (g/t)	Gold contained (oz)
Eleonore	E1	3,436,956	92,536	3.16	9,389
	E2	12,103,885	572,527	5.33	98,132
	E3	2,053,961	78,504	4.86	12,278
	E4	7,871,838	715,359	4.04	92,967
	E5	19,316,801	805,820	2.97	76,839
	E6	4,278,097	213,942	2.21	15,220
	E7	22,020,215	815,311	3.04	79,679
	E8	1,366,511	52,393	5.13	8,641
	E9	2,122,812	154,691	1.53	7,625
	E10	629,596	64,760	1.59	3,300
Lily	LILY	14,553,500	3,204,867	0.92	94,831
Sophie	S1A	2,948,156	395,152	2.06	26,118
	S1B	2,065,921	233,893	1.87	14,062
	S2	16,116,200	1,578,786	1.29	65,281
	S3A	223,399	41,928	0.77	1,039
	S3B	127,613	7,306	3.33	781
	S3C	679,599	111,971	1.13	4,078
TOTAL		**111,915,060**	**9,139,745**	**2.08**	**610,260**

The open pit design includes 9.14 M tonnes of measured, indicated and inferred mineral resources. In order to mine these resources, 111.92 M tonnes of waste material will need to be mined and stockpiled in waste dumps on the property. Table 16-6 summarises the resources by mining zone.

Table 16-6: Summary of In-Pit Resources by Zone

Zone	Tonnes (diluted)	Grade Au (g/t)	Gold Contained (oz)
Eleonore	3,565,843	3.52	404,070
Lily	3,204,867	0.92	94,831
Sophie1&2	2,369,035	1.46	111,359
TOTAL	**9,139,745**	**2.08**	**610,260**

3 (*) Cautionary statement NI 43-101: The PEA was prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"). Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

16.3 Waste Rock Dump Design

The current waste rock disposal locations selected for the Report is depicted in Figure 16-5. The dump footprint selected targets zones of low potential for mineralization and also in close proximity to mining zones to reduce haulage distance for waste. Waste rock dumps were designed with an overall slope of 20°. A swell factor of 35% was considered in the design of the waste rock dumps this accounts for the increase in volume associated with blasting in-situ material and then compacting this material on the dump. Table 16-7 summarises volume capacities and footprint area for each of the waste dumps designed for the Project. Total maximum height considered for a dump is 60 m.

Table 16-7: Summary of Waste Dump Capacities

Waste Stockpile	Volume (m^3)	Footprint (m^2)
EN	8,736,750	378,108
ES	29,597,810	770,750
LILY	8,153,000	393,151
S1	1,944,338	273,032
S2	9,854,476	391,056

16.4 Mine Planning

This mine plan forms the basis of the mine capital and operating cost estimate presented in Section 21. The mine plan was established annually for the first four (4) years of production at a rate of 2,976 t/d, followed by a four (4) year period at 4,500 t/d, for a total life of mine of just over 7 years.

16.4.1 Mine Planning Parameters

Work Schedule

The owner will operate seven (7) days per week, twenty-four (24) hours per day, twelve (12) months a year with two shifts per day.

Annual Production Requirements

The annual production target will be 1,086,240 tonnes for Phase 1 and 1,642,500 tonnes for Phase 2. (with 138,805 tonnes in Year 8).

Mine Production Schedule

Table 16-8 presents the mine production schedule that was developed for the approximately 8-Year life of the Project. This schedule includes a pre-production phase of six (6) months which is required for waste stripping and also allows for road construction and pit development. During this period, 2,902,573 tonnes of waste will be mined.

The total material mined per year during the 8-years period is not constant as certain periods of high stripping will need to be considered in order to satisfy the targeted mining rate (Phases 1 and 2). Table 16-8 presents the mine plan for the Project quarterly for the first year and annually for the remainder of the life of mine. Figure 16-8 graphically displays the tonnes of gold mined per period from the three mining zones, Eleonore, Sophie and Lily.

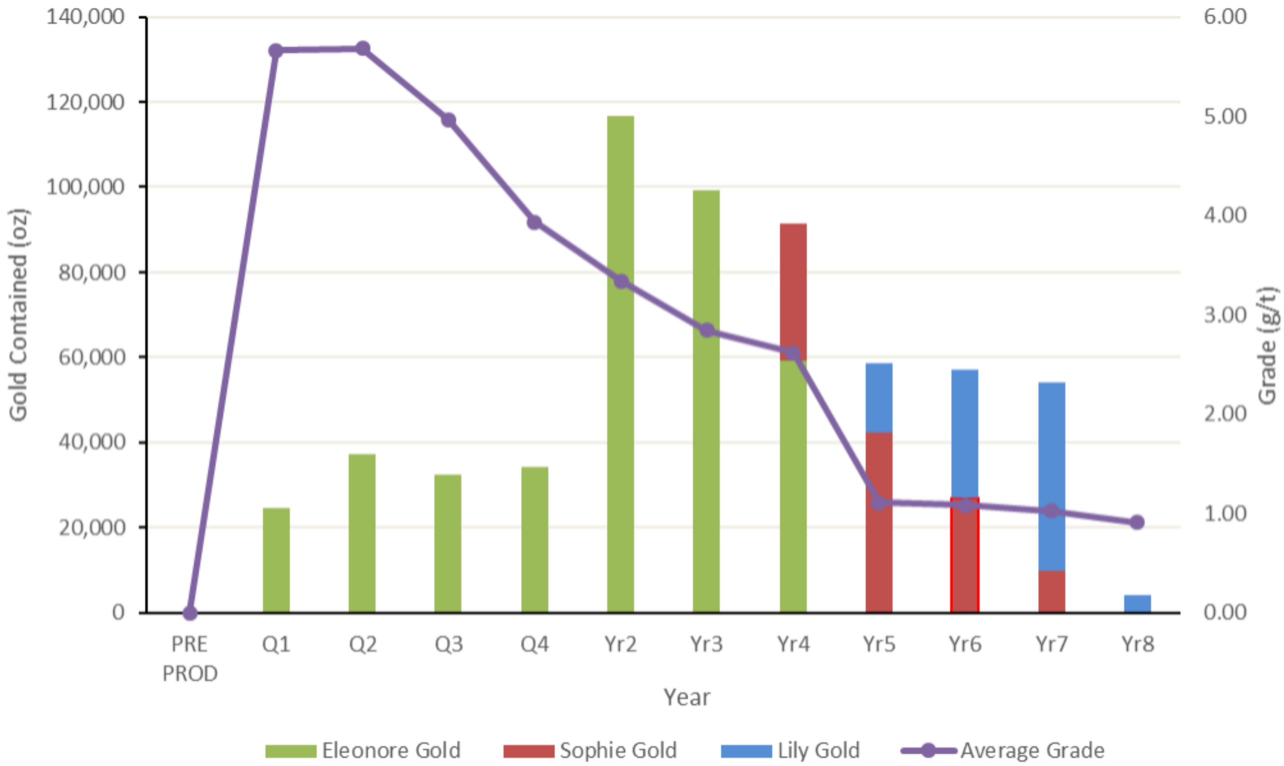


Figure 16-8: Gold Contained in In-Pit Resource per Mining Zone

Table 16-8: Mine Production Schedule

		Phase 1								Phase 2				
Description	**Units**	**Pre-Prod**	**Year 1**				**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Total**
			Q1	**Q2**	**Q3**	**Q4**								
ROM to Plant (diluted)	**t**	**0**	**135,782**	**203,679**	**203,678**	**271,523**	**1,086,228**	**1,086,196**	**1,086,255**	**1,642,610**	**1,642,528**	**1,642,461**	**138,805**	**9,139,745**
Au (Diluted)	g/t	0.00	5.66	5.68	4.97	3.93	3.34	2.85	2.62	1.11	1.08	1.03	0.91	2.08
Elenore ROM (diluted)	**t**	**0**	**135,782**	**203,679**	**203,678**	**271,523**	**1,086,228**	**1,086,196**	**578,757**	**0**	**0**	**0**	**0**	**3,565,843**
Au (Diluted 10%)	g/t	0.00	5.66	5.68	4.97	3.93	3.34	2.85	3.18	0.00	0.00	0.00	0.00	3.52
Sophie + Lily ROM (diluted)	**g/t**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**507,498**	**1,642,610**	**1,642,528**	**1,642,461**	**138,805**	**5,573,902**
Au (Diluted 5%)	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.97	1.11	1.08	1.03	0.91	1.15
Total Waste	**kt**	**2,903**	**2,128**	**3,475**	**3,787**	**4,662**	**27,328**	**26,448**	**9,765**	**18,662**	**8,643**	**4,068**	**48**	**111,915**
Total Material Moved	**kt**	**2,903**	**2,264**	**3,678**	**3,990**	**4,933**	**28,414**	**27,534**	**10,851**	**20,305**	**10,286**	**5,710**	**186**	**121,055**
Stripping Ratio	**w/o**	**n/a**	**15.7**	**17.1**	**18.6**	**17.2**	**25.2**	**24.3**	**9.0**	**11.4**	**5.3**	**2.5**	**0.3**	**12.2**
Period	**days**	**182.5**	**91.3**	**91.3**	**91.3**	**91.3**	**365.0**	**365.0**	**365.0**	**365.0**	**365.0**	**365.0**	**35.0**	

16.5 Mine Equipment Fleet and Manpower

The following Section discusses equipment selection and fleet requirements in order to carry out the mine plan. Table 16.6 identifies the Caterpillar equivalent of equipment required in Year 2 of operation to provide the reader an appreciation for the size of each machine although the specific equipment selection will be done at a later stage of the Project.

Table 16-9: Mining Equipment Fleet

Equipment	Typical Model	Description	Units
Major Equipment			
Haul Truck	Dramis	Payload – 55 t	22
Hydraulic Excavator	CAT 6015	Bucket – 8 m^3	3
Production Drill	Atlas D65	153 mm hole	6
Support Equipment			
Wheel Loader	988K	373 - 393 kW	1
Track Dozer	D7R	235 - 265 kW	1
Track Dozer	D8R	245 kW	2
Road Grader	CAT14M3	160 - 170 kW	2
Excavator	CAT349	425 HP	1
Water Truck	Peterbilt 365	20,000-litre	1
Light Plant	MLT3080	6 kW	6
Fuel and Lube Truck	Peterbilt365	330 kW	2
Mechanic Truck	Peterbilt 348	250 kW	2
Boom Truck	CT660	354 kW	1
Tire Handler	Kalmar	DCF330-12	1
Pickup Truck	Ford F250	300 kW	8

16.5.1 Haul Trucks

The haul truck selected for the Project is a rigid frame quarry truck with a payload of 55 tonnes. This size truck was selected since it matches well with the production requirements and results in a manageable fleet size. Due to the need to mine at different mines simultaneously, DRA/Met-Chem estimated a smaller size trucks truck fleet. The following parameters were used to calculate the number of trucks required to carry out the mine plan. These parameters result in 5,277 working hours per year for each truck as is presented in Table 16-9.

- Mechanical Availability – 85%;
- Utilization – 90% (non-utilized time is accrued when the truck is not operating due to poor weather, blasting, excavator relocation and no operator available);
- Nominal Payload – 55 tonnes;

- Shift Schedule – Two (2) twelve (12) hour shifts per day, seven (7) days per week;
- Operational Delays – 838 hours/year;
- Job Efficiency – 90% (54 min/h; this represents lost time due to queuing at the shovel and dump as well as interference on the haul road);
- Rolling Resistance – 3%.

Table 16 9: Truck Hours (h/y)

Description	Hours	Details
Total Hours	8,760	7 days per week, 24 hours per day, 52 weeks per year
Down Mechanically	1,314	15% of total hours
Available	7,446	Total hours minus hours down mechanically
Standby	745	10% of available hours (represents 90% utilization)
Operating	6,701	Available hours minus standby hours
Operating Delays	838	90 min/shift
Net Operating Hours	5,864	Operating hours minus operating delays
Working Hours	**5,277**	**90% of net operating hours (reflects job efficiency)**

Haul routes were generated for each period of the mine plan (per pit, per material mined) to calculate the truck requirements. These haul routes were imported in Talpac©, a commercially available truck simulation software package that DRA/Met-Chem has validated with mining operations. Talpac© calculated the travel time required for a 55-tonne haul truck to complete each route. Table 16-10 shows the various components of a truck's cycle time. The load time is calculated using a hydraulic excavator with an 8.1 m^3 (14.6-tonne) bucket as the loading unit. This size excavator which is discussed in the following section loads fresh ROM and Waste into a 55-tonne haul truck in four (4) passes, for weathered material, five (5) passes are required.

Table 16-10: Truck Cycle Time

Activity	Duration (sec)
Spot @ Excavator	30
Load Time[1]	80
Travel Time	Calculated by Talpac©
Spot @ Dump	30
Dump Time	30

[1] Four (4) Passes @ 20 sec/pass

Haul productivities (tonnes per work hour) were calculated for each haul route using the truck payload and cycle time. Truck hour requirements were calculated by applying the tonnages hauled to the productivity for each haul route. A fleet of four (4) trucks is required during pre-production and should be increased to (22) trucks in Years 2 to 3.

16.5.2 Excavator and Loader

The main loading machine selected for the Project is a hydraulic excavator (backhoe) with an 8.1 m^3 bucket. To maximize loading productivity, the excavator will be setup in a backhoe configuration, with the truck sitting at the bottom of the muck pile. Based on an 85% mechanical availability, a 90% utilization and 90 minutes per shift in operating delays, it was estimated that three (3) excavators can manage the tonnages of mineralised material and waste rock in the mine plan, an 8.4 m^3 loader was added to the fleet to allow for versatile as there will be multiple active pits working simultaneously to meet production targets.

16.5.3 Drilling and Blasting

Production drilling will be carried out with a diesel-powered track mounted down the hole (DTH) drill. Using the following parameters; 85% mechanical availability, 85% utilization and a penetration rate of 25 m/h, DRA/Met-Chem calculated that one (1) drill is sufficient in pre-production, this increases up to six (6) drills in Years 2 to 3 as mining in fresh rock will increase to meet production requirements as presented in the mine plan (Table 16-8). Table 16-11 presents the drilling and blasting parameters for production holes that have been designed for the Report.

Table 16-11: Drilling and Blasting Parameters

Parameter	Units	Production
Bench Height	m	5.0
Hole Diameter	m	0.153
Burden	m	4.0
Spacing	m	4.0
Subdrilling	m	1.0
Stemming	m	2.5
Explosives Density	g/cm^3	1.25
Powder Factor	kg/t	0.37

16.5.4 Mine Manpower Requirements

The mine workforce requirements range from 117 in pre-production and increases to 251 by Year 2 when the additional trucks are required for stripping waste. Manpower was estimated considering four (4) crews, each operator working an average of 2,190 hours per year.

17. RECOVERY METHODS

The process recovery methods included in the flowsheet are presented in this Section. The basis of the selected design is presented in Section 17.2 - Process Design Criteria. Process flow diagrams has been developed for Phase 1 and Phase 2 as well as a mechanical equipment list. A general plant layout presenting both phases of the Project is presented in Appendix IV. Two trade-off studies were produced during the process plant development. The first compared three different grinding circuit scenarios and the second compared two different grind sizes (75 microns vs 105 microns) for Phase 1 in order to select the optimal conditions for the process flowsheet.

17.1 Flowsheet Selections

The process plant includes the following unit process steps and associated steps:

- Primary crushing;
- Crushed mineralised material storage in the Coarse mineralised material Stockpile;
- Coarse mineralised material reclaiming;
- Single Stage SAG (SSSAG) milling circuit (Phase 1), with conversion to a SAG/Ball Mill (SAB) circuit (Phase 2);
- Gravity concentration;
- Leaching and CIL adsorption;
- Elution desorption;
- Intensive leaching of the gravity circuit concentrate;
- Electrowinning and gold room;
- Carbon regeneration;
- Tailings thickening and filtration;
- Tailings Storage;
- Reagent storage and distribution;
- Water services (process water, treated water, fire water, gland water)
- Potable water treatment and distribution;
- Air services.

17.2 Process Design Criteria

The process plant was developed for two distinct phases of production. In the first phase of production, the plant was designed for an average daily capacity of 2,976 tonnes of run of mine (ROM) processed from the higher-grade deposits, mostly located in the Eleonore mining zone. After approximately four years of operation, the plant will undergo an expansion in order to process 4,500 average tonnes per day of ROM processed from the lower-grade deposits, located in the Lily and Sophie mining zones. For both phases, the process plant consists of traditional gravity and Carbon-In-Leach (CIL) circuits, followed by electrowinning to produce doré bars.

The Phase 1 average plant design feed grade is of 3.33 g/t Au, while in Phase 2, the average plant feed grade decreases to 1.07 g/t Au. Yearly gold production will vary between 50,000 oz. and 123,000 oz. of doré bars depending on the plant feed rate and grade. The Main Process Design Criteria are shown in Table 17-1.

Table 17-1: Main Process Criteria

Criteria	Units	Phase 1	Phase 2
Plant Capacity – Daily Average	t/d	2,976	4,500
Crushing Circuit – Availability	%	70	70
Process Plant – Availability	%	92	92
Crushing Circuit – Nominal Hourly Feed	t/h (dry)	177	268
Process Plant – Nominal Hourly Feed	t/h (dry)	135	204
Average Feed Grade	g/t	3.33	1.07
Recovery – Gravity Circuit	% of fresh feed	44.8	38.6
Recovery – CIL and Elution Circuit	% of fresh feed	51.2	55.0
Recovery - Overall	% of fresh feed	96.0	93.6
Average Annual Gold Production	oz./y	111,661	52,888
ROM specific gravity	-	2.85	2.88
Bond Ball Mill Work Index for Mill Sizing	kWh/t	18.1	18.1
Target Grind Size (P_{80})	microns	105	75

17.3 Mechanical Process – Crushing, Grinding and Gravity Concentration

17.3.1 Crushing Circuit

ROM mineralised rock is delivered by 55 tonne haul-type trucks to the crusher feed stockpile area. The rock is delivered directly into the crushing circuit feed hopper or stored in one of the two stockpiles (divided into high grade and low grade material). ROM rock stored in the stockpiles can be blended if necessary to stabilize the plant feed grade when multiple pits are being mined simultaneously. Each stockpile has a live capacity of 6,000 tonnes.

The high and low grade stockpiles are located within the same area as the crushing circuit equipment. The stockpiles are reclaimed by front-end-loader and fed into the crushing circuit feed hopper when the haul trucks are not directly dumping into this hopper.

ROM dumped into the crushing circuit feed hopper is fed to a jaw crusher. A variable speed apron feeder transfers the crushed mineralised rock from the crusher onto the crushed rock conveyor. The speed of the apron feeder is used to control the crushing circuit throughput.

The crushing circuit equipment are sized to handle both Phase 1 and Phase 2 throughputs. During Phase 1 operation, the open side setting of the jaw crusher is set to 100 mm. This will produce a crushed mineralised material with an 80% passing size about 98 mm. For Phase 2, the jaw crusher open side setting is increased to 150 mm, producing crushed material with an 80% passing size of about 143 mm. The Phase 2 grinding circuit can handle larger feed material since it uses a two-stage grinding arrangement, as described in the next sections. The Phase 1 grinding circuit requires finer feed material as it consists of a single grinding stage. A 132 kW jaw crusher has been selected for this flowsheet.

The crushing circuit is equipped with a bag house-type dust collector to control dust emissions.

Crushed material at the crusher discharge is conveyed to the Coarse Mineralised Rock Stockpile via the crushed rock conveyor. The stockpile has a live capacity of 4,500 tonnes, which will provide up to a day of feed material to the grinding circuit during Phase 2, and as much as a day and half during Phase 1. The stockpile ensures the processing plant operates independently of the mining and crushing activities, providing constant feed to the grinding circuit.

17.3.2 Grinding and Gravity Circuits

During the first phases of process plant design, a trade-off study was performed comparing different options for the grinding circuit.

Crushed Mineralised Rock stored in the Crushed Rock Stockpile is reclaimed by two variable speed apron feeders (one operating and one standby). The speed of the apron feeders will be used to control the processing plant throughput. From the apron feeders, the crushed mineralised rock is fed to the Semi-Autogenous (SAG) grinding mill by a belt conveyor. The coarse mineralised rock is ground as slurry in the SAG mill with the addition of process water. The SAG mill will operate in closed circuit with a cluster of hydrocyclones. A re-circulating load of 400 % was selected for equipment sizing. The targeted cyclone overflow size has an 80% passing size of 105 microns. The selected SAG mill has an installed power of 3300 kW and was selected to operate in both the Phase 1 and Phase 2 circuits.

SAG mill discharge passes over a trommel screen to remove oversize material (pebbles), which are recycled internally in the SAG mill with a water jet system. Phase 1 operation does not include the use of a pebble crusher. Trommel screen undersize flows by gravity to a pump box from where it is pumped by one of two pumps (one operating and one standby) to the hydrocyclone cluster.

The hydrocylcone overflow is the final product of the grinding circuit and gravitates to the CIL circuit. The targeted hyrdocyclone overflow density is 40 %w/w as per the selected design criteria. The hydrocyclone underflow will be divided into two streams. The first stream, equivalent to the solids fresh feed to the grinding circuit, gravitates to the gravity separation circuit. The balance is returned to the SAG mill feed to for further grinding and liberation.

The gravity concentration circuit consists of a trash and oversize screen and a gravity concentrator. The trash and oversize screen will prevent any oversized material from entering the gravity concentrator and blocking the fluidization openings. This oversized material is returned the hydrocyclone feed pump box. The trash and oversize screen undersize material will flow into the gravity concentrator. This equipment uses centrifugal forces to separate the liberated coarse gold material from the unliberated mineralised material. It operates on a semi-continuous basis.

During the “concentration cycle”, feed slurry enters the gravity concentrator which spins at high velocity, concentrating the free gold. The unliberated material discharges from the bottom of the concentration bowl continuously during the concentration cycle. This tailings material is returned to the hydrocyclone feed pump box for further liberation. The free gold will accumulate on the walls of the concentrator during the concentration cycle. Fluidization water is added through holes in the spinning bowl to remove fine low specific gravity mineralised material. On a periodic basis, feed slurry to the gravity concentrator is stopped. This slurry will bypass the concentrator and return to the hydrocyclone feed pump box. Flush water is admitted into the concentrator to push the free gold to the concentrate discharge. The concentrated free gold then gravitates to the Intensive Leach Reactor circuit and slurry feed to the gravity concentrator is resumed.

As discussed previously, Phase 2 of operations will see an increase in the average grinding circuit throughput, from 2,976 t/d to 4,500 t/d, and a reduction in the target of the 80% passing size of the grinding circuit from 105 microns to 75 microns. In order to achieve this, several changes will be made to the grinding and gravity circuits.

To accommodate the higher plant throughput, an additional reclaim feeder will be required. The new reclaim arrangement will consist of two operating apron feeders, each providing 50 % of the plant feed, and a third apron feeder on standby. The SAG mill feed conveyor is already sized to handle the higher throughput.

Several modifications will be made to the grinding circuit to change it from a single stage grinding circuit to a two-stage grinding circuit. The following equipment will be added to the circuit:

- A pebble cone crusher;
- A ball mil;
- A larger capacity hydrocyclone cluster; and
- An additional gravity concentrator.

The SAG mill will be operated in an open circuit-configuration. Crushed mineralised rock will be fed to the SAG mill along with water. Ground rock from the SAG mill trommel undersize will gravitate to the hydrocyclone feed pump box.

A pebble crusher will be added to the SAG mill circuit in Phase 2 to crush pebbles generated from the increased throughput and reduced grinding action in the SAG mill. SAG mill trommel oversize is conveyed to the cone crusher and returned to the SAG mill, also via a belt conveyor. The selected cone crusher will have an installed power of 185 kW.

The hydrocyclone feed pump box is already sized to handle the Phase 2 throughput. However, the hydrocyclone feed pumps (one operating and one standby), and the hydrocyclone cluster itself, will be replaced to handle the new throughput and decreased grind size.

The new hydrocyclone will operate in closed circuit with a ball mill. The selected ball mill will have an installed power of 3,300 kW. This size has been selected in part to meet the throughput and grind size requirements, but also to maximize the interchangeability of parts between the SAG and ball mills. The ball mill trommel undersize is directed to the hydrocyclone feed pump box, while the overflow is simply discarded in bins.

Finally, the gravity concentration circuit will operate in the same fashion as during Phase 1. However, a second trash and oversize screen and gravity concentrator will be added to the circuit to accommodate the higher throughput of material.

17.4 Gold Recovery – Intensive Leach, CIL, Elution and Electrowinning

17.4.1 Carbon-In-Leach

The ground mineralised rock recovered at the overflow of the grinding circuit hydrocyclone cluster gravitates to the Carbon-In-Leach (CIL) circuit via the trash screen. The trash screen will remove any oversized material from the slurry before leaching. This will ensure that minimize blockage with the intertank carbon screens.

The leach and CIL circuit selected consists of seven tanks: one pre-aeration tank, one leach tank and five CIL tanks. The total selected circuit residence time for the Phase 1 is 48 hours, while a reduced residence time of 30 hours is required for Phase 2. The residence times include two hours of pre-aeration ahead of leach and CIL. Despite the increased slurry throughput in Phase 2, because the required residence time is greater in Phase 1, more total volume is required in this first Phase. As a result, no modifications to the circuit will be required to implement Phase 2. Using these criteria to size the equipment, a 910 m^3 pre-aeration tank, a 2,000 m^3 leach tank and five 2,000 m^3 CIL tanks were selected.

Air is injected to the bottom of the tanks to ensure the target dissolved oxygen level is maintained. Hydrated lime slurry is added to the pre-aeration and leach tanks to ensure the slurry pH remains above 10 in the circuit. This is a critical operation as dangerous hydrogen cyanide gases will form should the pH ever drop below this level. Sodium cyanide is added to the leach and first CIL tank. The sodium cyanide will dissolve the gold in the feed slurry. The CIL tanks contain activated carbon. The carbon is advanced from one tank to the next counter-currently to the slurry flow on an intermittent basis using carbon advance pumps. Dissolved gold is adsorbed onto the surface of the carbon in the CIL tanks. The carbon in the fifth CIL tank will be mostly barren. Fresh or barren carbon is returned to this tank. Gold-loaded carbon is pumped from the first CIL tank to the elution circuit when an elution batch is initiated.

Barren slurry gravitates from the fifth CIL tank to the tailings thickening circuit via a safety screen. This safety screen ensures that any fine carbon that will contain gold is not sent to the tailings storage facility. The screen oversize is bagged and will be reprocessed. The screen undersize gravitates to the tailings thickener.

17.4.2 Elution and Carbon Regeneration

Gold laden carbon is pumped to the elution circuit for gold recovery. The selected elution circuit is of the AARL type. As with the Leach and CIL circuit, the elution and carbon regeneration circuits are sized to handle both Phase 1 and Phase 2 operations. A 3-tonne acid wash column and a 3-tonne elution carbon column have been selected.

When an elution batch is initiated, loaded carbon is pumped from the first CIL tank into the acid wash column via the loaded carbon screen. This screen will separate loaded carbon from any CIL slurry entrained with the carbon, as well as de-water the carbon. The screen undersize is returned to the first CIL tank. The screen oversize falls into the acid wash column. Hydrochloric acid is used to dissolve carbonate scale that has formed on the surface of the carbon. This process will ensure that the maximum of carbon surface area is available for gold deposition, reducing the consumption of fresh carbon.

The washed carbon is then transferred to the elution column where the elution process takes place. Pre-soak solution prepared using sodium hydroxide and sodium cyanide is heated and circulated through the carbon bed in the column for a pre-set time period. At the end of the pre-soak cycle, heated water is circulated through the carbon to desorb the gold from the carbon, creating a pregnant solution.

A diesel-fired indirect heating system will be used to heat the barren solution. The elution circuit requires adequate water quality for the AARL process. Total dissolved solids content of the water should not exceed 500 mg/L, and ideally is below 350 mg/L. A water treatment system is required to ensure this water quality is achieved. The use of anti-scalents is also necessary to maintain correct water quality.

The pregnant solution generated by the elution process is stored in the pregnant solution tank for use in the electrowinning circuit. When an elution cycle is complete, the circuit is ready to initiate a new acid wash and elution cycle. It is expected that an elution cycle will be performed approximately once a day during Phase 1 and twice a day during Phase 2. Electrowinning barren solution is pumped to the CIL circuit.

At the end of an elution cycle, the barren carbon remaining in the elution carbon is transferred to the carbon regeneration circuit. This circuit consists of rotary kiln that will heat the carbon to about 700 ˚C, re-activating the surfaces of the carbon. The regenerated carbon is then cooled with water, and fresh carbon is added to the carbon slurry, as required. Finally, the regenerated carbon is returned to the CIL circuit in the fifth CIL tank to complete a new adsorption/desorption cycle.

17.4.3 Intensive Leaching

A separate leaching circuit is used to treat the free gold concentrate produced by the gravity concentrators. Similarly to the elution circuit, the Intensive Leach Reactor (ILR) operates on a batch basis. A single unit will be able to accommodate both Phase 1 and Phase 2 operations simply by modifying the ILR batch frequency.

In the ILR circuit, the free gold concentrate is leached into solution using sodium hydroxide, sodium cyanide, hydrogen peroxide (as needed) and a leaching aid. Each batch will generate two products: the ILR tailings, and the ILR pregnant solution. The ILR tailings slurry is returned to the grinding circuit for further liberation and recovery in the CIL circuit. The ILR pregnant solution is stored in the ILR pregnant solution tank for use in the electrowinning circuit.

The selected ILR unit has a 6-tonne batch capacity. During Phase 1, approximately 4 ILR batches per week will be required. During Phase 2, approximately 6 ILR batches per week will be required.

17.4.4 Electrowinning and Gold Room

The electrowinning circuit consists of two independent cells, one dedicated to the elution pregnant solution and one dedicated to the ILR pregnant solution. An electric current is applied across the cells, causing gold to deposit on the surface of the cathodes. After an electrowinning cycle, the deposited gold is washed off the cathodes and dewatered in a manually operated filter press. The dewatered gold is dried in an oven and then mixed with the flux. Finally, the mixture is fed to the diesel-fired furnace where the gold is melted and poured in bars. No changes are required to the electrowinning and gold room circuits to implement Phase 2, only the frequency of the electrowinning and gold pours will change.

After the electrowinning process, barren solution from the elution circuit is returned to the elution circuit for processing during the next elution cycle. A portion of this solution will be purged to the CIL circuit to prevent the buildup of contaminants. The ILR circuit barren solution is pumped to the CIL circuit.

17.4.5 Tailings Dewatering

The Project is using a dried stacked tailings storage approach. To do this, the final tailings need to be thickened, then filtered.

Slurry from the fifth CIL tank is fed to the tailings thickener via the safety screen. An 18 m diameter high-rate thickener has been selected to thicken the tailings slurry from 40 %w/w to more than 60 %w/w. Flocculant will be used to accelerate the thickening process. The thickener overflow is recovered as process water and re-used in the process. Recycling the remaining cyanide in the recovered water will help reduce overall cyanide consumption.

The thickener underflow will be pumped to a vacuum disc filter for further filtration. A disc filter with 175 m^2 of filtration area has been selected. The filter cake will undergo a washing step to displace and recover any cyanide-containing water using treated water. The recovered filtrate and wash water is combined with the thickener overflow and recycled in the process. The filter cake is stored in a stockpile at the plant site from where it is loaded into trucks by front-end loaders and transported to the tailings storage facility. Filter cake moisture of less than 15 %w/w is targeted.

In order to transition to Phase 2 of the project, some modifications will be made to the tailings dewatering circuit. The tailings thickener has been sized to accommodate the increased throughput of Phase 2 without modification. However, the thickener underflow pumps will have to be replaced for higher capacity pumps.

In addition, a second vacuum disc filter, with a filtering area of 140 m^2, is added to the circuit.

Finally, the process water pumps will also have to be replaced to handle the larger consumption during Phase 2.

17.4.6 Consumables and Reagents

The consumables and reagents required for the mechanical and chemical treatment of the ROM can be summarized as follows:

- Hydrated Lime ($Ca(OH)_2$) – used to control the pH in the leach and CIL circuit;
- Sodium Cyanide (NaCN) – used as the main gold leaching reagent in the CIL circuit and in the ILR circuit. It is also used to prepare the barren liquor in the gold desorption (elution) circuit;
- Sodium Hydroxide (NaOH) – used to control pH in the elution, IRL and cyanide preparation circuits;
- Hydrochloric Acid (HCl) – used in the acid wash circuit to remove scale formation on the carbon;
- Activated carbon – used in the CIL circuit to adsorb dissolved gold;
- Flocculant – used as a thickening aid in the Tailings Thickener;
- Leach aid – used in the ILR circuit to improve the free gold leaching process;
- Antiscalant – to reduce the formation of scale in the elution and electrowinning circuits equipment, and on the activated carbon itself;

- Flux – used as a cleaning agent during gold smelting;
- SAG mill media – grinding media required in the SAG mill;
- Ball mill media - grinding media required in the ball mill;
- Crusher and grinding mills liners; and
- Filter cloths.

18. PROJECT INFRASTRUCTURE

The details for the infrastructure is presented for the period of construction and for operation. In some cases, the construction infrastructure will be adapted or reused to meet the needs of operations. Operational infrastructure must be functional for commissioning tests. A general site layout could be found in Annex V

18.1 Infrastructure (Construction)

18.1.1 Access Roads (Construction)

Temporary access roads connecting the industrial zone, the three main operating areas, the base camp, and the tailings stockpile will be constructed at the beginning of construction. The quantities are based on the distances of these routes and depend on the final design.

It should be noted that the widths of the access roads will vary according to their use. In fact, the roads used by heavy-haul mining trucks will be 12 metres wide and designed with heavy load bearing capacity. Access roads between mining pits and the processing plant site and haul roads are treated as mining infrastructure.

Roads used by vehicles hauling lower tonnage will be 6 metres wide with 1.5 metre shoulders.

It should be noted that during construction, certain portions of the existing 130 km road between the site and National Highway 2 (RN-2) will need to be upgraded.

18.1.2 Water Supply (Construction)

The water supply for the exploration camp comes from a commercial drilling site located 80 km southwest of the existing base camp and is transported to the site by tanker truck. Until commissioning of the Project's brackish water boreholes for process purpose, it will be considered that tanker water transportation will still be in operation.

18.1.3 Site Excavation (Construction)

Earthworks for the site will be executed as soon as the first construction companies are mobilized.

Since the plant site will be located on flat land, earthworks will result in low volumes. For this study, the bearing capacity of soils is considered to be in the order of 250 to 350 KPa. This means that superficial foundations will be adequate.

The superficial foundations form a type of base that can be put in place on soils with good soil bearing capacity, to be able to support the building loads with minimal settlement. Their simplicity of construction and their low cost make this type of foundation the most common structure.

18.1.4 Laydown Areas (Construction)

The following storage areas will be prepared during the first earthworks:

- Storage areas for receiving containers and equipment and materials;
    - In order to receive materials and equipment during the construction period, a handling area is planned that will cover a total area of 15,000 m². This area will be fenced.
- Central Warehouse Storage Area;
- Storage area for central maintenance workshop and parking for heavy and light vehicles on the site;
- Steel storage area, rebar bending, and on-site prefabrication.

Figure 18-1: Typical Container Storage Area for Equipment and Materials

Figure 18-2: Typical Area for Storing Steel and for Rebar Bending

18.1.5 Concrete Production (Construction)

A company will be responsible for concrete production as well as aggregate supplies, water additives and cement.

It is assumed that aggregates are available within a radius of 10 km. Suitability analyses should be performed to confirm this assumption.

Figure 18-3: Typical Concrete Production Plant

It is assumed that the perimeter fence during construction period will be maintained for operations.

18.1.6 Base Camp (Construction)

A base camp will be built to accommodate workers during construction. The infrastructure during the construction period includes:

- Exploration Camp with a capacity of 100 people and associated infrastructure
- Temporary water supply
- Power supply
- Warehouses
- Maintenance garage for vehicles and equipment including drilling equipment.

The base camp during construction will be built in the same place as the permanent base camp. Additional infrastructure used during construction that can not be reused for the operation phase will be dismantled.

The construction camp will be sized to house:

- Client team;
- EPCM team;
- Construction and service company personnel;
- Visitors and other representatives including suppliers;
- Kitchen, cleaning, and maintenance staff for site services;
- Security personnel;
- Health care staff.

A preliminary analysis of the labour force curve during construction shows that the number of workers required will be between 250 and 300 with peaks of up to 400-500 people over a period of 18 months.

18.2 Permanent Infrastructure

Permanent infrastructure is either on-site or off-site.

18.2.1 Permanent infrastructure On-Site

18.2.1.1 Access Roads On-Site (Operation)

The following temporary access roads will be converted into permanent roads on the operation’s site:

- Road between waste dumps and tailings stockpile;
- Road between the base camp and the processing plant;
- Road between the base camp and the explosives production area;

18.2.1.2 Site Buildings (Operation)

In developing the site infrastructure, some assumptions have been done and are presented in the following section.

18.2.1.3 Process Buildings

Typically, buildings linked to the process are:

Warehouse for reagents that is divided according to each product:

- Flocculant warehouse (flocculant)
  - A warehouse will be dedicated to the storage of flocculants. Flocculants are used to promote the thickening of residues. The type of flocculants required and the quantities required will be determined once the mineralogical tests are finalized.
- Storage of hydrochloric acid (HCl)
  - Hydrochloric acid will be delivered to the site by drums and stored in a tank on-site.
  - HCl is used for carbon cleaning in the elution circuit;
  - Drums will be kept in dry storage to protect the product.
- Storage of caustic soda (NaOH)
  - The caustic soda will be delivered to the site in barrels or bags and ultimately stored in a tank. The tank will be elevated on a slab of reinforced concrete and surrounded by a dike for spill containment;
  - Caustic soda (NaOH) is used to control the pH in the elution circuit and in the leaching of the gravity concentrate;
  - Bags and barrels will be kept in dry storage to protect the product.
- Storage of sodium cyanide (NaCN)
  - Sodium cyanide will be delivered to the site in barrels, bags or isotainers and ultimately stored in a tank. The tank will be elevated on a slab of reinforced concrete and surrounded by a dike for spill containment;
  - Sodium cyanide (NaCN) is used for leaching gold from the mineralized material in the leach circuit (CIL and gravity circuits) and extracting the gold from the carbon during the elution process;
  - Bags, barrels, and isotainers will be kept in dry storage to protect the product.
- Storage of hydrated lime ($Ca(OH)_2$)
  - The lime will be delivered to the site in barrels or bags and stored in a small dedicated warehouse before use;
  - Lime is used for pH control in the leach circuit. It may also be introduced at other points of the circuit, if required;
  - Bags and barrels will be kept in dry storage to protect the product.
- Activated carbon storage.
  - Activated carbon will be delivered to the site in barrels or bags and stored in a small dedicated warehouse prior to use;
  - Activated carbon is used for the adsorption of gold in the leach circuit;

  - Bags and barrels will be kept in dry storage to protect the product
- Process control room
  - The process control room will be located in the nerve center of the plant.
- Laboratory
  - The laboratory building provides analytical services during the operating period.
- Gold Room
  - The smelter is a secure building that allows the recovery of gold at the end of the process to produce doré bars;
  - Entry to the building is through a secure airlock;
  - This building also houses a room that is actually a safe for storing doré bars.
- Buildings controlling power distribution and mechanical, electrical and instrumentation controls
  - These buildings are prefabricated units (MCC) which are located in several locations and near transformers that are protected by simple shelters.

18.2.1.4 Non Process Buildings

A) Administrative Building

The administrative building will be used for the mine, process plant, and site administration.

Modular buildings will be grouped to provide a sufficient number of offices for the following departments:

- Finance;
- Human Resources;
- Department of the Environment, Health, Safety and Community Relations;
- General Administration and a conference room;
- Procurement;
- Mine and a conference room;
- Geology and a room for geological maps;
- Processing Plant;
- Maintenance.

In addition to offices, a room is required for printing and photocopying. A printer for large format drawings (A1) is an advantage at a remote site.

Two rooms will be required for training as well as a library. Training is an important part of the Tijirit project strategy.

B) Main Entrance, Security, and Training (Operation)

A gatehouse building will be provided at the main entrance to the site. The main function of the gatehouse is to control entrances and exits.

The site security supervisor's office will be located in this building.

A room is also reserved for induction training for site visitors.

C) Maintenance Workshops (Operation)

One building will house all maintenance workshops:

- Welding workshop;
- Machine shop;
- Carpentry shop;
- Painting shop;
- Workshop for battery maintenance;
- Workshop for the maintenance of office equipment and the base camp;
- Workshop for the maintenance of plant electric motors and other equipment including solar power equipment;
- Workshop for the maintenance of electromagnetic controls;
- Diagnostic workshop and special tools;
- Maintenance workshop for light vehicles.

D) Maintenance Workshop for Mine Heavy Equipment (Operation)

A building will be dedicated to the maintenance of mine equipment and more particularly heavy-haul transport trucks. This first section will be equipped with cranes and a tire change area.

A second section will be reserved for the maintenance of the excavators of the mine.

A third section will be multipurpose including the maintenance of heavy vehicles such as graders, wheel loaders and compactors.

All sections will be served by compressed air distributors and oil and grease dispensers.

An area will be reserved for waste oil collection and temporary storage.

A washing area will be built including a slab and a water-oil separator.

A truck will be equipped to offer a mobile breakdown service for heavy equipment.

E) Main Warehouse (Operation)

The main warehouse will be located near the two workshops and the administrative building. Thus the parts required for the maintenance of heavy and light vehicles can be distributed quickly. Inventories will be frequently monitored to ensure efficient inventory management and supply availability to mine and process plant site users.

This warehouse will be erected quickly at the beginning of construction so that it can be used during construction.

A prefabricated steel type building is planned but adapted for the climate of Mauritania so insulated against heat. It will also be scalable, that is to say it can be expanded through the addition of modules.

It is expected that the interior of this building will be divided according to uses, including:

- Storage and protection of materials and equipment on shelves;
- Air-conditioned storage areas;
- Office for the control of entrances and exits.

F) Fuel Storage and Distribution (Operation)

An area is planned for diesel fuel supply and on-site distribution for both heavy and light vehicles.

Finally, another area is reserved for the storage and distribution of gasoline. Some equipment and some vehicles run on gasoline.

All these facilities will be characterized by:

- Elevated concrete foundations;
- Double wall tanks with a capacity of 30 days of consumption;
- Watertight retention dikes with a retention capacity equivalent to 110% of the volumes stored;
- Peripheral drainage;
- Fence and gatehouse;
- Fire protection;
- Distribution of fuel to users will be done through a prefabricated station with electromagnetic card access control.

G) Core Shack Sample Preparation and Storage

- A warehouse will be dedicated to the storage of drill cores and the preparation of samples.

Figure 18-4: Typical Core Shack and Sample Preparation Building

H) Explosives Storage and Magazine

- Ammonium nitrate explosives will be used for blasting.
- Ammonium nitrate will be stored in a secured (fenced) area at a safe distance from other facilities at the mine site.
- The detonators will be stored separately in the explosives magazine at a distance of at least 100 metres from the ammonium nitrate storage area.

I) Permanent Camp Lodging (Operation)

The capacity of the new base camp will be a permanent camp that can accommodate more than 400 workers. This camp will be a prefabricated modular type that offers the following advantages:

- Easy and quick installation;
- Suitable for arid and hot climates;
- Possibility of expansion.

The construction camp modules will be reused as much as possible for the permanent camp with minor refurbishing as required and additional modules and functionality added to be able to provide service amenities to workers and visitors for the life of the mine. Additional infrastructure used during construction that can not be reused for the operation phase will be dismantled.

The base camp will be designed to meet the needs of workers during the operating life. It will include the kitchen and dining hall, housing, laundry, gym, recreation area and prayer area. It will be located approximately 5 km from the industrial zone to minimize the inconvenience caused by the noise and dust generated by mining and processing activities.

It will also include a clinic with an infirmary and an ambulance to evacuate to Nouâdhibou or Nouakchott.

The first aid building will also be equipped with an ambulance shelter and a shelter for the fire truck.

18.2.2 Permanent Infrastructure Off-Site (Operation)

18.2.2.1 Off Site Road Access (Operation)

The following access road will be maintained throughout the life of the project:

- Access road connecting the National Road 2 (RN-2) to the site: 130 km.

Portions of the existing 130 km roadway between the site and the Nouakchott-Nouâdhibou National Highway 2 will need to be upgraded. The road will need to be leveled at certain locations to allow access for light vehicles and trucks for personnel transportation and site supplies during the construction and operation phase routes.

18.2.2.2 Gatehouse for Security and Control to Water Intakes (Operation)

The construction of a gatehouse to the right of brackish water access field is planned.

18.3 Utilities (Operation)

18.3.1 Power (Operation)

Several project sites must be powered by electrical energy, i.e.:

- The site for process plant and the base camp;
- The site of water well pumping station;
- The waste management site.

18.3.1.1 Power Supply to the Processing Plant and Base Camp

The assumption is that the Tijirit mine site can be fed from the Mauritanian network from the future 225 kV line which will link Nouakchott to Nouâdhibou.

This assumption supposes that that a 90-kV line from the Chami substation to Tasiast will be built by a third-party and that the Tijirit mine will be feed from the Tasiast substation through a 33 kV line that will have a length of approximately 40 km.

The installed capacity of the plant is 7.03 MW for Phase 1 and 11.01 MW for Phase 2. For this study, it is assumed the total consumption is 7.5 MW in Phase 1 and 11.5 MW in Phase 2.

18.3.2 Compressed Air (Operations)

One building will house two high-pressure air compressors (one in operation and the other on stand-by) to provide compressed air for the process and dried compressed air for instruments.

18.3.3 Water Supply (Operation)

It is planned to feed the plant with brackish water for Phase 1 and for Phase 2 with an expected length of 70 km.

Water well will be drilled and located to intercept the aquifer in order to provide brackish water to the site at a rate of 700 to 1100 cubic metres per day (m^3/day).

Water treatment requirements include the following water requirements:

- Drinking water supply and service for the base camp (to be supplied by tanker truck);
- Softened water for the process plant;
- Treated water (demineralized) for the elution process.

Table 18-1 presents the water consumption planned for the two phases of the Project.

Table 18-1: Water Requirement

Phase	Daily Consumption (m^3/day)	Duration (years)	Annual Consumption (m^3/yr)	Total Consumption for both Phases (m^3)
I	724	4	265,000	1,060,000
II	1063	3	390,000	1,202,500
			Total	**2,262,500**

Well pumps and electromechanical controls will be powered by a local generator under cover. Water pumped from each well will be directed to a reservoir near the pumping station. From the pumping station water will be pumped back through the 70 km pipeline to the water tank at the plant site.

A reverse osmosis water treatment unit will also be required at the plant site.

It is expected that an operator will be assigned to monitor the operation of brackish water boreholes. He will also be responsible for monitoring and ensuring the good condition of the pipeline.

Figure 18-5: Typical Infrastructure for Deep Well Boreholes in the Region

18.3.3.1 Supply of Power to Deep Wells

A pumping station will be located near the brackish water supply wells.

At the pumping station, the power supply will be provided by a 350 kW power generator. The power generation system will include a tank for storing diesel fuel. This, as well as the connections for filling the tank, will be installed inside a waterproof dike having a retention capacity equivalent to 110% of the volume of the tank.

The volume of diesel stored at the diesel fuel storage tank will represent the average consumption for 30 days of operation.

18.3.4 Fire Protection Equipment (Operation)

The fire protection equipment on site will respect local laws of Mauritania and comply with the requirements of Algold's insurers. The fire protection system will include a water tank for fire protection, pumps and fire hydrants and detection systems located in areas that pose a risk to workers and assets.

In areas of high risk, dedicated fire extinguishers will be installed. In places with potential for exposure to toxic and irritating chemicals, amenities are provided such as emergency showers and other specific safety equipment.

A service building will house the following pumps:

- Diesel fire pumps;
- Electric fire pumps.

A fire water network will supply the fire hydrants to cover the entire plant site.

18.3.5 Storage of Dangerous Materials

Hazardous materials, other than hydrocarbons, such as ammonium nitrate and chemicals, will be used on site. The hazardous materials storage area will be identified and fenced. Liquid materials will be stored in watertight areas protected from precipitation with retention dikes.

A storage area separate from other hazardous materials will be provided for the storage of sodium cyanide (received in solid form). The floor of the storage area will be concrete.

Cyanide leaching will be conducted in an area of the plant designed to contain and direct any spill to secondary containment. This area will be equipped with emergency showers and hydrogen cyanide (HCN) detectors.

18.3.6 Solid Waste Management (Operation)

Different types of waste will be generated throughout the life of the project by the mining operations, the processing plant and the base camp.

The waste generated during the operation will be sorted into four categories:

- Recyclable materials;
- Combustible domestic waste;
- Non-combustible household waste;
- Hazardous Materials.

In particular, the waste generated from the production (mine and plant), but are not limited to:

- Waste rock;
- Tailings;
- Lubricants and associated waste;
- Batteries;
- Packaging;
- Tires;
- Laboratory waste.

Waste from activities that are not related to production, but are not limited to:

- Perishable domestic waste;
- Non-perishable household waste (plastic, glass, metal);
- Waste from clerical activities, i.e. paper, cardboard, electronic equipment;
- Biomedical waste;
- Waste from maintenance and maintenance workshops (gas cylinders, used parts, scrap metal).

Waste management methods will be in line with current good practice and standards including:

- Environmental Code in Mauritania (Law 2000-045);
- IFC EHS Guides - Section 1.3 - Wastewater Discharges to the Environment (see next subsection);
- IFC EHS Guides - Section 1.6 - Waste Management;

- IFC EHS Guides for Mines - Waste Management and Non-Hazardous Waste;
- International Code for the Management of Cyanidation.

18.3.6.1 Waste Generation

The following table summarises the types of waste generated by the Project.

Table 18-2: Type of Wastes

Origin	Type of Waste
Base Camp- Lodging/Kitchen	Organic waste from food preparation, packaging, oil / grease / glass / plastics / paper / wipes / AC equipment
CIL Plant and Process Facilities	Containers of reagents, drums, used filters (anti-pollution), solvents, oils and greases, contaminated soils, oily rags, spare parts, conveyor belts
Generator	Used electrical equipment, used filters (anti-pollution), solvents, oils and greases, contaminated soils, oily rags, spare parts
Wastewater Treatment and Water Treatment	Used filters, activated sludge, oils and greases, used electrical and mechanical equipment
Incinerator	Ashes
Maintenance and Workshops Building	Batteries, oils and lubricants, oily rags, used glycol filters, various drums, pneumatic tires, rubber parts, used electrical and mechanical parts, scrap metal
Administration Building	Paper, cardboard, glass, perishable kitchen waste, office equipment and electronics, printer cartridges
Medical Facilities	Biomedical waste
Laboratory	Laboratory waste, solvents, reagents, oil and grease
Brackish water deep wells	Waste generated at pumping sites, oils and greases, used parts, used mechanical and electrical parts, used filters

18.3.6.2 Waste Characterization

All waste is either dangerous or non-hazardous. The characterization is in accordance with the definitions of the Environmental Code in Mauritania (Law 2000-045).

18.3.6.3 Waste Management Site

Waste management will be carried out in an area reserved for this activity which will be located about 2 km from the plant site.

It should be noted that solid waste will need to be managed during the construction period and the operating period. It is therefore planned that the development of this site starts at construction. It is during the construction period that the largest volumes of waste will be produced including the construction camp waste and packaging waste from equipment and materials.

Garbage trucks and waste bins for collection are planned.

The solid (non-hazardous) waste generated by the activities on the site will be sorted at the source. Recyclable materials (metal, paper and cardboard, plastic) will be compacted, bundled and sent to the recycling facilities.

The non-combustible waste will be buried in a pit designed for this purpose.

Other materials (organic waste and non-recyclable material) will be burned. An incineration area will be located at a safe distance from other facilities. A recycling plant can be set up with local artisans to recycle plastic, wood, etc.

Hazardous waste, such as used oil and grease, will be stored short-term in metal drums stored inside a secured area and shipped out on a regular basis to Nouakchott to a recycling center.

18.3.6.4 Waste Water Management

A domestic sewage system will direct the sewage to a buffer tank.

The gray water from the kitchen will be subject to special recycling before being directed, if necessary, to the buffer tank.

From this reservoir, the sewage will be directed to a septic tank and to a septic field.

18.4 Heavy and Light Vehicles

A preliminary list has been established for the evaluation of the number of light and heavy vehicles required for construction and operations.

The acquisition of this mining equipment depends on the strategy adopted by Algold. There are several choices, including but not limited to:

- Purchase of vehicles during construction and transfer to operations;
- Rental of vehicles over the life of the Project.

An execution strategy that works well and in agreement with Algold, is the management by the EPCM of a fleet of machines belonging to the Owner. The operators of these machines are trained on the job by the trainers. Thus for operations, the operators are ready and competent.

18.4.1.1 Mining Equipment

Please refer to Section 16.5 of the Report for mining equipment.

18.4.1.2 List of Equipment for the Processing Plant and Site Maintenance

The equipment list for the Processing Plant and Site Maintenance is shown in Table 18-3.

Table 18-3: Processing Plant and Site Maintenance Equipment List

Equipment	Model	Quantity
Auxiliary Fleet		
Water Truck	CAT 730B, HYDEX WT30	1
Hydraulic Excavator	CAT 385CL	1
Hydraulic Excavator with Hammer	CAT 390D L	1
Backhoe loader	CAT 430F	1
Compactor	CAT CS56	1
Skid steer loader	CAT 252B3 & CAT226B3	2
Bulldozer (for all stockpiles)	Komatsu U-dozer, model D155A	2
Fuel Tanker Truck	Mack Granite GU713	1
Maintenance		
Boom Truck	Boom Truck_ALTEC AC23-95B & TEREX BT 3870	2
Vacuum Truck	Sterling L7500 truck VT	1
Welding Truck	MAINTCORP 4X4 truck WT	1
Mechanical Service Truck	MAINTCORP 4X4 truck ST	2
Fuel/Lube Truck	2015 International 4400 truck PM	1
Telehandler	Telehandler_CAT TL943C	1
Articulated Boom Lift	Genie Lift (Terex) - Z™-45/25JRT (52 ft)	1
Articulated Boom Lift	Genie Lift (Terex) - Z™-135/70 (141 ft)	1
Scissor Lift	Genie Lift (Terex) - GS™-2669RT (32 ft)	1
Scissor Lift	Genie Lift (Terex) - GS™-5390RT (59 ft)	1
Forklift	Toyota 4FD200	2
Flat Bed Truck	Volvo VHD64B200	2
Mobile Compressor	Doosan XHP1170WCU	2
Personnel Transport Fleet		
Passenger Bus	Mercedes Benz / Mitsubishi - 40 seater	2
Passenger Mini Bus	Mercedes Benz / Mitsubishi - 12 seater	2
Pick-up Truck, Crew Cab	Toyota hilux pick-up truck	3
Pick-up Truck, Single Cab	Toyota hilux pick-up truck	3
Other Equipment		
Ambulance	Ambulance_Mercedes Benz	1
Fire Truck	Fire Truck_Mercedes Benz/DEMERS	1
Lighting Tower	SMC TL-90 Mobile Lighting Tower	2

Equipment	Model	Quantity
Dewatering Pump	Weir- Multiflo® MF360	1
Mobile Pump	Mobile Pump	1
Garbage Truck	CAT CT660S	1
Emergency generator	CAT XQ30	2

18.5 Tailings Management and Rock Waste Dump

The Tailings Storage Facility (TSF) will be located close to the processing plant to reduce transportation costs of the tailings from the plant to the TSF. It is estimated that 9.14 Mt of tailings (approximately 6.1 Mm^3 at 1,500 kg/m^3) will be produced over the life of the mine.

The method chosen for the management of the tailings is dry stack tailings (DST). In this method, the tailings are filtered into cakes at the processing facility in order to produce a tailing with less than 15% moisture by weight. This process permits the maximum recovery and the lowest losses of the process water and additives. It also reduces the amount of cyanide sent to the tailings storage facility. Finally, it removes or reduces the needs for capital costs related to dam construction and water management at the tailings storage facility since the tailings are not generally fluid and can be stacked in piles.

Two other tailings management strategies were also considered, conventional and thickened tailings, but have the disadvantages to send significant process water to the tailings storage facility (approximately 25 to 75 % water), most of which will evaporate or be trapped in the tailings and not be available to be recirculated back into the processing stream. The arid conditions of the site and the high rate of evaporation results in high water losses from these types of tailings storage facilities and significant makeup water requirements from groundwater or other sources. In addition, both these methods require the construction of containment dams and polishing ponds to handle the slurry or paste, and the decant water adding to the capital costs.

The tailings cake will be transported to the TSF by haul truck. A bulldozer will spread the tailings in layers and compacted in the same manner as a waste rock pile. For the current study, the tailings will be place directly on a suitable foundation soil without an impermeable membrane. This choice is to be reviewed in detail during the PFS. Small starter berms constructed of waste rock or other suitable material may be required to provide stability.

The tailings piles can be progressively placed in individual smaller cells and raised to their maximum height before start of the next cell. Each cell can be covered during construction and once it is completed. This progressive restauration will reduce the post-closure financial security and reduce the fugitive dust from the pile.

A bulk density of 1,500 kg/m^3 was chosen for the placed filtered tailings and is considered conservative for early volume estimation purposes. Proctor and consolidation tests can be conducted to obtain closer to field values and will likely result in higher density values, thus lower volume requirements for the tailings piles. Approximately 6.1 Mm^3 will be required for the 9.14 Mt of tailings. The storage area can be easily adapted to any adjustments to the volume.

Assuming side slopes of 3H:1V and a maximum height of 40 m, the piles will occupy a space of about 25 Ha. These slope angles and heights will require a geotechnical investigation and stability analysis to confirm the stability of the piles. The piles will be developed in three (3) phases following the estimates in the table below.

Table 18-4: Dry Stack Tailings Cells – Dimensions and Capacity

	Length (m)	**Width (m)**	**Volume (Mm^3)**
Phase I	270	370	2.1
Phase II	200	370	2.0
Phase III	200	370	2.0
Total	**670**	**370**	**6.1**

Phases 2 and 3 will overlap one face of the preceding phase. Runoff water from precipitation, the majority of which falls in the months around September, will be collected by polishing ponds located near the toe of each pile. The polishing ponds will be lined with geomembrane to prevent seepage and water losses. This water will be recirculated back to the plant for use in processing. Access to the polishing ponds will be restricted by fences.

The piles will be progressively covered, with protection added to the slopes as the piles are raised to avoid excessive erosion and heavy sediment loads reporting to the polishing ponds. Silt fences may be required on slope faces that cannot be immediately covered by waste rock or other protective material.

The presence of cyanide in the process water, and therefore the tailings interstitial and runoff water, justifies the concept of a tailings storage facility with a foundation of low permeability material. However, due to the method of management, which includes washing of the filter cake to displace and recover any cyanide and filtering the tailings to a moisture content less than 15 % moisture by weight, there is the possibility of placing the dry stacked tailings directly on the existing terrain without impermeabilization. This management method will be reviewed during the PFS when more testworks results will be available.

18.5.1 Acid Generation Potential

Some preliminary tailings geochemical characterization have been done by Lakefield in 2016 on 4 composites samples. In summary, these preliminary results indicate that the Sophie I tailings (7 % of the LOM tonnage) are likely acid generating, the tailings coming from Eleonor and Lily deposits (74 % of the LOM tonnage) may be acid generating, and the tailings from Sophie II (19 % of the LOM tonnage) are likely non-acid generating. As such, the leaching potential is still uncertain since the simple static tests conducted only indicate potential and are not conclusive. In addition, no acid generation potential tests have been conducted on the waste rocks.

At the feasibility level, kinetic testing (humidity cells, column tests, TCLP, etc.) would be required to verify the true acid generating potential of each areas and adapt the storage strategy accordingly. For the PFS, a more comprehensive static testing program is required as an initial screen of the acid generation and neutralization potential. Tailings and waste rock not yet characterised would need to be included in the static tests. A detailed review of the existing mineralogy and geology is needed to plan the selection of the static tests at the PFS level.

In light of the above information, the proposed management is currently based on the following scenario.

In relation with the relative tonnage mined from each deposit, there are possibilities that some portion of the tailings might be acid generating. The cyanide content is also a concern for the tailings since cyanide destruction is not currently planned for the tailings leaving the processing stream. Both of these facts equally could justify the need for additional protective measure at the tailings storage facility and associated water management infrastructure. These aspects will be closely look at during the PFS.

However, due to the lack of receptors through a groundwater path in the Tijirit desertic environment, which is currently assumed to be limited and of poor quality for consumption if it exists, and also the low moisture content of the dry stack tailings being the chosen management solution, compared to conventional saturated slurry tailings, the retained solution so far would be the placement of the tailings directly on the existing terrain without the need for a membrane.

It is recommended to review this choice more closely in the PFS phase of the project following additional geochemical testing, but the overall economic impacts to the project are minor (less than $500k) if the membrane is added to the current design.

Due to the “dry” nature of the placed tailings, fugitive dust would need to be managed to limit exposure of the mine personnel. It is expected that a crust will form on the tailings, as is the often case when tailings in arid region dry, however, this aspect will be reviewed for low moisture content tailings in upcoming phases of the project.

For the proposed runoff collection ponds around the TSF, a membrane is recommended, both to conserve as much runoff water as possible for recirculation, and also due to the higher risk associated with the saturated tailings reporting to the runoff ponds during the limited periods of intense “monsoon” rains. This is a minor cost which was not included in the current study due to the limited hydrology data available at this time.

Finally, the tailings reporting to the TSF have not been treated by cyanide destruction, but the filter cake will be washed and the solution containing cyanide will be recovered and recycled into the process. For the above assumptions to be viable, the cyanide content reporting to the TSF should not be in excess of 50mg/L, which could potentially be achieved by incorporating this rinse into the processing cycle that would have the added economic benefit of recovering cyanide for additional processing. This aspect will also be looked at during the PFS to asses the pro and con of such a strategy.

18.5.2 Waste Rock

The waste rock produced by the site preparation works and mining operations will be transported by haul truck towards the waste rock piles sites located close to each open pit. They will have a maximum height of 60m and approximate side slopes of 20 degrees. The waste rock piles will have a total volume of 58 Mm^3 and cover a combined surface of approximately 220 ha. The current assumption is that the waste rock is not problematic from a geochemical standpoint, and it will not be required to confine them, or capture the seepage water. This hypothesis is to be validated in the PFS by static geochemical tests on representative samples.

However, if test results show that some portions may be problematic on a geochemical standpoint, it will require additional management of the material, including potential confinement and contingency for the runoff and seepage. This will limit their use in construction of infrastructure and require additional considerations for environmental remediation. An increase in the associated costs would need to be realised.

In addition, the results of the geochemical characterization of the waste rock will be required to decide if it can be used as construction material for tailings covers, roads, berms, foundations, etc. If the material is found to be non-acid generating and not leachable, it may be a potential source for construction material. If they are found to be not usable, a borrow pit investigation will be required to locate alternative sources. Alternatively, it may be possible to source the material at the construction sites, making use of cut and fill if the materials are suitable. This option needs to be validated by the geotechnical investigation planned for the TSF location.

In conclusion, in order to reduce the above uncertainties in the tailings and waste rock management strategies, it is recommended to complete detailed geochemical characterizations of the material while proceeding to the next phase of development. Static testing is recommended for the PFS while starting kinetic testing is recommended before commencement of the FS as they require several months (3 to 6 months on average) for completion after material is received by the analytical laboratory.

19. MARKET STUDIES AND CONTRACTS

The economic analysis is based on a gold price of $1,250 US per ounce which reflect Algold conservative outlook of the future market for gold. This could be compared to a last 3 years average of $1,230 US, a last year average of $1,288 US and a last 6 months average of $1,304 US. All number calculated up to March 9, 2018.

A refinery/transportation cost of $15 US per ounce have also been deducted from that projected price which is considered to be a fair approximation of that cost.

19.1 Contracts

No contracts relevant to the PEA have been established by Algold. Algold has not hedged, nor committed any of its production pursuant to an off-take agreement.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental and Social Impact Assessment Study

In 2016, Algold gave the mandate to AECOM to prepare an Environmental and Social Impact Assessment Study (ESIA) on Tijirit Project

This study covers:

- The legal and institutional framework;
- A detailed description of the Tijirit Project;
- A portrait of the initial state of the receiving environment;
- A communication plan with a public consultation program developed for the Project with the results obtained;
- An assessment of the environmental and social impacts with appropriate mitigation measures;
- An assessment of cumulative impacts;
- An environmental and social management plan;
- The broad lines of the Resettlement Action Plan
- The broad lines of the Environmental Emergency Response Plan
- The broad lines of the site rehabilitation and closure plan.

As part of the consultation process, AECOM conducted additional investigations with stakeholders to collect information and to have further discussions. Meetings were held with Algold employees and with representatives of the wilayas of l'Inchiri and Dakhlet-Nouâdhibou; from Chami moughataa as well as Chami and Tmeïmichat Townships.

The study was given to the Mauritanian Government at the end of 2016 and at the beginning of 2017, the Mauritanian Ministry of Environmental and Sustainable Development has provided a conclusive acceptance and opinion regarding the ESIA Study Report.

20.2 Artisanal Mining

Some illegal artisanal mining is conducted in the Tijirit area since 2016. The environmental impacts of their works will be further evaluated in 2018.

20.3 Permitting

In June 2017, a thirty-year mining permit (2480C2) have been granted to Tirex by the Government of Mauritania giving them the exclusive right of prospecting, seeking and exploiting gold within the limit of this permit.

According to Mauritanian Legislation, no other permits are required to proceed with the development of the Project.

21. CAPITAL AND OPERATING COSTS

21.1 Extent and Scope of the Capex Estimates

The purpose of this Basis of Estimate is to outline the methodology used for the development of the capital expenditure (Capex) estimate forming part of the PEA for the execution of the Project.

The extent of the Capex estimates includes all Projects' direct and indirect costs to be incurred during the implementation of the Project, inclusive of further studies as well as the execution phases complete with basic and detailed engineering. The Capex is deemed to cover the period starting at the approval by Algold of a final Feasibility Study and finishing after commissioning is achieved. It should hence be understood that this Capex excludes transfers to Algold operations, performance test, start-up, ramp up and operations.

The scope of work of the Capex estimates includes the following process related areas:

Area	Major Scope Element: Phase 1	Major Scope Element: Phase 2
Mine	Open pit pre-production development Maintenance facilities, workshop, warehouse and laydown Drill core storage Mine administration and mine dry Explosives magazines and detonators	
Crushing	Primary crushing Crushed mineralised material stockpile and grinding mill feed conveyor	Secondary crushing (pebble);
Grinding	Primary grinding (SAG Mill) Gravity separation	Primary grinding (Ball Mill)
Treatment Plant	Gravity concentration CIL and carbon regen. Washing and elution Electrowinning and smelting Reagents storage and prep. Process control Laboratory Tailings management	

All infrastructure and utilities necessary for proper plant operations are included in the scope of work. The vast majority will be built during Phase 1.

Infrastructure included are:

- Plant administration;
- Gate house, security and training;
- Maintenance facility and workshop;
- Warehouse;
- Fuel storage and distribution;
- Permanent camp, also used during the construction phase;
- Medical clinic

Utilities included are:

- Brackish water, sourced from a well field 70 km away, for all water needs;
- Power from Tasiast through an overhead 33 kV power line, including connections to an existing substation;
- Compressed air;
- Fire protection and detection;
- Waste management;
- Sewage water collection, treatment and disposal;
- Storage and distribution of fuel;
- Telecommunications;
- Security.

Finally, all construction field indirect costs are included.

21.1.1 Capex Estimate Presentation

All capital costs are expressed in United States Dollars (USD). Currency exchange rates are dated 2Q 2018. No inflation is included in the estimates. The total capital cost has been estimated at $145.5M USD for Phase 1 and at $16.3M USD for Phase 2.

A cost summary of the Project is presented in Table 21-1:

Table 21-1: Cost Summary

Area #	Area description	Phase 1 ('000 USD)	Phase 2 ('000 USD)
1000	Mine Infrastructure	3,060	
	Initial Mine Equipment & Pre-Development	16,633	
2000	Crushing	8,728	2,759
3000	Grinding	10,014	5,345
4000	Process Plant	28,901	1,785
5000	Infrastructure	25,152	0
6000	Utilities	3,925	134
	Sub-total – Direct Costs	**96,413**	**10,023**
9100	Owner's Costs	8,000	0
9200	EPCM	9,031	1,203
9300	Construction Field Indirect Costs	14,125	1,801
9400	Contingency	17,935	3,257
9500	Inflation (Excluded)		
9600	Risks (Excluded)		
	Sub-total – Indirect Costs	**49,091**	**6,261**
	TOTAL:	**145,504**	**16,283**

21.1.1.1 Capex Estimate Accuracy

The accuracy of the Capex estimate for Phase 1 is established at ±30 %. For Phase 2, the accuracy is established at ±50 %.

21.1.1.2 Deliverables

The Capex estimate was developed based on the following list of deliverables:

- Project description;
- Mechanical equipment list;
- Overall general arrangement plan.
- Preliminary Mining Plan

21.1.1.3 Estimate Coding

All estimate line items were coded using the Project Work Breakdown Structure. Also, discipline codes were used to group the various activities enabling the standardization of unit hours and material rates.

21.1.1.4 Currency Exchange Rates

All costs were expressed in their native currency. Currency exchange rates were based on Oanda's web site. The following table lists the currencies used for the estimate along with currency exchange rates dated April 22nd, 2018.

Table 21-2: Exchange Rates

Source Currency	Description	Base Currency	Currency Exchange Rate	Phase 1 ('000 USD)	Phase 2 ('000 USD)
USD	United States Dollar	USD	1.0000	103,865	8,918
MRO	Mauritanian Ouguiya	USD	0.0028	22,115	3,264
CAD	Canadian Dollar	USD	0.7828	15,080	3,054
EUR	EURO	USD	1.2284	3,946	1,047
AUD	Australian Dollar	USD	0.7666	498	0
				145,504	16,283

21.1.1.5 Estimating Software

The Capex estimate was developed using MS Excel.

21.1.2 Methodology

21.1.2.1 Data Sources

A) Mining Capital Costs

The Capital costs associated with the development of the Tijirit project are specified in Table 21-3. The bulk of the Capital costs for the project are summarized into the purchase of equipment required for the operation, the replacement of equipment as sustaining Capital and the capital associated with the development of the project (during pre-production). DRA/Met-Chem received budgetary pricing from the equipment suppliers for each of the major pieces of mining equipment including support equipment such as dozers, wheel loaders and graders. Service and smaller support equipment capital was estimated using DRA/Met-Chem database. The level of precision used for the Tijirit project is ±30% which is in line with a PEA Study.

Table 21-3 – Summary of Capital Cost Estimate (Mining)

Description	Units	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Major Equipment	USD	5,046,185	14,594,738	5,046,185	0	2,085,000	5,560,000	32,332,108
Support Equipment	USD	4,236,224	750,000	71,804	0	0	0	5,058,028
Service Equipment	USD	1,310,163	0	324,376	0	0	0	1,634,539
Mine Development Cost	USD	6,040,488	0	0	0	0	0	6,040,488
Total		**16,633,060**	**15,344,738**	**5,442,365**	**0**	**2,085,000**	**5,560,000**	**45,065,163**

Mine Fleet Requirements

The major mining equipment, including haul trucks, excavators and production drills will be purchased on an as required basis in order to reduce the initial capital cost of the Project. Additional equipment that is required throughout the mine plan and equipment that is purchased to replace the initial fleet when it reaches its expected life is treated as sustaining capital. The current mine life of the Project is estimated at approximately 8 years of operation. As a result, the replacement of most major, support and service equipment was not considered, with the exception of haul trucks. DRA/Met-Chem considered the useful life of a haul truck to be 22,000 hours. The purchase and replacement schedule of mining equipment is represented graphically in Figure 21-1.

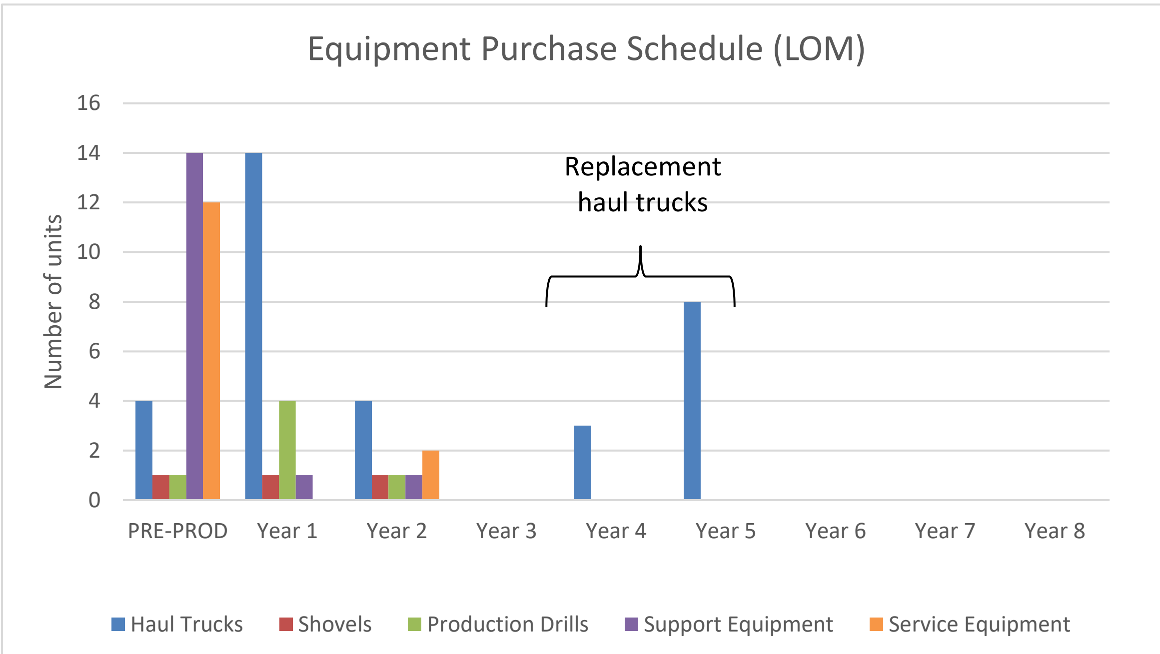


Figure 21-1: Equipment Purchase Schedule

Mine Development Cost

The open pit mine development cost accounts for the activities that will be carried out during pre-production to prepare the mine for operations. These activities include; Road construction, waste stripping in preparation of several mineralised material faces and creating a Primary mill feed stockpile. In order to estimate the mine development cost, DRA/Met-Chem determined the cost to operate the mining fleet during the pre-production phase. The total development cost is $ 6.04 M.

Haulage Road Construction

Mine haulage road construction costs were considered at a rate of $150K US/km. These costs were incurred based on the development of the mining zones as per the mine plan presented in Section 16.

Support and Service Equipment

The quantity of support and service equipment was estimated based on similar projects given amount of personnel and major equipment estimated on-site.

B) Plant Equipment and Bulk Quantities and Material Costs

A mechanical equipment list was developed by Ausenco's Process Group. All bulk materials were factored from the mechanical equipment costs and compared against benchmarks from projects similar in nature and size.

Plant equipment costs were generally developed based on an internal database. Some equipment costs were estimated when no relevant data was available. Rates for bulk material were developed based on recent in-house data obtained from similar projects in Mauritania.

Some allowances were generated to ascertain the entire scope coverage.

Table 21-4 present a summary of bulk materials quantity used:

Table 21-4: Summary of Bulk Materials

Discipline	U. of M.	Ratio over Mechanical	Phase 1	Phase 2
Earthworks	M^3	10 %	62,300	3,500
Concrete	M^3	10 %	2,175	350
Steel	T	10 %	661	110
Platework (excludes hoppers and tanks, taken from MEL)	T	30 %	234	27.5
Pipeline	M	Direct MTO	70,000	0
Piping	M	15 %	12,899	3,774

Costs for electrical equipment and bulk materials as well as instrumentation and controls were factored from mechanical equipment costs at 40% and 15% respectively.

C) Labour Costs

Labour manhours were developed internally, for each site activities. A productivity factor was defined based on recent in-house data obtained from similar projects in Mauritania. It has been defined at 2.4, which takes into consideration the labour skill, in-plant movement, extended work week and rotation schedule.

Labour rates were also obtained from the same source; labour rates are inclusive of contractors' indirect costs, namely mob and demob, small tools, construction equipment, consumables, PPE, temporary site establishments, supervision and administration as well as overhead and profit.

Rotational transportation costs are included as part of construction field indirect costs.

D) EPCM Services

While the Project may not ultimately be executed via the EPCM model, the cost estimate was structured on that basis. EPCM services consist of the following:

- EPCM team salaries, fringes, uplifts, recruitment, overhead, etc.;
- EPCM team expenses (i.e. business travelling, room & board, accommodation, etc.);
- Home office support and expenses (communications, IT services, IT equipment, courier, printing, office space, furniture, consumables, stationaries, etc.).

EPCM services costs for Phase 1 are estimated at 10% of the direct costs, exclusive the overhead power line as well as contract mining costs. For Phase 2, they are estimated at 12% of the direct costs, based on the unknows pertaining to an operating plant environment.

E) Construction Field Indirect Costs

Site construction indirect costs are included as a percentage of direct costs, excluding the overhead power line. Indirect costs are inclusive of:

- Site preparation for all temporary infrastructure and buildings, construction facilities, laydown areas, temporary services, etc.;
- Temporary roads, walkways, parking areas and fencing, c/w signage and including temporary lighting, complete with maintenance;
- Temporary buildings/construction facilities (offices - for EPCM and Owner's staff, camp, cafeteria, laundry facilities, medical clinic, security gate/office, etc.), complete with mobilization, demobilization, rental, operations and maintenance. It should be noted that contractors will be responsible for the provision of their own temporary facilities.
- Temporary infrastructure for the supply of power, fuel, gas, water and communications. It should be noted that contractors will be responsible for their own temporary infrastructure.
- Temporary infrastructure for the management of sewerage and construction waste (dry and wet, hazardous and non-hazardous), including collection, treatment and disposal. It should be noted that contractors will be responsible for their own requirements.
- Pad preparation and fencing – only – of contractor's pads are included in the construction field indirect costs;
- Field office supply (IT equipment, courier, printing, office space, furniture, consumables, etc.);
- Access control and monitoring;
- Temporary lay down and storage areas, as well as warehousing, complete with, but not limited to, materials management and materials handling equipment;
- Mobilization and demobilization of all above listed temporary site establishments and restoration back to original site conditions;
- Site surveying
- Site security;
- Light vehicles;
- First aid and medical services;
- General and final clean-up.

Construction field indirect costs for Phase 1 are estimated at 8% of the direct costs, excluding the overhead power line.

For Phase 2, construction field indirect costs are estimated at 6% of the direct costs, taking into consideration the existing facilities and infrastructures implemented during the Phase 1.

F) Room and Board

Costs for room and board are included as 2 separate line items, i.e. permanent camp as well as catering services. It is assumed that the permanent camp will be made available for the construction period; no additional costs are included to cover for the restoration of the camp to its nominal condition.

G) Owner's Costs, by Algold

Owner's costs were estimated at $8.0M USD by Algold. They were integrated in the overall estimate as obtained. It is assumed that Owner's Costs for Phase 2 are included as part of the Opex.

H) Freight

Freight costs, namely transport and logistics, are estimated at 11% of all permanent equipment and bulk materials, excluding earthworks and concrete. Freight costs are inclusive of overseas land and ocean transportation as well as local Mauritanian land transport.

I) Other Costs

Other costs, such as spare parts, vendor representatives, first fills, special tools, etc. are included as an overall percentage of 5% of all permanent equipment.

J) Project Contingency

For Phase 1, the project contingency was assessed at 15% of all costs.

For Phase 2, based on the unknowns pertaining to operating plant environment, the contingency was assessed at 25% of all costs.

K) Inflation

Inflation beyond this Capex estimate base date is explicitly excluded.

L) Risks

Risks, complete with mitigation plans, are explicitly excluded from this Capex estimate.

21.1.3 Qualifications

All estimates are developed within a frame of reference defined by assumptions and exclusions, grouped under estimate qualifications. Assumptions and exclusions are listed below.

A) Assumptions

The following items are assumptions concerning the Capex:

- Estimate is based on rotations schedule of 4 and 2, i.e. 4 weeks in and 2 weeks R&R;
- Estimate is based on 7 days at 10 hours per day workweek;
- Estimate is based on travel time during the 2 weeks R&R;
- Estimate assumes that labour skills will range from medium to high, i.e. no unskilled nor low skill labour;
- Estimate assumes fresh water trucked to site does not need any treatment to be used for concrete mix, leak/hydro testing, flushing, cleaning, etc.;
- Estimate assumes EPCM and Owner's teams will be in sufficient quantity so as not to delay contractors;
- Estimate assumes a mix of manual labour which will be sourced locally with a combination of expats from neighbouring countries (i.e. Morocco, Senegal, etc.) and from Europe (i.e. Spain, Portugal, etc.);
- Estimate assumes no underground obstructions of any nature;
- Estimate assumes no delay in Client's decision making;
- Normal peak workforce;
- Engineering progress sufficient to avoid rework.

B) Exclusions

The following items are not included in the Capex:

- Currency fluctuations
- Oxygen plant;
- Power generation via solar panels;
- Any and all scope changes;
- Allowances for disruption due to underground obstruction;
- Inflation beyond the Capex estimate base date;
- Risk;
- Financing charge;
- Delays resulting from community relation, permitting, project financing, etc.;
- Any and all taxes, customs charges, excises, etc.

21.2 Operating Cost Estimate

21.2.1 Basis of Estimation

The project operating costs presented in this section include, amongst other things, the mining operating costs, the processing plant operating costs, the transportation of tailings material to the Tailings Storage Facility as well as the site infrastructure and administration costs.

More specifically, the operating costs include the following areas:

- Mining infrastructure and operating costs;
- Explosives storage;
- Heavy vehicle workshop;
- Processing plant infrastructure and operating costs;
- Auxiliary infrastructure including offices, workshops, warehouses, canteen and ablution rooms;
- Camp operating costs;
- Laboratory operating costs,
- Storage and distribution of fuel, and
- Road maintenance costs.

The following items are not covered in the operating cost estimate, but have generally been included in the financial model:

- Cost of capital and financing;
- Depreciation;
- Exchange rate fluctuations;
- Royalties and taxes;
- Inflation;
- Income taxes;
- Algold central office and corporate costs;
- Exploration;
- Contingency;
- Commissioning and start-up costs, such as:
    - Working capital;
    - Critical spare parts;
    - First fills.

21.2.2 Operating Cost Summary

Operating costs were developed on the basis of the Phase 1 and Phase 2 mining schedule, including the established run-of-mine (ROM), plant feed grade, gold recovery and doré production, as described in previous sections of this Report.

Two key cost parameters are the price of electricity and diesel. The electrical cost was supplied by the Mauritanian government. It was provided as a fixed annual cost of $50,000 USD and a variable cost of 8 c/kWh. A diesel cost of 1.13 USD/L was selected.

Table 21-5 and Table 21-6 summarise the project operating costs for the Life of Mine (LOM) and for the two distinct operating phases. Unit costs are presented as a function of plant run-of mine (ROM) material.

A detailed breakdown of the operating costs is provided in Table 21-7. Finally, Figure 21-2 and Figure 21-3 show the variation in OPEX over the 7 Year and 1 month LOM.

Table 21-5: Operating Cost Summary

Operating Cost Summary	Life of Mine (M USD)	Unit Cost (USD/t processed)
OPEX – Mining (incl. tailings transportation)	171.5	18.77
OPEX – Processing (incl. water management)	126.5	13.84
OPEX – G&A	45.8	5.01
Total OPEX (excl. royalties)	**343.8**	**37.62**

Table 21-6: Operating Cost Summary by Phase

Operating Cost Summary by Phase	Phase 1 Years 1 to 4			Phase 2 Years 5 to 8		
	Total Cost M USD	Unit Cost USD/t processed	Unit Cost USD/ oz Au	Total Cost M USD	Unit Cost USD/t processed	Unit Cost USD/ oz Au
OPEX – Mining (incl. tailings transportation)	111.6	27.40	266.99	59.9	11.83	367.98
OPEX – Processing (incl. water management)	61.1	15.01	146.24	65.4	12.90	401.35
OPEX – G&A	25.7	6.30	61.42	20.1	3.98	123.66
Total OPEX (excl. royalties)	**198.4**	**48.71**	**474.66**	**145.4**	**28.71**	**892.99**

Table 21-7: Operating Cost Breakdown

	Total LOM OPEX (M USD)	LOM Unit Cost (USD/t processed)
Mining		
Loading	11.0	1.21
Hauling	33.3	3.64
Drill & Blast	82.2	9.00
Support & Services	20.7	2.26
Road Construction	2.7	0.29
Labour	21.6	2.36
Total Mining	**171.5**	**18.77**
Processing		
Power	30.9	3.37
Reagents	30.9	3.38
Consumables (incl. water treatment consumables)	5.8	0.63
Diesel (for process plant only)	2.7	0.29
Mobile equipment	28.9	3.16
Maintenance materials	7.0	0.77
Labour	20.3	2.22
Total Processing	**126.5**	**13.84**
General & Administration		
Camp operations	19.6	2.15
Management and accounting	20.3	2.22
Employee services	4.0	0.44
Laboratory consumables	1.9	0.21
Total G&A	**45.8**	**5.01**
Total Operating Cost	**343.8**	**37.62**

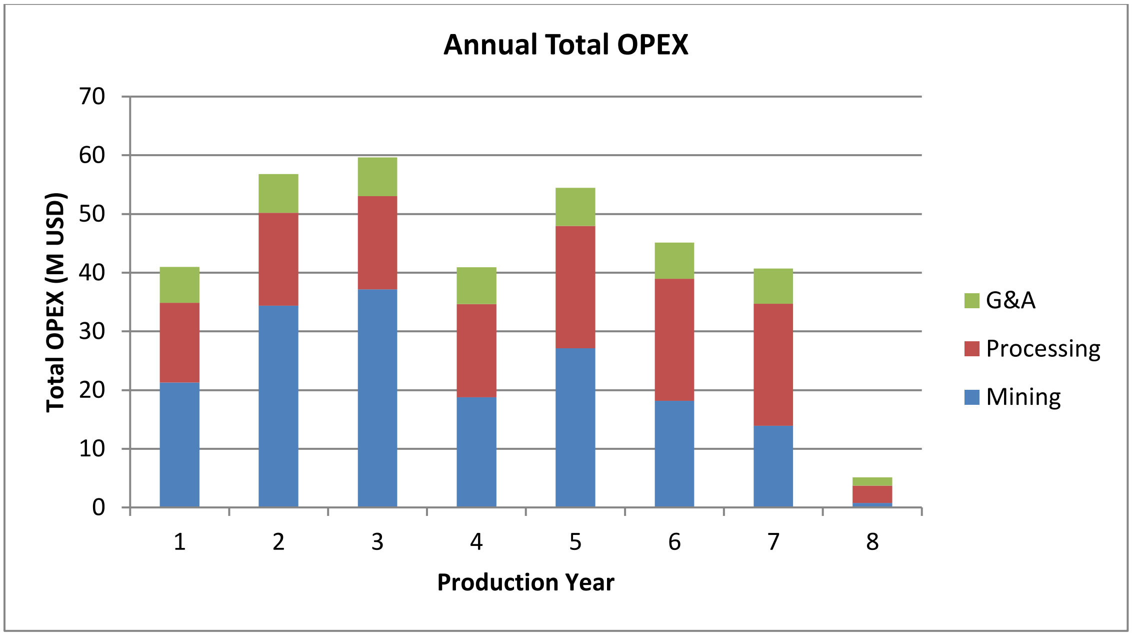


Figure 21-2: Total OPEX by Year

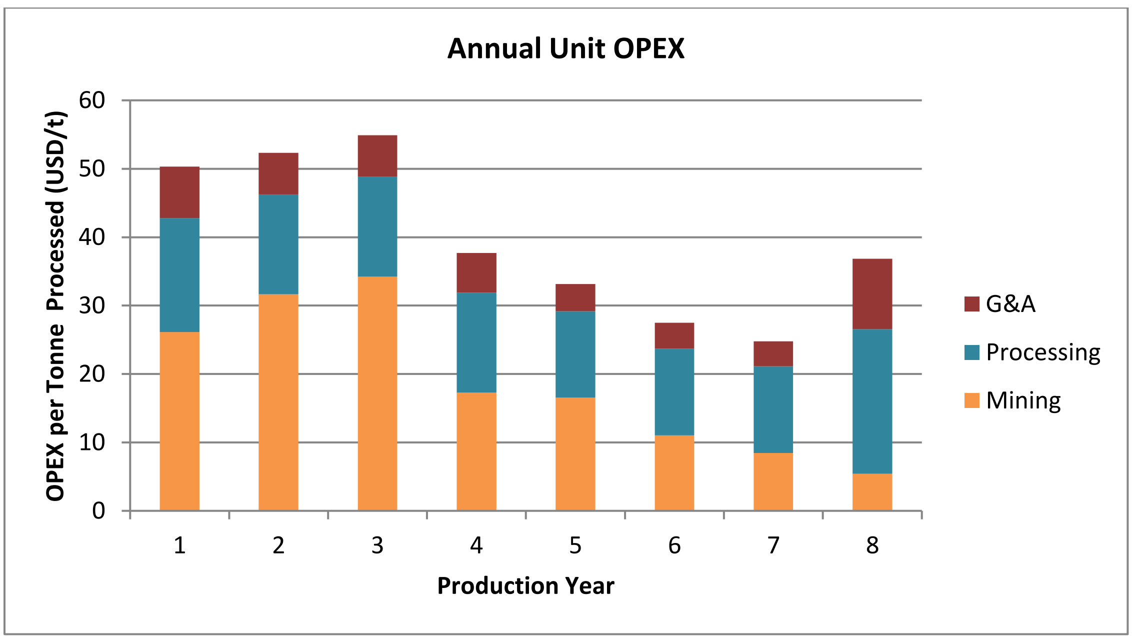


Figure 21-3: Unit OPEX by Year

21.2.3 Mine Operating Costs

The mine operating cost were estimated with a precision of ±30%. The sources of information used to develop the operating costs include in-house database and supplier budgetary estimates particularly for the estimation of operating cost for major mining equipment (Hauler, Loading equipment and production drills).

The following parameters used to estimate operating costs for the Project:

- Operator salary based on manpower rates provided by Algold (based on local rates) – salaries range between 6,000 and 18,000 USD/year;
- Diesel cost of 1.13 USD/litre;
- Emulsion unit cost – 0.51 USD/t.

Based on the above unit costs above combined with the mine plan described in Section 16, the mine operating cost was estimated at an average of $1.42 US/t mined for the life of the Project. The mine operating cost per year are summarized in Table 21-8.

Table 21-8: Mine Operating Cost Per Year

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Major Equipment	$	7,473,183	13,952,857	15,552,285	6,056,595	9,884,316	5,813,591	3,851,744	160,516	62,745,086
Support Equipment	$	2,658,436	2,792,859	2,792,859	2,792,859	2,792,859	2,792,859	2,792,859	267,808	19,683,397
Service Equipment	$	391,491	525,914	525,914	525,914	525,914	525,914	525,914	50,430	3,597,403
Other	$	10,765,414	17,090,863	18,312,586	9,397,503	13,944,027	9,013,946	6,719,168	273,848	85,517,354
Mine OPEX	**$**	**21,288,524**	**34,362,492**	**37,183,643**	**18,772,870**	**27,147,115**	**18,146,309**	**13,889,684**	**752,602**	**171,543,239**
$/t mined	**$/t**	**1.43**	**1.21**	**1.35**	**1.73**	**1.34**	**1.76**	**2.43**	**4.04**	**1.42**

* Totals may not add up due to rounding

21.2.3.1 Mine Operating Cost Breakdown

The following Tables and Figures present the total operating cost breakdown for the Project. These Opex costs are represented in USD/t mined and USD/t processed and as percentage based on Total Operating Cost (LOM).

Table 21-9: Operating Cost Breakdown by Activity

Activity	Costs $/t Mined	Costs $/t Processed	Total %
Loading	0.09	4.30	6%
Hauling	0.28	0.71	19%
Drilling and Blasting	0.68	4.40	48%
Support and Service	0.17	6.70	12%
Road Construction	0.02	2.36	2%
Manpower	0.18	0.29	13%
Total	**1.42**	**18.77**	**100%**

* Totals may not add up due to rounding

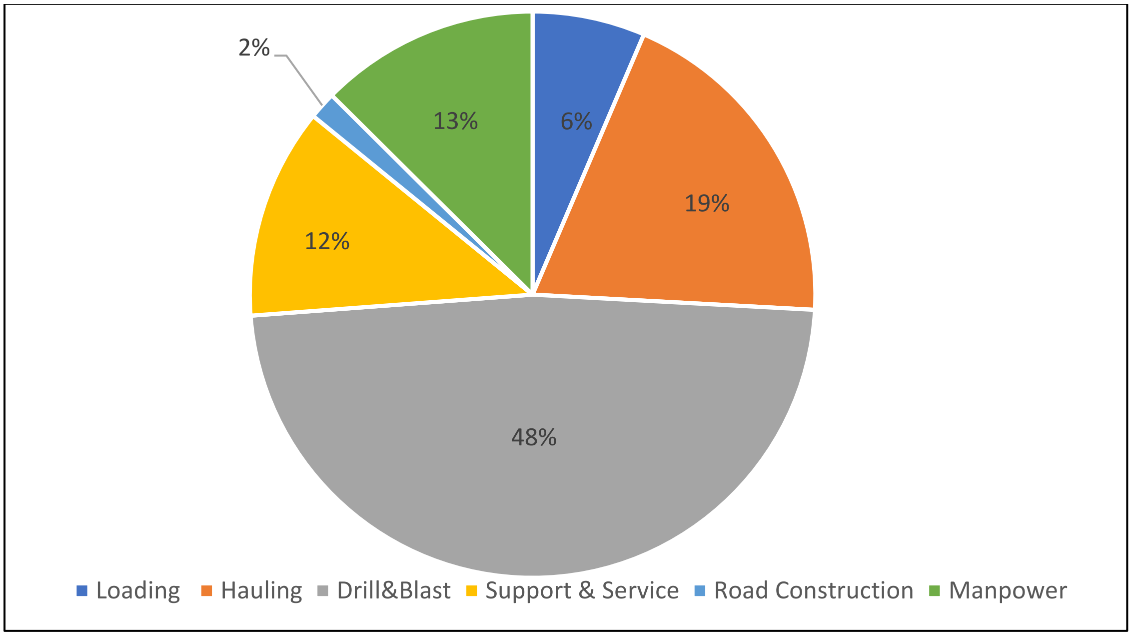


Figure 21-4: OPEX Breakdown by Activity

Table 21-10: Operating Costs Breakdown (Consumables and Manpower)

Consumables / Manpower	Costs $/t Mined	Costs $/t Processed	Total %
Fuel	0.32	1.21	23%
Tires	0.05	3.64	4%
Repair / Parts	0.33	9.00	23%
Explosives	0.51	2.26	36%
Manpower	0.18	0.29	13%
Other	0.02	2.36	1%
Total	**1.42**	**18.77**	**100%**

* Totals may not add up due to rounding

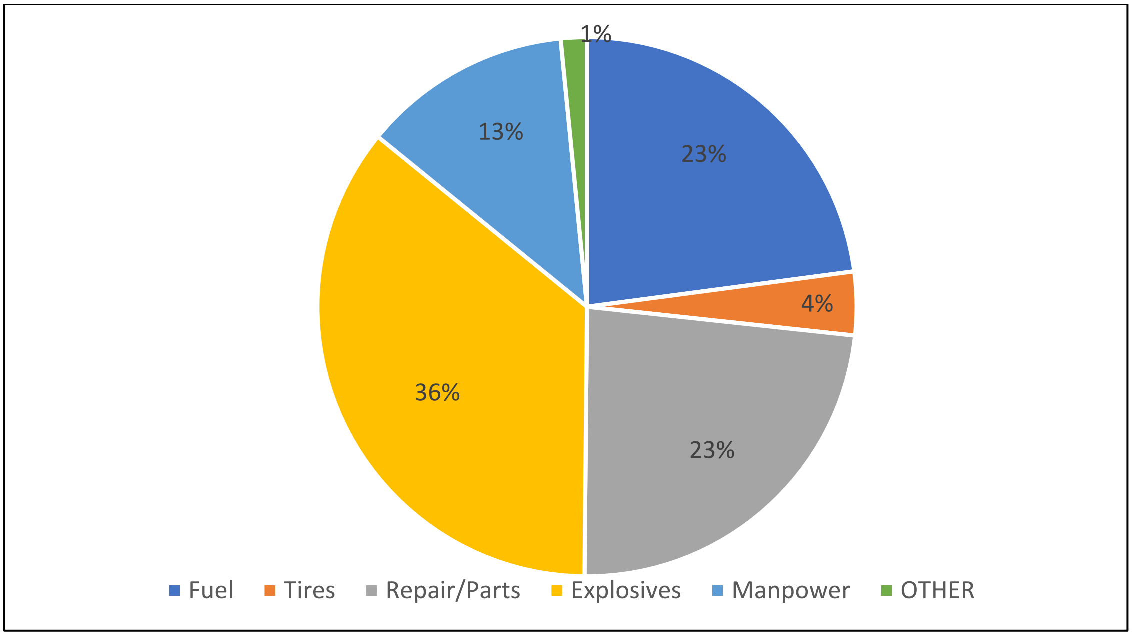


Figure 21-5: Operating Costs Breakdown (Consumables and Manpower)

21.2.3.2 Operating Cost Breakdown by Major Components

The hourly operating cost for the major mining equipment was supplied by the equipment suppliers and manufacturer. These were used to develop the operating costs as described above. For certain equipment where hourly operating cost estimates were not obtained, DRA/Met-Chem used its internal database. Table 21-11 provides a detailed breakdown of the hourly operating cost for each piece of equipment considered in the mining fleet.

Table 21-11: Equipment Hourly Operating Costs

Equipment	Description	Fuel $/h	Tires $/h	Parts $/h	Total $/h
Major Equipment					
Haul Truck	Payload – 55-tonne	21.20	10.00	25.00	**56.20**
Shovel	Bucket – 5 m³	76.59	n/a	50.66	**127.25**
Shovel	Bucket – 8.1 m³	84.80	n/a	57.12	**141.92**
Production Drill	153 mm hole - 13.6m mass	53.00	n/a	95.44	**148.44**
Support Equipment					
Track Dozer	CAT D7R	37.10	n/a	30.72	**67.82**
Track Dozer	CAT D8R	41.61	n/a	36.83	**78.44**
Road Grader	CAT 16M3	29.57	2.65	30.89	**63.12**
Wheel Loader	CAT 988K	41.87	9.54	38.98	**90.39**
Excavator	CAT 349K	46.85	n/a	50.76	**97.61**
Water Truck		21.20	2.50	9.00	**32.70**

Equipment	Description	Fuel $/h	Tires $/h	Parts $/h	Total $/h
Lighting Plant	MAGNUM MLT3080	2.65	n/a	0.50	**3.15**
Service Equipment					
Fuel and Lube Truck		21.20	2.50	9.00	**32.70**
Mechanic Truck		21.20	2.50	9.00	**32.70**
Boom Truck		15.90	0.65	2.50	**19.05**
Tire Handler		15.90	0.65	2.50	**19.05**
Pickup Truck		6.36	0.15	1.00	**7.51**

21.2.4 Processing Operating Costs

Processing operating costs were developed by Ausenco on the basis of the LOM schedule, the processing plant mass balance and the processing plant equipment and load list. The operating cost estimate was calculated using internal database information, with costs escalated to the first quarter of 2018 (Q1 2018).

The estimate includes all operating costs associated with the production of doré bars by the Tijirit processing plant. The operating costs do not include downstream product transportation, marketing or corporate overheads.

The processing plant operating cost for Phase 1 is 15.01 USD/t processed and 12.90 USD/t processed for Phase 2. The overall processing plant operating cost over the LOM is 13.84 USD/t.

Figure 21-6 shows a pie chart illustrating the relative importance of the different processing costs over the LOM.

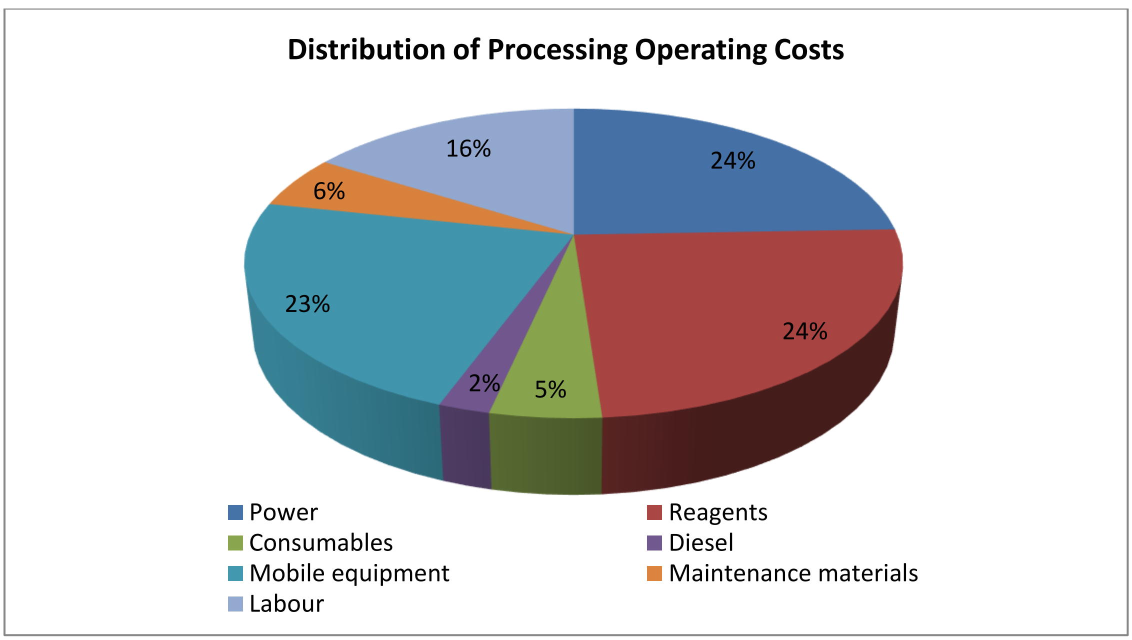


Figure 21-6: Distribution of Processing Operating Costs

21.2.5 General & Administration Operating Costs

General and administration operating costs were developed by Ausenco. The operating cost estimate was calculated using internal database information, with costs escalated to the first quarter of 2018 (Q1 2018).

The Project site is considered remote and will require an operator camp and other remote-site facilities. The operating costs of the camp were calculated using the mining and processing plant labour roaster. Other G&A costs were generally assumed to be fixed per unit of time (independent of variations in production feed). Management and accounting operating costs includes items such as communications services, commercial costs, travel costs, insurance costs, IT equipment and software, service contracts and consultant and supplier fees. Employee services costs include items such as medical equipment and supplies, training costs and safety equipment (PPE) costs.

The G&A operating cost for Phase 1 is 6.30 USD/t processed and 3.98 USD/t processed for Phase 2. The overall cash operating cost over the LOM is 5.01 USD/t.

Figure 21-7 shows a pie chart illustrating the relative importance of the different processing costs over the LOM.

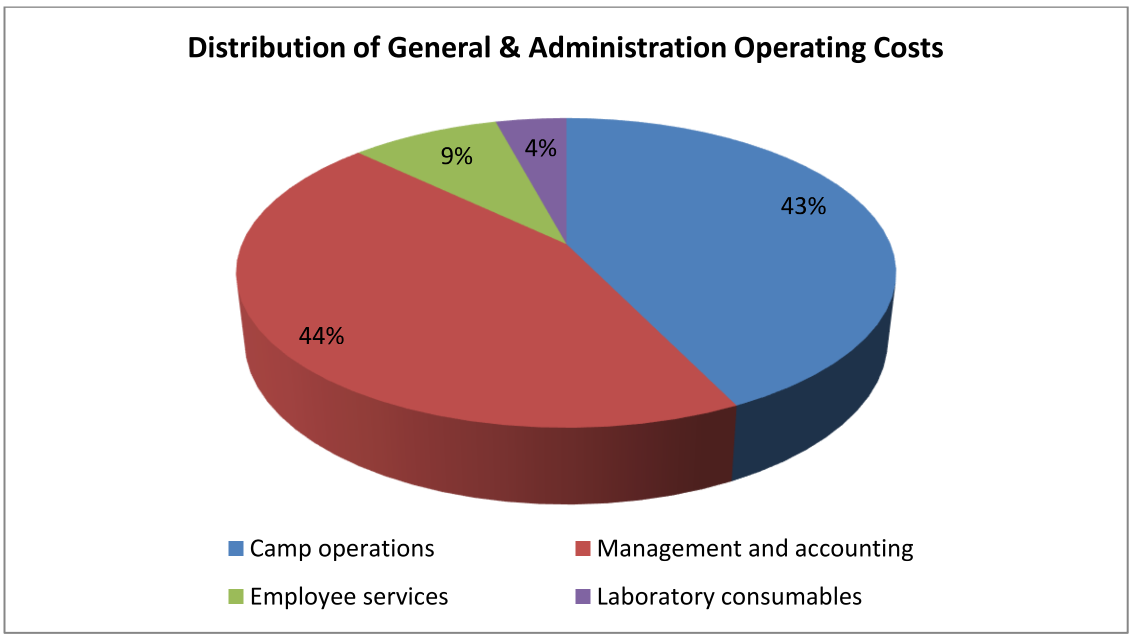


Figure 21-7: Distribution of General & Administration Operating Costs

22. ECONOMIC ANALYSIS

The economic/financial assessment of the Project of Algold is based on Q2-2018 price projections and cost estimates in U.S. currency. No provision was made for the effects of inflation. The evaluation was carried out on a 100 %-equity basis. Current tax regulations in Mauritania were applied to assess the project's state royalty liabilities. However, as it is unsure at this time whether the state will impose outright the mining code regulations associated with corporate income taxes on the benefits of the Project, these were ignored for the purpose of this preliminary assessment.

The financial indicators under base case conditions are:

Table 22-1: Financial Model Indicators

Base Case Financial Results	Unit	Value
Pre-State Royalty (PSR) NPV @ 8 %	M USD	94.9
After-State Royalty (ASR) NPV @ 8 %	M USD	69.0
PSR IRR	%	28.4
ASR IRR	%	23.5
PSR Payback Period	years	1.6
ASR Payback Period	years	1.8

A sensitivity analysis reveals that the Project's after-state royalty viability is moderately vulnerable to variations in capital and operating costs, within the margins of error associated with Preliminary Economic Analysis (PEA) estimates. However, the Project's viability remains more vulnerable to the larger uncertainty in future gold market prices.

22.1 Assumptions

22.1.1 Macro-Economic Assumptions

The main macro-economic assumptions used in the base case are given in Table 22-2. The price forecast for gold is based on observed prices over the past three (3) years. Details on the derivation of this price forecast are given in Section 19 of this Report. The sensitivity analysis examines a range of prices 30 % above and below the base case forecast.

Table 22-2: Macro-Economic Assumptions

Item	Unit	Base Case Value
Gold Price Forecast	USD/oz.	1,250
Marketing and Refining Charges	USD/oz.	15
Discount Rate	% per year	8
Discount Rate Variants	% per year	10 and 12

The Mauritanian fiscal regime for mining projects consists of a State Royalty that varies with the mineral commodity and associated market price (4 - 6.5 % of London Fix for gold), and corporate taxes at a rate of 25% of taxable income. A three-year corporate tax holiday is applicable. Depreciation of mine development expenses other than tangible assets is on a straight-line basis over a 2-year period while depreciation of all other capital expenses is on a straight-line basis over a period of 3 years. Depreciation need not be claimed during the tax holiday. State Royalty payments are deductible for the purpose of determining taxable income and operating losses can be carried forward for a period of up to five (5) years.

A Minimum Tax provision (currently 1.25% of revenues FOB Mine) applies on completion of the tax holiday period. Due to the short mine life and the lower quality of the mineral resources in the last 3½ years of production (beyond the corporate tax holiday), it is uncertain at this time whether the State will impose outright corporate taxes on the Project. For this reason, the corporate taxes (payable beyond the 3-year tax holiday) have been ignored in this evaluation, i.e., only the State Royalty payments are assessed.

The assessment was carried out on a 100 %-equity basis. Apart from the base case discount rate of 8 %, two (2) variants of 10 and 12 % were used to determine the Net Present Value of the Project. These discount rates represent possible costs of equity capital.

22.1.2 Royalty and Impact and Benefit Agreements

This Project incorporates a third-party “NSR” royalty agreement. This agreement calls for annual payments of 1.5% of revenues FOB mine. The mineral property is not subject to any agreement with local communities.

22.1.3 Technical Assumptions

The main technical assumptions used in the base case are given in Table 22-3.

Table 22-3:Technical Assumptions

Item	Unit	Value
Open Pit Resource Mined	k tonnes	9,140
Average Mill Head Grade	g/t Au	2.08
Design Milling Rate (Years 1-4 / 5-8)	k tonnes/year	1,100 / 1,650
Average Stripping Ratio	w:o	12.245
Mine Life	years	8.1
Process Recovery (Years 1-4 / 5-8)	%	96.0 /93.6
Average Gold Production (Years 1-4 / 5-8)	ounces/year	104.5 / 53.0
Average Mining Costs	($/tonne milled)	18.77
Average Processing Costs	($/tonne milled)	13.84
Average General and Administration Costs	($/tonne milled)	5.01
Average Third-party Royalty Payments	($/tonne milled)	1.18
Average Total Costs (excludes third-party royalty)	($/oz.)	591.95

The first production year consists of a ramp-up period of 3 months followed by 9 months during which production exceeds 60% of full capacity (as defined by the Canada Revenue Agency). The start of commercial production corresponds to the beginning of this nine-month period.

Infrastructure such as the pumping station, and the campsite will be turned over to the local community. The costs associated with the rehabilitation of other production-related infrastructure will be compensated by the sale of equipment and other assets.

22.2 Financial Model and Results

Figure 22-1 illustrates the after-state royalty (after royalty) cash flow and cumulative cash flow profiles of the Project for base case conditions. The Year -2 cash flow covers a 6-month period and the Year 8 cash flow, a period of 36 days. The intersection of the after-royalty cumulative cash flow curve with the horizontal dashed line represents the payback period (not adjusted for the 3 month ramp-up).

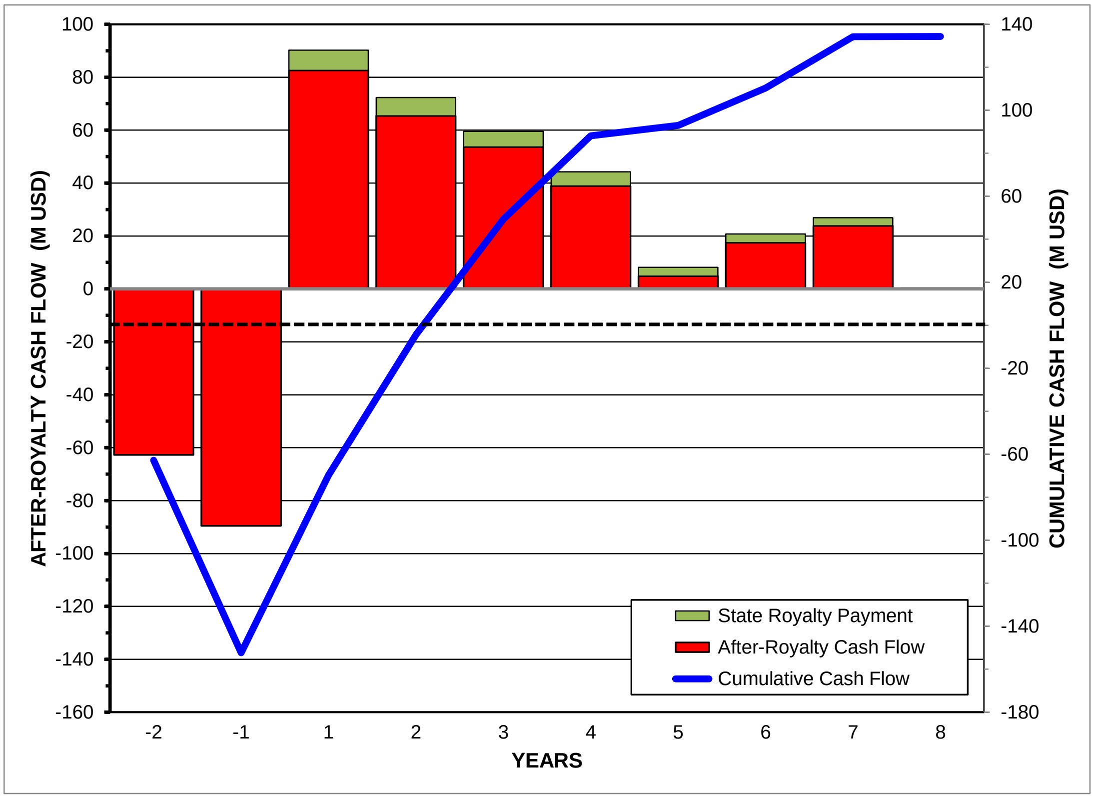


Figure 22-1: After-Royalty Cash Flow and Cumulative Cash Flow Profiles

A summary of the evaluation results is given in Table 22-4 and Table 22-5 gives the cash flow statement, both for base case conditions.

The summary and cash flow statement indicate that the total pre-production (initial) capital costs were evaluated at $ 161.8 M (including Phase 2). The sustaining capital requirement was evaluated at $ 30.8 M. Mine closure costs are assumed to be compensated by the sale of salvageable equipment and other assets.

The cash flow statement shows the estimated capital spending schedule over the 18-month pre-production period of the Project (the Year -2 column covers a 6-month period). Working capital requirements were estimated at two (2) months of total annual operating costs. Since operating costs vary annually over the mine life, additional amounts of working capital are injected or withdrawn as required.

The total revenue derived from the sale of the gold was estimated at $ 717.4 M, or on average, $ 78.49/tonne milled. The total operating costs were estimated at $ 343.9 M (excludes third-party royalty), or on average, $ 37.62/tonne milled.

The financial results indicate a pre-state royalty Net Present Value ("**NPV**") of $ 94.9 M at a discount rate of 8 %. The pre-state royalty Internal Rate of Return ("**IRR**") is 28.4 % and the payback period is 1.6 years. The payback period is measured from the start of commercial production and consequently excludes the ramp-up period in production year 1.

The after-state royalty NPV is $ 69.0 M at a discount rate of 8 %. The after-state royalty IRR is 23.5 % and the payback period is 1.8 years.

Table 22-4: Project Evaluation Summary – Base Case

Item	Unit	Value
Total Revenue	M USD	717.4
Total Operating Costs (excludes third-party royalties)	M USD	343.9
Total Third-party Royalty Payments	M USD	10.8
Initial Capital Costs (excludes Working Capital)	M USD	161.8
Sustaining Capital Costs	M USD	30.8
Total Pre-state Royalty (PSR) Cash Flow	M USD	170.2
PSR NPV @ 8 %	M USD	94.9
PSR NPV @ 10 %	M USD	80.8
PSR NPV @ 12 %	M USD	68.1
PSR IRR	%	28.4
PSR Payback Period*	Years	1.6
Total After-state Royalty (ASR) Cash Flow	M USD	134.3
ASR NPV @ 8 %	M USD	69.0
ASR NPV @ 10 %	M USD	56.8
ASR NPV @ 12 %	M USD	45.8
ASR IRR	%	23.5
ASR Payback Period*	Years	1.8

* Measured from the start of commercial production

Note: The mid-period convention is assumed in the calculation of Net Present Values.

Table 22-5: Cash Flow Statement – Base Case

TIJIRIT PROJECT – Algold Resources Ltd.

All monetary values in M USD	-2	-1	1	2	3	4	5	6	7	8	Total
Gold Production (troy ounces)			123,643	111,998	95,410	86,974	54,785	53,563	50,736	3,791	580,901
Revenues			**152.7**	**138.3**	**117.8**	**107.4**	**67.7**	**66.2**	**62.7**	**4.7**	**717.4**
Mining Costs			21.3	34.4	37.2	18.8	27.1	18.1	13.9	0.8	171.5
Processing Costs			13.6	15.8	15.8	15.9	20.8	20.8	20.8	2.9	126.5
General & Administration Costs			6.1	6.6	6.6	6.3	6.5	6.2	6.0	1.4	45.8
Third-Party Royalty Payments			2.3	2.1	1.8	1.6	1.0	1.0	0.9	0.1	10.8
Total Operating Costs			43.3	58.9	61.4	42.6	55.5	46.1	41.7	5.2	354.6
Operating Profit			**109.4**	**79.4**	**56.4**	**64.9**	**12.2**	**20.0**	**21.0**	**-0.5**	**362.8**
State Royalty Payments			7.6	6.9	5.9	5.4	3.4	3.3	3.1	0.2	35.9
Capital Costs	62.8	82.7				16.3					161.8
Sustaining Capital Costs			16.5	6.6		2.1	5.6				30.8
Working Capital		6.8	2.6	0.5	-3.1	2.3	-1.6	-0.7	-5.9	-0.9	0.0
CASH FLOW (pre state royalties)	**-62.8**	**-89.6**	**90.2**	**72.3**	**59.5**	**44.2**	**8.2**	**20.7**	**26.9**	**0.3**	**170.2**
CASH FLOW (after state royalties)	**-62.8**	**-89.6**	**82.6**	**65.4**	**53.6**	**38.9**	**4.8**	**17.4**	**23.8**	**0.1**	**134.3**

22.3 Sensitivity Analysis

A sensitivity analysis has been carried out, with the base case described above as a starting point, to assess the impact of changes in total pre-production capital expenditure ("**Capex**"), operating costs ("**Opex**") and product price ("**Price**") on the Project's NPV @ 8 % and IRR. Each variable was examined one-at-a-time. An interval of ±30 % with increments of 10 % was used for the three (3) variables.

The pre-royalty results of the sensitivity analysis, as shown in Figure 22-2 and Figure 22-3, indicate that, within the limits of accuracy of the cost estimates in this Study, the Project's pre-royalty viability does not seem significantly vulnerable to the under-estimation of capital and operating costs, taken one at-a-time. As seen in Figure 22-2, the NPV is more sensitive to variations in Opex than Capex, as shown by the steeper slope of the Opex curve. As expected, the NPV is most sensitive to variations in price. The NPV becomes negative at a price variation of about -19 %. This corresponds to a break-even gold price of about $1,020/oz.

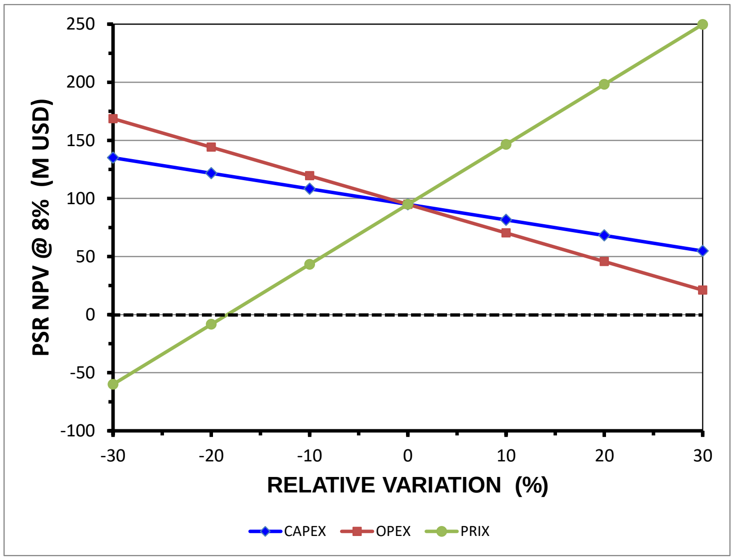


Figure 22-2: Pre-state Royalty NPV$_{8\ \%}$:
Sensitivity to Capital Expenditure, Operating Cost and Price

Figure 22-3, showing variations in internal rate of return, provides the same conclusions. The horizontal dashed line represents the base case discount rate of 8%. Because of the different timing associated with Capex versus Opex, the IRR is more sensitive to variations in Capex than Opex for negative variations, but remains less sensitive to Capex for positive variations. The IRR becomes significantly sensitive to variations in price at the lower end of the price interval.

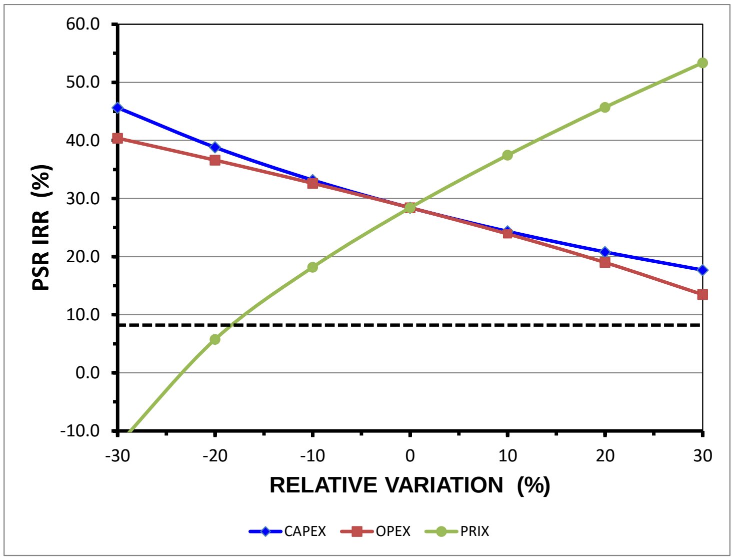


Figure 22-3: Pre-state Royalty IRR: Sensitivity to Capital Expenditure, Operating Cost and Price

The same conclusions can be made from the after-royalty results of the sensitivity analysis as shown in Figure 22-4 and Figure 22-5. Figure 22-4 indicates that the Project's after-tax viability is mostly vulnerable to a price forecast reduction, while being less affected by the under-estimation of capital and operating costs. The NPV becomes negative at a price variation of about -14% and marginally negative at the upper limit of the operating cost variation interval. These variations correspond to a break-even gold price of about $1,070/oz. and an operating cost exceeding $48.20/t milled (excluding third-party royalty) as compared to the base case value of $37.62/t milled.

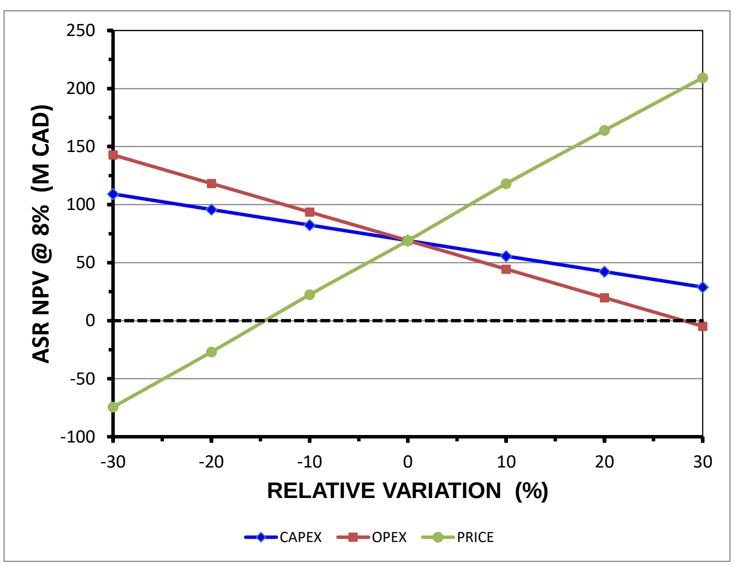


Figure 22-4: After-state Royalty NPV$_{8\ \%}$:
Sensitivity to Capital Expenditure, Operating Cost and Price

Figure 22-5, showing variations in internal rate of return, provides the same conclusions.

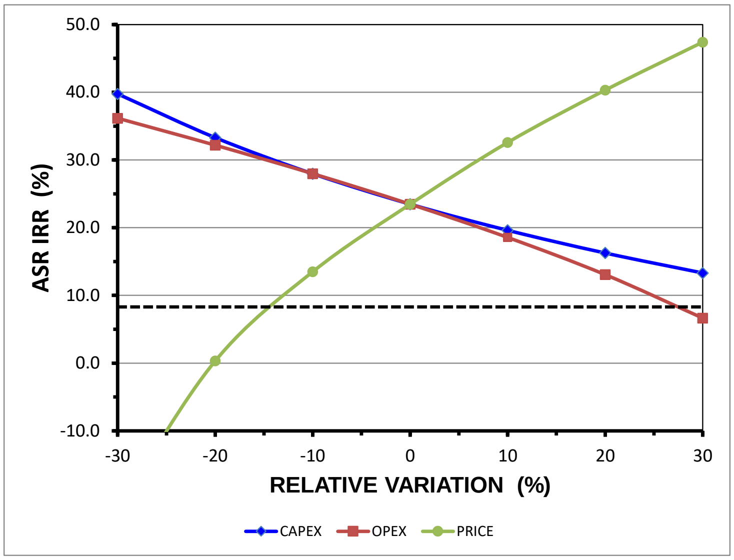


Figure 22-5: After-state Royalty IRR:
Sensitivity to Capital Expenditure, Operating Cost and Price

23. ADJACENT PROPERTIES

According to RIM mining license database for 2017, three exploration permits are adjacent to the Project, Table 23-1. Two of the permits belongs to Mauritanian local owner and one to Kinross through Tasiast Mauritania. The properties are located in the granitic-volcanic complexes of the southwestern Reguibat Archean Ridge and are partly covered by a thick sand cover of Azéfal dune bar to the northwest and of Akchâr to the southeast of Tijirit.

437B2 - Kinross conducted exploration works on the northern part of 437B2 permit only (Sims, 2014), but the southern portion adjacent to the Tijirit permit is almost entirely covered with sand prohibiting any exploration works.

2194B2 - the Gulf Mining permit shows NNE-trending continuity with the Archean meta volcanic, sedimentary and igneous assemblages of the eastern Tijirit. The sands of Azéfal and Akchâr cover the northern and southern parts of the permit, Figure 23-1.

2503B2 - the Quark 74 permit shows no geological continuity with Tijirit's 1117B2 permit to the north. It is almost half covered by of the sand of the Akchâr.

At the time of this Report, apart from Kinross (Tasiast Mauritania), no information on the various exploration work carried out by the different owners was available.

Table 23-1: Adjacent Properties and Permit Owners

Permit Nb	Permit Owner	Area km^2	Sand Cover %
437B2	Tasiast Mauritania	1478	30
2194B2	Gulf Mining	460	29
2503B2	Quark 74	498	45

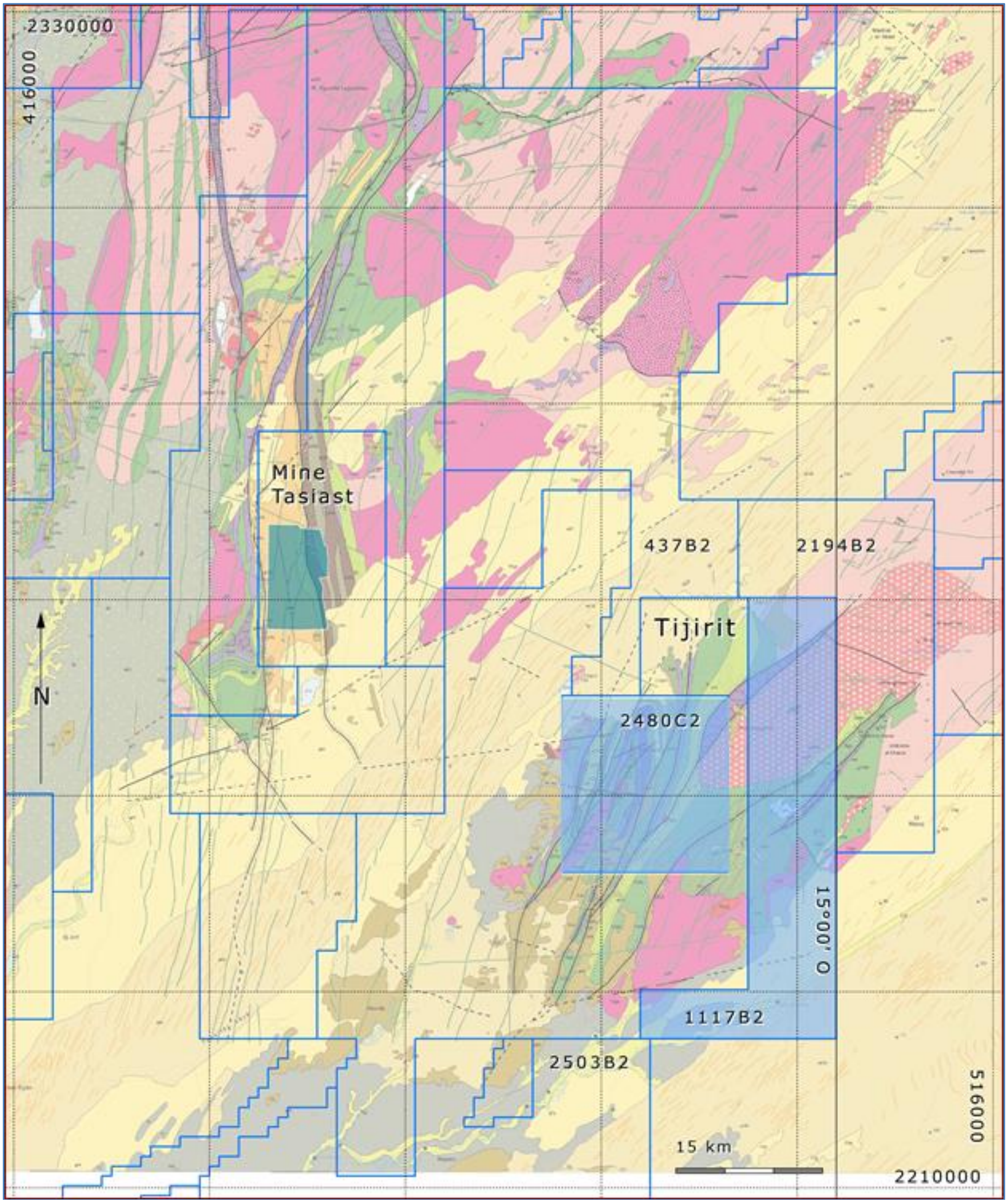


Figure 23-1: Tijirit (2480C2) and Tijirit-East (1117B2) permits and Adjacent Permits on the 1:200,000 Geological Map. RIM Mining & Exploration Licenses as of 2017

24. OTHER RELEVANT DATA AND INFORMATION

There are no other relevant data or information to present.

25. INTERPRETATION AND CONCLUSIONS

The Tijirit project is comprised of the Tijirit mining concession 2480C2, covering 306 km^2, and of the adjacent exploration permit 1117B2, covering 460 km^2. The project is located in western Mauritania, in the southwestern Precambrian Reguibat shield and shows mostly Archean granite greenstone terrains. The project shows sheared and folded NNE-trending Archean metasedimentary and intermediate to ultramafic volcanic rocks in contact with porphyry, syn-tectonic granitoids and basement quartzo-feldspathic gneiss. The project was acquired by Shield Mining in 2007 after the discovery of gold mineralization and transferred to Gryphon Minerals in 2010 and acquired by Algold in March 2016.

A fair amount of exploration works was carried out by past owners including a large soil survey, an airborne magnetic and radiometric survey, trenches, rock samples, auger drilling and 37,703 m of RC and 3,814 m of core drilling. Large gold soil anomalies were revealed in five prospect areas and a limited IP survey was done on the Sophie prospect.

Since the acquisition of the project, Algold has completed over 50,000 metres of core drilling and reverse circulation drilling in the Tijirit project. Algold has also prepared an environmental and social impact study that was submitted to the Mauritanian government in 2016. SGS Lakefield was commissioned to implement two metallurgical test programs in 2016 and 2017. In February 2017, the Minister Environment and Sustainable Development of Mauritania has agreed to the environmental feasibility of the Tijirit gold mining project.

To date, Algold has been able to substantially increase resources on the project, especially on Eleonore and to identify promising new areas (Salma, Eleonore East, Nour, Southeast). Efforts over the next few months will be required to better define these new areas to hopefully grow the overall resources and to increase drilling density on already known areas in order to transform the majority of existing inferred resources into indicated or measured resources.

Mining

The current Tijirit project results in an annual mining production of 1.1 Mt of ROM material per year at 3.3 g/t of gold during Phase 1 and approximately 1.6 Mt of ROM material per year at 1.1 g/t of gold during Phase 2. The total In-Pit Resources, spread between the Eleonore, Sophie and Lily zones, include 9.14 Mt of Mineralized Material grading 2.08 g/t of gold. The ROM material and Waste rock will be mined at 5 m high benches drilled, blasted and loaded into rigid quarry haul trucks with hydraulic excavators.

Mineral Processing

To date, two (2) metallurgical testwork campaigns have been performed on the Tijirit gold project. Both campaigns demonstrated that all deposits within the Tijirit property responded well to a traditional grinding – gravity concentration – cyanidation process. The process plant design criteria have been selected according to the metallurgical testwork results. The resulting process plant has been designed for a throughput of 1.09 Mt/y in Phase 1, with an expansion to 1.64 Mt/y in Phase 2. Gold recovery is expected to be 96.0% for Phase 1 and 93.6% for Phase 2, generating between 50,0000 and 120,000 oz of doré per year. Additional testwork should be performed in subsequent project phases to refine reagents and air consumption as well as thickening and filtration parameters.

Algold has also started a research program for water to supply the Project as well as a program of geotechnical studies to better define the parameters related to the stability of the slopes of the pits of the future mining operation.

There are no uncertainties or risks related to the information inspected by the authors other than standard risks in the mining industry like price drops and such.

Many studies and additional work will have to be carried out to validate and optimize the scenarios presented in this Report. The main steps to be completed before the start of production will be:

- Updating the resource estimation;
- Fieldwork;
- Geological, metallurgical, geotechnical and hydrogeological studies;
- Analysis of variants;
- Pre-Feasibility and Feasibility Studies;
- Updating the ESIA;
- Grant and fulfilment of an EPCM mandate;
- Basic Engineering (FEED) and early works;
- Construction;
- Pre-commissioning, Commissioning, Start-up, and Ramp up to full Production.

In conclusion, the work done to date has demonstrated the potential of the Tijirit Project and the preliminary economic analysis foresees positive outcome. Several recommendations are made (Section 26) to continue the development of this Project in order to fully assess its potential.

26. RECOMMENDATIONS

Considering the positive outcome of the PEA, it is recommended to pursue the definition of the Project through various aspects in order to get sufficient data to produce a Pre-Feasibility Study (PFS).

26.1 Resources

The following development plan is recommended for the Project:

- Complete infill drilling to cover current resources. The aim is to increase the quantity of indicated resources (and possibly measured);
- Focus on in-pit resources first as they are closer to surface, probably easier to extract, and they require shorter drillholes so much less costly drilling;
- Explore new recent discoveries that are not at the resource development stage including mapping, structural study, additional soil survey, trenching, additional geophysics, hydrological and geotechnical studies, and preliminary drilling;

It is assumed that between 20,000 to 50,000 metres will be required for the next drilling phase.

26.2 Mining

The following items are recommended for consideration in the future studies:

- Consider the use of potential drill and blast of mining contractor in order to lower the initial capital requirement
- Perform a more detailed dilution study, especially for the Eleonore mineralized zone
- The Eleonore zone mineralization being open at depth, an underground operation could be considered if confirmed by further drilling

26.3 Geotechnical

The following recommendations are provided for consideration for future studies:

- It is recommended that a 3D geological and lithological model, including identification of pit size and regional structural features such as contact, faults and shears, be completed prior to future geotechnical studies.
- It is recommended that the assumptions for the structural regime and orientation of structures be verified with future geotechnical data collection. This could include line mapping of trenching to validate the orientations of foliation and joint discontinuities, as well as to assess any additional structural discontinuities (faults/shears). Due to the lack of this data, there is uncertainty in the 3D geotechnical model and associated pit design domains.
- It is recommended that rock strength data collection be completed for direct shear, compressive strength and triaxial strength tests to add to the geotechnical dataset. Rock strength testing should also collect data associated with density of various lithologies. Rock strength, cohesion and friction angles for all rock types were assumed for this study based on empirical studies and field descriptions.

- For future data collection purposes, detailed descriptions of all discontinuity conditions (veins, foliation, fractures and faults/shears) including spacing of defects, thickness of alteration should be detailed.
- Overburden soils logging and collection including in situ densities, sieve tests, and depth should be completed.
- Collection of fracture frequency and spacing is required to provide a better understanding of the quantity and variability of discontinuities.
- Location and size of any waste dumps associated with the planned pits should be included in future stability assessments to determine the effect of surface loading in proximity to the pits.
- Continued collection of hydrogeological data, including recording lack of water presence in drillholes to fully characterize expected hydrogeological conditions should be completed.

Stability of the final bench faces can be improved using controlled blasting techniques, scaling of loose rock from the bench faces, and removal of loose on benches after blasting. This design has an operational constraint which includes controlled blasting considerations.

Extensive monitoring of pit slopes and ongoing commitment to data collection throughout the life of the Project is required to ensure design appropriateness and to validate assumed design parameters.

A more detailed geological understanding of the location and continuity of weak or altered zones, including weathered zones within the pit are required to fully assess the pit stability. Further drilling, as well as Televiewer (geocamera) studies or oriented core collection and geological interpretation are required.

26.4 Processing Plant and Process

The development of the Tijirit process plant at this phase of development has led to several recommendations for project improvements in subsequent phases. A comparative study related to the grinding circuit was carried out by Ausenco. This study considered three different grinding circuits, for the two phases of the project:

- A conventional circuit that was selected for this phase of the project and which includes a semi-autogenous mill with the addition of a ball mill for the second production stage.
- An alternate circuit that included a ball mill with the addition of a high pressure grinding rolls press to increase mill capacity in the second production phase.
- A modified alternative circuit that included a smaller SAG mill than the reciprocating circuit for Phase 1, and the addition of a high pressure grinding rolls and a second ball mill for Phase 2.

The conclusion of this study was to maintain the conventional circuit at this stage of the Project but to continue to explore alternative scenarios in the next steps of the project aiming for greater cost accuracy. In fact, the net present values of the three scenarios proposed are very similar at this study level, slightly higher in the case of the alternative scenario and the modified alternative scenario. Given the accuracy of the current study, it is not possible to conclude that one scenario is more economically advantageous than the other.

Ausenco believes that using a high-pressure grinding rolls could have several benefits for the project, including:

- A possible reduction in plant water consumption in Phase 2, HPGRs operating almost entirely dry.
- Few modifications to Phase 1 facilities would be required to proceed to Phase 2 as opposed to the conventional scenario. This would reduce the transition time between the two phases which could be important for a short-term project like this one.

In order to better evaluate this technology, it will be necessary to test this equipment to determine the capacity and specific energy requirements for this equipment processing Tijirit ore.

The metallurgical tests performed at SGS, and the development of the process by Ausenco, have also led to several other process recommendations:

- The addition of a flotation circuit on the tailings discharge of the gravity circuit. Preliminary tests for this purpose have already been carried out on Eleonore material and have demonstrated recoveries above 90%. More testing of all project zones should be undertaken to better understand the potential of this process for the project.
- Additional metallurgical tests at the CIL circuit in order to optimize the residence time of this process, determine the optimal oxygen and reagent consumptions of this circuit.
- The preparation of a comparative study on the use of thickeners in the process. For the moment, a tailings thickener has been included in the process. A thickener could also be used upstream of the CIL circuit (Pre-Leach Thickener) which could reduce the size of the CIL tanks.
- Conduct sedimentation and filtration tests to optimise sizing of the process thickeners and tailings disc filters. These facilities have been sized on the basis of similar projects in the region.
- Samples for future test programs should be based on the major lithological units of each deposit as defined in the resource and block models. A review of the geology and mineralization of each lithological unit should be conducted to determine the range of variability within each unit to select possible variability samples.

26.5 Tailings Stockpile and Waste Dumps

The following are recommendations for the management strategies to be reviewed and the investigations to complete for the tailings storage facility:

- Review the location of the tailings site in light of dominant winds and consider relocating it to reduce exposure of mine personnel to fugitive dust.
- Conduct geotechnical and borrow pit investigation at the future tailings and waste rock sites to determine suitability of foundation ground and locate potential construction materials
- Determine maximum design flood for runoff collection and channel sizing.
- Proctor testing on filtered tailings to obtain optimum bulk density and maximum density at filtered moisture contents.
- Review fugitive dust management and crust/hard-pan forming behaviour of filtered tailings
- ARD screening: Comprehensive static geochemical tests (NAG, some sequential NAG, modified Sobek ABA,) on tailings and waste rock material using a suitable number of representative samples selected by an environmental engineer or geologist.
- Revisit the requirements for membranes and seepage drainage below the waste rock and tailings piles following review of geochemical results.

26.6 Proposed Work Program Budget

The expected cost to complete the above-mentioned exploration, drilling and sites costs, and metallurgical tests, is between $5 and $8.3 M US; a high-level budget recommendation is outlined in Table 26-1.

Table 26-1: Recommended Budget

Description	$US
Drilling Costs (additional infill program): Total of between 20,000 and 50,000 m of RC and DDH drilling with assays and logging (75 % RC, 25 % DDH). This drilling should be enough to cover current resources with infill drilling. This will help for future technical/economical studies.	2,200,000 to 5,500,000
Site Costs: Including camp costs, salaries and transportation costs	1,000,000
Metallurgical Tests: Heap Leach amenability test Gravimetric and cyanidation tests Comminution tests Environmental Testwork	500,000
Pre-Feasibility Study (PFS) including: Resources Update Process engineering Mine engineering	500,000
Other Suggested Budgets: Mapping, structural study, additional soil survey, trenching, additional geophysics, hydrological and geotechnical study	800,000
TOTAL (varies depending on the infill drilling required to delineate enough resources to an indicated or measured resources)	5,000,000 to 8,300,000

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28. ABBREVIATIONS

The following abbreviations may be used in this Report.

Table 28-1: List of Abbreviations

Abbreviation	Term or Unit
µg/m³	Microgram per Cubic Metre
µm	Microns, Micrometre
'	Feet
"	Inch
$	Dollar Sign
$/m²	Dollar per Square Metre
$/m³	Dollar per Cubic Metre
$/t	Dollar per Metric Tonne
%	Percent Sign
% w/w	Percent Solid by Weight
°	Degree
°C	Degree Celsius
AI	Abrasion Index
AMSL	Above Mean Sea Level
ARD	Acid Rock Drainage
ASL	Above Sea Level
Au	Gold
az	Azimuth
BGS	British Geological Survey
BIF	Banded Iron Formation
BRGM	*Bureau de recherches géologiques et minières*
BSG	Bulk Specify Gravity
BTU	British Thermal Unit
BWI	Bond Ball Mill Work Index
CAD	Canadian Dollar
$Ca(OH)_2$	Hydrated Lime
Capex	Capital Expenditures
CDP	Closure and Decommissioning Plan
cfm	Cubic Feet per Minute

Abbreviation	Term or Unit
CFR	Cost and Freight
CIF	Cost Insurance and Freight
CIL	Carbon in Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	Centimetre
COG	Cut-Off Grade
CRM	Certified Reference Materials
d	Day
d/w	Days per Week
d/y	Days per Year
DDH	Diamond Drill Hole
deg	Angular Degree
DMS	Dense Media Separator
DT	Davis Tube
DWI	Drop Weight Index
DWT	Drop Weight Test
E	East
EBS	Environmental Baseline Study
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EOH	End of Hole
EPCM	Engineering, Procurement and Construction Management
ESBS	Environmental and Social Baseline Study
ESIA	Environmental and Social Impact Assessment
FOB	Free on Board
ft	Feet
g	Grams
G&A	General and Administration
g/l	Grams per Litre
g/t	Grams per Tonne
gal	Gallons
GCW	Gross Combined Weight

Abbreviation	Term or Unit
GPS	Global Positioning System
Gr	Granular
H	Horizontal
h	Hour
h/d	Hours per Day
h/y	Hour per Year
H_2	Hydrogen
ha	Hectare
HCl	Hydrochloric Acid
HDPE	High Density Polyethylene
HG	High Grade
HP	Horse Power
HQ	Drill Core Size (6.4 cm Diameter)
HVAC	Heating Ventilation and Air Conditioning
Hz	Hertz
I/O	Input / Output
ID	Identification
ILR	InLine Leach Reactor
in.	Inches
IRR	Internal Rate of Return
KE	Kriging Efficiency
kg	Kilogram
kg/l	Kilogram per Litre
Kg/t	Kilogram per Metric Tonne
kl	Kilolitre
km	Kilometre
km/h	Kilometre per Hour
kPa	Kilopascal
KSR	Kriging Slope Regression
kt	Kilotonne
kV	Kilovolt
kVA	Kilovolt Ampere
kW	Kilowatt

Abbreviation	Term or Unit
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per Metric Tonne
L	Line
l	Litre
l/h	Litre per hour
lbs	Pounds
LG-3D	Lerchs-Grossman – 3D Algorithm
LIMS	Low Intensity Magnetic Separator
LOI	Loss on Ignition
LOM	Life of Mine
LV	Low Voltage
m	Metre
m/h	Metre per Hour
m/s	Metre per Second
m^2	Square Metre
m^3	Cubic Metre
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/y	Cubic Metre per Year
mA	Milliampere
masl	Meter above sea level
MCC	Motor Control Center
mg/l	Milligram per Litre
MI	Mineralized Intervals
min	Minute
min/h	Minute per Hour
min/shift	Minute per Shift
ml	Millilitre
ML	Metal Leaching
MLA	Mineral Liberation Analyzer
mm	Millimetre
mm/d	Millimetre per Day
Mm^3	Million Cubic Metres
MMER	Metal Mining Effluent Regulation

Abbreviation	Term or Unit
MMU	Mobile Manufacturing Units
MOLP	Multiple Objective Linear Programming
MOU	Memorandum of Understanding
Mt	Million Metric Tonnes
Mt/y	Million of Metric Tonnes per year
MV	Medium Voltage
MVA	Mega Volt-Ampere
MW	Megawatts
MWh/d	Megawatt Hour per Day
My	Million Years
N	North
NaCN	Sodium Cyanide
NAG	Non Acid Generating
NaOH	Sodium Hydroxide
Nb	Number
NE	Northeast
NI	National Instrument
Nm^3/h	Normal Cubic Metre per Hour
NPV	Net Present Value
NSR	Net Smelter Return
NW	North West
OB	Overburden
O/F	Overflow
OK	Ordinary Kriging
Opex	Operating Expenditures
oz	Troy ounce (31.1035g)
P&ID	Piping and Instrumentation Diagram
PAPs	Project Affected Persons
PEA	Preliminary Economic Assessment
PF	Power Factor
PFS	Pre-Feasibility Study
ph	Phase (electrical)
pH	Potential Hydrogen

Abbreviation	Term or Unit
ppb	Part per Billion
ppm	Part per Million
PRISM	*Projet de Renforcement Institutionnel du Secteur Minier*
PR	*Permis de recherche*
psi	Pounds per Square Inch
PVC	Polyvinyl Chloride
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
RAP	Resettlement Action Plans
RC	Reverse Circulation
RCMS	Remote Control and Monitoring System
RIM	*République Islamique de Mauritanie*
ROM	Run of Mine
rpm	Revolutions per Minute
RQD	Rock Quality Designation
RWI	Bond Rod Mill Work Index
S	South
S/R	Stripping Ratio
SAB	SAG Mill and Ball Mill
SAG	Semi-Autogenous Grinding
scfm	Standard Cubic Feet per Minute
SE	South East
sec	Second
Set/y/unit	Set per Year per Unit
SG	Specific Gravity
SGS Geostat	SGS Canada Inc.
SGS Lakefield	SGS Lakefield Research Limited of Canada
SMC	SAG Mill Comminution
SNIM	*Société Nationale Industrielle et Minière*
SOP	Standard Operating Procedures Manual
SPI	SAG Power Index
SPLP	Synthetic Precipitation Leaching Procedure
SR	Stripping Ratio

Abbreviation	Term or Unit
SSSAG	Single Stage SAG mill circuit
SW	South West
t	Metric Tonne
t/d	Metric Tonne per Day
t/h	Metric Tonne per Hour
t/h/m	Metric Tonne per Hour per Metre
t/h/m²	Metric Tonne per Hour per Square Metre
t/m	Metric Tonne per Month
t/m²	Metric Tonne per Square Metre
t/m³	Metric Tonne per Cubic Metre
t/y	Metric Tonne per Year
TCLP	Toxicity Characteristic Leaching Procedure
TER	*Travail d'Études et de Recherches*
ton	Short Ton
tonne	Metric Tonne
ToR	Terms of Reference
U/F	Under Flow
USA	United Stated of America
USD	United States Dollar
USGPM	Us Gallons per Minute
UTM	Universal Transverse Mercator
V	Volt
W	Watt
W	West
X	X Coordinate (E-W)
y	Year
Y	Y coordinate (N-S)
Z	Z coordinate (depth or elevation)

29. CERTIFICATES OF QUALIFIED PERSONS

Ausenco

CERTIFICATE OF QUALIFIED PERSON

Thomas Zwirz, P. Eng.

To Accompany the Report entitled, "Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania" prepared for Algold Resources Ltd. and dated July 4, 2018.

I, *Thomas Zwirz, P. Eng.,* do hereby certify:

1. I am a Senior Study Manager at Ausenco Engineering Canada, 555, boul. René-Levesque Ouest, Suite 200 Montréal, QC, H2Z 1B1.
2. I hold a Bachelor's degree in Chemical Engineering from McGill University of Montreal, Quebec, Canada.
3. I the am registered as a Professional Engineer with Ontario and Quebec.

   I have worked for more than *25* years in the mining industry in various positions continuously since my graduation from university.
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 by 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 have not visited site.
6. I have participated in the preparation of this Technical Report and am responsible for Sections 2 to 3, 18 (except for Section 18.5) to 20, 21.2 (except for 21.2.3 and 21.2.4), 22, and 24 and parts of Sections 1 and 25 to 27.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of July 2018

"Original document signed and sealed"

Thomas Zwirz, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Tommaso Roberto Raponi, P. Eng.

To Accompany the Report entitled, “Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania” prepared for Algold Resources Ltd. effective date March 9, 2018 and dated July 4, 2018.

I, *Tommaso Roberto Raponi, P. Eng.,* do hereby certify:

1. I am a Senior Mineral Processing Specialist at Ausenco Engineering Canada Inc., 11 King St West, Suite 1550, Toronto, ON, M5H 4C7.
2. I hold a Bachelor's degree in Geological Engineering from University of Toronto, Toronto, Ontario, Canada.
3. I am registered as Professional Engineers in Ontario and British Columbia. I have worked for more than *34* years in the mining industry in various positions continuously since my graduation from university. I have worked as an independent consultant since 2016;
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 by 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 have not visited site.
6. I am responsible for Sections 13, 17, 21.2.4, and parts of Sections 1 and 25 to 27 of the Technical Report.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of July 2018

“Original document signed and sealed”

Tommaso Roberto Raponi, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

David Sims, Géo.

To Accompany the Report entitled, “Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania” prepared for Algold Resources Ltd. and dated July 4, 2018.

I, *David Sims, Géo.,* do hereby certify:

1. I am a Principal at David Sims Inc., 45, 5E Ave sud, Montreal, QC, H8Y 2T6.
2. I hold a Bachelor's degree in Science from University of Victoria, British Columbia, Canada.
3. I am registered as a Professional Geoscientist in Quebec. I have worked as a geoscientist for more than 15 years in the mining industry in various positions. I am currently the President of David Sims Inc., as hydrogeologist and water management specialist, and working under a service agreement contract with Ausenco Engineering Canada. Prior to David Sims Inc., I was a Hydrogeologist at Norda Stelo and AMEC.

   My relevant experience for the purpose of the Technical Report is in tailings management for mines throughout the United States and Canada.
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 by 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 have not visited site.
6. I am the QP responsible for Sections 18.5 and contributed to Section 26 of the Technical Report;
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of July 2018

“Original document signed and sealed”

David Sims, Géo.

CERTIFICATE OF QUALIFIED PERSON

Yann Camus, P. Eng.

To Accompany the Report entitled, “Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania” prepared for Algold Resources Ltd. and dated July 4, 2018.

I, *Yann Camus, P. Eng.,* do hereby certify:

1. I am a Mineral Resource Engineer for SGS Canada Inc. - Geostat with an office at 10, boul. de la Seigneurie Est, Suite 203, Blainville Quebec Canada, J7C 3V5.
2. I am a graduate of the École Polytechnique de Montréal (B.Sc. Geological Engineer, in 2000).
3. I am a member of good standing, No. 125443, of the l'Ordre des Ingénieurs du Québec (Order of Engineers of Quebec).

   My relevant experience includes continuous mineral resource estimation since my graduation from University including many gold projects.
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 by 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. My most recent visit of the site is from August 14 to August 18, 2017.
6. I am responsible for Sections 4 to 12, 14, 23, and part of Sections 1 and 25 to 27 of the Technical Report.
7. My prior involvement with the project is the preparation of maiden resources and supporting technical report dated August 4, 2016. I also prepared a resource update and supporting technical report named “Tijirit Property NI 43-101 Technical Report with Resource Estimate Update, Tijirit, Mauritania”, dated June 15, 2017 with two effective dates: March 17, 2017 for Eleonore zone and November 4, 2016 for Sophie and Lily zones. Aa well as, the technical report entitled “Tijirit Project NI 43-101 Technical Report with Resource Estimate Update” dated April 10, 2018 (the “Technical Report”) with an effective date of January 19, 2018.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of July 2018

“Original document signed and sealed”

Yann Camus, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Patrick Pérez, P. Eng.

To Accompany the Report entitled, "Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania" prepared for Algold Resources Ltd. and dated July 4, 2018.

I, *Patrick Pérez, P. Eng.,* do hereby certify:

1. I am Senior Mining Engineer with Met-Chem, a division of DRA Americas Inc, with an office at 555 René-Lévesque Blvd. West, 6th Floor, Montreal, Canada.
2. I am a graduate from "Ecole Nationale Supérieure de Géologie de Nancy", in France, with a M.Sc. in Geological Engineering in 2003.
3. I am a registered member of APEGS (Association of Professional Engineers and Geoscientists of Saskatchewan), membership #16131.

   I have worked as a Mining Engineer or Project Manager continuously since my graduation from university.
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 by 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 have not visited the site.
6. I am responsible for Section 16 (except for 16.2.2 - Geotechnical Pit Slope Parameters), 21.2.3, and part of Sections 1, 25, and 26.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of July 2018

"Original document signed and sealed"

Patrick Pérez, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Martin Saint-Amour, P. Eng.

To Accompany the Report entitled, "Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania" prepared for Algold Resources Ltd. and dated June 29, 2018.

I, *Martin Saint-Amour, P. Eng.,* do hereby certify:

1. I am Senior Estimator with Met-Chem, a division of DRA Americas Inc, with an office at 555 René-Lévesque Blvd. West, 6th Floor, Montreal, Canada.
2. I hold a Bachelor's degree in Engineering from École Polytechnique de Montréal, Montreal, Quebec, Canada.
3. I am a member of the Ordre des Ingénieurs du Québec (affiliation #116377).
4. I have worked as an engineer for over 20 years in the mining and metallurgy industry in various positions. Prior to joining Met-Chem, I was a lead estimator at Ausenco Engineering Canada and SNC-Lavalin Inc. My relevant experience for the purpose of the Study is in estimating on projects' execution as well as studies in for the mining and metallurgy industry.
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 by 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 not visited the site.
7. I am responsible for Section 21.1 and part of Section 1.18.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
11. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of July 2018

"Original document signed and sealed"

Martin Saint-Amour, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Jean-Sébastien Houle, P. Eng.

To Accompany the Report entitled, “Technical Report for the Preliminary Economic Assessment of the Tijirit Project in Mauritania” prepared for Algold Resources Ltd. and dated July 4, 2018.

I, *Jean-Sébastien Houle, P. Eng.,* do hereby certify:

1. I am Project Manager (Geomechanical and Mining Environment) with WSP Canada Inc. at 171, rue Léger, Sherbrooke, Quebec, J1L 1M2 Canada.

   I am a graduate from Université Laval, Québec, Canada, with a Coop Bachelor’s degree in Mining and Mineral Processing Engineering obtained in 2000. I am a member in good standing of the Ordre des Ingénieurs du Québec (no. 129263). My relevant experience includes 14 years of experience in the mining industry with many projects involving geotechnical studies.

2. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that due to my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
3. I have visited the site November 21 to 24, 2017.
4. I am responsible for Section 16.2.2 and part of Sections 1, 2, 25, and 26.
5. I have not had prior involvement with the property that is the subject of the Technical Report.
6. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
7. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
8. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of July 2018

“Original document signed”

Jean-Sébastien Houle, P. Eng.

30. ANNEX I – LIST OF DIAMOND DRILLHOLES ON THE PROJECT

(A complete list of the DDH Holes for the Project is available on Sedar in the Report titled: **Tijirit Project NI43-101 Technical Report with Resource Estimate Update**)

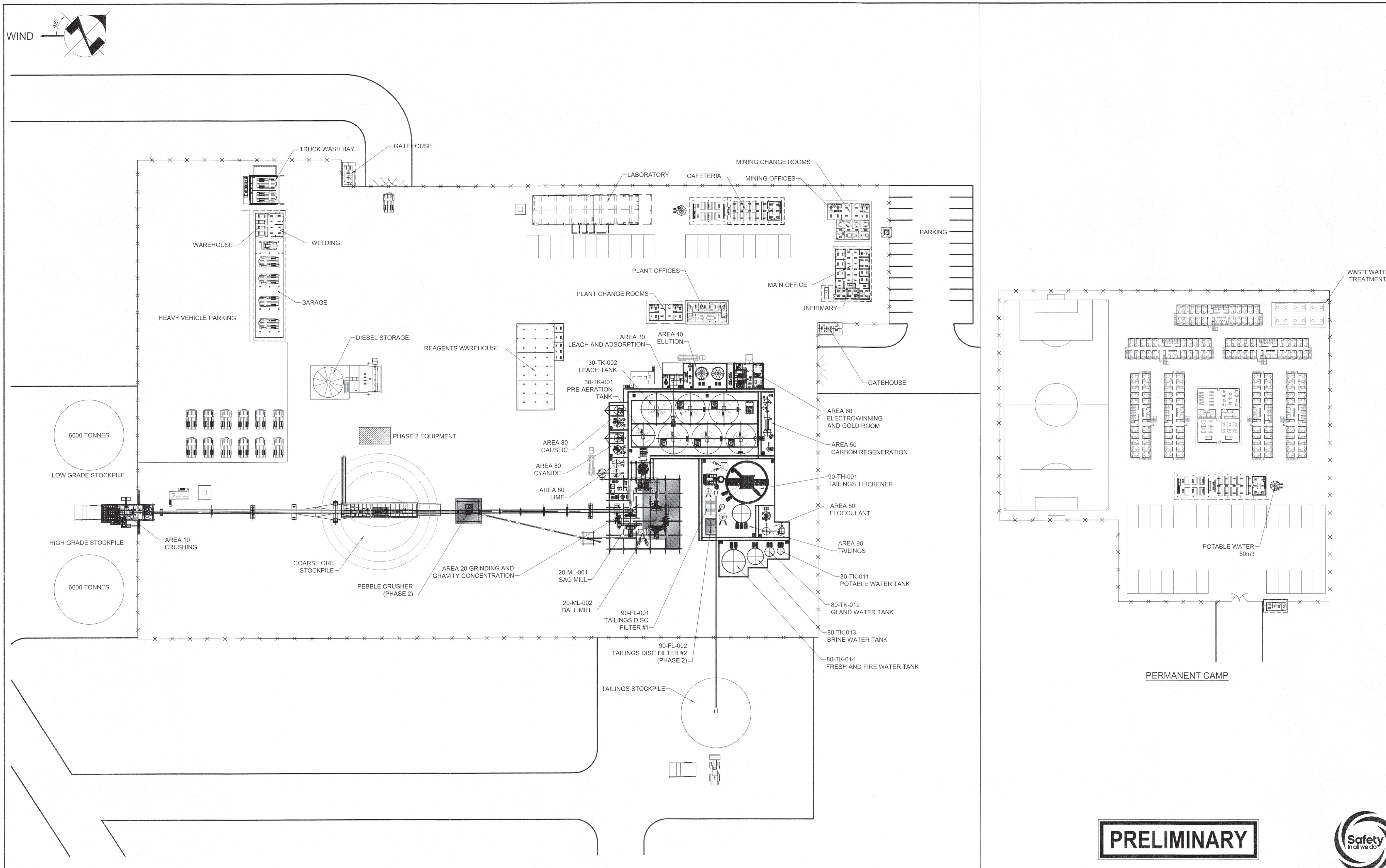
31. ANNEX II – LIST OF REVERSE CIRCULATION HOLES ON THE PROJECT

(A complete list of the RC Holes for the Project is available on Sedar in the Report titled: **Tijirit Project NI43-101 Technical Report with Resource Estimate Update, April 10th 2018**)

32. ANNEX III – DRILLHOLES INTERSECTIONS ABOVE 0.27 G/T AU

(A complete list of the Drillholes Intersection above 0,27 g/t Au for the Project is available on Sedar in the Report titled: **Tijirit Project NI43-101 Technical Report with Resource Estimate Update, April 10th 2018**)

33. ANNEX IV – PLANT LAYOUT



PRELIMINARY

Safety in all we do

ALGOLD RESOURCES LTD

DRAWING No.	REFERENCE DRAWING

No	BY	DATE	REVISION DETAILS	CHKD	ENG	APPR
B	B.I	2018-05-16	ISSUED FOR CLIENT APPROVAL	G.C	G.C	T.Z
A	B.I	2018-05-07	ISSUED FOR CLIENT REVIEW	G.C	G.C.	T.Z

DRAWN	B. IMPERIOLI	2018-05-16
CHECKED	G. CLAYTON	2018-05-16
DESIGNED	S. VENNE	2018-05-16
DES. APPR	G. CLAYTON	2018-05-16
PROJ APPR	T. ZWIRZ	2018-05-16

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CLIENT ALGOLD Resources Ltd.

TITLE TIJIRIT PEA

PLANT GENERAL LAYOUT
PLANT AND CAMP VIEWS

PROJECT No 102533 | SCALE NONE | SIZE A1

DRAWING No 102533-0000-M-100 | REV B

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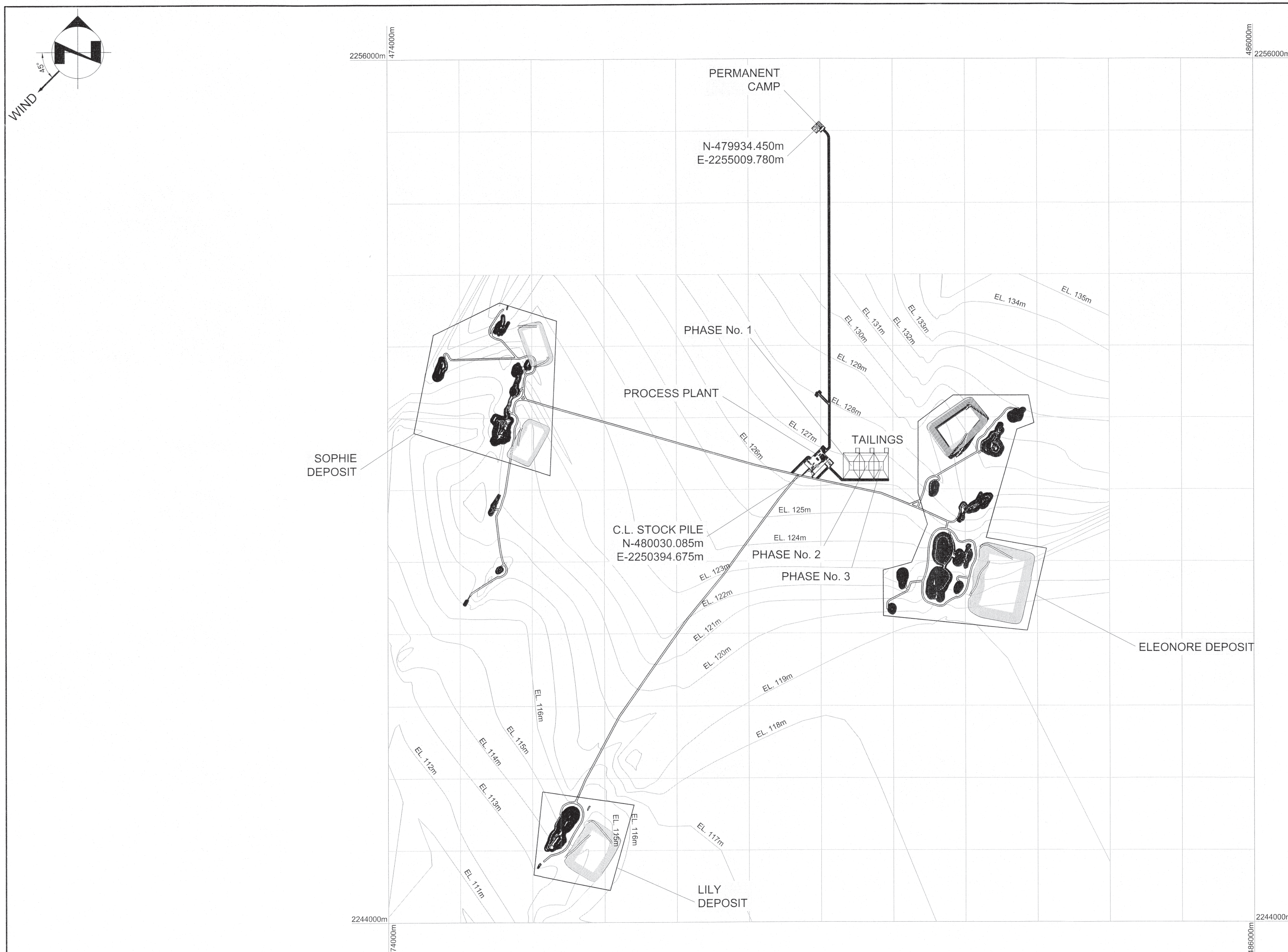
1731

34. ANNEX V – SITE LAYOUT



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CHECKED	G. CLAYTON	2018-05-16
DESIGNED	S. VENNE	2018-05-16
DES. APPR	G. CLAYTON	2018-05-16
PROJ APPR	T. ZWIRZ	2018-05-16

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CLIENT ALGOLD Resources Ltd.

TITLE TIJIRIT PEA

SITE LAYOUT
PLANT VIEW

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PROJECT No	SCALE	SIZE
102533	1:30000	A1

DRAWING No	REV
102533-0000-G-100	B

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Tijirit