IAMGOLD Corporation

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TECHNICAL REPORT ON THE ESSAKANE GOLD MINE CARBON-IN-LEACH AND HEAP LEACH FEASIBILITY STUDY, SAHEL REGION, BURKINA FASO

NI 43-101 Report

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1 SUMMARY

1.1 EXECUTIVE SUMMARY

IAMGOLD Corporation (IAMGOLD) has prepared a Feasibility Study (FS) for the mill optimization of a carbon-in-leach (CIL) circuit and the addition of a heap leach (HL) facility for its Essakane Gold Mine (Essakane or the Project) located in the Sahel region of Burkina Faso, West Africa. The results of the FS support an investment in a lower capital CIL plant de-bottlenecking project, which could increase current CIL plant throughput of 10.8 million tonnes per annum (Mtpa) rock equivalent by 6% to 11.7 Mtpa of 100% fresh (hard) rock CIL plant feed. The CIL crushing circuit would be used for the heap leach project at the end of CIL operations. The purpose of this Technical Report is to disclose the results of the FS and to support the disclosure of the August 31, 2019 Essakane Gold Mine Mineral Resource and Mineral Reserve estimate and the May 25, 2018 Mineral Resource estimate for the satellite Gossey deposit which is located approximately 12 km northwest of the Project within the Essakane exploration properties. All currency in this report is US dollars (US$) unless otherwise noted.

This Technical Report was prepared by IAMGOLD and incorporates the work of IAMGOLD, Kappes, Cassiday & Associates (KCA), SRK Consulting (U.S.) Inc. (SRK), and G Mining Services Inc. (GMSI) Qualified Persons (QPs).

IAMGOLD is a mid-tier mining company with four operating gold mines and several exploration properties on three continents.

Essakane consists of one mining permit (the Essakane Mining Permit), which contains the Essakane main zone (EMZ) deposit and the Falagountou West and Wafaka deposits. In April 2008, the Essakane Mining Permit was granted to Essakane S.A., a Burkinabé company created for the purpose of developing and operating the Project. In addition, there are six exploration permits, all located on contiguous ground (the Essakane Exploration Permits), which contain the Gossey deposit. IAMGOLD, through its wholly-owned subsidiary Essakane S.A., owns 90% of Essakane with the government of Burkina Faso (the government) holding the remaining 10%.

Essakane has been in operation since May 2009 and attained commercial production in July 2010. Mining is carried out using a conventional drill, blast, load, and haul surface mining method with an owner fleet. The annual mining rate was 50.7 million tonnes (Mt) in 2018 with a stripping ratio of 3.05 including 13.0 Mt of ore at an average grade of 1.18 g/t Au, for a total of 450,000 ounces of gold on a 100% basis.

Essakane ore is currently processed using two stages of crushing, semi-autogenous grinding (SAG), ball mill grinding, pebble crusher grinding (SABC), gravity concentration, and a CIL gold plant.

As part of the FS, a number of investigations to upgrade the existing Essakane milling operation have been carried out by identifying equipment throughput constraints at various plant capacities. The grinding circuit will be designed to accommodate a CIL plant feed of 11.7 Mtpa of fresh rock equivalent, in terms of the total specific energy required, through modifications to the primary screening circuit, gravity circuit, pebble crushing circuit, and cyclone underflow distribution system.

Following the end of the existing CIL process plant operations, the heap leach operation will process 8.5 Mtpa of ore from a stockpile with a current estimated life of mine (LOM) for the heap leach operation of approximately five years.

The heap leach facility (HLF) consists of the heap leach pad; a pregnant solution pond, an excess solution pond, and surface water management controls.

Mineral Resources and Mineral Reserves have been prepared in accordance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) definitions).

The Essakane Mineral Resource estimate at August 31, 2019 is summarized in Table 1-1 and is reported on a 100% basis. The Essakane Mineral Resource estimate was prepared by Essakane S.A. and is inclusive of Mineral Reserves.

TABLE 1-1 ESSAKANE MINERAL RESOURCE ESTIMATE – AUGUST 31, 2019

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Measured	-	-	-
Indicated	154,854	0.98	4,878
Total Measured + Indicated	**154,854**	**0.98**	**4,878**
Inferred	12,823	1.10	454

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade which varies between 0.25 and 0.55 g/t Au depending on material type and pit.
3. Mineral Resources are estimated using an average long-term gold price of US$1,500/oz.
4. A minimum mining width of 10 m was used for Falagountou and 10 m for EMZ.
5. Bulk density is estimated by Ordinary Kriging (OK) by weathering type.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Numbers may not add due to rounding.

The Essakane Mineral Reserve estimate at August 31, 2019 is summarized in Table 1-2 and is reported on a 100% basis. The Essakane Mineral Reserve estimate was prepared by IAMGOLD's Longueuil Technical Services team.

TABLE 1-2 ESSAKANE MINERAL RESERVE ESTIMATE – AUGUST 31, 2019

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Proven	-	-	-
Probable	129,299	0.96	3,985
Total	**129,299**	**0.96**	**3,985**

Notes:
1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves estimated assuming open pit mining methods.
3. Mineral Reserves are estimated using an undiluted cut-off grade which varies between 0.31 and 0.61 g/t Au depending on material type and pit.
4. Mineral Reserves are estimated using an average long-term gold price of US$1,200/oz.
5. Average weighted CIL process recovery of 92.1% and heap leach process recovery of 67.0%.
6. Mineral Reserves are reported on a 100% basis.
7. Mineral Reserves include material from EMZ and Falagountou pits.
8. Numbers may not add due to rounding.

In addition to the Essakane Mineral Resource estimate, IAMGOLD appointed GMSI to complete an independent Mineral Resource estimate for the Gossey deposit located approximately 12 km northwest of the Project inside the Essakane Exploration permits. The objective of the Gossey Mineral Resource estimate was to feed into future mine planning activities and to assist in the targeting of extensions to mineralization along strike and at depth.

The Gossey Mineral Resource estimate at May 25, 2018 is summarized in Table 1-3 and is reported on a 100% basis. The Gossey Mineral Resource estimate was prepared by GMSI. The Gossey deposit does not contain a Mineral Reserve.

TABLE 1-3 GOSSEY MINERAL RESOURCE ESTIMATE – MAY 25, 2018

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Measured	-	-	-
Indicated	10,454	0.87	291
Total Measured + Indicated	**10,454**	**0.87**	**291**
Inferred	2,939	0.91	85

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade which varies between 0.33 and 0.47 g/t Au depending on material type.
3. Mineral Resources are estimated using an average long-term gold price of US$1,500/oz.
4. Bulk density is estimated by Inverse Distance Squared (ID^2) by weathering type.
5. Mineral Resources are inclusive of Mineral Reserves.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

1.1.1 CONCLUSIONS

1.1.1.1 ESSAKANE GEOLOGY AND MINERAL RESOURCES

IAMGOLD has the following conclusions and observations on the EMZ, Falagountou West, and Wafaka deposits:

- Mineral Resources have been prepared in accordance with the CIM (2014) definitions.
- Work completed to date by the geological staff is appropriate.
- The geological model employed by Essakane S.A. geologists is reasonably well understood and is well supported by field observations in both outcrop and drill intersections.
- The resource model has been prepared using appropriate methodology and assumptions. These parameters include:
    - o Treatment of high assays.
    - o Compositing length.
    - o Search parameters.
    - o Bulk density.

- o Cut-off grade.
- o Classification.

- The block model has been validated using a reasonable level of rigor consistent with common industry practice.
- The current drill spacing in the EMZ deposit is judged adequate to develop a reasonable model of the mineralization distribution and to quantify its volume and quality with a good level of confidence in all three areas of the Project.
- Based on visual verification, the models (Rock Type, Density, and Au Grade) were found to be globally representative of the known geological and structural controls of mineralization at the EMZ deposit.
- Statistical analysis demonstrates that the block model provides a reasonable estimate of the Mineral Resources for the EMZ deposit.
- Validation of the block model using different interpolation methods indicated that tonnages, grades, and gold contents are similar.
- Swath plots for Indicated and Inferred Mineral Resources by vertical sections for the EMZ and North Satellite areas indicate that peaks and lows in gold content generally match peaks and lows in composite grades; no bias was found in the resource estimate in this regard.
- The information stored in the Falagountou West and Wafaka database was reviewed and found it to be in good standing.
- Drill hole spacing on the Falagountou West and Wafaka deposits are judged adequate to develop a reasonable model of the mineralization distribution and to quantify its volume and quality with an acceptable level of confidence.
- The Ordinary Kriging (OK) method was judged to be the most suitable to replicate composite grades throughout the Falagountou West and Wafaka deposits
- Sampling and assaying have been carried out following standard industry quality assurance and quality control (QA/QC) practices. These practices include, but are not limited to, sampling, assaying, chain of custody of the samples, sample storage, use of third-party laboratories, standards, blanks, and duplicates.

1.1.1.2 GOSSEY DEPOSIT GEOLOGY AND MINERAL RESOURCES

GMSI has the following conclusions and observations on the Gossey deposit.

- Mineral Resources were classified into Indicated and Inferred categories according to the CIM (2014) definitions.
- The geological interpretation for the Gossey deposit is based primarily on diamond drilling (DD) data and geological interpretations by representatives of Essakane S.A. The geology of the deposit is relatively well understood.
- The mineralization is found mainly in the arenitic lithologies, and occasionally found within the diorite intrusive. The strongest mineralization is found along the contacts between these two lithologies. The mineralization controls of the deposit are well understood.

- The protocols followed to collect sample data are considered sufficient for National Instrument 43 101 Canadian Standards of Disclosure for Mineral Projects (NI 43 101) purposes. Stringent protocols are in place to ensure that sampling and assaying of drill samples are undertaken to a high standard, and that QA/QC data is checked frequently to identify any errors that may arise. Sampling has been undertaken based on geological logging and is adequate for the mineralization style and size of the deposit.
- QA/QC samples submitted as part of the 2017 drilling campaign returned values within expectations. QA/QC data from previous drilling campaigns were also reviewed and were found to be in good standing. All analyses were undertaken by the on-site laboratory at the Essakane mine site, with frequent umpire checks submitted to an external laboratory (SGS Ouagadougou). GMSI considers all matters relating to QA/QC in line with NI 43-101 requirements.
- Réjean Sirois, P. Eng. and James Purchase, P. Geo, from GMSI, observed the reverse circulation (RC) drilling campaign during a site visit on March 27-31, 2018 and a laboratory tour was also undertaken. GMSI found the drilling methods and sample recoveries acceptable.
- The geological model was undertaken in Leapfrog GEO where 3D wireframe solids of lithologies and weathering profiles were produced and are representative of the style of deposit observed at Gossey.
- Mineral Resources were estimated within the lithology domains using GEOVIA GEMS from 2.5 m long composites using four interpolation passes of Inverse Distance Cubed (ID^3). Each search ellipse was incrementally larger than the previous, and dimensions were based on drill hole spacing.
- The block model was validated against the drill hole composites through global and local validation methods, including visual comparisons, descriptive statistics, swath plots, and Q:Q plots. No production data was available to validate the accuracy of the model to true known grade. Block grades were found to reproduce composite grades sufficiently in the block model.
- The Mineral Resources are reported within a Lerchs-Grossman open pit shell (based on Indicated and Inferred Mineral Resources) and are effective as of May 25, 2018. A 0.33 g/t Au cut-off for saprolite and laterite material, 0.42 g/t Au for transitional material, and 0.47 g/t Au for fresh rock were used to report Mineral Resources. A gold price of US$1,500/oz was used for the pit optimization. The pit constrained Mineral Resource for the Gossey deposit is as follows:
    - Indicated Mineral Resource is estimated at 10.4 Mt at an average grade of 0.87 g/t Au, totalling 291,000 ounces of gold.
    - Inferred Mineral Resource is estimated at 2.9 Mt at an average grade of 0.91 g/t Au, totalling 85,000 ounces of gold.
- No mining buffer zone was applied around the nearby Gossey Village. Should a 350 m buffer zone be maintained around the Gossey Village a portion of the Gossey Mineral Resource will be sterilized. Approximately half of the contained gold ounces would be lost due to the buffer zone restraint, however, the resulting strip ratio is slightly less.

1.1.1.3 MINING AND MINERAL RESERVES

IAMGOLD has the following conclusions and observations:

- The mine design and Mineral Reserve estimate have been completed to a level appropriate for an FS.
- The economic assumptions and methodology used for estimation of the Mineral Reserves are appropriate.
- The Mineral Reserve estimate is consistent with the CIM (2014) definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources, and do not include any Inferred Mineral Resources.

1.1.1.4 METALLURGICAL TESTING AND MINERAL PROCESSING

Kappes, Cassiday & Associates (KCA) has the following conclusions and observations:

- The results of the metallurgical test programs indicate that the ore types tested are amenable to standard heap leaching methods. The available test results are sufficient to support this FS. Gold recovery is estimated to be 67% and reagent requirements are low.
- Heap leach metallurgical testing has been carried out by KCA. KCA has identified the following risks that may affect the economics of the heap leach project:
    - No metallurgical test work has been completed on the material in the stockpiles and the effects of weathering is unknown.
    - Due to the low grade of the heap leach ore and the presence of coarse gold, individual tests give ranges of assays and there could be some error in recovery estimates.
    - Some of the ore at Essakane is preg-robbing which can have long term effects if placed in the lower lifts of the heap.
- KCA has identified the following opportunities that may affect the economics of the heap leach project:
    - It may be possible to vary the cement addition rate by lift, dependant on the ultimate load of additional lifts stacked on top.

1.1.1.5 ENVIRONMENT

IAMGOLD has the following conclusions and observations:

- No outstanding technical issues were identified for environment and permitting.
- Communications with communities were initiated in 2018 during the geological investigation campaign. In light of the growing influx of people who came to settle in the Gossey Project area to benefit from a possible resettlement action plan, the mayor of the commune of Gorom-Gorom, issued a decree fixing the deadline for settlement as May 10, 2018. Beyond this date, no new installation will be taken into account in the inventory of affected property and people.
- There has not yet been a study of the environmental and social impacts of the Gossey Project.

1.1.2 RECOMMENDATIONS

The FS recommends initiating the detailed engineering for the CIL plant upgrade to 11.7 Mtpa. The project schedule is estimated to be 12 months and is expected to be commissioned in Q3 2020. For the HL, study assumptions will be validated on a yearly basis during the LOM process. Some additional test work will also be initiated to evaluate low grade transition material within the CIL Mineral Reserve that may be amenable to HL with the addition of agglomeration.

As construction of the HL facility is not required until 2025, the business retains the option of re-evaluating the economics of the construction project at that point in time. Given that the CIL process generates higher recovery and doesn't require additional capital investment, it is recognized that there may be a case where the existing stockpiled ore planned for the HL process generates superior economics by processing through the existing CIL circuit, especially in scenarios with higher gold prices than have been used for the current study.

1.1.2.1 ESSAKANE GEOLOGY AND MINERAL RESOURCES

IAMGOLD has the following additional recommendations for the EMZ, Falagountou West, and Wafaka deposits:

- A more complex structural model should be integrated in the next update in order to have a better understanding of mineralization features at a smaller scale.
- The area covered by the pit shell in this study reached areas with lower confidence in the geological model (west flank and lower layer). Diamond drilling should be carried out in the upcoming year in order to improve the geological model.

1.1.2.2 GOSSEY DEPOSIT GEOLOGY AND MINERAL RESOURCES

GMSI has the following additional recommendations for the Gossey deposit:

- Additional exploration work should be carried out as the Gossey deposit remains open to the northeast, where some of the best intersections of the 2018 drilling campaign were returned (23 m at 1.18 g/t Au, 26 m at 2.26 g/t Au, and 14 m at 1.17 g/t Au). The most northern drill line is located on the northeast margin of the Gossey Village, and artisanal workings continue further northeast for 1 km. There is scope for significant additional tonnage to be discovered along this highly prospective trend.
- Increase the proportion of DD drilling (currently 14%) within the pit constrained Mineral Resource to increase confidence in the gold grades and provide more bulk density data.
- Periodically review the RC drill sample splitting procedure. During the site visit it was noted that the riffle splitters were often bias towards one side (poorly aligned), therefore affecting the ability to obtain a representative sample. In addition, sample splitting was

undertaken outside in windy conditions resulting in a loss of fine material during splitting.

- A budget of $1.0 million is proposed to drill the northeast extensions of the deposit.

1.1.2.3 METALLURGICAL TESTING AND MINERAL PROCESSING

- A recent metallurgical study indicated a risk of lower gold recovery related to the amount of graphitic ore present in future mining zones, according to the LOM. Essakane S.A. has undertaken a mitigation plan that needs to be completed. Additionally, a geometallurgy survey, which is currently ongoing, will help determine where the graphitic ore originates and serves as a basis for better mill feed sequencing in order to optimize CIL plant operating parameters as a function of graphitic carbon concentration in the feed.
- KCA has the following recommendations:
  - Metallurgical testing should be conducted on stockpile material to check if weathering has any effect on recoveries.
  - Test transition material.
  - Testing with reduced cement addition.
- SRK Consulting (Canada) Inc. (SRK) has the following recommendation:
  - For the HLF additional loaded permeability (USBR 5600) interface shear and liner integrity testing of the final overliner material selected should be completed as part of detailed design to ensure satisfactory interface shear strength and drainage of the overliner layer relative to the agglomerated ore.

1.2 ECONOMIC ANALYSIS

The Project has been evaluated using discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including sustaining and expansion capital costs, operating costs, taxes, and royalties. These are subtracted from revenues to arrive at the annual cash flow projections. Cash flows are taken to occur at the end of each period.

The appropriate discount rate can depend on many factors, including the type of commodity, the cost of capital for the firm, and the level of project risks (e.g. market risk, technical risk, and political risk) in comparison to the expected return from the equity and money markets. The base case discount rate for this Technical Report is 6%, which has been used to evaluate the Essakane mine projects. The discounted present values of the cash flows are summed to arrive at the Project's net present value (NPV).

For the purposes of the financial analysis, the assumed gold price for the LOM is $1,350/oz. The gold price was the consensus forecast of the following sources: bank analysts' long-term forecasts; historical metal price averages; and prices used in recent publicly-disclosed comparable studies.

Table 1-4 summarizes the financial results with the NPV 6% highlighted. The after-tax NPV at 6% for the remaining LOM (2019 to 2031) is $874 million. Only Essakane Probable Mineral Reserves are mined in the LOM. No mining of the Gossey deposit is included in the LOM.

The LOM total cash cost per ounce of gold sold is US$778 while the all-in sustaining cost (AISC) per ounce of gold sold is US$949. Note that AISC, as reported, is based solely on costs associated with this mine site and does not take into account head office or any other corporate costs not directly associated with the Project.

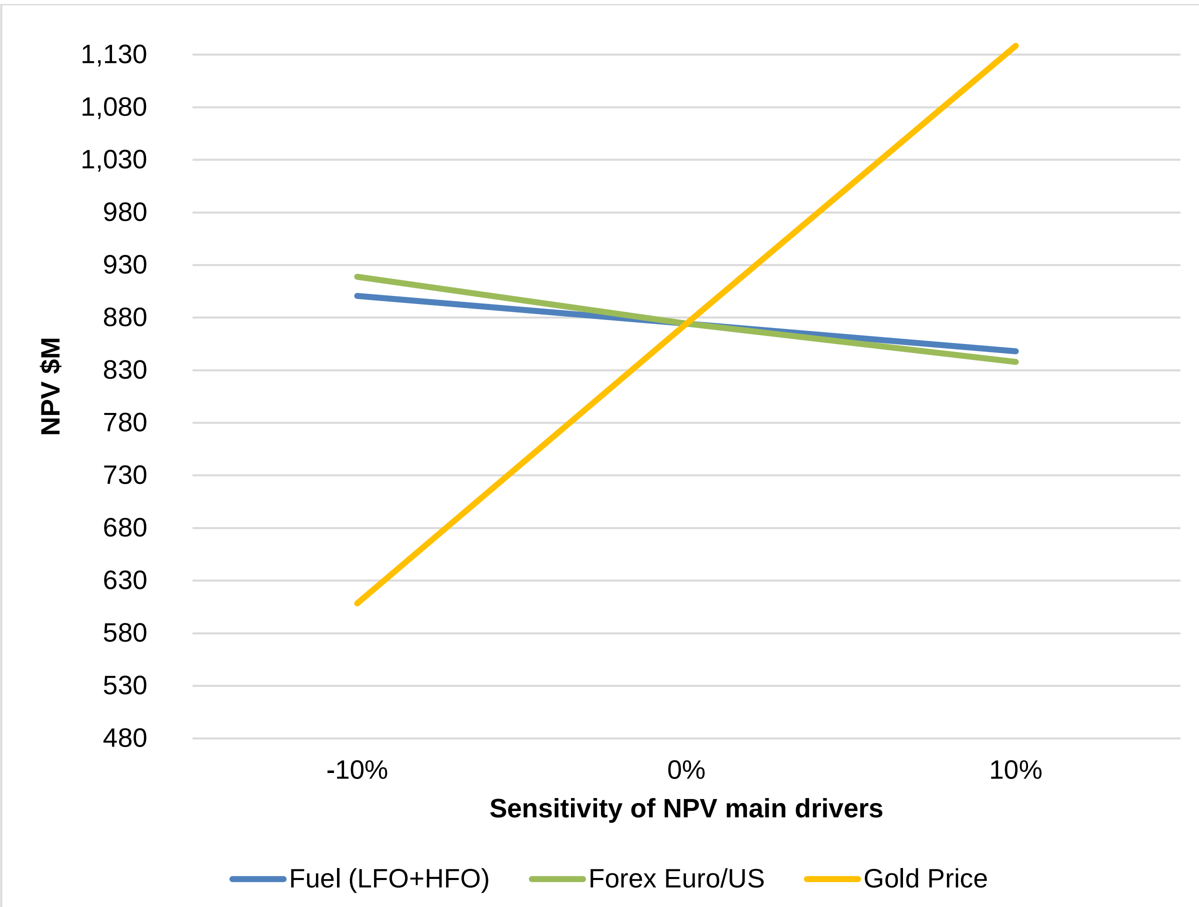
TABLE 1-4 SUMMARY OF FINANCIAL MODEL

Key Performance Indicators		T39						LOM								R4
		2019	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2020-2032**
Mining Statistics																
Ore Mined	(000 t)	18,423	16,328	11,849	14,755	12,042	14,262	19,476	11,543	-	-	-	-	-	-	**100,256**
Waste Mined	(000 t)	25,962	25,167	20,774	15,741	13,490	33,597	21,442	4,814	-	-	-	-	-	-	**135,025**
Capital Waste Mined	(000 t)	11,726	24,024	24,437	26,580	30,693	4,180	-	-	-	-	-	-	-	-	**109,915**
Total Tonnes Mined	**(000 t)**	**56,110**	65,520	57,060	57,076	56,225	52,040	40,919	16,356	-	-	-	-	-	-	**345,196**
Milling Statistics - CIL																
Throughput	(000 t)	12,841	12,599	11,829	12,500	12,143	11,870	10,802	11,396	-	-	-	-	-	-	**83,140**
Gold Production - CIL	**oz**	**420,215**	**424,751**	**419,666**	**410,441**	**411,778**	**400,053**	**530,769**	**435,588**	**-**	**-**	**-**	**-**	**-**	**-**	**3,033,045**
Unit Costs																
Mining Cost / t Mined	$/t	3.10	2.82	2.84	2.81	2.93	3.03	3.40	4.47	-	-	-	-	-	-	**3.02**
Mining Cost / t Milled (net of ore stockpiled & deferred stripping)	$/t	9.09	7.51	8.50	6.43	6.55	11.02	10.63	7.67	-	-	-	-	-	-	**9.61**
Milling Cost / t Milled	$/t	7.17	7.04	7.26	6.89	6.99	7.04	7.26	7.12	-	-	-	-	-	-	**7.08**
Power Cost / t Milled (CIL only)	$/t	4.65	4.37	4.67	4.52	4.62	4.57	4.96	4.74	-	-	-	-	-	-	**4.63**
G&A Cost / t Milled	$/t	4.14	4.19	4.44	4.12	4.21	4.28	4.51	4.13	-	-	-	-	-	-	**4.27**
Total Cost / t Milled	**$/t**	**25.05**	**23.12**	**24.87**	**21.96**	**22.37**	**26.91**	**27.37**	**23.66**	**-**	**-**	**-**	**-**	**-**	**-**	**25.58**
Heap Leach Operation																
Tonnage Processed	(000 t)	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**8,500**	**8,498**	**8,500**	**8,500**	**9,348**	**-**	**43,346**
Gold Production (HL)	oz	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**80,734**	**84,944**	**70,243**	**62,899**	**69,426**	**-**	**368,246**
Heap Leach Processing Cost	$/t									4.35	4.35	4.35	4.35	4.35		**4.35**
Total Combined Production & Sales (CIL+HL)																
Total Gold Production	oz	**420,215**	**424,751**	**419,666**	**410,441**	**411,778**	**400,053**	**530,769**	**435,588**	**80,734**	**84,944**	**70,243**	**62,899**	**69,426**	**-**	**3,401,291**
Total Gold Sales	oz	**430,003**	**424,751**	**419,666**	**410,441**	**411,778**	**400,053**	**530,769**	**435,588**	**102,083**	**84,944**	**70,243**	**62,899**	**69,426**	**-**	**3,422,641**
Gold Price	$/oz	1,282	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	**1,350**
Silver Price	$/oz	16.64	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	**18.00**
Total Cash Costs ($ / oz)	**$/oz**	**820**	**755**	**770**	**738**	**729**	**867**	**626**	**688**	**1,075**	**1,022**	**1,160**	**1,252**	**1,251**	**-**	**778**
TOTAL CAPEX	**US$ 000**	**110,611**	**124,357**	**101,364**	**101,863**	**117,547**	**33,485**	**77,179**	**66,739**	**4,259**	**13,406**	**4,215**	**3,757**	**4,230**	**-**	**652,400**
										-	-	-	-			
All-In Sustaining Costs ($ / oz sold)	**$/oz**	**982**	**993**	**1,045**	**1,006**	**998**	**989**	**712**	**746**	**1,044**	**1,107**	**1,246**	**1,338**	**1,340**	**-**	**949**
Free Cash Flow																
Operating Cash Flows	US$ 000	172,606	230,960	232,747	226,506	233,128	160,883	344,124	200,890	55,376	73,665	27,806	4,588	41,190	10,199	**1,842,054**
Sustaining Capital Expenditures	US$ 000	(49,258)	(79,823)	(95,394)	(91,120)	(92,401)	(33,485)	(19,744)	(9,304)	(4,259)	(4,957)	(4,215)	(3,757)	(4,230)	-	**(442,689)**
Non-sustaining Capital Expenditures	US$ 000	(61,353)	(44,534)	(5,970)	(10,743)	(25,146)	-	(57,435)	(57,435)	-	(8,449)	-	-	-		**(209,711)**
Lease Principal Payments	US$ 000	(952)	(6,272)	(6,566)	(6,372)	(6,833)	(3,540)	(493)	0	0	-	-	-	-		**(30,075)**
Lease Interest Payments	US$ 000	(1,132)	(1,749)	(1,464)	(1,020)	(560)	(130)	(12)	(0)	(0)	-	-				**(4,935)**
Restricted Cash	US$ 000	313	(3,454)	(3,454)	(3,454)	(3,454)	(3,454)	(3,454)	-	4,000	4,000	4,000	4,000	4,000	24,354	**23,630**
Dividends Paid to Government	US$ 000	-	-	-	-	-	-	-	-	-	-	-				**-**
Free Cash Flow	US$ 000	**60,224**	**95,128**	**119,898**	**113,797**	**104,734**	**120,275**	**262,985**	**134,151**	**55,117**	**64,259**	**27,592**	**4,831**	**40,960**	**34,553**	**1,178,274**

NPV		2019-2032
NPV (US$ 000)	6%	**$874,331**

Three important parameters (fuel price, foreign exchange (Forex), and gold price) greatly impact the NPV. A simulation was performed to understand the impact on the site NPV by fluctuating these parameters by 10%. As expected, the NPV reacts the most with the variation of the price of gold (Figure 1-1). The impact of the gold price increasing by 10% is $264 million. It is followed by Forex by more than $45 million when the Euro becomes stronger compared to the US Dollar. Finally, the fuel price (HFO and LFO) impacts the NPV by $26 million when the unit price varies by 10%.

FIGURE 1-1 NPV SENSITIVITY ANALYSIS



Payments of $76.5 million are estimated for the closure and reclamation cost of Essakane. This amount was updated in March 2019 to reflect the updated projected disturbance until the end of the mine.

1.3 TECHNICAL SUMMARY

1.3.1 PROPERTY DESCRIPTION AND LOCATION

The Project spans the boundary of the Oudalan and Seno provinces in the Sahel region of Burkina Faso and is approximately 330 km northeast of the capital, Ouagadougou. The property's latitude and longitude are 14º 23' N and 0º 04' E.

The mining and exploration permits comprising the Project are subject to Burkina Faso's 2015 Mining Code No.3 036-2015/CNT, dated June 26, 2015 (the Burkina Faso Mining Law). The Project consists of one mining permit (the Essakane Mining Permit), which contains the EMZ deposit and the Falagountou West and Wafaka deposits. The mining permit is surrounded by six exploration permits (the Essakane Exploration Permits) belonging to Essakane Exploration SARL, the exploration subsidiary of IAMGOLD working in the region of the Project. The Gossey deposit is located within the Essakane Exploration Permits approximately 12 km northwest of the Essakane EMZ deposit.

1.3.2 LAND TENURE

In April 2008, the 100.2 km^2 Essakane Mining Permit was granted to Essakane S.A., a Burkinabé company created for the purpose of developing and operating the Project. The mining permit is valid for a period of 20 years and is renewable every five years until Mineral Reserves have been depleted.

According to the Mining Law of Burkina Faso, a mining convention must be negotiated between the mining permit owner and the government before operations can start. The mining convention describes the governmental commitments, operational tax regime, and obligations of the company to Burkina Faso. The mining convention between Essakane S.A. and the government was signed on July 14, 2008.

IAMGOLD owns a 90% interest in Essakane S.A., while the government has a 10% free-carried interest. In addition, the government receives a 3% royalty on the revenues from mineral production if the gold price is below US$1,000/oz, 4% if the gold price is between US$1,000/oz and US$1,300/oz, and 5% if the gold price is greater than or equal to US$1,300/oz. The government also collects various taxes and duties on the imports of fuels, supplies, equipment, and outside services, as specified by the Burkina Faso Mining Law.

The Essakane Mining Permit is surrounded by the Essakane Exploration Permits, which currently cover a total area of 906.12 km^2. The Essakane Exploration Permits are presently in good standing and Essakane S.A. has been issued with Certificate #1587/2007 (Issue date 04/10/2007) by Mr. Seydou Balama at the Office Notarial in Ouagadougou.

Surface rights in the area of the Essakane Mining Permit belong to the State of Burkina Faso. Utilization of the surface rights is granted by the Essakane Mining Permit under the condition that the current users are properly compensated. All the taxes relating to Essakane S.A.'s Mining Rights have been paid to date and the concession is in good standing. IAMGOLD has all required permits to conduct the proposed work on the property.

1.3.3 HISTORY

Essakane, especially the EMZ deposit, has been an active artisanal mining site since 1985. At its peak, up to 15,000 artisanal miners worked at the EMZ deposit.

In 1991, the Essakane Mining Exploration Permit was granted to Compagnie d'Exploitation des Mines d'Or du Burkina (CEMOB). In 1992, CEMOB constructed an HLF which produced 18,000 ounces of gold in 1993 but averaged between 3,000 ounces and 5,000 ounces of gold per year thereafter. Due to low gold prices and operational problems, CEMOB went into liquidation at the end of 1996 and Coronation International Mining Corporation (CIMC) secured title. In July 2000, six new Essakane licences were granted to CIMC. CIMC carried out an exploration program and drilling of oxide resources. In 2002, CIMC merged with Orezone Resources Inc. (Orezone Resources) and Orezone Resources became 90% owner of Essakane S.A. Orezone Resources was the operator of Essakane until Gold Fields Essakane (BVI) Limited (GF BVI) assumed management responsibilities in January 2006.

In April 2007, Orezone Resources, Orezone Inc., Orezone Essakane Limited, GF BVI, Gold Fields Orogen Holding (BVI) Limited, and Essakane (BVI) Limited (Essakane BVI) entered into a member's agreement and eventually formed a joint venture. GF BVI earned a 60% interest in Essakane (BVI) after the Essakane Definitive Feasibility Study (DFS) was completed in September 2007.

In October 2007, Orezone Resources entered into an agreement with GF BVI to acquire its 60% interest in the Project. On November 26, 2007, Orezone Resources became the operator and owner of a 100% interest in the Project, subject to the interest of the government.

In April 2008, the Essakane Mining Permit was granted over an area of 100.2 km^2 containing the EMZ deposit and the Falagountou West and Wafaka deposits. An updated FS (the 2008 UFS) was completed on June 3, 2008 and readdressed to IAMGOLD in 2009 after IAMGOLD acquired Orezone Resources and Essakane was transferred to IAMGOLD Essakane S.A. Commercial production started on July 16, 2010.

1.3.4 GEOLOGY AND MINERALIZATION

Burkina Faso is extended over two geological terrains: the Paleoproterozoic Baoulé-Mossi Domain, which corresponds to the eastern portion of the West African craton, and the sedimentary cover of lower Precambrian to recent age.

The Project lies in the Oudalan-Gorouol greenstone belt. The area is underlain by the Birimian sedimentary and volcano-sedimentary sequences. The western part is made of granitic and gneissic rocks. The Markoye Shear zone separates Paleoproterozoic rocks to the east from older granite-gneiss terranes to the west.

The Oudalan-Gorouol greenstone belt is bounded and/or crosscut by several north-northeast to northeast trending shear zones. These shear zones are related to the crustal-scale steeply east-dipping Markoye Shear Zone.

The gold mineralization is generally hosted in the hanging wall of northeast trending faults and/or northwest trending folds in meta-siltstone, sandstone, and shale sequences and can be classified as orogenic gold deposits under the sub-class of “intrusion-related” due to their proximity to plutonic masses. Gold is either disseminated or concentrated in quartz veins.

1.3.5 EXPLORATION STATUS

Essakane has been explored since the 1990s by geochemical sampling, mapping, trenching, Aster/Landsat image analysis and interpretation, geophysical surveys, and drilling. Exploration efforts at Essakane were initially focussed on identifying the potential of the entire area of the mine. In the mid 1990s, a widely spaced drilling program was carried out on the EMZ deposit followed by infill drilling.

Orezone Resources started resource definition drilling at the EMZ deposit in February 2003. Orezone Resources and GF BVI drilled 20,364 m of oriented HQ (63.5 mm) diameter core between September 2005 and June 2006 for the project development and the 2007 FS program.

RC and DD drilling has been conducted by Essakane S.A.'s Resource Development Group since January 2010. Up to December 31, 2018, a total of 1,773 RC holes (218,513 m), 44 holes pre-collared with RC then completed by DD (RCD) (12,507 m), and 969 DD holes (268,834 m) had been drilled at the EMZ, Falagountou West, and Wafaka deposit areas.

Essakane S.A.'s drilling objectives include infill drilling to upgrade Inferred Mineral Resources, expand the resource inventory, gain a better understanding of the geology and controls of mineralization to advance geological modelling, and improve the quality of assay samples.

At the EMZ deposit, most DD holes targeted Inferred Mineral Resources below the EMZ pit and along the deposit's northern, southern, and down-dip extensions.

DD results were positive on the EMZ deposit with continuity of mineralization demonstrated at depth along the east limb of the deposit in the northern sector of the pit, as well as in the southeast end of the pit. EMZ deposit mineralization is oriented north-northwest. The DD results were incorporated into the updated resource model as reported at August 31, 2019.

An infill RC and DD program conducted at the Falagountou West deposit confirmed lateral continuity of mineralization oriented mostly north-south as well as down dip extension which remained open.

In 2011, a drilling program on the Gossey deposit consisting of nine RC holes (1,072 m) and ten DD holes (2,508 m) was executed by IAMGOLD. This program resulted in the discovery of the Gossey-SE mineralized zone and the extension of the Gossey main zone. During the same year, 48 RC holes and 12 DD holes totalling respectively 5,723 m and 2,846 m successfully tested the southern extensions of the mineralized trend.

During the second quarter of 2017, an infill RC drilling program of 15,000 m was proposed and implemented to upgrade the classification of the resource at the Gossey deposit.

In 2018, a second infill drilling program of 14,300 m commenced testing for strike extensions of the Gossey deposit, to test grade continuity to a vertical depth of approximately 100 m and convert Inferred Mineral Resources into Indicated Mineral Resources.

1.3.6 MINERAL RESOURCES

The Essakane Mineral Resource estimate for the EMZ, Falagountou West, and Wafaka deposits was prepared in accordance with CIM (2014) definitions and is reported in accordance with the NI 43-101 guidelines. Classification, or assigning a level of confidence to Mineral Resources, has been undertaken with strict adherence to CIM (2014) definitions. In the opinion of the responsible QP, the resource estimation reported herein is a reasonable representation of the Mineral Resources delineated at Essakane as of August 31, 2019.

The Mineral Resource estimate at August 31, 2019 for Essakane is summarized in Table 1-1 and is reported on a 100% basis. The Mineral Resource estimate is inclusive of Mineral Reserves.

Since the previous Essakane Mineral Resource estimate as of June 5, 2018, the EMZ resource model has been updated with new drilling information. The modelling work was completed by Essakane S.A. personnel. As of August 31, 2019, the EMZ deposit pit constrained Indicated Mineral Resource is estimated to be 149.4 Mt at an average grade of 0.97 g/t Au for a total of 4,662,000 ounces of gold, including 361,000 ounces of gold stored in stockpiles. The EMZ deposit open pit Inferred Mineral Resources are estimated to be 10.8 Mt at an average grade of 1.05 g/t Au for a total of 365,000 ounces of gold.

The August 31, 2019 Falagountou West and Wafaka deposits Mineral Resource estimate is a major update from the June 5, 2018 model, since the mineralization zones were completely remodelled by the Essakane geology team in 2018. As of August 31, 2019, Falagountou West and Wafaka pit constrained Indicated Mineral Resources are estimated to be 5.4 Mt at an average grade of 1.24 g/t Au for a total of 216,000 ounces of gold. In addition, Inferred Mineral Resources are estimated to be 2.0 Mt at an average grade of 1.36 g/t Au for a total of 89,000 ounces of gold.

The Mineral Resource estimate at May 25, 2018 for the Gossey deposit is summarized in Table 1-3 and is reported on a 100% basis. The Gossey Mineral Resource estimate was prepared by GMSI. As of May 25, 2018, the Gossey pit constrained Indicated Mineral Resource is estimated to be 10.4 Mt at an average grade of 0.87 g/t Au for a total of 291,000 ounces of gold. The Gossey pit constrained Inferred Mineral Resource is estimated to be 2.9 Mt at an average grade of 0.91 g/t Au for a total of 85,000 ounces of gold.

The Gossey deposit is located near the Gossey village for which future development of some portions of the deposit may require either: the development of a re-location program, similar to the one that was completed previously at Essakane; or the consideration of a mining buffer zone to restrict mining activities proximal to the village. Should a 350 m buffer zone be maintained around the Gossey Village a portion of the Gossey Mineral Resource will be sterilized. Approximately half of the contained gold ounces would be lost due to the buffer zone restraint, however, the resulting strip ratio is slightly less.

Aside from the proximity of the Gossey village to the resource area, the responsible QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

1.3.7 MINERAL RESERVES

The Mineral Reserve estimate at August 31, 2019 for Essakane is summarized in Table 1-2 and is reported on a 100% basis. The Essakane Mineral Reserve estimate was prepared by IAMGOLD's Longueuil Technical Services team.

Mineral Reserves were separated based on CIL and heap leach processing types to account for the distinct processing mill recoveries. The Falagountou deposits (Falagountou West and Wafaka) have no Mineral Reserve attributed to the heap leach process as the ore is not confirmed to be suitable for this process.

As of August 31, 2019, there were 82.4 Mt of CIL Probable Mineral Reserves defined in the EMZ pit design and within stockpiles, at an average grade of 1.25 g/t Au for a total of 3,299,000 ounces of gold. The heap leach Probable Mineral Reserves are estimated to be 43.1 Mt in the EMZ pit design and within stockpiles, at an average grade of 0.40 g/t Au for a total of 549,000 ounces of gold.

Additionally, as of August 31, 2019, there were 3.8 Mt of CIL Probable Mineral Reserves defined in the Falagountou pit designs and within stockpiles, at an average grade of 1.12 g/t Au for a total of 137,000 ounces of gold.

IAMGOLD's 90% attributable Probable Mineral Reserves at Essakane total 116.4 Mt of ore and 3,587,000 ounces of gold.

Essakane is in operation and the mine design and Mineral Reserve estimate have been completed to an operational detailed level. The Mineral Reserve estimate stated herein is consistent with the CIM (2014) definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources.

There are no Mineral Reserves for the Gossey deposit.

The responsible QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate.

1.3.8 MINING METHOD

Mining is carried out using a conventional drill, blast, load, and haul surface mining method with an owner fleet. The annual mining rate was 50.7 Mt in 2018 with a stripping ratio of 3.05 including 13.0 Mt of ore at an average grade of 1.18 g/t Au for a total of 450,000 ounces of gold.

Essakane consists of several operating sites. The Essakane main pit is mined in several mining phases and accounts for the majority of the production. The Falagountou and Essakane North satellite pits provide additional ore and operational flexibility. Grade control is accomplished by RC drilling and sampling of the mineralized zone on a 10 m x 10 m pattern, or tighter as required. For sterile sections of the pit, the grid may be widened out based on the nature of the contacts and/or other geological occurrences.

A fleet of four drill rigs are used for the 229 mm (9.0 inch) production blast holes. All blasting activities on site are executed by an explosives supplier. Holes are loaded with bulk explosive matrix and initiated with electronic detonators.

Grade movement during blasting is a critical issue at Essakane. For this reason, blast movement monitors (BMMs) are systematically used when blasting mineralized areas in order to measure vertical and horizontal displacement which allows for the adjustment of the post blast ore packets.

The Project's loading fleet currently consists of three O&K Terex RH-120 shovels, five CAT 993K wheel loaders, and two Komatsu PC 2000 390 excavators.

The Project's hauling fleet currently consists 26 CAT 785C haul trucks, eight CAT 785D haul trucks, five CAT 777F haul trucks, and three CAT 777F water trucks.

Mine haul roads are 30 m wide and are constructed by the mining department to support the mine haul trucks.

Waste material is being stored in the waste rock dumps (WRDs) located east of the main pit.

Other mining infrastructure includes a mine office complex (mine offices, change houses, and canteens), equipment workshop with overhead cranes integrated with the main warehouse, and external wash bays, blasting and explosives compound including magazines, diesel storage and dispensing facility, and a drill core storage facility.

There are no current mining plans for the Gossey deposit.

1.3.9 MINERAL PROCESSING

1.3.9.1 EXISTING CIL CONCENTRATOR

Essakane ore is currently processed using two stages of crushing, SABC, gravity concentration, and a CIL gold plant. The 2008 UFS proposed a process plant throughput rate of 7.5 Mtpa. During construction, some debottlenecking improvements were made to the design, resulting in a revised nameplate capacity of 9.0 Mtpa based on processing 100% saprolite ore. Due to further operational improvements, plant throughput has increased beyond the constructed design capacity.

Fresh rock CIL plant feed has gradually increased from 2012 onwards. To maintain gold production levels, with increasing proportions of fresh rock in the CIL plant feed, an expansion was completed in 2014. The objective was to double the fresh rock processing capacity from 5.4 Mtpa on a 100% fresh rock basis to 10.8 Mtpa. The expansion consisted of the addition of a secondary crushing circuit and a second process line (grinding, gravity concentration, and leach) in the CIL plant.

The CIL plant expansion was commissioned in February 2014, and effectively doubled the fresh rock processing capacity.

No material from the Gossey deposit has been milled in the CIL gold plant.

1.3.9.2 MODIFICATIONS TO EXISTING CONCENTRATOR

IAMGOLD has conducted a number of investigations to upgrade the existing Essakane CIL plant by identifying equipment throughput constraints at various plant capacities. The areas of focus were the crushing and grinding areas, with additional investigations into the CIL and tailings handling areas, the water balance, and the lime addition circuits.

The grinding circuit will be designed to accommodate 11.7 Mtpa of fresh rock equivalent in terms of the total specific energy required through modifications to the primary screening circuit, gravity circuit, pebble crushing circuit, and cyclone underflow distribution system.

1.3.10 HEAP LEACH FACILITY

Following the end of the existing CIL plant operations, the heap leach operation will process 8.5 Mtpa of ore from a stockpile with a current estimated LOM for the heap leach operation of approximately five years.

The HLF consists of the heap leach pad; a pregnant solution pond, an excess solution pond, and surface water management controls. The heap leach pad will be constructed in two phases for a total of 830,000 m^2. The leach pad base liner system will be a compacted 0.3048 m (12-inch) thick, low-permeability subgrade layer overlain by a single geomembrane liner.

Re-handled ore from a stockpile will be delivered to a local feed bin by a haul truck to feed a three-stage crushing plant consisting of an existing first and second stage crushing area and a new high pressure grinding roll (HPGR) tertiary crushing circuit. Cement will be added to the crushed ore via a rotary valve and screw conveyor suspended from a large overhead storage silo. The cement will also act as an alkalinity control reagent while agglomerating the crushed ore. The agglomerated crushed ore will be conveyed via a series of grasshopper conveyors and stacked onto the leach pad with a radial stacker.

The leach pad will be constructed in an up-gradient manner with ore stacked in multiple lifts. Cyanide bearing solution will be applied on the leach pad for approximately 180 days prior to draining. The solution will be applied counter current to the placement of new ore using a network of piping. Both sprinklers and drip emitters will be used for uniform distribution of the solution. Gold bearing pregnant solution will be drained from the leach pad via gravity to a pregnant solution pond prior to being pumped to the carbon-in-column (CIC) circuit.

The pregnant solution will be processed using one train of six CIC stages with an 8.5 tonne carbon capacity for each carbon adsorption column. Loaded carbon will be stripped with hot cyanide-caustic solution in an existing pressure Zadra elution circuit designed for 17 tonnes of carbon. Each CIC train will advance 8.5 tonnes of carbon every other day to the existing elution facility and the loaded carbon will be processed there once a complete batch of 17 tonnes is reached. Gold concentrated sludge from electrowinning will be dried in an oven and smelted in a furnace to pour into doré bars.

1.3.11 PROJECT INFRASTRUCTURE

General services are an essential component to the success of the mine operation. Due to the remoteness and complex logistics of the mine coupled with the limited services available in Burkina Faso, the personnel of the general services department required to support production is extensive.

Mine infrastructure consists of a mine office complex (mine and administrative offices, change houses, and canteens), equipment workshop with overhead cranes integrated to the main warehouse and external wash down bays, blasting and explosives compound including magazines, diesel storage and dispensing facility, and a drill core storage facility.

The initial mine infrastructure and support facilities constructed between 2009 and July 2010 have been modified and/or adapted for the expansion phase which was carried out from 2012 to 2014. Modifications have been made to the mine truck shop and warehouse, site and mine roads, communication system and information technology (IT), fuel oil storage, exploration building, mine camp, assay and metallurgical laboratories and mill office, river deviation, power generation and distribution, administration buildings, and potable water and treatment facilities.

Some dedicated infrastructure is required to support the heap leach operations. Heap leach project construction will be carried out by contractors under the supervision of IAMGOLD.

Existing workshops and a construction warehouse will be reused for the heap leach project. A construction laydown will be installed at close proximity to the warehouse. The work area will also be fenced. The construction project will benefit from the existing supply chain and logistics system already in place at Essakane.

There is no current infrastructure at the Gossey deposit.

1.3.12 MARKET STUDIES AND CONTRACTS

Gold is the principal commodity at Essakane and is freely traded at prices that are widely known so that prospects for sale of any production are virtually assured. All gold produced by IAMGOLD is in the form of doré bars, which is then shipped to a refiner who refines the doré into bullion. The bullion is then sold directly on the open market to gold trading institutions at prevailing market prices.

1.3.13 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

In 2007, prior to the beginning of construction work, an Environmental and Social Impact Assessment (ESIA) was conducted by Knight Piésold Consulting and submitted to the government of Burkina Faso. The ESIA included an Environmental and Social Management Plan (ESMP) for the Project. The ESIA was completed following a public consultation with key stakeholders, as prescribed under Burkinabé law. In 2008, and following the changes made during construction, an addendum to the ESIA (2008 addendum) was submitted to the authorities of Burkina Faso. A mining permit over an area of 100.2 km² was approved by Burkinabé authorities in late 2007.

In order to increase the annual gold production, IAMGOLD initiated a mine expansion FS in 2011. As part of the mine expansion work in 2012 and 2013, a new addendum to the ESIA and the 2008 addendum was prepared in February 2012 (the February 2012 addendum). The

February 2012 addendum covers the expansion phase of the main pit and CIL plant infrastructures, a new satellite pit east of the mine, and the Gorouol River diversion. It includes an updated ESMP incorporating the necessary adjustments to the initial ESMP to include the expansion changes and to consolidate, in one document, all social and environmental commitments of IAMGOLD. The February 2012 addendum was approved in 2013.

IAMGOLD Essakane S.A. implemented two resettlement plans consistent with Burkinabé laws and best practices recommended by international organizations (World Bank). The first plan started in 2008 (13,000 individuals and 2,981 households affected) and the second plan started in 2012 (3,208 individuals and 555 households affected).

As part of the community investment plan, socio-educational infrastructures are being built (wells, medical centres, schools, etc.). Programs to fight malaria and HIV/AIDS, and increase road safety awareness, were developed for the benefit of neighbouring populations. Rural development activities (agriculture, animal husbandry, etc.) are primarily undertaken as part of the livelihood restoration program and through the community investment program. Since 2014, a community investment program has been financing community projects through communal development plans.

A program for environmental monitoring (groundwater quality, fauna, and dam stability inspection) and progressive rehabilitation of the tailings site is in place, at and around, the tailings site. This program encompasses water quality monitoring, air quality, soil, biodiversity (fauna and flora), noise, vibration, weather, and follow up and assessment of the community investment program (health, education, potable water access, agriculture, animal husbandry, etc.).

The heap leach project will be another expansion of the mine. The ESIA report will need to be completed prior the commencement of the detailed engineering phase. This will be the first step towards obtaining the environmental and social feasibility notice.

The heap leach project triggers an Environmental Impact Statement (EIS) in accordance with Article 4 of Decree No. 2015-1187/PRES-TRANS/PM/MERH/MATD/MME/MS/MARHASA/MRA/MICA/MHU/MIDT/MCT of October 22, 2015 laying down conditions and procedures for carrying out and validating the strategic environmental assessment, the study, and the environmental and social impact notice. Field works to collect baseline data were performed during January and February 2018. Consultations on targeted groups (such as women, elders, youths, gold diggers, farmers, etc.) were performed during the field work. Since the heap leach project incurred significant changes in schedule and location compared to the Prefeasibility study, it is expected that part of the field work, as well as community consultations, will have to be redone.

The ESMP resulting from this study will have to be integrated into the general ESMP of the mine in order to obtain an aggregated ESMP, which will include all the environmental studies carried out within the framework of the exploitation of the mine.

Communications with communities were initiated in 2018 during the geological investigation campaign. In light of the growing influx of people who came to settle in the Gossey Project area to benefit from a possible resettlement action plan, the mayor of the commune of Gorom-Gorom, issued a decree fixing the deadline for settlement as May 10, 2018. Beyond this date, no new installation will be taken into account in the inventory of affected property and people. The inventory of properties and people began immediately after the announcement of the deadline. The Gossey Project area was surveyed almost entirely, but the inventory was then suspended, and the communities were informed that the Project was postponed.

There has not yet been a study of the environmental and social impacts of the Gossey Project.

A conceptual rehabilitation and closure plan (PRF) was developed in 2009 and last updated in 2013 and 2018. Closure costs are updated annually or whenever the mining development plan is amended. A progressive mining rehabilitation process commenced in 2011, shortly after the start of production.

A closure plan PFS will be conducted three years prior to mine closure. A closure plan FS must be conducted two years prior to the closure of the mine and must be approved by the relevant authorities.

1.3.14 CAPITAL AND OPERATING COST ESTIMATES

Total capital spending over the remaining LOM (2020-2031) amounts to $652 million, representing $5.16/t processed (including heap leach) or $191/oz of Au sold.

The total sustaining capital spending planned, excluding capital waste stripping (cash portion) in 2020, is $41.4 million out of a total of $199.4 million over the LOM (2020-2031). In 2020, the Project's capital cost, including capital waste stripping, is $124.4 million or $293/oz Au sold.

The LOM mine sustaining capital costs are mainly related to the acquisition of mobile equipment, equipment capital spares, and equipment purchases ($23.8 million in 2020, $26.8 million in 2021, and $23.1 million in 2022), with the aim of renewing the aging fleet and supporting production until the LOM ends in 2031.

Total expansion capital is estimated at $138.4 million. A total of $15.1 million is included in 2020 for the CIL plant upgrade and tailings dam capital expenditures. The heap leach project capital expenditures total $57.4 million in 2025, $57.4 million in 2026, and include a sustaining capital cost of $8.5 million in 2028 for the heap leach pad extension.

Capital waste stripping is the largest capital element estimated at $314.5 million or $92/oz of Au sold over the LOM and represents 48% of the LOM capital cost. Capital waste stripping continues until 2024 after which all mining will be in ore until the end of the heap leach.

No capital costs have been estimated for the Gossey deposit.

A summary of life of mine capital expenditures is presented in Table 1-5.

TABLE 1-5 LIFE OF MINE CAPITAL EXPENDITURES

Items		2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2020-2032
Expansion/Process Improvement	(US$ M)	15.1	-	-	-	-	57.4	57.4	-	8.5	-	-	-	-	**138.4**
Expansion - Cap. Strip	(US$ M)	29.4	6.0	10.7	25.2	-	-	-	-	-	-	-	-	-	**71.3**
Sustaining Capital	(US$ M)	38.8	31.9	27.2	27.6	20.8	19.7	9.3	4.3	5.0	4.2	3.8	4.2	-	**196.9**
Resource Development	(US$ M)	2.6	-	-	-	-	-	-	-	-	-	-	-	-	**2.6**
Capitalized Stripping	(US$ M)	38.4	63.5	63.9	64.8	12.7	-	-	-	-	-	-	-	-	**243.2**
Total Capex	**(US$ M)**	**124.4**	**101.4**	**101.9**	**117.6**	**33.5**	**77.2**	**66.7**	**4.3**	**13.4**	**4.2**	**3.8**	**4.2**	**-**	**652.4**

The Project's operating costs are estimated on the basis of the physical quantities of the mine plan, realistic equipment productivity assumptions, overall equipment efficiencies, and updated consumable prices. Average operating costs over the LOM and over the Five Year Plan (2020 to 2024) are shown in Table 1-6.

TABLE 1-6 LIFE OF MINE AND FIVE YEAR PLAN OPERATING COSTS

Area	Units	LOM Average	Five Year Plan (2020-2024)
Mining	US$/t mined	3.02	2.88
CIL Processing	US$/t milled	11.71	11.59
Heap Leach Processing	US$/t milled	4.35	-
G&A	US$/t milled	4.27	-

The LOM total cash cost per ounce of gold sold is US$778 while the AISC per ounce of gold sold is US$949.

No operating costs have been estimated for the Gossey deposit.

1.3.15 PROJECT EXECUTION PLAN

The CIL plant upgrade project is directly managed by the Essakane project team. The project was approved in July 2019 and is expected to be completed in Q3 2020.

For the heap leach project, the project execution will be directly managed by the IAMGOLD project management team. The engineering will be contracted out to qualified firms. The construction work will be mainly contracted out to local and regional contractors under the supervision of the project management team. Project control functions such as scheduling, cost control, procurement, project logistics, and site supervision will be executed directly by the IAMGOLD project management team.

An Owners' Steering Committee will be formed to oversee the Project. The Project will be initiated in 2025 and completed in 2026. Production is expected to start in 2027.

2 INTRODUCTION

IAMGOLD Corporation (IAMGOLD) has prepared a Feasibility Study (FS) for the mill optimization of a carbon-in-leach (CIL) circuit and the addition of a heap leach (HL) facility for its Essakane Gold Mine (Essakane or the Project) located in the Sahel region of Burkina Faso, West Africa. The results of the FS support an investment in a lower capital CIL plant de-bottlenecking project, which could increase current CIL plant throughput of 10.8 million tonnes per annum (Mtpa) rock equivalent by 6% to 11.7 Mtpa of 100% fresh (hard) rock CIL plant feed. The CIL crushing circuit would be used for the heap leach project at the end of CIL operations. The purpose of this Technical Report is to disclose the results of the FS and to support the disclosure of the August 31, 2019 Essakane Gold Mine Mineral Resource and Mineral Reserve estimate and the May 25, 2018 Mineral Resource estimate for the satellite Gossey deposit which is located approximately 12 km northwest of the Project inside the Essakane exploration properties. All currency in this report is US dollars (US$) unless otherwise noted.

IAMGOLD is a mid-tier mining company with four operating gold mines and several exploration properties on three continents.

Essakane consists of one mining permit (the Essakane Mining Permit), which contains the Essakane main zone (EMZ) deposit and the Falagountou West and Wafaka deposits. In April 2008, the Essakane Mining Permit was granted to Essakane S.A., a Burkinabé company created for the purpose of developing and operating the Project. In addition, there are six exploration permits, all located on contiguous ground (the Essakane Exploration Permits), which contain the Gossey deposit. IAMGOLD, through its wholly-owned subsidiary Essakane S.A., owns 90% of Essakane with the government of Burkina Faso (the government) holding the remaining 10%.

Essakane has been in operation since May 2009 and attained commercial production in July 2010.

2.1.1 SOURCES OF INFORMATION

This Technical Report was prepared by IAMGOLD and incorporates the work of IAMGOLD, Kappes, Cassiday & Associates (KCA), SRK Consulting (U.S.) Inc. (SRK), and G Mining Services Inc. (GMSI) Qualified Persons (QPs). The dates of personal inspections of the Project by the QPs are provided in Section 29 of this Technical Report.

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

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

2.1.1.1 LIST OF ABBREVIATIONS

Units of measurement used in this report conform to the metric system. All currency in this report is US dollars (US$) unless otherwise noted.

A	annum	kWh	kilowatt-hour
A	ampere	LIMS	Laboratory Information Management System
AACE	American Association of Cost Engineers	L	litre
AAS	atomic absorption spectroscopy	lb	pound
AC	air core	LFO	light fuel oil
Ai	Bond Abrasion work Index	LOM	life of mine
AISC	all-in sustaining cost	L/s	litres per second
bbl	barrels	m	metre
BLEG	bulk leach extractable gold	M	mega (million); molar
BMM	blast movement monitor	m^2	square metre
btu	British thermal units	m^3	cubic metre
BWi	Bond ball mill work index	μ	micron
BWS	bulk water storage	MASL	metres above sea level
°C	degree Celsius	mbgs	metres below ground surface
C$	Canadian dollars	MCC	motor control centre
cal	calorie	μg	microgram
CDF	cumulative distribution function	m^3/h	cubic metres per hour
cfm	cubic feet per minute	mgbs	metres below ground surface
Cg	graphitic carbon	mi	mile
CI	medium plasticity	min	minute
CIC	carbon-in-column	μm	micrometre
CIL	carbon-in-leach	mm	millimetre
CL	low plasticity	Mm^3	million cubic metres
cm	centimetre	MOA	Memorandum of Understanding
cm^2	square centimetre	MPa	mega pascal
COG	cut-off grade	mph	miles per hour
CoV	coefficient of variation	Mt	million tonnes
CRM	certified reference material	Mtpa	million tonnes per year
CWi	Bond Crusher Impact	MWMT	meteoric water mobility testing

d	day
DD	diamond drilling
dia	diameter
DFS	Definitive Feasibility Study
DGPS	differential global positioning system
dmt	dry metric tonne
dwt	dead-weight ton
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Plan
°F	degree Fahrenheit
FOS	factor of safety
FSE	fundamental sampling error
ft	foot
ft^2	square foot
ft^3	cubic foot
ft/s	foot per second
g	gram
G	giga (billion)
Gal	Imperial gallon
GC	grade control
g/L	gram per litre
Gpm	Imperial gallons per minute
GSE	grouping and segregation error
g/t	gram per tonne
gr/ft^3	grain per cubic foot
GRG	gravity recoverable gold
gr/m^3	grain per cubic metre
ha	hectare
HDPE	high-density polyethylene
HFO	heavy fuel oil
HL	heap leach
HLF	heap leach facility
hp	horsepower
HPGR	high pressure grinding roll
hr	hour
Hz	Hertz
ID^2	Inverse Distance Squared
ID^3	Inverse Distance Cubed
IFRS	International Financial Reporting Standards
in.	inch
in^2	square inch
IP	induced polarization
IRA	inter ramp angle
IRR	internal rate of return

MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
NGO	non-government organization
NPV	net present value
NN	Nearest Neighbour
OCR	off channel reservoir
OK	Ordinary Kriging
oz	Troy ounce (31.1035g)
oz/st, opt	ounce per short ton
PMC	Community Management Program
ppb	part per billion
ppm	part per million
PPM	pore pressure grid
PRF	rehabilitation and closure plan
PSD	particle size distribution
psia	pound per square inch absolute
psig	pound per square inch gauge
PV	photovoltaic
PVC	polyvinyl chloride
QA/QC	quality assurance and quality control
QXRD	quantitative x-ray diffraction analyses
QP	Qualified Person
RAB	rotary air blast
RAP	
RC	reverse circulation
RCD	holes pre-collared with RC then completed by DD
RL	relative elevation
RQD	rock quality designation
s	second
SABC	semi-autogenous grinding, ball mill grinding, pebble crusher grinding
SAG	semi-autogenous grinding
st	short ton
STD	standard deviation
stpa	short ton per year
stpd	short ton per day
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
TRS	tailings reclaim sump
TSF	tailings storage facility
UCS	unconfined compressive strength
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt

J	joule	VTEM	Versatile Full Waveform Time-Domain Electromagnetic
k	kilo (thousand)	VWP	vibrating wire piezometer
kcal	kilocalorie	W	watt
kg	kilogram	WAD	weak acid dissociable
km	kilometre	WBS	work breakdown structure
km^2	square kilometre	wmt	wet metric tonne
km/h	kilometre per hour	WRD	waste rock dump
kPa	kilopascal	wt%	weight percent
kV	kilovolt	yd^3	cubic yard
kVA	kilovolt-amperes	yr	year
kW	kilowatt		

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by IAMGOLD and incorporates the work of IAMGOLD, KCA, SRK, and GMSI QPs. For the purpose of this report, IAMGOLD QPs have relied on in-house non-QP personnel for property ownership information which has been disclosed in Section 4 and the Summary of this Technical Report.

4 PROPERTY DESCRIPTION AND LOCATION

Essakane is located in Burkina Faso at the boundary of the Oudalan and Seno provinces in the Sahel region and is approximately 330 km northeast of the capital, Ouagadougou. It is situated approximately 42 km east of the nearest large town and the Oudalan capital of Gorom-Gorom and near the village of Falagountou to the east (Figure 4-1). The property's latitude and longitude are 14º 23' N and 0º 04' E.

Essakane consists of one mining permit (the Essakane Mining Permit), which contains the Essakane main zone (EMZ) deposit, the Falagountou West deposit, and the Wafaka deposit. The mining permit is surrounded by six exploration permits (the Essakane Exploration Permits) belonging to Essakane Exploration SARL, the exploration subsidiary of IAMGOLD working in the region of the Project.

The satellite Gossey deposit is located inside the Essakane Exploration SARL properties. The Gossey deposit is located on the Koriléna and the Lao Gountouré 2 permits approximately 12 km northwest of the Essakane EMZ deposit.

4.1 MINING PERMIT

The mining and exploration permits comprising the Project are subject to Burkina Faso's 2015 Mining Code No.3 036-2015/CNT, dated June 26, 2015 (the Burkina Faso Mining Law).

In April 2008, following the filing by Orezone Resources Inc. (Orezone Resources) of the 2007 Essakane Definitive Feasibility Study (DFS), an Environmental and Socio-Economic Impact Assessment (ESIA), and the obtaining of the Essakane Environmental Permit, the government granted to Essakane S.A. the Essakane Mining Permit over an area of 100.2 km^2 containing the EMZ deposit, the Falagountou West deposit, and the Wafaka deposit. The Essakane Mining Permit is valid for a period of 20 years and is renewable every five years until the Mineral Reserves have been depleted.

The Essakane Mining Permit's perimeter is defined by UTM coordinates of the corner posts as listed in Table 4-1 and the permit's limit is shown in Figure 4-2.

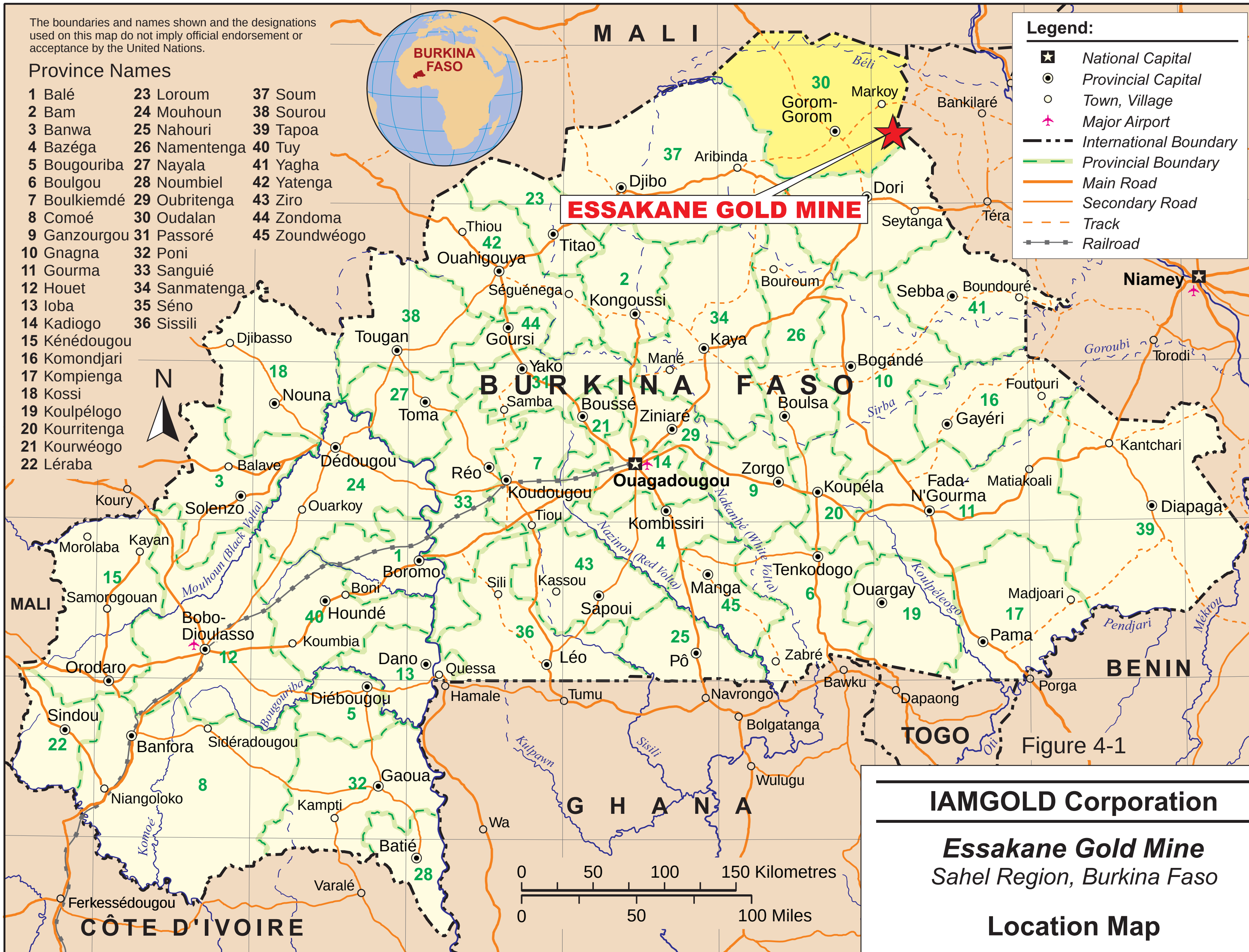
TABLE 4-1 ESSAKANE MINING PERMIT BOUNDARY COORDINATES

Points	Datum	Zone	X	Y
A	Adindan BF	31P	177,115	1,592,488
B	Adindan BF	31P	180,607	1,592,488
C	Adindan BF	31P	180,607	1,594,564
D	Adindan BF	31P	188,770	1,594,564
E	Adindan BF	31P	188,770	1,592,379
F	Adindan BF	31P	194,430	1,592,379
G	Adindan BF	31P	194,367	1,587,187
H	Adindan BF	31P	181,104	1,587,187
I	Adindan BF	31P	181,104	1,589,186
J	Adindan BF	31P	177,115	1,589,186

According to the Mining Law of Burkina Faso, a mining convention must be negotiated between the mining permit owner and the government before operations can begin. The mining convention outlines the governmental commitments, operational tax regime, and obligations of the company to Burkina Faso. Once executed, the mining convention cannot be changed without the mutual agreement of both parties. If tax law changes are promulgated, the mining company may choose to continue with the current terms of the mining convention or adopt the new terms if they are deemed more favourable. The mining convention between Essakane S.A. and the government was signed on July 14, 2008.

The new Burkina Faso Mining Code was approved by the transitional government and came into effect on June 16, 2015, however, an application decree is required for the Mining Code to be operational. IAMGOLD does not expect the new Mining Code to have a material impact on Essakane S.A., as it has fiscal stability clauses in its existing mining convention.

Essakane S.A. is a Burkinabé company created for the purpose of developing and operating the Project. IAMGOLD owns a 90% interest in Essakane S.A., while the government has a 10% free-carried interest. In addition, the government receives a 3% royalty on the revenues from mineral production if the gold price is below US$1,000/oz, 4% if the gold price is between US$1,000/oz and US$1,300/oz, and 5% if the gold price is greater than or equal to US$1,300/oz. The government also collects various taxes and duties on the imports of fuels, supplies, equipment, and outside services, as specified by the Burkina Faso Mining Law.



IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Location Map

January 2020

Source: Map No. 4230, United Nations, 2004.

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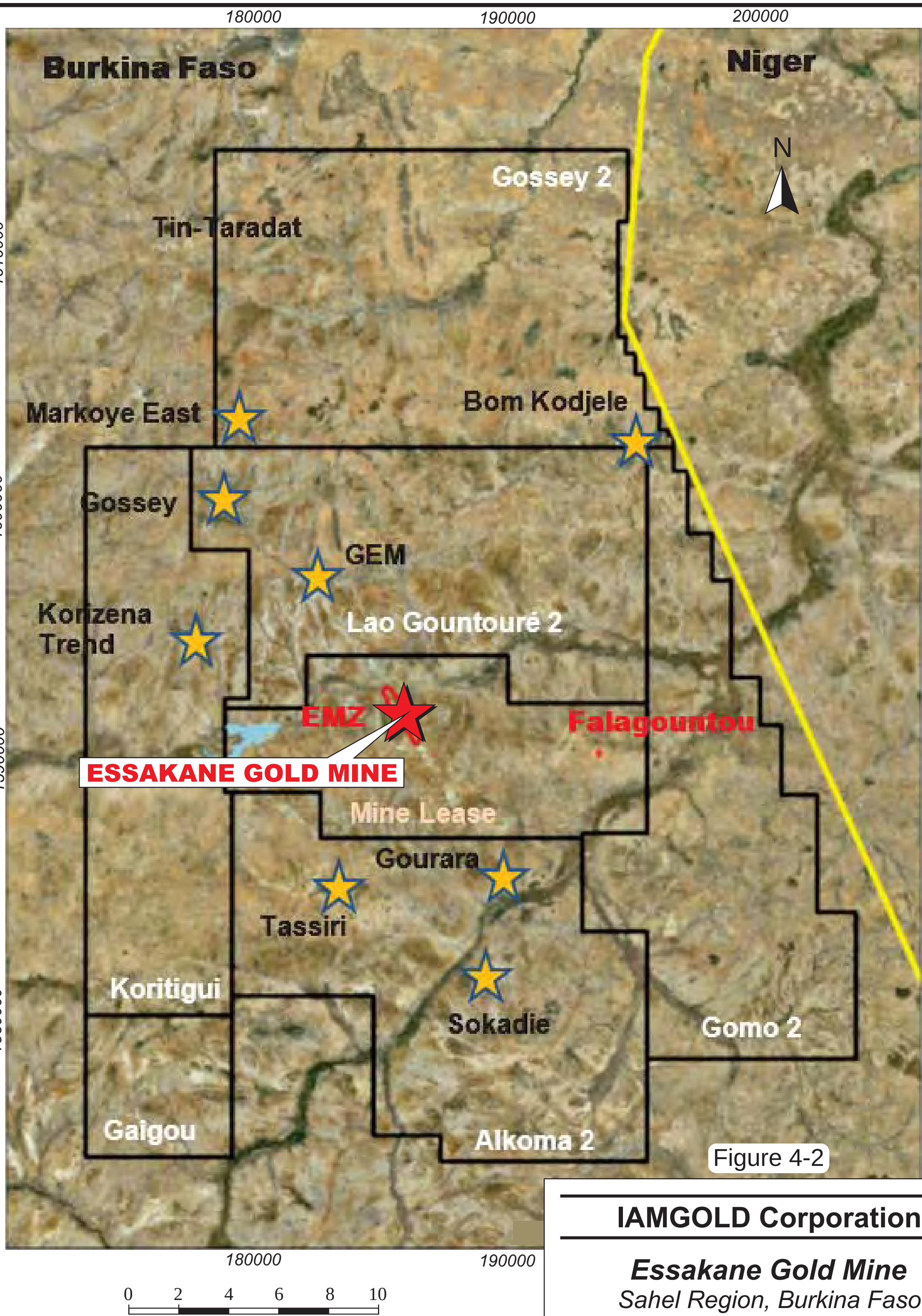

Figure 4-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Essakane Mining and Exploration Permits

January 2020

Source: IAMGOLD, 2019.

4.2 EXPLORATION PERMITS

The Essakane Mining Permit is surrounded by the Essakane Exploration Permits, which currently cover a total area of 906.12 km^2.

The Burkina Faso Mining Law gives the exploration permit holder the exclusive right to explore for the minerals requested on the surface and subsurface within the boundaries of the exploration permit. Exploration permits are guaranteed by the law and its associated arrêtés (decrees) provided that the permit holder complies with reporting requirements and annual exploration expenditures totalling 270,000 CFA francs per km^2, or approximately $650/$km^2$.

The exploration permit also gives the holder the exclusive right, at any time, to convert the exploration permit into a mining permit, in accordance with the law. Each mining permit application requires a separate FS, however, there are precedents in Burkina Faso for variations to this rule (e.g., Etruscan's Youga Project, now owned by Avesoro).

Exploration permits are valid for a period of three years from date of issue and may be renewed for two more consecutive terms of three years each for a total of nine years; however, on the second renewal, at least 25% of the original area must be relinquished.

The Essakane Exploration Permits have been granted by the Minister of Mines, Quarries, and Energy (MMCE) as a decree under Burkina Faso's 2003 Mining Code (Code Minier, No. 31–2003/AN dated May 8, 2003). Five of the seven Essakane Exploration Permits were granted by the MMCE in November 2009 for an initial three year term ending in November 2012 and were approved for renewal by the MMCE for the first three year term on December 18, 2012. The request for a second renewal was submitted to the MMCE on August 18, 2015. For three exploration permits (Dembam 2, Gomo 2, and Alkoma 2), 25% of the initial surface area was relinquished, whereas for two (Gossey 2 and Lao Gountouré 2), a special request was submitted to the MMCE to keep the original surface area.

In September 2018, a request for the exceptional extension of the second renewal for another three years period was submitted to the MMCE for Alkoma 2, Gomo 2, Lao Gountouré 2, and Gossey 2. The different grant decrees were approved on May 2, 2019. The Dembam 2 permit was returned to the government.

The sixth Essakane Exploration Permit (Korizéna permit) was approved for renewal for a second three year term on December 18, 2012, and 25% of the original surface area covered by that permit was relinquished. An application for a new permit on the relinquished area was subsequently filed and approved by the MMCE on May 6, 2013. On August 18, 2015, a request for extending the actual surface area of the Korizéna permit for another three year period was submitted to, and eventually approved by, the MMCE. In September 2018, a new permit request was submitted for Koritigui (former Korizéna) to the MMCE for approval. The request was made under IAMGOLD Exploration Mali. As of the date of this Technical Report, the grant decree is pending.

The seventh permit (Gaigou permit) was granted on May 6, 2013 by Ministerial Decree 2013/000076/MME/SG/DGMGC, and subsequently renewed in late 2016. On March 25, 2019, the second renewal of Gaigou was submitted to Minister for approval. A total of 25% of the original surface was relinquished. The grant decree is pending.

At the completion of the renewal process the total surface area of the Essakane Exploration Permits is 906.12 km^2.

The Gossey deposit is located on the Korizéna and the Lao Gountouré 2 permits.

The exploration permits are presently in good standing and Essakane S.A. has been issued with Certificate #1587/2007 (Issue date 04/10/2007) by Mr. Seydou Balama at the Office Notarial in Ouagadougou.

The decree numbers and expiry dates are listed in Table 4-2 and the Essakane Exploration Permit coordinates are listed in Table 4-3. Since December 31, 2018, an application for a decree was published on May 24, 2012 that stipulates a new geodesic and altimetry referential for Burkina Faso (decree: 2012 - 443/PRES/PM/MHU/MID/MEF). The coordinates of Essakane Exploration Permits listed in Table 4-3 are already in the new system.

TABLE 4-2 ESSAKANE EXPLORATION PERMIT DETAILS

Permit Name	Decree Granted	Date Granted	Status	Surface area (km²)	Decree Renewed	Renewal Date	Expiry Date
Alkoma 2	09/262/MCE/SG/ DGMGC	24/11/2009	Extension exceptional Second renewal	186.57	2019-034/MMC/SG/ DGCM	25/11/2018	24/11/2021
Dembam 2	09/263/MCE/SG/ DGMGC	24/11/2009	Returned to Burkina Faso government				
Gaigou	2013/000076/MME/SG/ DGMGC	06/05/2013	Second renewal	35.45	Pending		
Gomo 2	09/261/MCE/SG/ DGMGC	24/11/2009	Extension exceptional Second renewal	149.49	2019-035/MMC/SG/ DGCM	25/11/2019	24/11/2021
Gossey 2	09/260/MCE/SG/ DGMGC	24/11/2009	Extension exceptional Second renewal	215.00	2019-033/MMC/SG/ DGCM	25/11/2018	24/11/2021
Koritigui (Ex Korizéna)	06/135/MCE/SG/ DGMGC	21/11/2006	New permit	147.59	Pending		
Lao Gountouré 2	09/264/MCE/SG/ DGMGC	24/11/2009	Extension exceptional Second renewal	172.02	2019-036/MMC/SG/ DGCM	25/11/2018	24/11/2021
6 Permits				**906.12**			

TABLE 4-3 ESSAKANE EXPLORATION PERMIT COORDINATES

Permit Name	Points	Datum	X	Y	Surface Area (km²)
Alkoma 2	A	ITRF 2008	762 700	1 587 600	186.57
	B	ITRF 2008	766 400	1 587 600	
	C	ITRF 2008	766 400	1 585 700	
	D	ITRF 2008	777 300	1 585 700	
	E	ITRF 2008	777 300	1 581 800	
	F	ITRF 2008	780 000	1 581 800	
	G	ITRF 2008	780 000	1 572 200	
	H	ITRF 2008	771 400	1 572 200	
	I	ITRF 2008	771 400	1 573 300	
	J	ITRF 2008	768 600	1 573 300	
	K	ITRF 2008	768 600	1 579 100	
	L	ITRF 2008	762 700	1 579 100	
Gaigou	A	ITRF 2008	756 600	1578 300	35.38
	B	ITRF 2008	762 700	1578 300	
	C	ITRF 2008	762 700	1 578 300	
	D	ITRF 2008	756 600	1 572 500	
Gomo 2	A	ITRF 2008	780 000	1 602 000	149.49
	B	ITRF 2008	781 100	1 602 000	
	C	ITRF 2008	781 100	1 601 000	
	D	ITRF 2008	781 700	1 601 000	
	E	ITRF 2008	781 700	1 598 400	
	F	ITRF 2008	782 700	1 598 400	
	G	ITRF 2008	782 700	1 596 400	
	H	ITRF 2008	783 500	1 596 400	
	I	ITRF 2008	783 500	1 592 400	

Permit Name	Points	Datum	X	Y	Surface Area (km²)
	J	ITRF 2008	784 700	1 592 400	
	K	ITRF 2008	784 700	1 590 400	
	L	ITRF 2008	785 600	1 590 400	
	M	ITRF 2008	785 600	1 587 500	
	N	ITRF 2008	787 100	1 587 500	
	O	ITRF 2008	787 100	1 582 600	
	P	ITRF 2008	788 700	1 582 600	
	Q	ITRF 2008	788 700	1 576 500	
	R	ITRF 2008	780 000	1 576 500	
	S	ITRF 2008	780 000	1 581 800	
	T	ITRF 2008	777 300	1 581 800	
	U	ITRF 2008	777 300	1 585 900	
	V	ITRF 2008	780 000	1 585 900	
Koritigui (Ex Korizéna)	A	ITRF 2008	756 265	1 602 270	147.59
	B	ITRF 2008	760 932	1 602 270	
	C	ITRF 2008	760 932	1 597 658	
	D	ITRF 2008	763 286	1 597 658	
	E	ITRF 2008	763 286	1 591 576	
	F	ITRF 2008	762 465	1 591 576	
	G	ITRF 2008	762 465	1 590 955	
	H	ITRF 2008	762 529	1 587 659	
	I	ITRF 2008	762 529	1 578 244	
	J	ITRF 2008	756 265	1 578 244	
Lao Gountouré 2	A	ITRF 2008	761 000	1 602 000	172.02
	B	ITRF 2008	780 000	1 602 000	
	C	ITRF 2008	780 000	1 591 300	
	D	ITRF 2008	774 200	1 591 300	
	E	ITRF 2008	774 200	1 593 300	
	F	ITRF 2008	765 800	1 593 300	
	G	ITRF 2008	765 800	1 591 100	
	H	ITRF 2008	762 400	1 591 100	
	I	ITRF 2008	762 400	1 591 500	
	J	ITRF 2008	763 400	1 591 500	
	K	ITRF 2008	763 400	1 597 700	
	L	ITRF 2008	761 000	1 597 700	
Gossey 2	A	ITRF 2008	762 000	1 614 400	215
	B	ITRF 2008	779 200	1 614 400	
	C	ITRF 2008	779 200	1 611 400	
	D	ITRF 2008	778 800	1 611 400	
	E	ITRF 2008	778 800	1 606 600	
	F	ITRF 2008	779 100	1 606 600	
	G	ITRF 2008	779 100	1 605 900	
	H	ITRF 2008	779 500	1 605 900	
	I	ITRF 2008	779 500	1 605 000	

Permit Name	Points	Datum	X	Y	Surface Area (km^2)
	J	ITRF 2008	779 900	1 605 000	
	K	ITRF 2008	779 900	1 603 600	
	L	ITRF 2008	780 500	1 603 600	
	M	ITRF 2008	780 500	1 602 700	
	N	ITRF 2008	780 900	1 602 700	
	O	ITRF 2008	780 900	1 602 500	
	P	ITRF 2008	781 100	1 602 500	
	Q	ITRF 2008	781 100	1 602 000	
	R	ITRF 2008	762 000	1 602 000	
6 Permits					**906.12**

4.3 SURFACE RIGHTS

Surface rights in the area of the Essakane Mining Permit belong to the State of Burkina Faso. Utilization of the surface rights is granted by the Essakane Mining Permit under the condition that the current users are properly compensated. All the taxes relating to Essakane S.A.'s Mining Rights have been paid to date and the concession is in good standing.

4.4 PERMITTING REQUIREMENTS AND STATUS OF PERMITS

Table 4-4 provides a description of the environmental and mining permits required at Essakane and their respective status.

TABLE 4-4 ENVIRONMENTAL AND MINING PERMIT REQUIREMENTS AND STATUS

Legal References	Requirements	Status
Order No. 2001-342 on the scope, content, and procedure of the Environmental Impact Study Statement	ESIA (2007)	Completed
	Resettlement Plan 2007	Completed
	Public Consultation (2007)	Completed
	Environmental Feasibility Ministerial Order (2007)	Completed
	Addendum (2008)	Completed
	Addendum 2012	Completed
	Resettlement Plan 2012	Completed
	Public Consultation (2013)	Completed
	COTEVE Meeting	Completed
	Environmental Feasibility Ministerial Order (2014)	Completed
Order No. 2007-845/PRES/PM/MCE/MEF	Mining Environment Preservation	Completed

Legal References	Requirements	Status
of December 26, 2007 implementing the management of a Mining Environment Preservation and Rehabilitation Fund.	And Rehabilitation Fund	
Law No. 31/-2003/AN implementing the Mining Code in Burkina Faso	Mining Permit (2008)	Completed
	Rehabilitation and Closing	Ongoing
	Environmental Discharge	Ongoing

4.5 DISCUSSION

Other than the royalty on the revenues from mineral production to the government, IAMGOLD is not aware of any royalties, back in rights, payments, or other agreements and encumbrances to which the property is subject.

IAMGOLD is not aware of any environmental liabilities on the property. IAMGOLD has all required permits to conduct the proposed work on the property. IAMGOLD is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

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

5.1 TOPOGRAPHY, ELEVATION, AND VEGETATION

The Project area, specifically the area surrounding the EMZ deposit, is characterized by relatively flat terrain sloping gently towards the Gorouol River to the north of the EMZ deposit. The average elevation over the Project area is 250 metres above sea level (MASL). Vegetation consists mostly of light scrub and seasonal grasses. Deforestation has been significant, particularly in the area surrounding the original village of Essakane.

5.2 ACCESS AND PROXIMITY TO POPULATION CENTRES

Access from the capital city of Ouagadougou is via a 263 km paved road to the town of Dori, followed by approximately 63 km via a laterite road to Essakane. Access via the town of Gorom-Gorom, located 42 km to the west, is also possible. Within the Essakane Exploration Permits, access is via local tracks and paths, which are suitable for two wheel drive vehicles in the dry season, however, four wheel drive vehicles and trucks are required in the wet season. There is no operating railroad. An airstrip has been built on packed laterite within the fenced perimeter of the Project and daily flights are made between Essakane and Ouagadougou using an aircraft owned and operated by Essakane S.A.

There are no major commercial activities in the Project area and economic activity is confined to subsistence farming and artisanal mining.

5.3 CLIMATE AND LENGTH OF OPERATING SEASON

The Project is located in the northeast of Burkina Faso and the climate is typically Sahelian, hot, sunny, dry, and somewhat windy all year long. Temperatures range from 10°C to 50°C, with annual pan evaporation rates of 3,000 mm/year. The mean annual rainfall is 397.5 mm with an estimated 100 year maximum of 171 mm in a 24 hour period.

A wet season occurs between late May and September, and the mean annual runoff in the Gorouol River is conservatively estimated to be 91 million m^3/year. Rainfall is sporadic or absent throughout the rest of the year. Weather conditions have had minimal impacts on the mining operations thus far, however, proper planning is required to ensure an adequate water supply during the dry season.

5.4 SURFACE AREA AND PHYSICAL RESOURCES

The Essakane Mining Permit covers an area of 100.2 km^2 and has ample surface area for mining operations. Figure 5-1 shows the location of the EMZ, Falagountou West, and Wafaka deposits, CIL plant, tailings storage facility (TSF), and waste rock dumps (WRDs).

Electricity to the EMZ deposit is supplied by on-site diesel generators. A 26 MW power plant, fueled with heavy fuel oil (HFO), was built for the production phase. An additional 31 MW of capacity was added in 2013 to power the expanded milling circuit.

In 2018, a new photovoltaic solar farm was commissioned. This power plant provides 15 MW to the Project without any carbon emission and helps to reduce the Project's reliance on fossil fuels while also protecting the environment.

Satellite communication is available at the Project. The main sources of water are the Gorouol River during the rainy season and well fields around the Essakane pit and near the Gorouol River. Water is pumped from wells (boreholes) in sufficient quantities for exploration drilling and the mining camp.

Essakane S.A. initiated local training programs for artisans. Unskilled labour was sourced locally with skilled labour drawn from Burkina Faso at large. Approximately 90 to 150 expatriates from North America and Europe were required in the initial years of production, however, that number decreased as local Burkinabé workers acquired the expertise and experience to replace the expatriate employees.

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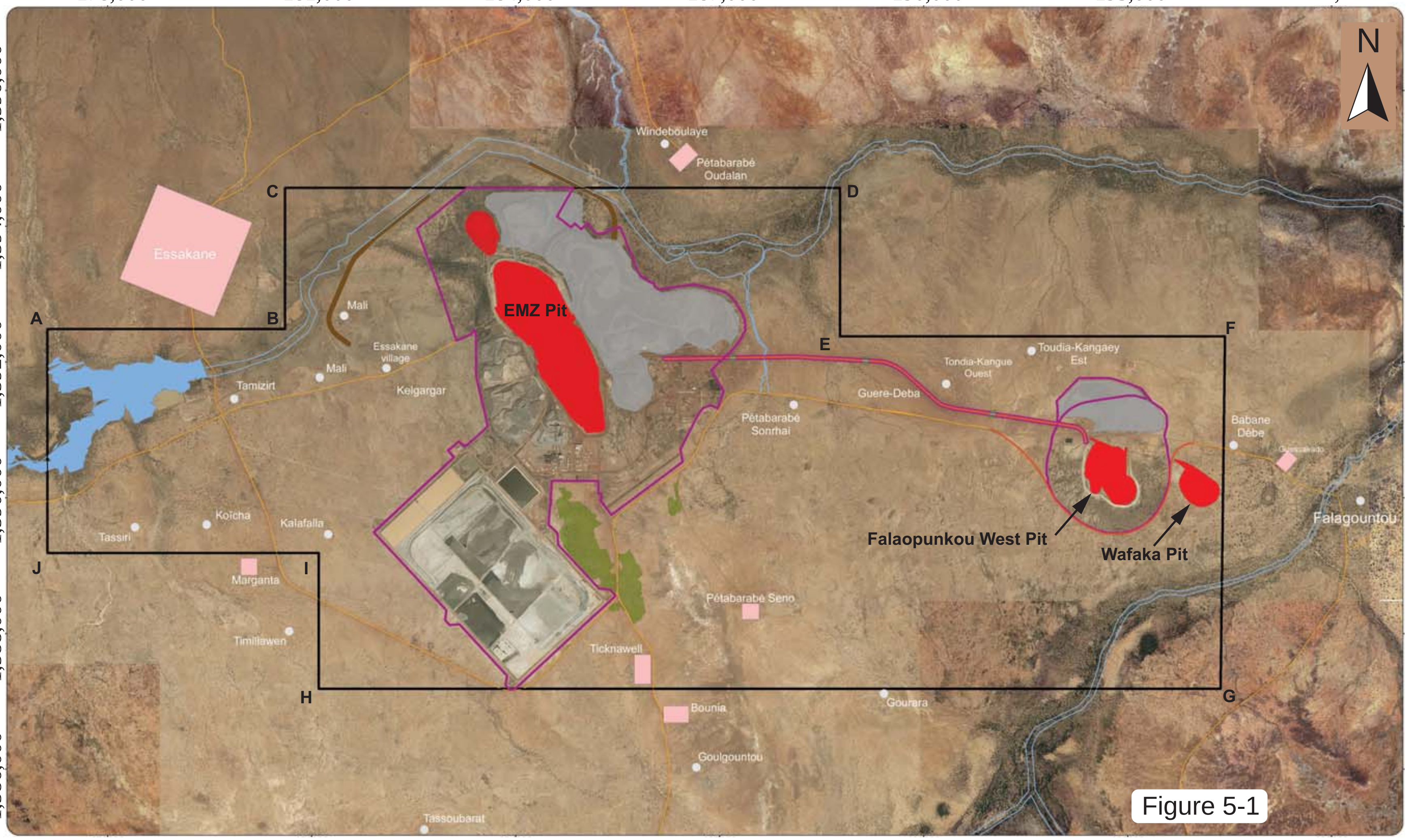

Figure 5-1

Note: For Mining Permit Boundary Coordinate see Table 4-1

Legend:

Mining Lease	*Village*
Pit	*Protection Dike*
Waste Dump	*Mining Road*
Resettlement Site	*Road*
Dam	*Departmental Road Diversion*
	Mining Fence
	River

0 1 2 3 4
Kilometres

UTM Zone 31, WGS 84

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Mine Infrastructure

January 2020

Source: IAMGOLD, 2019.

6 HISTORY

6.1 OWNERSHIP, EXPLORATION, AND DEVELOPMENT HISTORY

The EMZ deposit has been an active artisanal mining site since 1985. At its peak, up to 15,000 artisanal miners worked at the EMZ deposit.

The Bureau des Mines et de la Géologie du Burkina (BUMIGEB) undertook regional mapping and geochemical programs and arranged and financed a program of heap leach test work between 1989 and 1991. Compagnie d'Exploitation des Mines d'Or du Burkina (CEMOB) was granted the Essakane Mining Exploration Permit in 1991. The permit covered most of the area which is now included within the Essakane Mining Permit (excluding the Gomo permit).

CEMOB constructed a heap leach facility (HLF) in 1992 and produced 18,000 ounces of gold in 1993, however, production averaged between 3,000 and 5,000 ounces of gold per year thereafter. Efforts were also made to leach saprolite from the EMZ deposit, however, based on verbal accounts, leaching failed due to high cement consumption and solution blinding in the heaps.

BHP Minerals International Exploration Inc. (BHP) assisted CEMOB and explored the area from 1993 to 1996 under a proposed joint venture earn-in agreement. BHP excavated and sampled 26 trenches (4,903 m) along the EMZ deposit. Scout reverse circulation (RC) drilling was completed (including on the Falagountou and Gossey prospects), followed by RC drilling (7,404 m of vertical holes on a 100 m by 50 m grid) and a few diamond drill (DD) holes (1,462 m) in the main area of artisanal mining on the EMZ deposit.

Low gold prices and operational problems caused CEMOB to go into liquidation at the end of 1996 and BHP decided to withdraw from the project.

Upon CEMOB going into liquidation in 1996, Coronation International Mining Corporation (CIMC) secured the title and in July 2000, six new Essakane licences were granted to CIMC.

In September 2000, CIMC entered into an option agreement with Ranger Minerals (Ranger) pursuant to which Ranger undertook an exploration program, focusing on intensive rotary air blast (RAB) and RC drilling of an oxide resource between October 2000 and June 2001. RAB drilling (12,867 m) was used to locate drill targets at Essakane North, Essakane South, Falagountou, and Gossey. Follow up RC drilling at the EMZ deposit, amounting to 22,393 m, was completed along with 1,070 m of DD drilling on twins and extensions. Ranger mapped and sampled veins in the BHP trenches. In 2001, Ranger withdrew from the joint venture.

In 2002, CIMC merged with Orezone Resources. Orezone Resources became 90% owner of Essakane S.A.

Gold Fields Orogen Holding Ltd (Orogen), formerly known as Orogen Holdings (BVI) Limited, a subsidiary of GFL Mining Services Limited, entered into an Option Agreement with Orezone Resources in July 2002. Orezone Resources was the operator of Essakane until Gold Fields Essakane Limited (GF BVI) assumed management responsibilities in January 2006.

In 2006, GF BVI carried out an exploration program on the deposit which focussed on the quality of gold assays, geological modelling, and Mineral Resource estimation.

In April 2007, Orezone Resources, Orezone Inc., Orezone Essakane Limited, GF BVI, Orogen, and Essakane Limited (Essakane BVI) entered into a members agreement and eventually formed a joint venture.

GF BVI earned a 50% interest in Essakane BVI by spending the requisite $8 million on exploration. GF BVI increased its ownership to 60% of the Project by gaining a further 10% interest in Essakane BVI after Essakane BVI completed the Essakane DFS on September 11, 2007.

In October 2007, Orezone Resources entered into an agreement with GF BVI to acquire its 60% interest in Essakane. On November 26, 2007, Orezone Resources became the operator and owner of a 100% interest in Essakane, subject to the interest of the government.

In April 2008, after obtaining the Environmental Permit, and concluding a Memorandum of Understanding (MOA) with the local population, the Essakane Mining Permit was granted, which resulted in the transfer of the Project to Essakane S.A., a Burkinabé anonymous company, created for the purpose of owning and operating the Project. An updated Feasibility Study (2008 UFS) was completed on June 3, 2008.

In 2009, IAMGOLD acquired Orezone Resources and Essakane was transferred to IAMGOLD Essakane S.A. The 2008 UFS was readdressed to IAMGOLD. Commercial production started on July 16, 2010.

The ownership history is summarized in Figure 6-1.

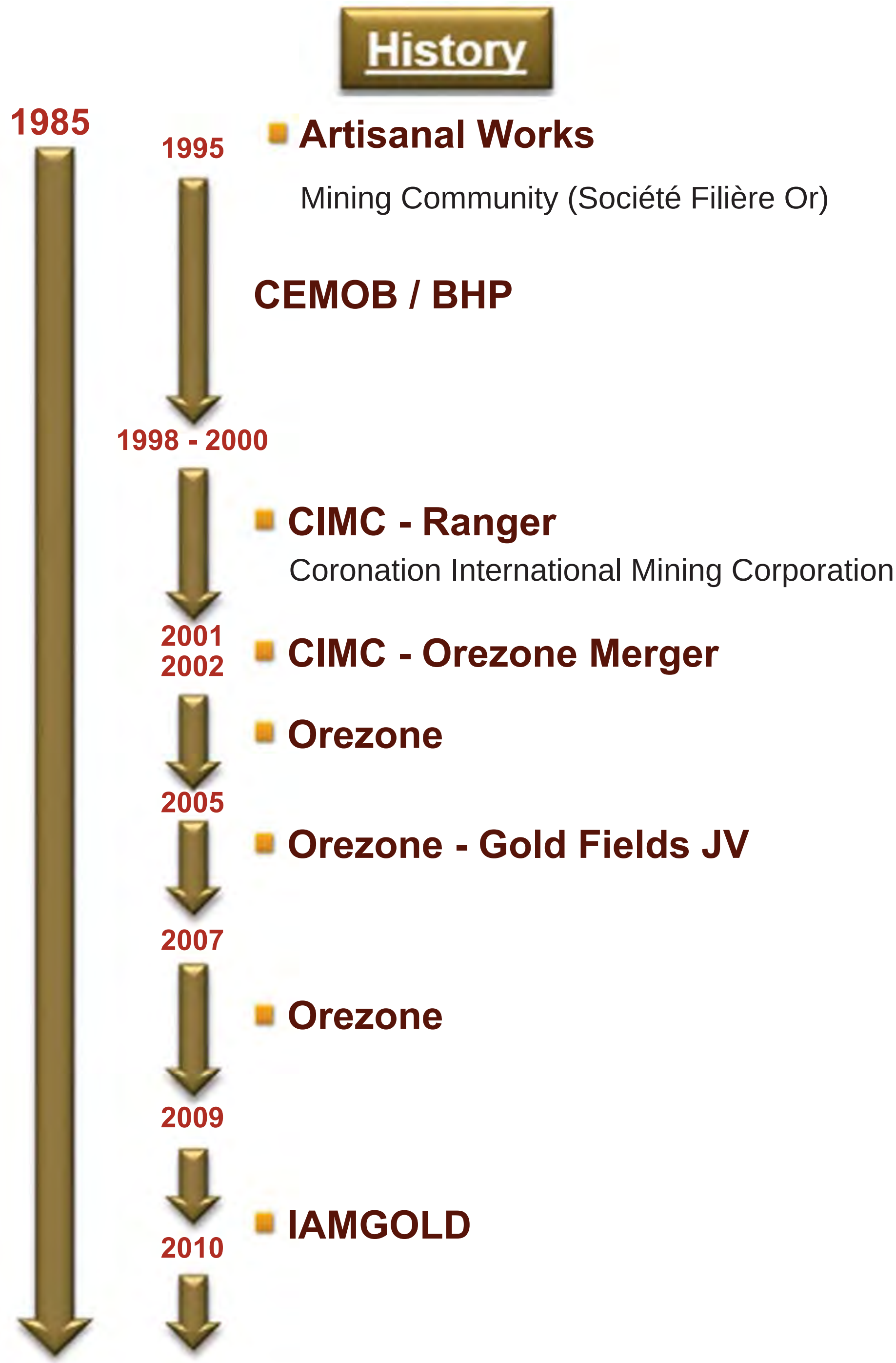

Figure 6-1

IAMGOLD Corporation

Essakane Gold Mine
Sahel Region, Burkina Faso
Ownership History

January 2020

Source: IAMGOLD, 2015.

6.2 HISTORICAL MINERAL RESOURCE ESTIMATES

Orezone Resources completed two JORC Mineral Resource estimates; the first was prepared by SRK (Cardiff) in 2004 and the second, by RSG Global (Perth) in 2006. Table 6-1 shows the historical Mineral Resources estimates. SRK (Cardiff) recognized that these tonnages were overstated by 15% due to incorrect allocation of densities to the weathering domains. In addition, some uncertainties were noticed about the quality of the historical assay data and poor quality assurance/quality control (QA/QC).

TABLE 6-1 HISTORICAL MINERAL RESOURCE ESTIMATES

Company	Cut-Off Grade (1 g/t Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
SRK (Cardiff) 2004	Indicated	30.5	2.0	1,910
	Inferred	4.4	2.0	290
RSG Global (Perth) 2006	Indicated	19.6	2.3	1,470
	Inferred	15.3	2.3	1,190

IAMGOLD is not treating these historical Mineral Resource estimates as current and they were superseded by the previous Mineral Resource estimate prepared for the 2008 UFS filed by IAMGOLD in March 2009, upon acquisition of the Project.

There are no historical Mineral Resource estimates for the Gossey deposit.

6.3 PAST PRODUCTION

From 1992 to 1999, heap leach processing of gravity rejects from the artisanal winnowing and washings was carried out by CEMOB. CEMOB placed a total of 1.01 million tonnes (Mt) of material on the heap leach pad at an average grade of 1.9 g/t Au and achieved 73% recovery during its ownership.

Table 6-2 shows CEMOB's heap leach plant production from 1992 to 1999 and Table 6-3 shows Essakane S.A.'s CIL plant production from 2010 to 2018. It is estimated that a total of 3.3 million ounces of gold has been produced since 1992.

Artisanal gold production continues currently, however, no reliable gold production statistics are available on the artisanal workings.

TABLE 6-2 CEMOB HEAP LEACH PLANT PRODUCTION 1992 TO 1999

Year	Tonnage (000 t)	Grade (g/t Au)	Ounces Produced (000 oz Au)
1992	42	4.5	6
1993	116	5.1	18
1994	157	1.7	8
1995	148	1.5	7
1996	257	1.0	8
1997	165	0.8	4
1998	72	1.4	3
1999	50	2.0	3
Total	**1,007**	**1.9**	**58**

TABLE 6-3 ESSAKANE MINE AND CIL PLANT PRODUCTION - 2010 TO 2018

Year	Tonnes Mined (000 t)	Grade Mined (g/t Au)	Tonnes Milled (000 t)	Grade Milled (g/t Au)	Ounces Produced (000 oz Au)
2010	10,097	1.05	2,973	1.49	136
2011	10,110	1.08	7,977	1.53	375
2012	9,562	1.04	10,762	1.10	350
2013	11,869	0.84	10,613	0.89	277
2014	12,580	0.98	11,897	1.06	369
2015	11,519	1.14	11,716	1.23	426
2016	11,374	1.21	12,005	1.22	419
2017	11,696	1.16	13,890	1.07	432
2018	13,866	1.12	13,031	1.18	450
Total	**102,673**	**1.07**	**94,864**	**1.16**	**3,234**

There is no past production on the Gossey deposit.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The geological setting of northeast Burkina Faso consists predominantly of Precambrian rocks of the Oudalan-Gorouol greenstone belt (Figure 7-1) which forms part of the Paleoproterozoic Baoulé-Mossi domain of the West African Craton and hosts numerous gold deposits including Essakane, Gossey, Korizéna, and Falagountou (Nkuna, 2009).

The Oudalan-Gorouol greenstone belt is bounded by intrusive granitic rocks belonging to the plutonic belt (Tshibubudze et al., 2010). Along its western edge, granitic–gneissic rocks are exposed in local tectonic thrust slices. The Birimian sedimentary and volcano-sedimentary sequences in the belt are dominated by metavolcanoclastic, greywacke, metaconglomerate, siltstone and shale, carbonate (dolomite), and volcanic unit pillowed basalts (Tshibubudze et al., 2009) (Figure 7-2).

The Oudalan-Gorouol greenstone belt is bounded and/or crosscut by several major north-northeast to northeast trending shear zones including the crustal-scale steeply east dipping Markoye Shear Zone, the Tin Takanet-Bellekcire Shear Zone, the Dori Shear Zone, and the Kargouna Shear Zone, etc. The Markoye Shear Zone located through the western portion of the belt trends north-northeast and separates Paleoproterozoic rocks on the east from older granite-gneiss terranes to the west (Tshibubudze et al., 2009).

Recent structural investigations in the northern part of the belt suggested that the Markoye Shear Zone has been affected by at least two phases of tectonic reactivation associated with two phases of regional deformation (Tshibubudze et al., 2009). The first deformation (D1) involved a northeast-southwest directed compression and resulted in the formation of north-northwest to northwest trending folds and thrusts during dextral-reverse displacement on the Markoye Shear Zone. This deformation predates the Eburnean Orogeny and is termed the Tangean Event dated at ca. 2170 Ma to 2130 Ma (Tshibubudze et al, 2009). The second deformation (D2) involved a period of northwest-southeast crustal shortening and sinistral-reverse displacement on the Markoye Shear Zone and is correlated to the ca. 2.0 Ga Eburnean Orogeny (Feybesse et al., 2006). D2 is characterized by northeast trending regional folds and a pervasive northeast trending foliation. D1 structures are compatible with pure-shear dominated transpression, while the D2 deformation is characterized by a switch to the strike-slip dominated east-west to west-northwest oriented transpressional regime (Tshibubudze et al., 2009; 2010) (Figure 7-4).

Gold mineralization is generally hosted in the hanging wall of northeast trending faults and/or northwest trending folds in metasiltstone, sandstone, and shale sequences and can be classified as orogenic gold deposits under the subclass of “intrusion-related” due to their proximity to plutonic masses (Nkuna, 2009). Gold deposits are most often related to transcurrent D2 shear zones and faults as these discontinuities have served as the main conduit for mineralized fluids. Gold is either disseminated or concentrated in quartz veins (Beziat et al., 2008). As with other Precambrian orogeny, the early fabrics were modified by the regional scale transcurrent shear zones D2, which acted as pathways during the gold mineralization events (Nkuna, 2009).

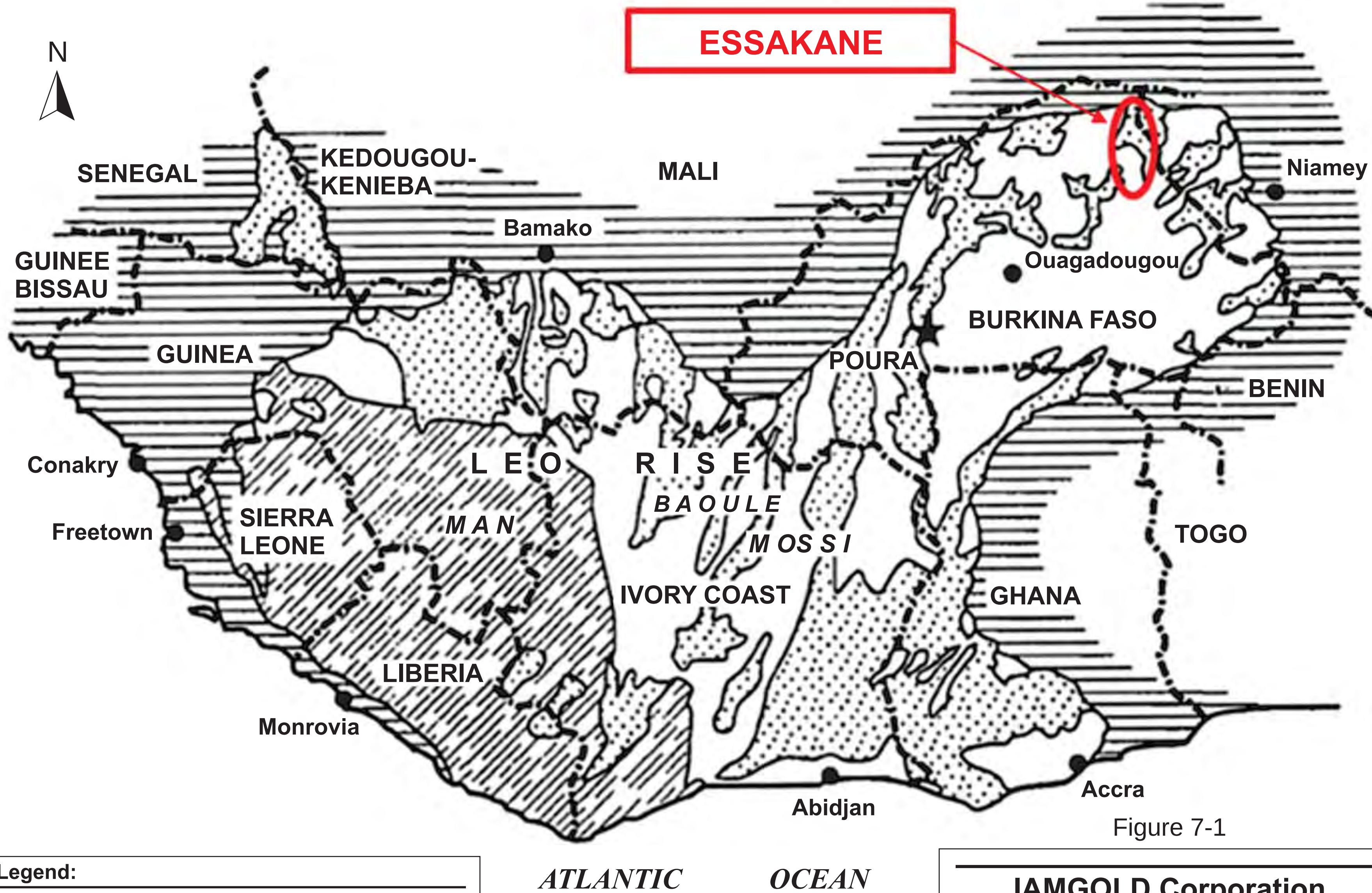

Figure 7-1

Legend:

- *Phanerozoic Cover*
- *Granites*
- *Bimimian Volcanic and Sedimentary Formations*
- *Archean*

0 100 200 300 400
Kilometres

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Location of Oudalan-Gorouol Greenstone Belt within West African Craton

January 2020

Source: Modified after Abouchami et al., 1990.

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Figure 7-2

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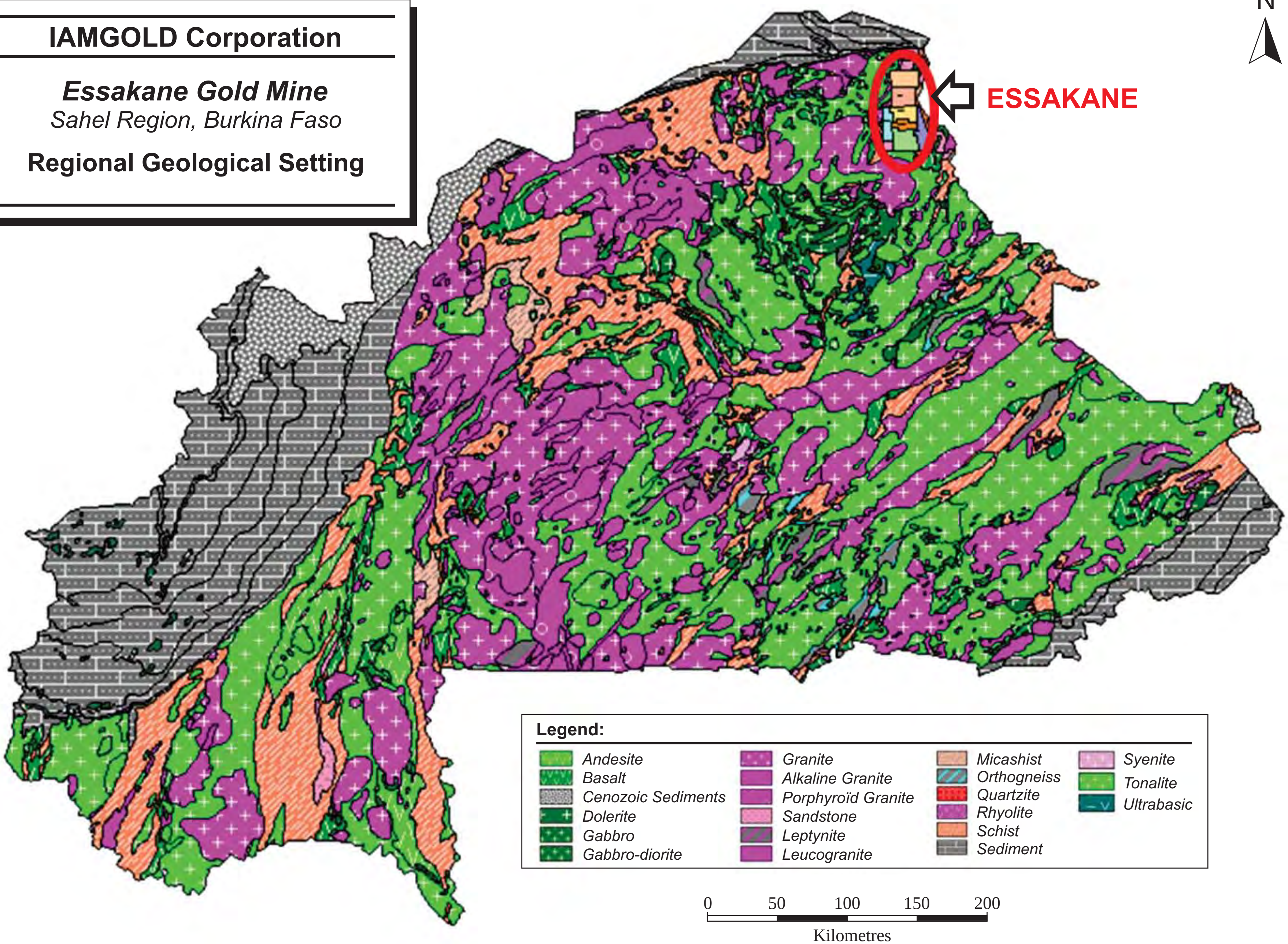
Essakane Gold Mine

Sahel Region, Burkina Faso

Regional Geological Setting



January 2020

Source: Modified after "carte géologique et minière du Burkina Faso à 1/1 000 000" BUMIGEB-BRGM, 2003).

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7.2 LOCAL GEOLOGY

Essakane lies in an outlier of folded sedimentary Birimian rocks, which are intruded in places by intermediate and mafic sills. The sediments in the district have been subdivided on the basis of lithology into deep water turbidites (the Birimian) and coarse clastic basin margin sequences (the Tarkwaian). The Birimian rocks consist of wackes, arenites and mudrocks (argillites), pebbly arenites, and minor tuffs, which have been metamorphosed to lower greenschist facies. Arenite is the dominant lithology. Intermediate intrusives occurring as sills are common and appear to predate all gold mineralization in the district. The Tarkwaian rocks are typically sandstones with thin intercalated bands of matrix-supported, polymictic conglomerates, however, they differ from the type of lithologies found in Ghana. In particular, the conglomerate matrices are not enriched in heavy minerals nor do they show the alteration mineral assemblages of Tarkwa and Iduapriem mines. Figure 7-3 shows the boundaries of the updated exploration permits comprising the property and the EMZ deposit current study area (highlighted in red) in context with the local geology. The bold red shape within the red perimeter is the crest line of the surface mine shell on the EMZ deposit.

The Birimian and Tarkwaian rocks are bounded to the west by the major north-northeast trending Markoye Fault and to the south by the Dori batholith. The Markoye Fault is thought to be a left-lateral wrench fault that was an active basin margin fault at the time of deposition of the sediments. Other regional faults in the district appear to trend northeast and west-northwest. Mesozoic age dolerite dykes are generally found in the latter. Fold axes within the Birimian rocks trend northwest and north except in the south where units are re-folded adjacent to the batholith (Figure 7-4).

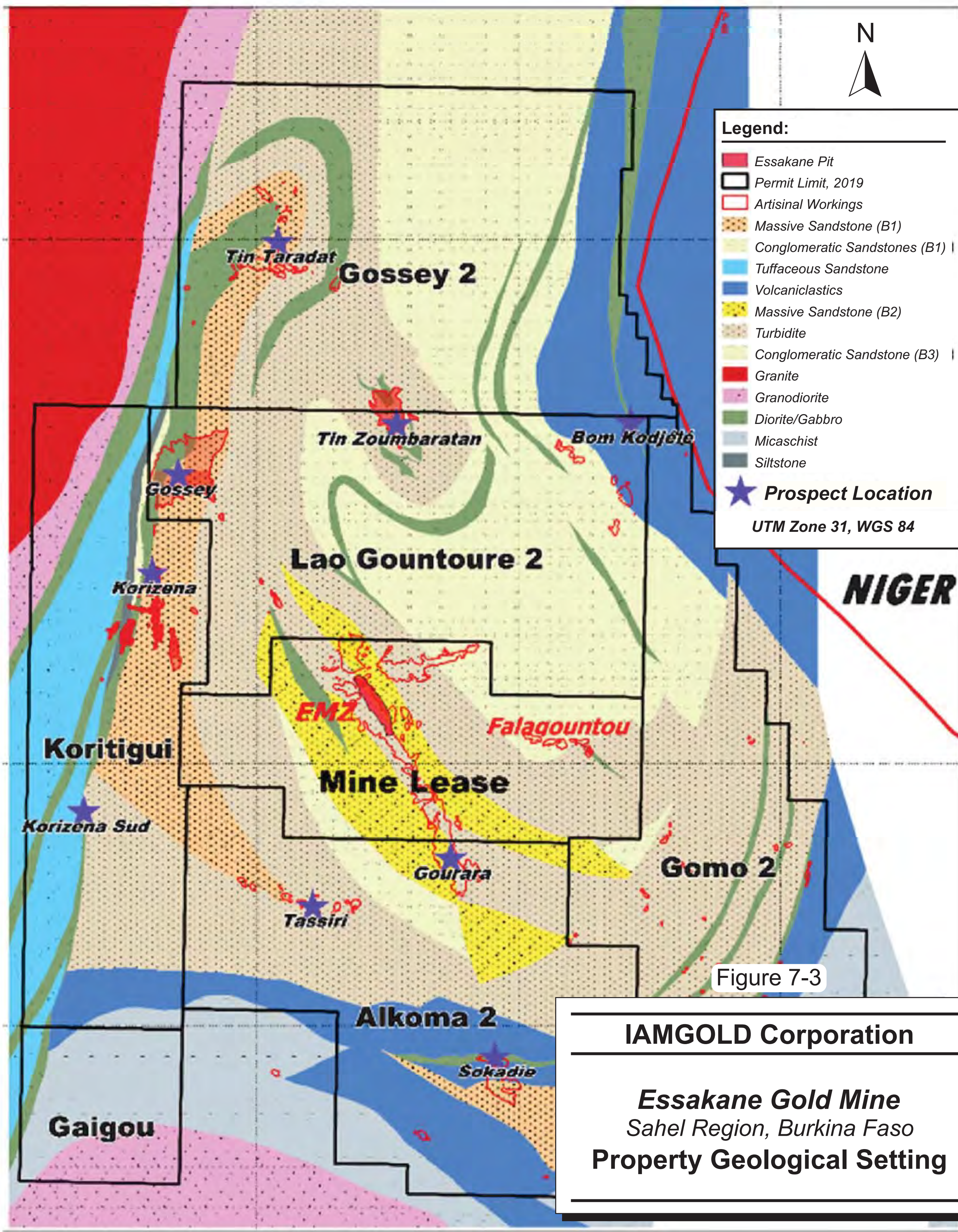


Figure 7-3

IAMGOLD Corporation

Essakane Gold Mine
Sahel Region, Burkina Faso
Property Geological Setting

January 2020

Source: IAMGOLD, 2019.

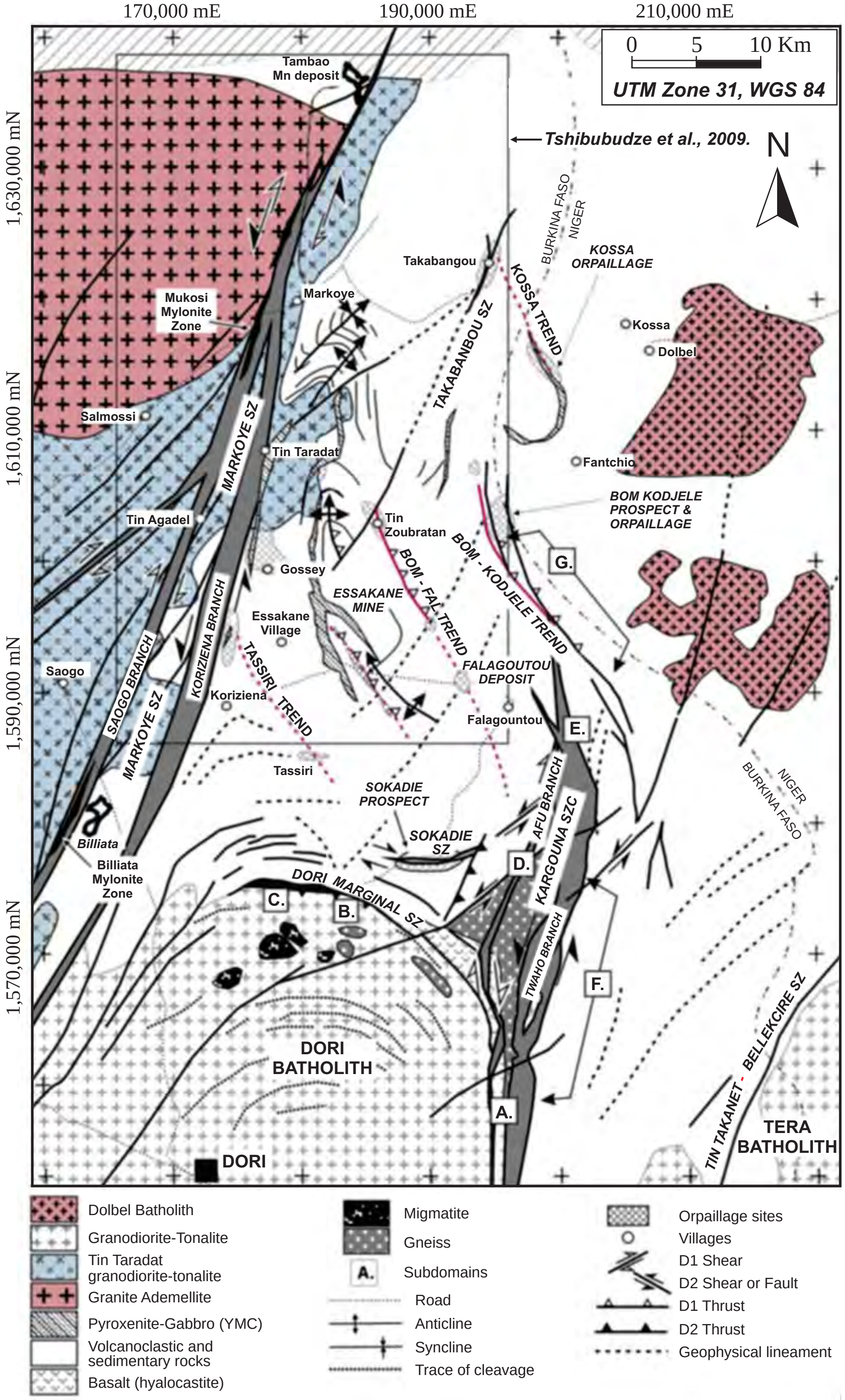


Figure 7-4

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Local Geological Map of the Oudalan-Gorouol Greenstone Belt

January 2020 ***Source: Tshibubudze et al., 2009.***

7.3 PROPERTY GEOLOGY

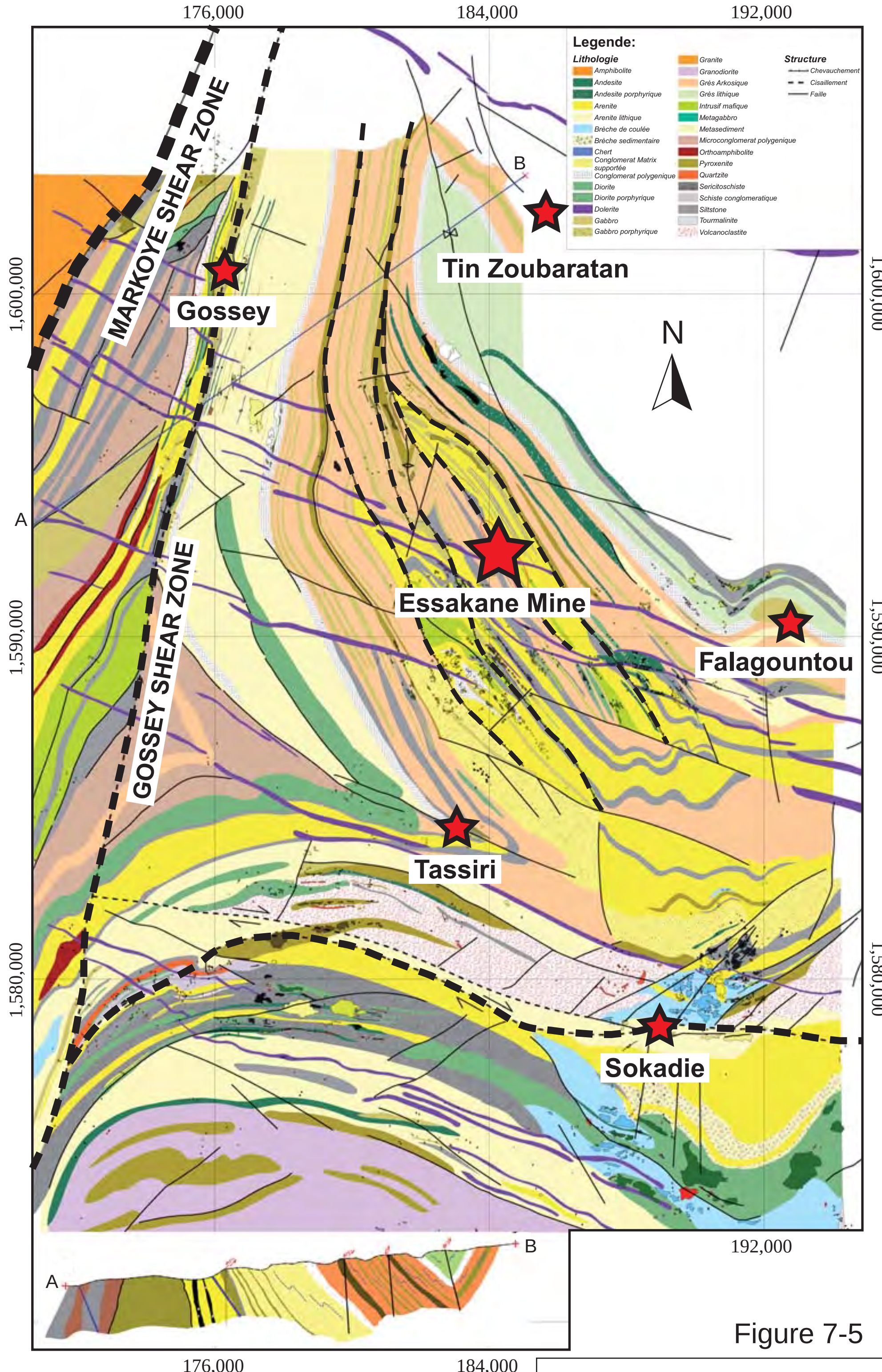
The Project occurs in the Paleoproterozoic Oudalan-Gorouol greenstone belt in northeast Burkina Faso. The stratigraphy can be subdivided into a succession of lower-greenschist facies metasediments (argillites, arenites, and volcaniclastics), conglomerate, subordinate felsic volcanics, and an overlying Tarkwaian-like succession comprised of siliciclastic metasediments and conglomerate. Each succession contains intercalated mafic intrusive units that collectively comprise up to 40% of the total stratigraphic section.

Gold prospects on the permits (shown as blue stars in Figure 7-3) occur exclusively in Birimian rocks and are generally associated with quartz veining on the margins of mafic and intermediate sills. Exceptions are the EMZ deposit and the Sokadie prospects (the latter on the Alkoma 2 permit). The EMZ deposit (Figure 7-3) is characterized by quartz veining in a folded turbidite succession of arenite and argillite. At the Sokadie prospect, the veins occur in a sheared volcaniclastic unit between undeformed andesite and metasediments. Gold has generally been found to occur with quartz veining on the contacts of rock units with contrasted competency and as filling of brittle fractures in folded sediments.

The region preserves evidence for at least two regional deformational events. D1 structural elements such as the Essakane-host anticline are refolded by a series of north-northeast trending F2 folds. Later localized deformation occurs near the margin of a calc-alkaline batholith in the south of the Project area. The Markoye fault trends north-northeast through the western portion of the Project area and separates the Paleoproterozoic rocks from an older granite-gneiss terrane to the west.

The Korizéna prospect is situated approximately 10 km west of the Essakane deposit and is the southern continuity of the Gossey deposit. Both have similar geology. For the purposes of this Technical Report, the Korizéna and Gossey prospects form the same mineralized trend.

The geology of the Gossey deposit includes sequences of detrital sedimentary rocks (quartz-arenites, quartz-feldspathic sandstones, fine to microconglomeratic lithic sandstones with polygenic clasts, lithic sandstones with pelitic fragments, greywackes, argillites/ graphitic siltstones) interbedded with igneous rocks (gabbro, diorite, gabbro-diorite, andesite) mainly arranged as sills and dykes (Allou et al. 2013). Structurally, this prospect is controlled by the Markoye fault especially its branch named the Gossey-Korizéna shear zone. The Markoye fault is a regional structure close to the prospect characterised by a predominantly NNE-SSW reverse directional sinistral shear corridor. The main deformation structures observed on this corridor are schistosity and shear planes. The effect of weathering makes it very difficult to measure these in the field. These measurements were mainly carried out on the oriented core and confirmed that the schistosity planes were parallel to the stratification. A more detailed analysis of these planes (stratification and schistosity) by zone reveals a progressive flexure of the orientations, going from the NNE-SSW with dipping an average 60° east in the north, towards the NE-SW with a dip subvertical and slightly inclined westwards to the south (Allou and Al. 2013). In addition to this schistosity, other structures are observed: asymmetrical sheared quartz veins (boudins), tension veins arranged in echelon and sigmoid clasts. This corridor is also marked by quartz veins of decametric size and oriented from N10° to N40°. Sometimes these veins are parallel to the shear corridor and have a brecciated structure characterized by crushed quartz taken from siliceous cement.



0 2 4 6 8

Kilometres

UTM Zone 31, WGS 84

Figure 7-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Property Geology

January 2020

Source: Modified after G. Iiboudo & J. Roger, 2012.

7.4 MINERALIZATION

7.4.1 EMZ DEPOSIT MINERALIZATION

The EMZ deposit is a greenstone hosted orogenic gold deposit. Specifically, it is a quartz-carbonate stockwork vein deposit hosted by a folded turbidite succession of arenite and argillite (Figure 7-6).

The laminated sedimentary units are part of turbidite sequences. The regular laminated unit is composed of very regular alternating sandstone, siltstone, and grey-black argillite. The lateral extension of this unit is limited. The irregular laminated unit is thicker than the regular bed and is mainly composed of an argillite unit (more than 65% of the whole rock). This irregular laminated unit is also made of an alternating sequence of sandstone, siltstone, and poorly sorted argillite.

Gold occurs as free particles within the veins and is also intergrown with arsenopyrite +/- tourmaline on vein margins or in the host rocks. Disseminated arsenopyrite in the host rock rapidly decreases away from the veins and is strongly associated with the gold mineralization. The same relationship is seen away from lithological contacts, which generally show higher densities of bedding parallel veining. Oriented diamond core drilling shows that significant concentrations of gold with arsenopyrite can be found in the arenite-argillite lithological contacts in association with quartz veining or in veinlets of massive arsenopyrite. Deeper below the main arenite unit, significant concentrations of gold are found in association with coarse arsenopyrite in the argillitic unit. The gold particles occur without sulphides in the weathered saprolite. The gold is free milling in all associations.

A cross section through the EMZ deposit model is shown in Figure 7-7. The model is based on the latest mine geology mapping and interpretation from extensive oriented diamond core drilling. It has been confirmed that the EMZ deposit is an anticlinal fold with flexural slips between layers and is westward thrusting along weakness planes parallel to bedding, with minor displacement.

The quartz veins fill brittle extension and shear deformation structures caused by the folding with at least three distinct sets of veins (Figure 7-8) and two phases of quartz veining and gold mineralization.

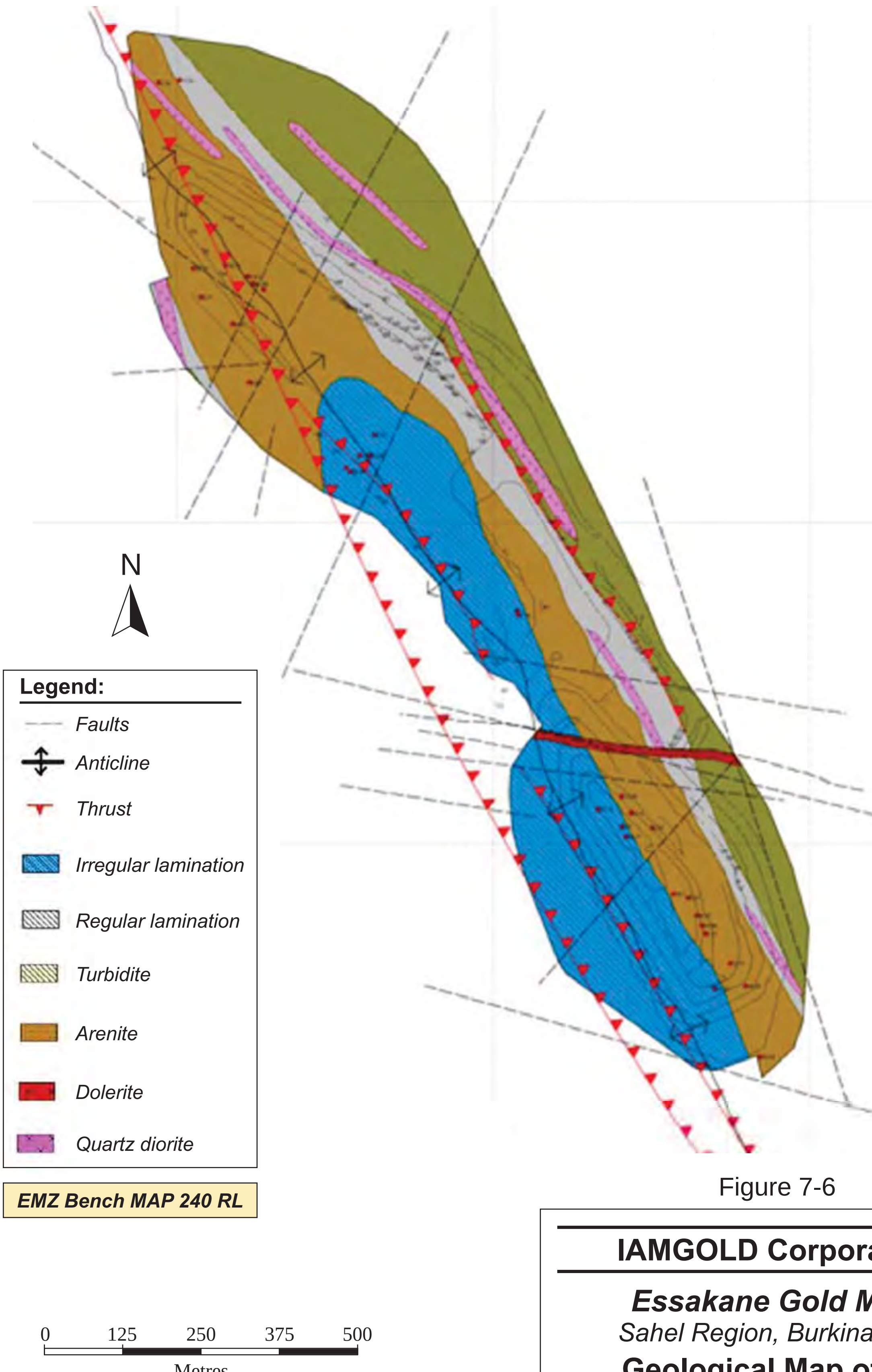


Figure 7-6

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Essakane Gold Mine

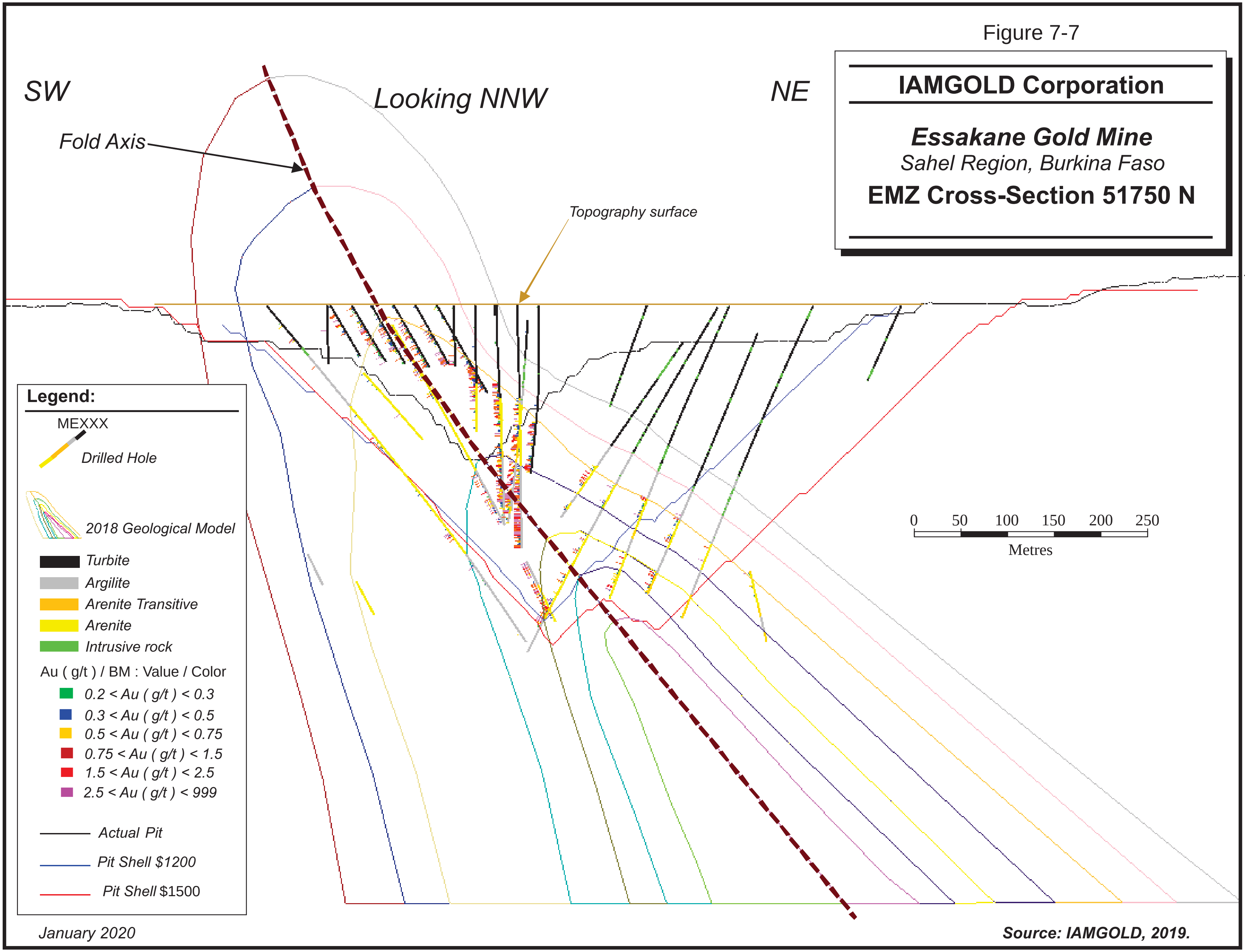
Sahel Region, Burkina Faso

Geological Map of the EMZ Deposit Level 240

January 2020

Source: IAMGOLD, 2019.

Figure 7-7



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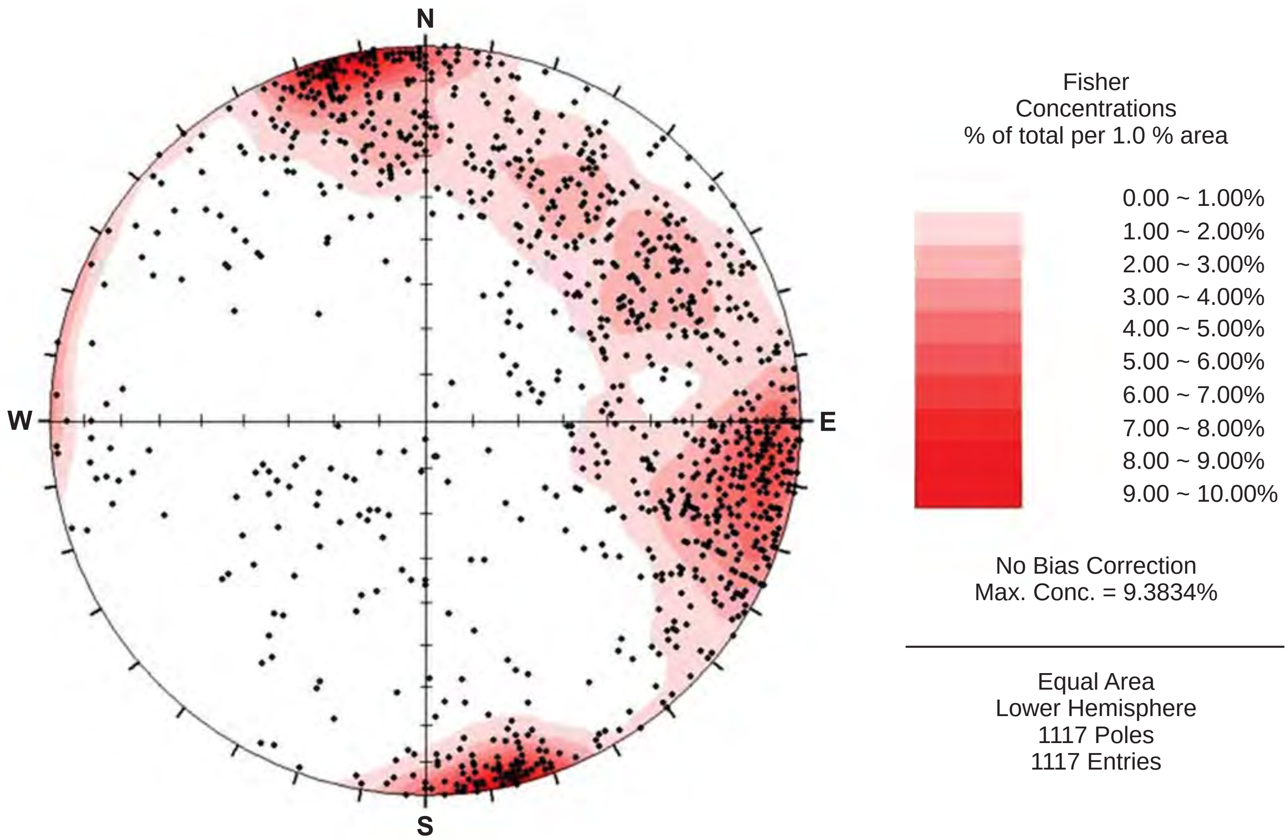


Figure 7-8

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Quartz Vein Orientations (from Pit Mapping)

January 2020

Source: IAMGOLD, 2015.

The vein arrays in the EMZ deposit are complex and consist of the following:

- Early bedding parallel laminated quartz veins caused by flexural slip and showing ptygmatic folding.
- Late, steep extensional quartz veins as vein filling in extension and shear joints formed by the folding.
- Axial-planar pressure solution cleavage (with pressure solution seams normal and parallel to bedding). All veins may be displaced by two sets of late opposing thrusts as shown in Figure 7-9.

The vein arrays occur in the east limb, fold hinge (or fold axis), and west limb lithostructural domains. The geology and economic potential of the EMZ deposit is dominated by the persistent east limb main arenite. The top contact of the east limb domain is a sharp, sheared contact with no significant gold mineralization above it. The shearing appears to be parallel to the bedding, however, some loss of vertical succession has occurred. The main arenite below this contact is the lower coarse grained part of a Bouma cycle. The locus of bedding-parallel deformation and alteration is within the east limb of the main arenite. Graphitic argillite occurs immediately above the contact. The deformation shifts into the hanging wall argillite unit to the north of the EMZ deposit.

Mineralization has been confirmed to over 550 m vertically below surface, however, the full depth extent in the fold hinge and east limb is still unknown. The geometry of the fold hinge zone is an anticlinal flexure that is easily recognized in the pit and oriented drill cores. The fold closure is sharp and sometimes truncated by thrusts and the transition from east limb to west limb takes place over a few metres. The position of the fold axis is often marked by a breccia in the arenite unit. The fold hinge zone in the argillite unit is marked by tight kink structures and sheath folds with rapid transitions from east dipping footwall rocks to near vertical west limb beds below the fold axial plane.

Hydrothermal alteration and meteoric weathering are pervasive through the east limb main arenite. It is generally associated with quartz veining and gold mineralization in the deformed main arenite. The alteration assemblage is sericite > carbonate > silica ± albite ± arsenopyrite ± pyrite. Disseminated tourmaline and rutile are found in accessory amounts. The main alteration minerals tend to occur in clearly defined veins and stringers.

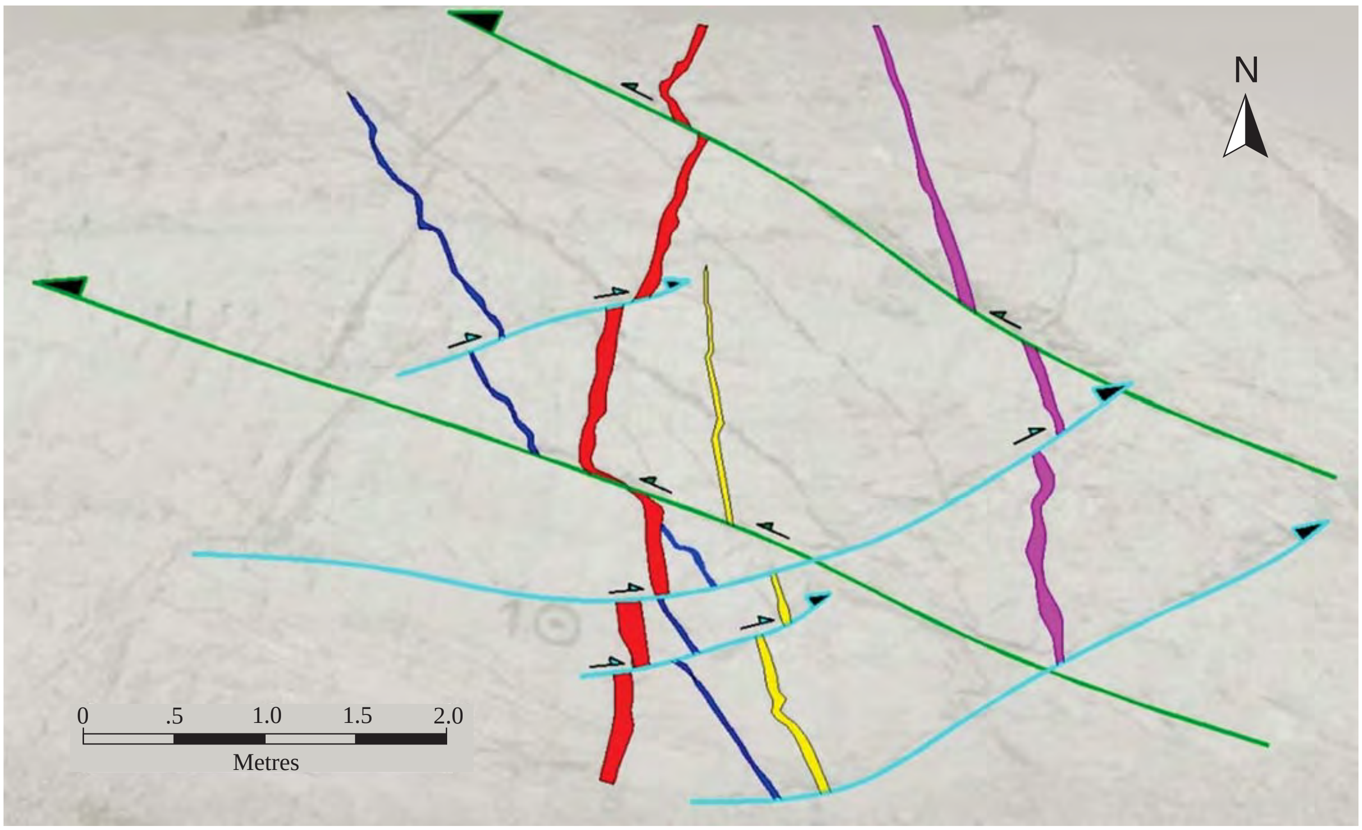


Figure 7-9

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Vein Displacements Along Minor Thrusts (West Wall EMZ Deposit)

January 2020

Source: IAMGOLD, 2015.

Arsenopyrite and pyrite occur within and adjacent to quartz veins as well as disseminated throughout areas of wall rock alteration. Traces of chalcopyrite, pyrrhotite, galena, and hematite occur with arsenopyrite. Minor amounts of tourmaline with rutile are found in the main arenite and in interbedded arenite stringers in the footwall argillite. Remobilized graphite can be found associated with tourmaline.

The fine-grained argillites can be strongly enriched in tourmaline and have also been subjected to quartz-carbonate, sericite, and quartz alteration. Fine needles of rutile are generally associated with the tourmaline. Sulphide mineralization preferentially occurs in the coarser arenaceous layers.

The EMZ deposit is characterized by multiple quartz and quartz-carbonate vein sets and stringers. Arsenopyrite and pyrite tend to be late and concentrated near the margins of the veins or in late cross-cutting stringers. The paragenetic sequence of veining is thought to be as follows:

- Early quartz-carbonate-albite-(sericite) veins
- Quartz veins with tourmaline and pyrite containing gold
- Diffuse quartz-albite-carbonate veins with arsenopyrite
- Later tourmaline-rutile-arsenopyrite stringers with gold
- Late skeletal pyrite and carbonate-quartz-pyrite stringers

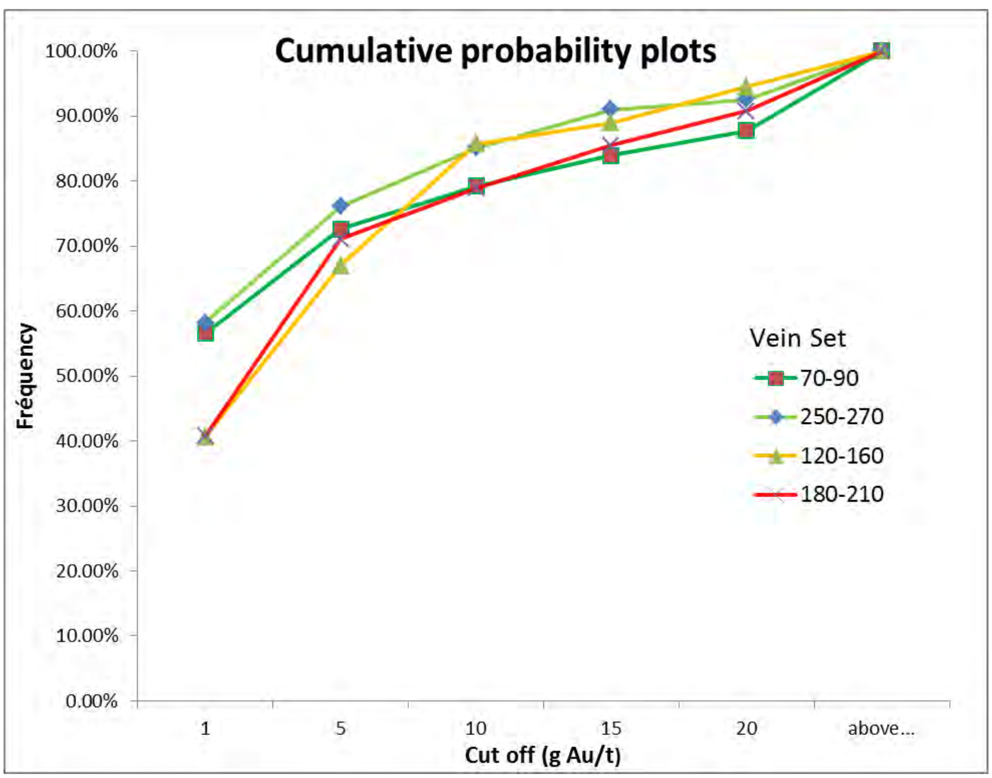
Except for vein sets located in the turbidite hosted Birimian sills, all recorded vein sets are mineralized. The east-west and north-south vein sets both appear to exhibit higher average gold grades than the other vein sets (Table 7-1), however, they are also more variable, with a higher proportion of the lower values also appearing in the same vein sets. Tests conducted in December 2010, over three areas inside the pit with oriented grade control drilling, have demonstrated that the grade difference between holes oriented 242° (old pattern) and 120° (new pattern that should intersect more of both sets) can be as high as 9%.

TABLE 7-1 GOLD GRADE DISTRIBUTION ACCORDING TO VEIN SETS

Vein Set	No. of Veins	% of Veins	No. of samples	% of Samples	Average Grade (g/t Au)	≤ 1g/t Au	1 g/t Au<x ≤ 5 g/t Au	≤ 5 g/t Au	5 g/t Au<x ≤10 g/t Au	> 10 g/t Au	>20 g/t Au
70°-90°	225	25%	106	25%	6.43	30%	17%	26%	17%	29%	35%
250°-270°	126	14%	67	16%	5.75	20%	12%	17%	14%	13%	14%
120°-160°	209	24%	91	22%	5.34	19%	24%	21%	40%	17%	14%
180°-210°	326	37%	152	37%	7.23	31%	46%	36%	29%	42%	38%
Total	**886**	**100%**	**416**	**100%**	**6.38**	**198**	**99**	**297**	**42**	**77**	**37**

Figure 7-10 shows the proportion of samples that have returned values below a series of cut-offs for each family of veins, as cumulative distribution functions (CDF), demonstrating how the vein sets compare with each other. The four sets can be split in two groups based on the CDF in the lower grade cut-offs: a higher proportion of low grade and lower variability in the 70° to 90° sets and the 250° to 270° sets as opposed to the other two sets. The higher variability and proportion of high grade lies within the 120° to 160° and 180° to 210° sets as demonstrated by the steeper CDF slopes overall, particularly above the 10 g/t Au cut-off. Updated compilations of vein sets in 2013 have confirmed the conclusions of the 2010 study.

FIGURE 7-10 CUMULATIVE DISTRIBUTION FUNCTIONS OF VEIN SETS AU GRADE



7.4.2 FALAGOUNTOU WEST DEPOSIT MINERALIZATION

Due to the intense artisanal mining ("orpaillage or orpailleur") activity, no detailed geological mapping has been carried out over the Falagountou West deposit. Observations, from visits within orpailleur workings, indicate that gold is located in smoky quartz veins injected in a sequence of fine to medium detrital sediments, similar to those found at the EMZ deposit that have been intruded and metamorphosed by shallow dioritic dykes.

Drill cutting and core observations have confirmed that the gold mineralization is structurally controlled, hosted in sheared and brecciated zones in the hanging wall contacts between sedimentary and intrusive rocks along a north-northwest to north trend. Gold is associated with quartz veins and is found disseminated in the wall rock, as well. There is a strong spatial relationship between the gold mineralization structures and the swarm of intrusive dykes that intrude the sedimentary sequence, suggesting that part of the fluid responsible for the gold deposition may have been exsolved from the dioritic magma during its emplacement. The alteration assemblage encountered is silica-calcite-chlorite. Pyrite and arsenopyrite are the main sulphide minerals observed to date, both in sedimentary rocks and in the dioritic dykes.

Most of the artisanal mining activity is located at the contacts between sedimentary and intrusive rocks. Airborne magnetic surveys suggest that other intrusive rocks are located to the southwest of the small scale artisanal miner pits and recent drilling results indicate that the western edge hosts gold mineralization (red dashed line shown in Figure 7-11).

7.4.3 WAFAKA DEPOSIT MINERALIZATION

At the Wafaka deposit, gold mineralization appears to be controlled by a series of shear zones and occurs in a network of parallel fracture systems associated with calcite and quartz within strongly deformed and hydrothermally altered turbidite rocks. The contact between the sedimentary sequences and the dioritic intrusion (dykes and sills) sometimes contains gold. Figure 7-11 shows the Wafaka deposit east of the Falagountou West deposit.

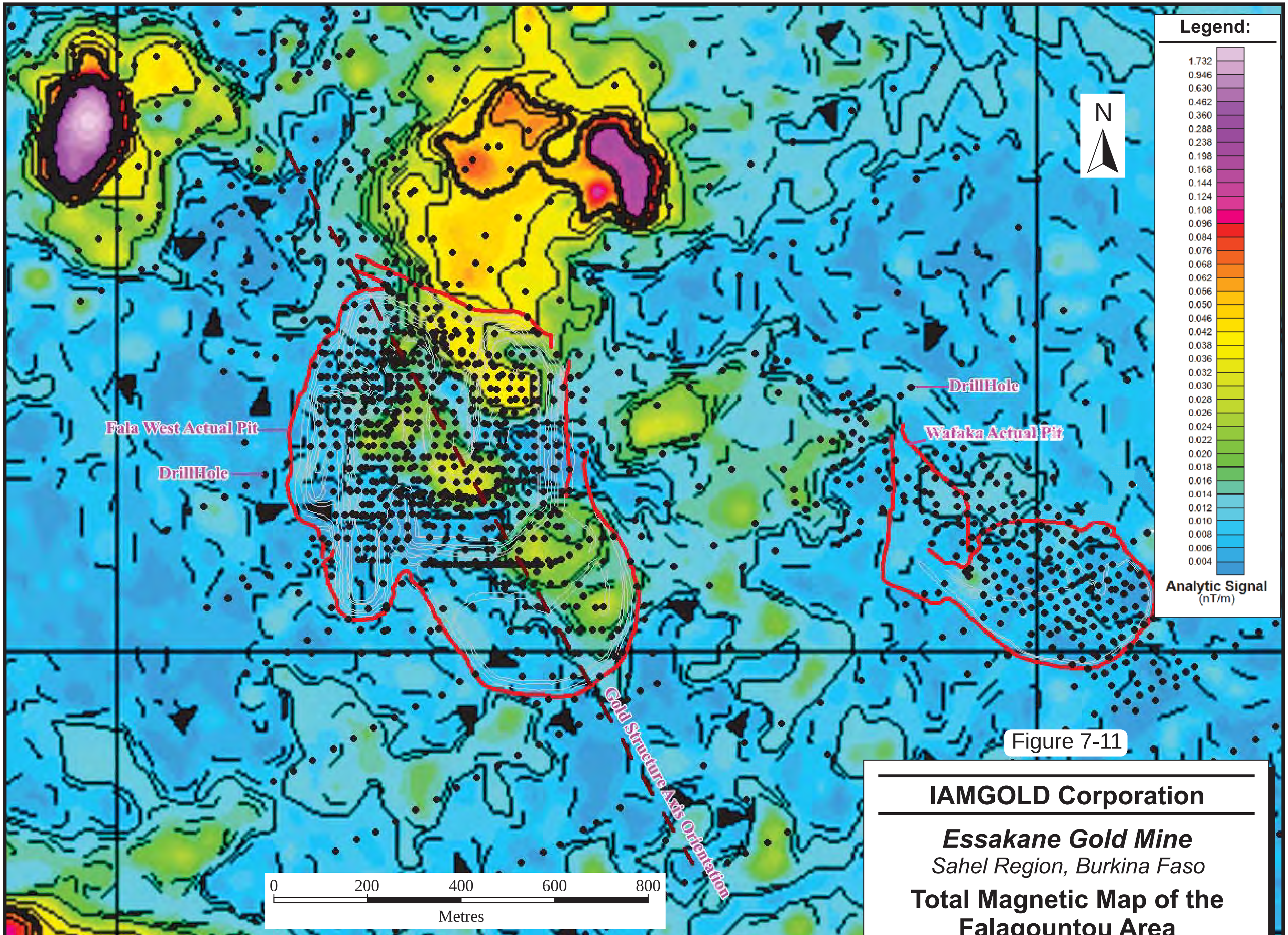


Figure 7-11

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Total Magnetic Map of the Falagountou Area

January 2020

Source: IAMGOLD, 2019.

7.4.4 GOSSEY DEPOSIT MINERALIZATION

Between 2010 and 2012, several regional geochemistry campaigns using vertical Air Core (AC) drilling were carried out on the Korizéna permit. The shape of the gold geochemical anomaly circumscribes perfectly the shear zone and seems to follow the flank of the anticline further north of the prospect at Tin Taradat (Figure 7-12). Following this geochemistry, several RC and DD campaigns have better defined this mineralization on the Gossey deposit and the Korizéna prospect (Allou et al., 2013) (Figure 7-13).

From composites made with a cut-off grade of 0.4 g/t Au over a minimum of 3 m and a maximum dilution thickness of 3 m, areas of great interest from the mineralization point of view could be delimited (Figure 7-14). These mineralized bodies are organized as lenses of quartz vein stockworks (millimetric to centimetric), and quartz-carbonates associated with pyrite, arsenopyrite, and more rarely, pyrrhotite. These lenses are subvertical or slightly inclined to the east, with hectometre lengths and of metric to decametric thicknesses. These mineralized structures follow a main direction of N10° and a secondary direction of N35°.

The mineralization is mainly hosted in sandstone to conglomeratic sedimentary formations along contacts with basic to intermediate intrusive dykes, and rarely within these intrusive units. Gold mineralization is also associated with quartz-vein (brecciated, banded, sheared, and as boudins) systems present in highly silicified zones and accompanied by sulphides.

The alteration associated with gold mineralization is underlined by sulphides (pyrite ± arsenopyrite), tourmaline (locally), and silicification of varying intensity. The gold mineralization of the Gossey deposit can be associated with the second phase of deformation (D2) during which there was reactivation of the pre-existing structures, generation of new structures, and then fluid injections in the different zones of weaknesses. The quartz veins and the alteration observed are the result of this hydrothermal circulation.

Following mineralogical analyzes by optical microscopy, electron microscopy, and Mineral Liberation Analyzer (MLA) carried out on core samples from the Gossey-Korizéna project by COREM, most of the interstitial gold (with traces of silver) was observed in the quartz grain edges and sulphides such as pyrite, arsenopyrite, and pyrrhotite. Many grains of gold occluded in these minerals are also observed. The size of the gold grains observed is variable (< 5 μm and from 10 to 30 μm).

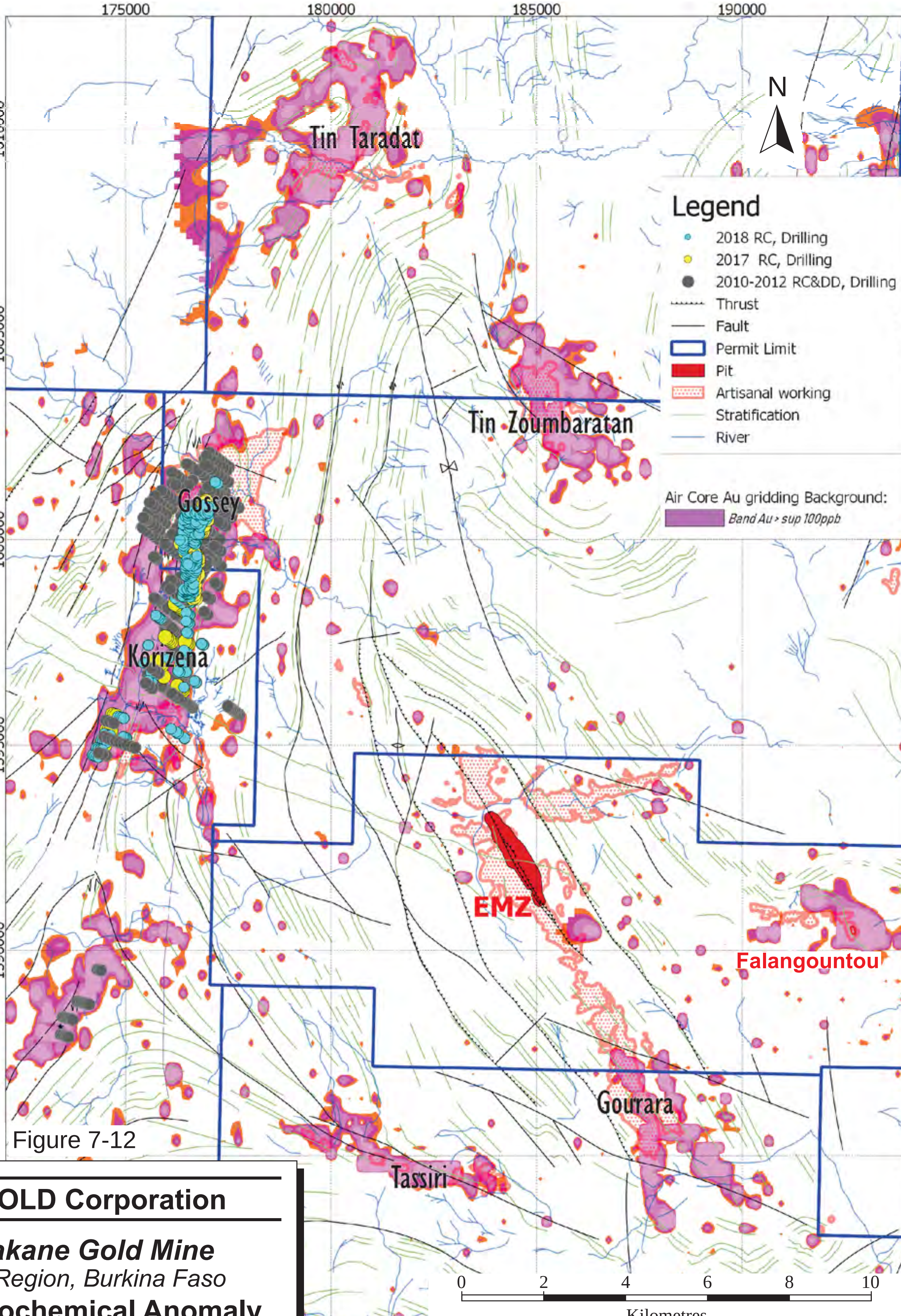


Figure 7-12

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Gold Geochemical Anomaly Map of the Gossey Region

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation

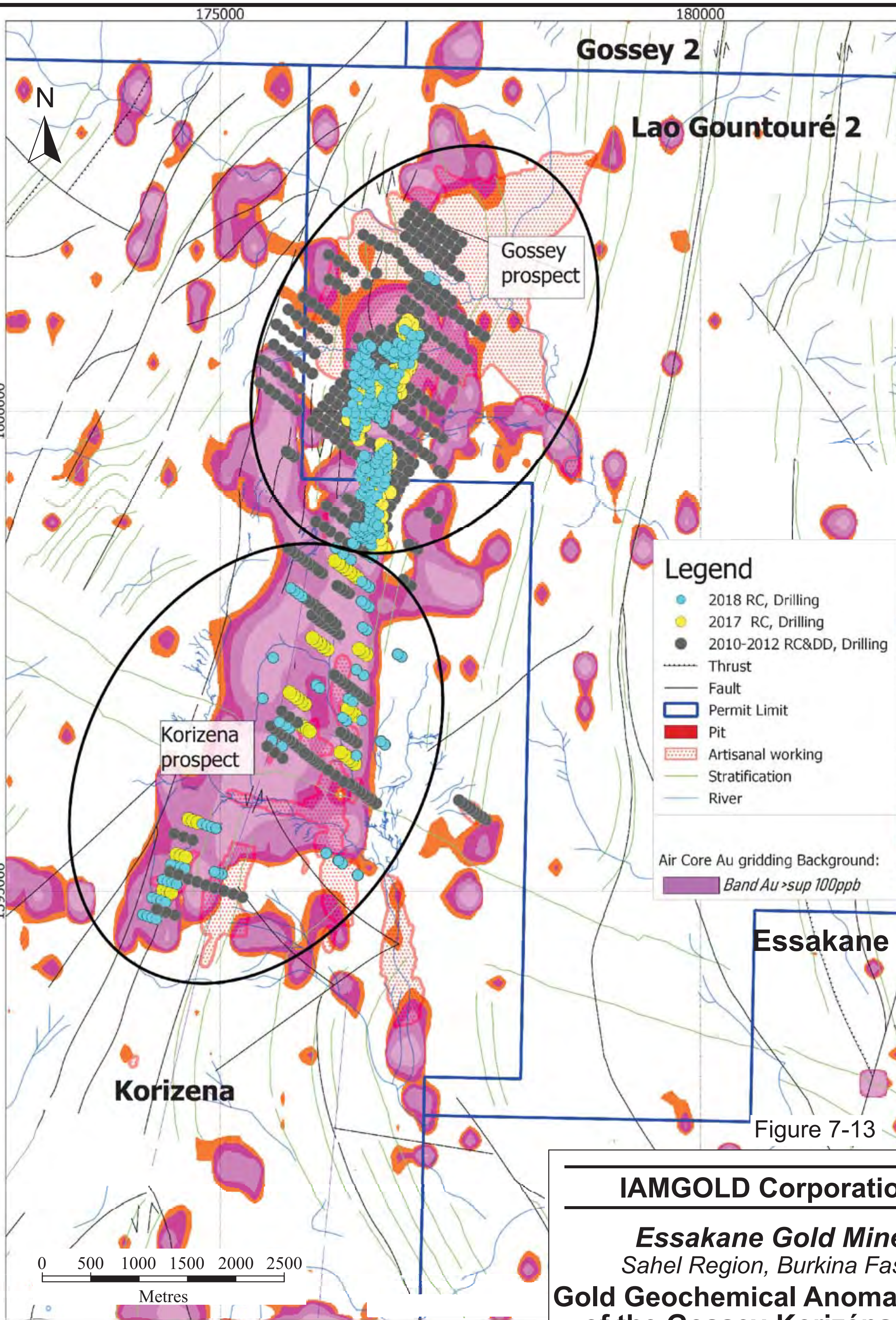


Figure 7-13

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Gold Geochemical Anomaly Map of the Gossey-Korizéna Area

January 2020

Source: IAMGOLD, 2019.

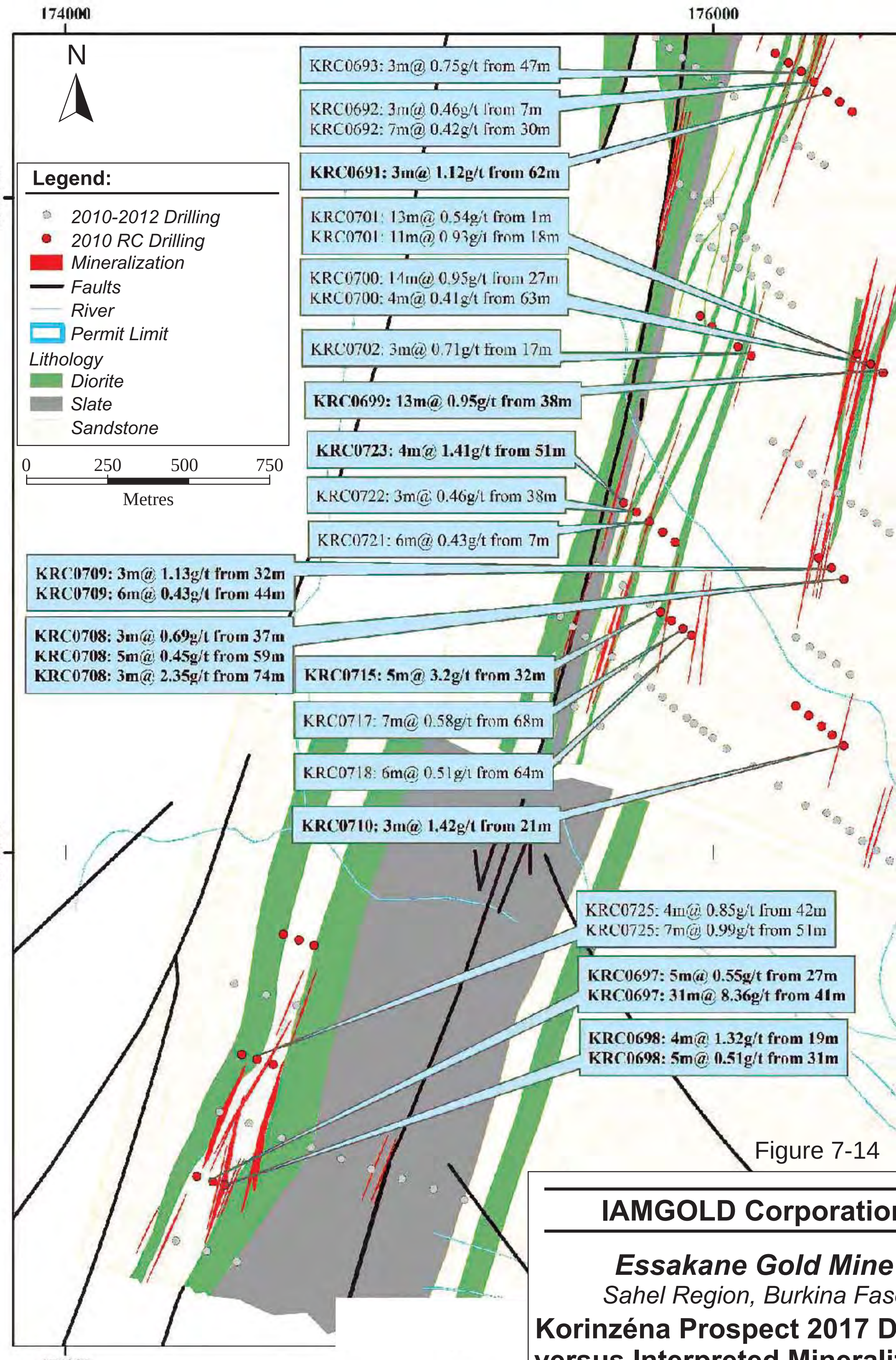


Figure 7-14

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Korinzéna Prospect 2017 Drilling versus Interpreted Mineralization

January 2020 ***Source: Allou, B. & Pratas, M., 2017.***

7.5 WEATHERING

Weathering of arenite and argillite by meteoric processes has produced a consistent, although very uneven weathering profile. The ability of drill core to absorb water and the rate of absorption was used from January 2006 to define the base of upper and lower saprolite (transition zone). This method was replaced by the use of Brown's hardness scale in early 2010 to better define the three main weathering profiles (saprolite, transition, and fresh rock) at Essakane.

Very little of the primary lithology can be recognized in the clay-rich saprolite near surface. The base of upper saprolite is easily recognized in drill core, particularly after the core is allowed to dry in the sun and the clay fraction disaggregates. In general, this is a fairly sharp contact and mining equipment is able to excavate this material without difficulty. Compared to the EMZ deposit, the saprolitic layer at the Falagountou West deposit is much thinner, sometimes less than a few metres. The base of transition (or top of fresh rock) is gradational and the contact is placed at the Brown's value of R3 (that is, the rock can be peeled by a pocketknife with difficulty; shallow indentations made by firm blow with point of geological hammer). Oxidation of sulphides on vein margins and joints can extend into fresh rock for some distance below this position.

7.6 GOLD MINERALOGY

The EMZ deposit is a coarse gold deposit. The rule of thumb definition for coarse gold is when particles are larger than 100 µm in diameter. Significant amounts of gold report to the +106 µm oversize despite the fine grind. Fifty per cent of the gold fraction is coarser than 106 µm in samples assaying greater than 5 g/t Au with a strong maximum between 60% and 80% in high grade samples. In lower grade samples, the proportion of gold coarser than 100 µm can vary from 5% to 80%. Strong heterogeneity would account for the sampling problems and imprecision in assaying the EMZ deposit samples. These observations have been mitigated by using a large sample (seven kilograms) for preparation and the use of the cyanide bottle roll leach (LeachWell) analysis method.

Visible gold particles have been recorded during core logging within and on the margins of quartz veins, intergrown with coarse arsenopyrite, and as isolated grains in the host rock. The usual associations are:

- gold particles in white, extensional, quartz-carbonate veins.
- on fractures or peripheral to late carbonate which has developed along quartz grain boundaries.
- associated with clusters of arsenopyrite grains. Mineralogical test work shows that the gold occurs:
  - on sulphide grain boundaries.
  - as small filamental grains concentrated along fractures within the sulphide, or as coarse flakes >100 µm in size and wholly occluded by the sulphide, and
  - interstitial to concentrations of tourmaline and arsenopyrite in the host rocks.

7.7 STRUCTURAL CONTROLS ON MINERALIZATION

The main structural features of the EMZ deposit are:

- The lithologies are folded into a west-verging anticline.
- There are competency contrasts between arenite and argillite, and flexural slip along bedding planes in a pervasive deformation style throughout the deposit.
- Early bedding-parallel, grey laminated quartz veins are related to flexural slip.
- Syn-deformational, steep extensional quartz veins with visible gold occur in the fold hinge and east limb domains.
- Axial-planar pressure solution seams are developed in the fold hinge.

Mine mapping and oriented core drilling have demonstrated that continuity of mineralization within the fold hinge domain is caused by conjugate vein sets. These vein sets have been repeatedly sealed and reactivated during a deformation history that saw a 40° clockwise rotation of the stress fields. Away from the hinge, dissemination of mineralization along flexural slips and lithological contacts are the more prevalent mechanisms of emplacement.

Pressure solution veining appears to be more common in the footwall argillite and provides grade continuity down the fold axis. The lengths of individual veins are usually short and only a few veins longer than ten metres are exposed in the pit. The vein density (number of veins in a given volume) is the most important factor to delineate favourable gold concentration. This pattern of mineralization extends into the east limb main arenite, with steep north-south veins supplemented by a lower frequency of east-west and 140° veins.

Grade continuity is best developed along the following lithological contacts:

- Upper part of the east limb main arenite
- The arenite-argillite contact at the base of the main arenite
- The gradational contacts between the footwall argillite and footwall arenite units
- The arsenopyrite-rich layers in the deep argillite

Continuity of mineralization in the steep west limb is poor. The mineralization is usually low grade due to the frequency of white, late-stage extensional quartz veins with visible gold, however, there are a few east-west extensional veins crosscutting the west limb which have been worked by the artisanal miners. Gold dissemination into the wall rock is rare, and gold is largely confined to the early stage, bedding parallel, and conjugate veins sets.

8 DEPOSIT TYPES

The EMZ deposit is a greenstone hosted orogenic gold deposit. Specifically, it is a quartz-carbonate stockwork vein deposit hosted by a folded turbidite succession of arenite and argillite. The original structural interpretation and gold settings have been confirmed by mining.

The Falagountou deposits (Falagountou West and Wafaka) are porphyry intrusive hosted, orogenic style, gold deposits. Gold is commonly located within smoky quartz veins injected along the contact of the dioritic dyke and a sequence of fine to medium grained detrital sediments. Gold is also disseminated into the rock. At both Guessakado (Falagountou's orpailleur pit area) and the Falagountou Southeast Zone, vein occurrence is prevalent at the contact of the intrusion and sedimentary rocks.

The Wafaka deposit also appears to be an orogenic gold deposit related to a series of shear zones and fractures oriented northwest to east-west. Gold is associated with northwest to east-west striking structures affecting the sedimentary sequence that is intruded locally by dioritic sills or dykes.

The Gossey deposit is also associated with intrusions and orogenic events similar to the styles observed at Falagountou. Gold mineralization is mainly concentrated in coarse sandstones, in microgranular granodiorite sills, and along the contacts between intrusions (gabbro, diorite, granodiorite) and coarse sandstones. Gold mineralization intensifies in the hinge of the anticline, where stronger gold anomalies are found. Mineralization is associated with quartz ± carbonate veins and silicification zones. The accompanying sulphides of gold are mainly pyrite and arsenopyrite disseminated in the rock or associated with quartz veins (Kinda and Sawadogo, 2016).

9 EXPLORATION

Essakane has been explored since the 1990s by geochemistry sampling, mapping, trenching, Aster/Landsat image analysis and interpretation, geophysical surveys, and drilling. Exploration prior to IAMGOLD's ownership is described in Section 6 of this Technical Report.

9.1 TRENCHING

In the early 1990s, CEMOB excavated five trenches for a total of 705 m. An additional 4,903 m of trenching was completed by BHP in 1993 to 1996.

9.2 GEOPHYSICS

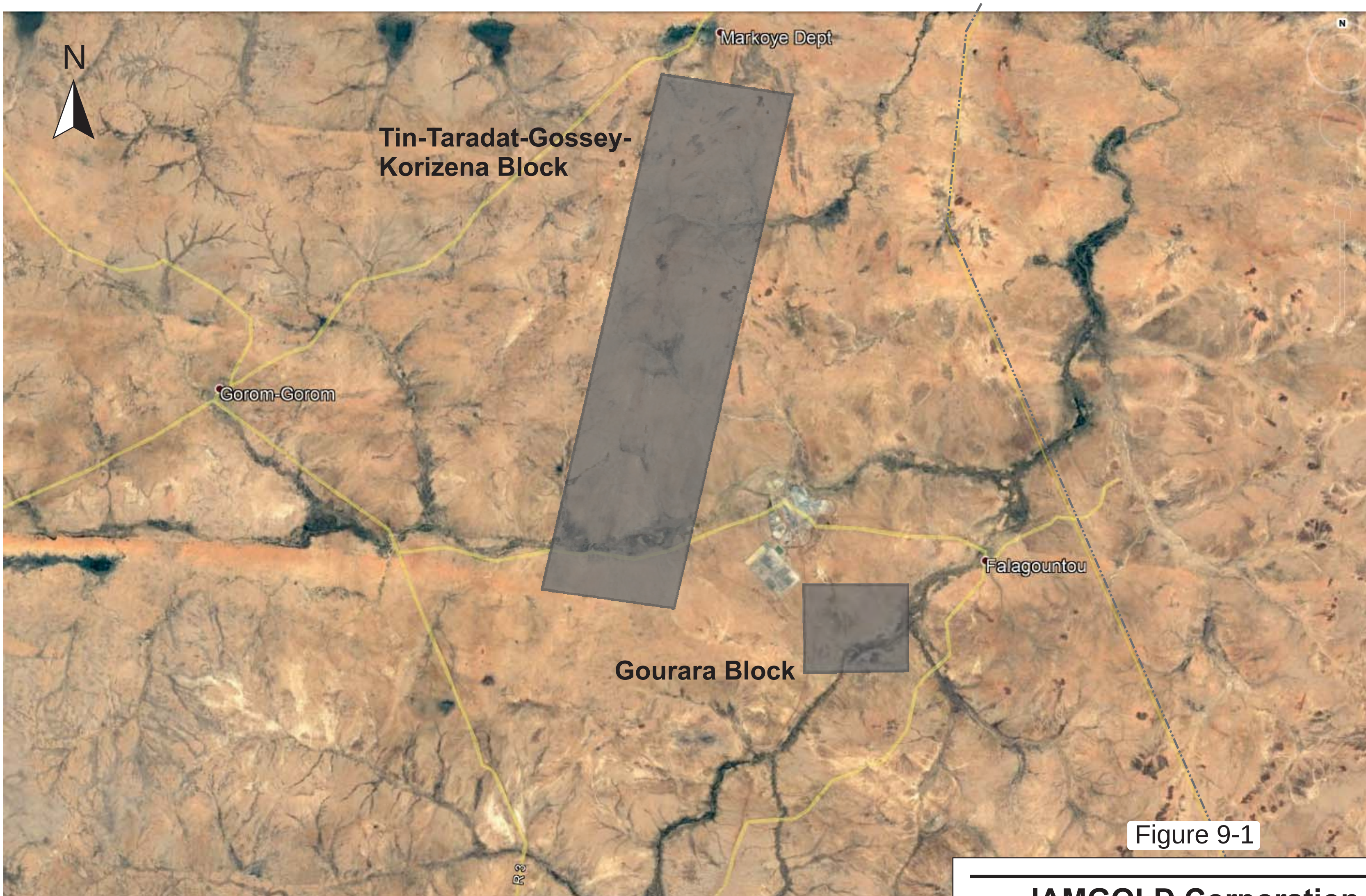
The first airborne geophysical survey reported in the area was an aeromagnetic/radiometric survey commentated by BHP over the both Exploration and Mining permit areas in 1995.

Between November 26, 2009 and February 10, 2010, a total of 30,407 line-km was flown over the Essakane Exploration Permits and the Essakane Mining Permit by South African contractor Xcalibur Airborne Geophysics for a high resolution magnetic/radiometric survey. Total and vertical gradient magnetics along with uranium/potassium/thorium (U/K/Th) radiometrics were recorded. Two induced polarization (IP) areas were surveyed by Sagax Geophysics in 2010: one immediately north of the EMZ deposit and the other immediately south.

During April 2017, two areas were covered by a helicopter borne geophysical survey of VTEM Plus (Versatile Full Waveform Time-Domain Electromagnetic) carried out by GEOTECH Airborne Geophysical surveys.

The two survey areas (Tin-Taradat-Gossey-Korizéna block and Gourara block) are located approximately four kilometres south and seven kilometres west of the Property (Figure 9-1). The survey areas were flown in an east-west (N100°E azimuth) direction for the Tin-Taradat-Gossey–Korizéna block and east-west (N90°E azimuth) direction for the Gourara block with traverse line spacing of 100 m. Tie lines were flown perpendicular to the traverse lines at a spacing of 1,000 m.

A total of 2,674 line-km covering 238 km^2 and 341 line-km covering 30 km^2 was surveyed over the Tin-Taradat-Gossey-Koriz√©na block and the Gourara block, respectively.



0 4 8 12 16
Kilometres

IAMGOLD Corporation

Corporation

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

VTEM Survey Area
Location on Google Earth

January 2020

Source: Essakane S.A., 2018.

9.3 GEOCHEMICAL SAMPLING AND REGOLITH MAPPING

Geochemical sampling, which involved assaying for gold and arsenic, conducted in the area successfully located potential targets for follow up pitting and drilling.

A regolith map was completed during the soil sampling process. Outcrop is limited and there is an extensive cover sequence of residual soils and transported material. The southern permits are characterized by a higher proportion of outcrop.

From 2001 to 2004, Orezone Resources collected pisolith samples over the major prospects of the Essakane area. A follow up of the anomalies by AC drilling was executed in 2007, after Goldfields joined Orezone Resources.

Since 2010, Essakane Exploration SARL has conducted several campaigns of regional shallow and deep follow up AC drilling over a large portion of the exploration permits with the aim of finding gold mineralization masked by transported material and were therefore not able to be located by conventional geochemical sampling.

9.4 SATELLITE IMAGERY INTERPRETATION

An interpretation of structural geology derived from Aster image and aeromagnetic data was carried out by the Orezone Resources exploration team. A number of fold axial traces observed have a spatial relationship with the main gold mineralization. These observations suggest that a significant proportion of the gold occurrences on the permits are associated with this folding event (Figure 9-2).

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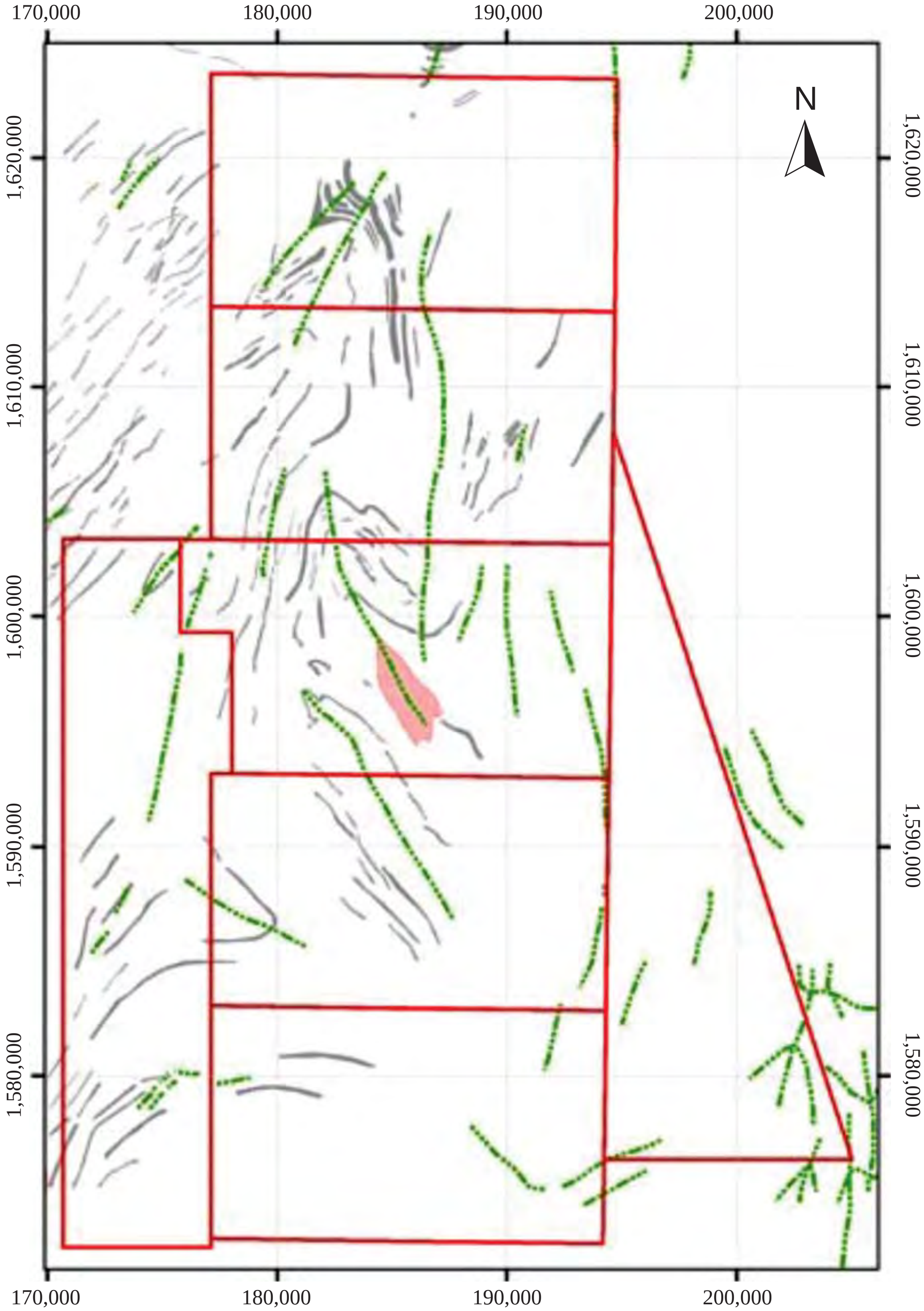


Figure 9-2

Note: Permit Boundaries Shown Current as of 2015

0 2 4 6 8 10
Kilometres

UTM Zone 31, WGS 84

Legend:

Permits	*Intrusion*
Inferred Fold Axis	*Stratigraphic Units*

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Essakane Structural Interpretation Map

January 2020

Source: Essakane S.A., 2015.

10 DRILLING

10.1 EMZ, FALAGOUNTOU WEST, AND WAFAKA DEPOSITS

Exploration efforts at Essakane were initially focussed on identifying the potential of the entire project area. In the mid 1990s, BHP undertook a widely spaced drilling program on the EMZ deposit that has been narrowed subsequently by Ranger.

Orezone Resources begun resource definition drilling at the EMZ deposit in February 2003. At the end of 2004, vertical RC drilling was performed on a nominal grid of 50 m x 25 m. The RC holes were drilled to the water table and sampled at one metre intervals. RC drilling was preferred over DD as it allowed an increase in the sample size, thus offset the coarse gold sampling issue.

In its early programs, Orezone Resources drilled a few HQ (63.5 mm) diameter DD tails of RC holes which had been stopped in the main arenite or in gold mineralization, in order to test for grade continuity at depth. Some of these tails returned significant gold assays in the footwall argillite. Systematic drilling of DD tails was started in May 2005 to evaluate the footwall units. IAMGOLD has continued to use this drilling method over most of the EMZ deposit area to a vertical depth of 400 m.

Orezone Resources and GF BVI drilled 20,364 m of oriented HQ diameter core between September 2005 and June 2006 for the project development and FS program.

RC, DD, and holes pre-collared with RC then completed by DD (RCD) drilling has been conducted by Essakane S.A.'s Resource Development Group since January 2010. As of December 31, 2018, a total of 1,773 RC holes (218,513 m), 44 RCD holes (12,507 m) and 969 DD holes (268,834 m) had been drilled at the EMZ, Falagountou West, and Wafaka deposit areas.

Essakane S.A.'s drilling objectives include infill drilling to upgrade Inferred Mineral Resources, expand the resource inventory, gain a better understanding of the geology and controls of mineralization to advance geological modelling, and improve the quality of assay samples.

At the EMZ deposit, most DD holes targeted Inferred Mineral Resources below the EMZ pit and along the deposit's northern, southern, and down-dip extensions.

DD results were positive on the EMZ deposit with continuity of mineralization demonstrated at depth along the east limb of the deposit in the northern sector of the pit, as well as in the southeast end of the pit. EMZ deposit mineralization is oriented north-northwest. The DD results were incorporated into the updated resource model as reported at August 31, 2019.

An infill RC and DD program conducted at the Falagountou West deposit confirmed lateral continuity of mineralization oriented mostly north-south as well as an extension down-dip, which remained open.

The drill programs are based on the targets and metreage proposed by the geology department during budget preparation. The drill programs are generally derived from the corporate objectives set earlier in terms of resource/reserve renewal and types of ore feed to mill. These translate into yearly drilling plans made up of individual hole information that are created and saved in GEOVIA GEMS mine modelling software (temporary hole-id, collar location, length, azimuth, and plunge).

Collar locations are then checked in the field by the senior technician to ensure that there is sufficient space for the drilling pad and a nearby water decant basin for DD holes that will collect the run-off water and drill cuttings.

The Essakane DD and RC drill holes as of December 31, 2018 are summarized in Table 10-1.

TABLE 10-1 ESSAKANE DRILLING PROGRAMS 1995 TO 2018

Year	Company	DD Metres (m)	DD No. of Holes	RC Metres (m)	RC No. of Holes	RCD Metres (m)	RCD No. of Holes	Total Metres (m)	Total No. of Holes
1995	BHP	1,511	9	7,404	120	-	-	8,914	129
2000	Ranger	69	1	3,952	52	222	2	4,242	55
2001	Ranger	113	1	17,380	179	1,728	11	19,221	191
2002	Orezone	-	-	-	-	-	-	-	-
2003	Orezone	288	2	12,126	176	724	6	13,138	184
2004	Orezone	819	4	20,310	227	8,818	48	29,947	279
2005	Orezone	13,200	84	46,030	459	29,980	184	89,210	727
2006	GF/Orezone	13,105	75	14,411	176	16,675	73	44,191	324
2007	Orezone	3,264	30	1,043	17	-	-	4,307	47
2008	IAMGOLD	10,992	70	2,822	38	-	-	13,814	108
2009	IAMGOLD	2,209	10	4,481	39	-	-	6,690	49
2010	IAMGOLD	38,849	150	32,220	230	1,032	3	72,101	383
2011	IAMGOLD	63,760	188	23,053	180	-	-	86,813	368
2012	IAMGOLD	50,008	119	40,040	307	-	-	90,048	426
2013	IAMGOLD	34,931	139	37,828	275	-	-	72,759	414
2014	IAMGOLD	33,296	153	14,280	148	-	-	47,576	301
2015	IAMGOLD	7,236	29	22,476	191	5,843	25	35,555	245
2016	IAMGOLD	861	6	30,286	274	-	-	31,147	280
2017	IAMGOLD	27,693	133	4,910	68	128	1	18,185	130
2018	IAMGOLD	12,200	52	13,420	100	5,504	15	31,124	167
	Total	**314,403**	**1,255**	**348,471**	**3,256**	**70,654**	**368**	**733,528**	**4,879**

Figure 10-1 shows the drill hole plan as of December 31, 2018 for the EMZ deposit. Figure 10-2 shows a typical cross section of the drilling on the EMZ deposit. Figure 10-3 shows the drill hole plan as of December 31, 2018 for the Falagountou West and Wafaka deposits. Figure 10-4 shows a typical cross section of the drilling on the Falagountou West deposit.

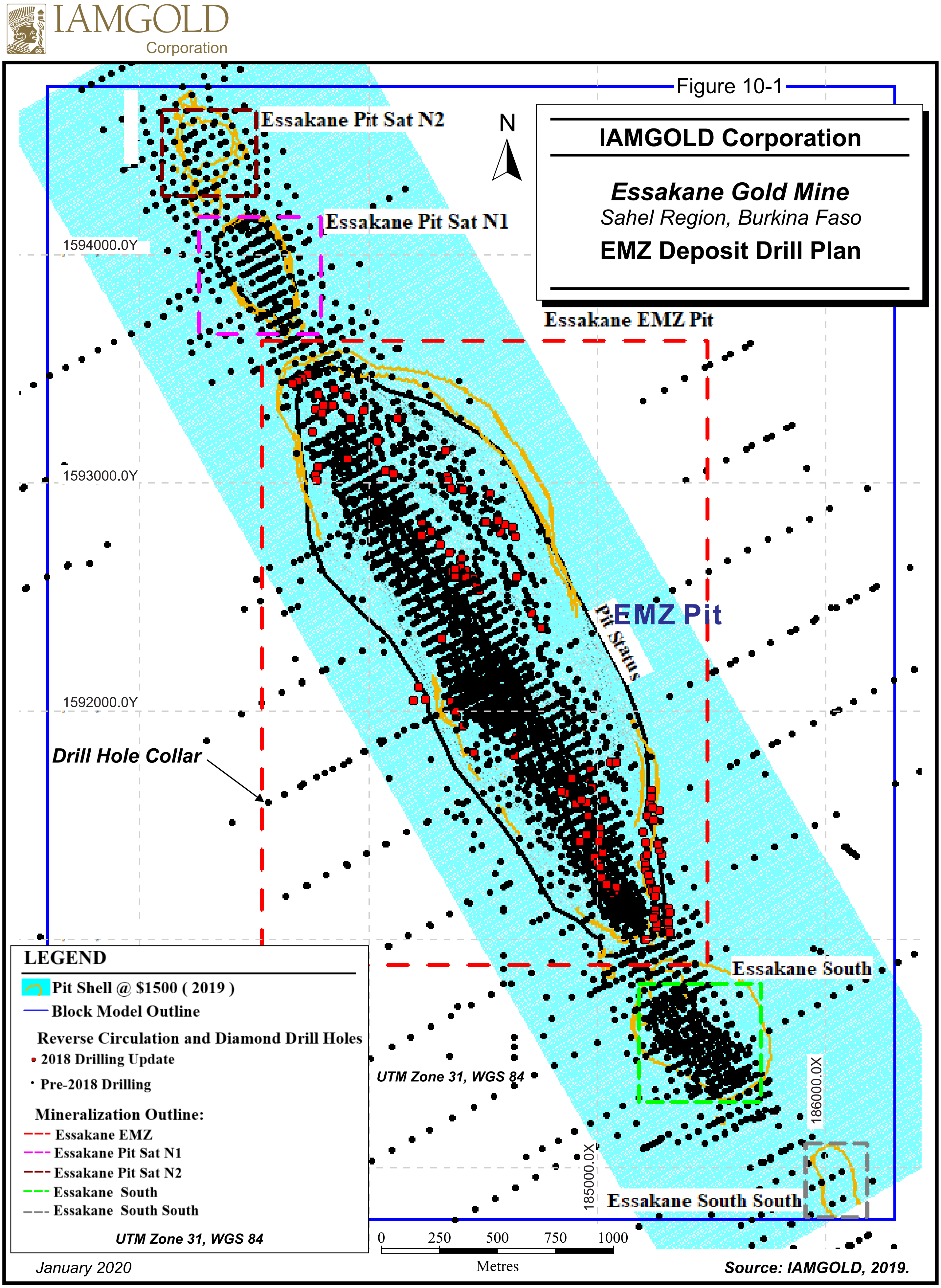


Figure 10-1
IAMGOLD Corporation
Essakane Gold Mine
Sahel Region, Burkina Faso
EMZ Deposit Drill Plan
Essakane Pit Sat N2
Essakane Pit Sat N1
Essakane EMZ Pit
EMZ Pit
Pit Status
Essakane South
Essakane South South
1594000.0Y
1593000.0Y
1592000.0Y
185000.0X
186000.0X
Drill Hole Collar
N
UTM Zone 31, WGS 84
LEGEND
Pit Shell @ $1500 (2019)
Block Model Outline
Reverse Circulation and Diamond Drill Holes
2018 Drilling Update
Pre-2018 Drilling
Mineralization Outline:
Essakane EMZ
Essakane Pit Sat N1
Essakane Pit Sat N2
Essakane South
Essakane South South
UTM Zone 31, WGS 84
0 250 500 750 1000
Metres
January 2020
Source: IAMGOLD, 2019.

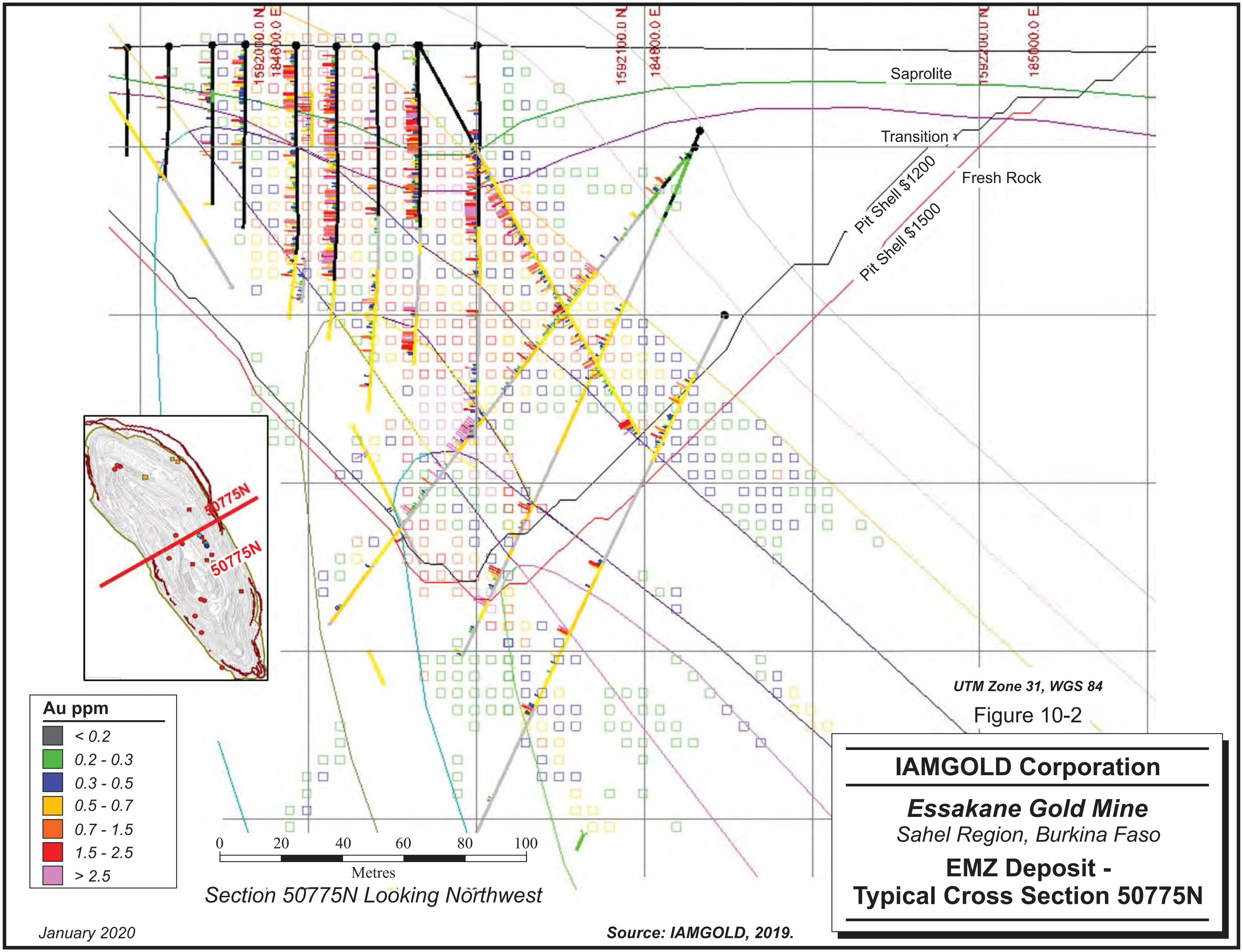


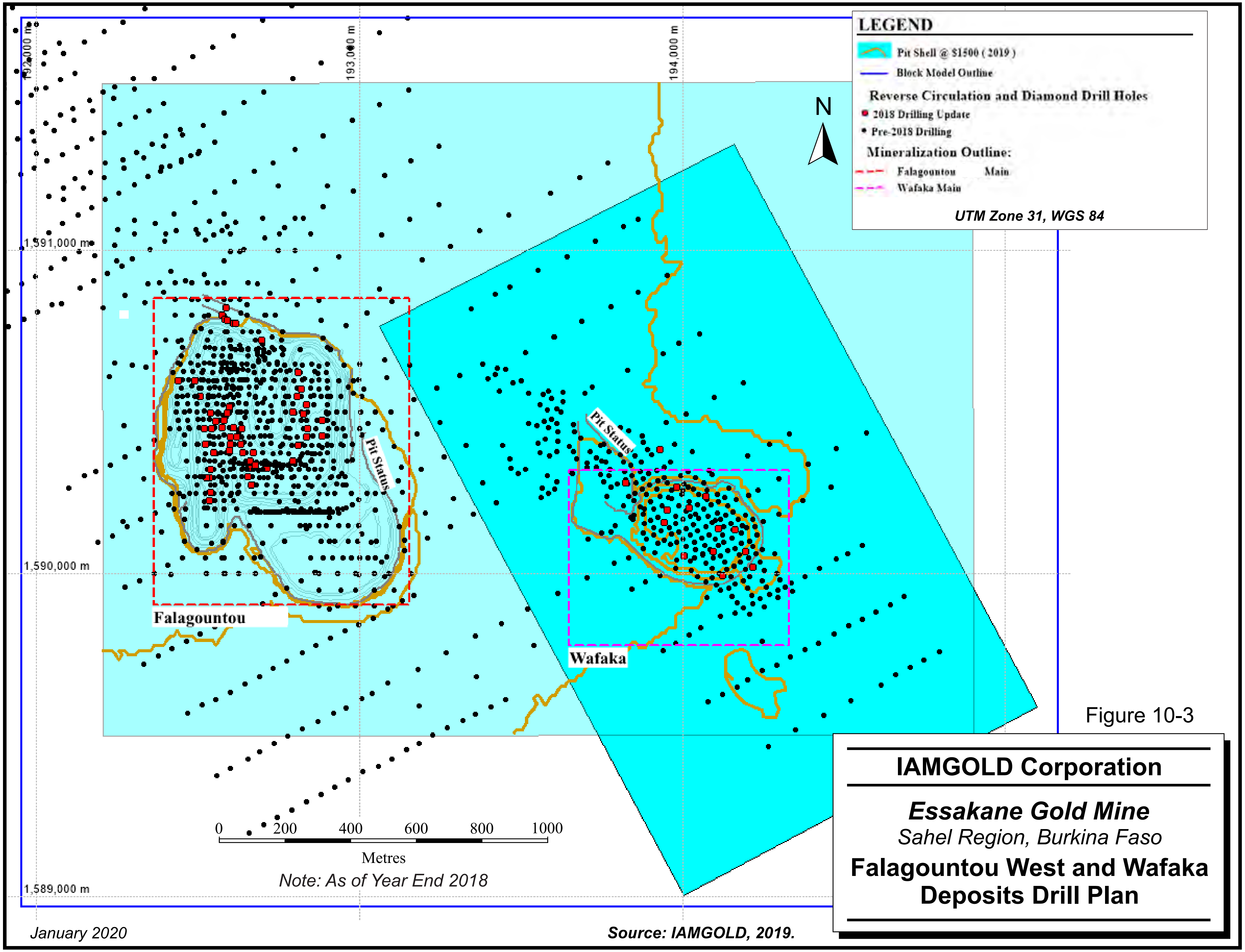
Figure 10-2

IAMGOLD Corporation

Essakane Gold Mine

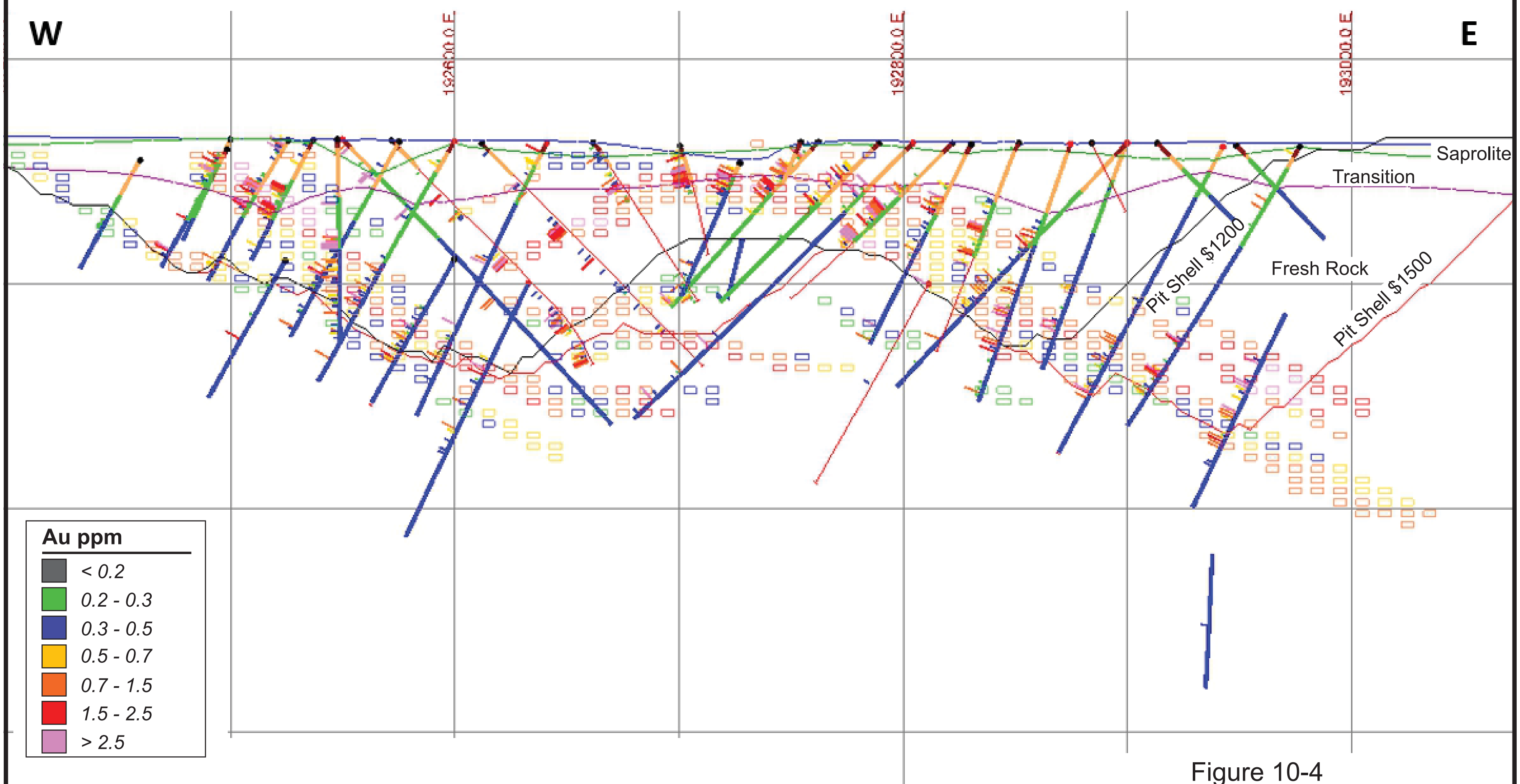
Sahel Region, Burkina Faso

EMZ Deposit -
Typical Cross Section 50775N



IAMGOLD Corporation

F0400N Section Looking North



0 40 80 120 160 200

Metres

UTM Zone 31, WGS 84

Figure 10-4

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Falagountou West Deposit - Typical Cross Section

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation

10.1.1 DIAMOND DRILLING

Since 2007, Essakane S.A. has contracted Boart Longyear Limited for all of its resource development DD. Beginning in 2014, contract drilling was carried out by Major Drilling Group International, Inc.

HQ core is drilled ten metres past the saprolite horizon and then reduced to NQ core (47.6 mm). The geologist may request that the hole be drilled HQ over a longer distance if hole deviation is an issue. Hexagonal core barrels and extended shells are often used to further reduce deviation. Core orientation is carried out using a downhole spear with wireline attachment. Drill core is placed in angle iron racks at the drill site and oriented by an Essakane S.A. technician. A continuous top node line is drawn along the length of the core in black indelible ink. The start and end depths of the drilled interval are written on the core along with the metre marks. Geotechnical information such as rock quality designation (RQD) is also recorded. The core is then packed into metal core trays at the drill site and transported to a dedicated logging facility within the secure mine perimeter. Wooden blocks are used to mark the start and end of drill runs. The borehole number, tray number, and from-to depths of the drilled interval are written on the core tray.

Efforts to properly core drill from surface through the upper saprolite often failed over the EMZ deposit due to loss of drilling fluid, caving of holes, or the washout of saprolite by entrained quartz fragments plugging the bit. All holes on the EMZ deposit are cased with either hard polyvinyl chloride (PVC) plastic or steel tubing which have to be pulled after downhole tests have been taken.

Due to the high ground and air temperature (> 35°C), the core is always dry when it is brought to the core shack. The core is logged by Essakane S.A. geologists with information recorded onto standard log sheets. After logging, each core tray is photographed on a jig that ensures the same picture quality. Previously, if the hole was located inside the Measured, Indicated, and Inferred (MII) Whittle shell, the entire core was bagged and sampled. Elsewhere, the core was cut in half by diamond saw and the one metre sample was placed in a plastic sample bag and brought to the Essakane laboratory managed by Essakane S.A. In 2013, the selection procedure was changed and one hole in five was split for archiving purposes. Exceptionally well mineralized holes were also kept. In 2014, the core was cut in half by diamond saw and a one metre sample in HQ size core and a 1.5 m sample in NQ-size core was placed in a plastic sample bag and submitted to the Essakane laboratory managed by Essakane S.A.

A shipping form listing all the samples that are ready to be analyzed is filled out and sent to the laboratory with a copy kept at the Resource Development office.

Downhole surveying is carried out by one of Essakane S.A.'s two VisionR instruments or by the drilling contractor's Reflex EZ-Shot. Survey results are checked by Essakane S.A. technicians. Survey readings are taken at downhole depths of three metres below the casing or at 12 m (whichever is the shallowest), and every 25 m thereafter. Since 2013, downhole surveys have been carried out using the drilling contractor's GYRO downhole survey tool that performs readings every five metres.

Drill hole collar positions are initially determined by a handheld global positioning system (GPS) on local grid lines by the Essakane S.A. geotechnicians. After drilling, the collar position is picked up by the surveying department using a differential global positioning system (DGPS). Away from mine workings, the collar positions are preserved by plastic pipe with written hole identifiers.

10.1.2 REVERSE CIRCULATION DRILLING

For RC drilling, a track-mounted Cat-Max rig is used with an attached cyclone unit that collects all the coarse (>50 µm) material over five metre runs starting at the collar. A seven kilogram sample split is collected at the cyclone's underflow using 50/50 single-stage riffle dividers. A 100 g sub-sample is taken from the split by the geologist for logging purposes and the rest of the sample is tagged and bagged before being sent to a secure sorting area near the core shack facility.

A shipping form listing all the samples that are ready to be analyzed is filled out and sent to the laboratory with a copy kept at the Resource Development office.

Downhole surveying is carried out in a similar manner to the DD holes, except that it uses a portable winch installed on the drill. Downhole survey readings are taken at a downhole depth of three metres below the casing or at 12 m (whichever is the shallower) and every 50 m thereafter. Collar locations are picked up by the surveying department using a DGPS.

10.1.3 LOGGING

Data capture for both DD and RC hole information has been formalized by procedures that detail the steps that must be taken to create consistent logs. The name and purpose of each description field in addition to their allowable codes and abbreviations are listed in each procedure. Data is entered directly into a laptop utilizing Maxwell GeoServices Pty Ltd.'s (Maxwell GeoServices) LogChief software and then transferred into the central database.

Data validation is completed by the geologist after the data entry stage and by the database geologist after the data has been transferred into Maxwell GeoServices' DataShed (DataShed) SQL database which constitutes Essakane S.A.'s central data repository for all grade control and resource development drilling information.

The log is transferred into the GEOVIA GEMS modelling database after it has been duly validated in DataShed and all the assays have been received and checked.

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

10.2 GOSSEY DEPOSIT

10.2.1 PRE-2017 DRILLING HISTORY

The distribution of historic drill holes at Gossey is presented in Figure 10-5, which includes BHP drilling from 1995, Orezone Resources drilling from 2003, and IAMGOLD drilling from 2011.

The first pass of conventional drilling over the Gossey deposit area was undertaken in 1995 by BHP who completed 11 shallow RC drill holes totalling 545 m with the objective to test geochemical anomalies and small-scale artisanal workings. This campaign identified several interesting gold anomalous sectors.

IAMGOLD Corporation

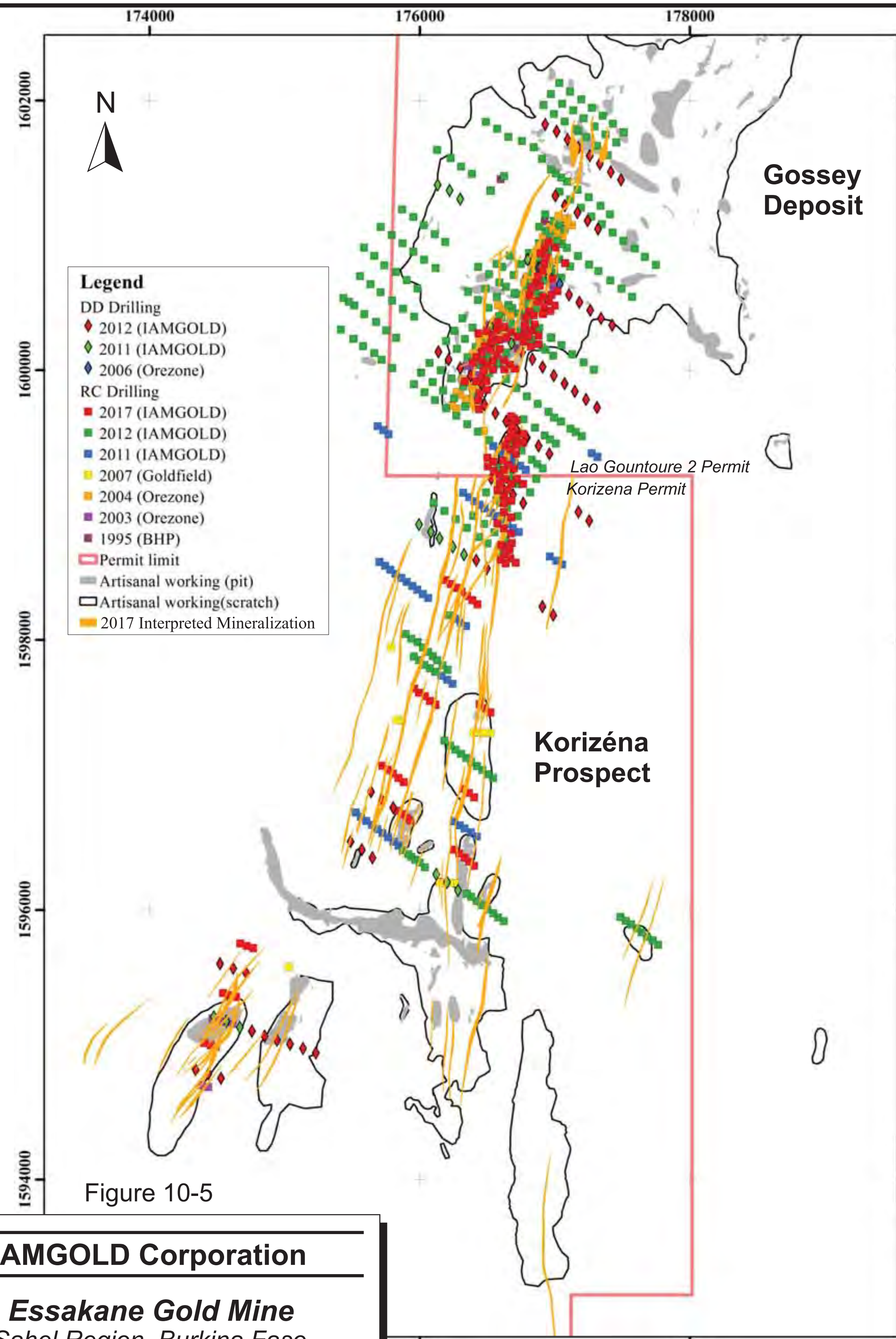


Figure 10-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Historic Drilling of Gossey Deposit and the Korizéna Prospect

January 2020

Source: Allou, 2018.

When Orezone Resources took over the Project in 2003, they continued with a second pass shallow RC drill program from November 2003 to December 2004 to test the lateral extensions of the previously identified gold mineralization. This program, which consisted of 81 RC holes totalling 5,907 m, defined two main gold mineralized zones over the Gossey prospect area. During the same period, seven additional RC shallow holes totalling 505 m were completed to the south (into the Koriténa Permit) to test the southern extensions of the Gossey mineralized system.

In 2006, the Orezone Resources / Goldfields partnership decided to evaluate the mineral potential of the exploration permits of the whole Essakane project area. This led to an extensive drill program targeting the already defined gold showings and existing artisanal workings. In the area of the Gossey prospect, the partners decided to investigate deeper with four DD holes totalling 680 m with the aim of evaluating the extensions at depth and also to increase the understanding of the structural and lithological control of gold distribution. The drill program continued in 2007 with 12 shallow RC holes totalling 909 m drilled to test a few geochemical anomalies to the south of Gossey, inside the Koriténa Permit.

In 2011, a drill campaign consisting in nine RC holes (1,072 m) and ten DD holes (2,508 m) was executed by IAMGOLD. The DD holes provided a better understanding of the structures along the mineralized corridors while the RC holes targeted a strong geochemical anomaly which was previously defined in the southeast of the Gossey main zone by IAMGOLD's exploration team. This program resulted in the discovery of the Gossey-SE mineralized zone and the extension of the Gossey main zone more than 600 m to the north. During the same year, the southern extensions of the mineralized trend and several geochemical anomalies were successfully tested in the Koriténa permit with 48 RC holes and 12 DD holes totalling 5,723 m and 2,846 m, respectively.

IAMGOLD pursued the exploration activities over the Gossey project in 2012 with an extensive drilling phase with three main objectives:

- Bring the Gossey main zone to a mineral inventory of approximately 500,000 ounces of gold.
- Add 500,000 additional ounces of gold to the known deposit by executing some infill drilling on the Gossey-SE prospect.
- Test the western contact of the large mafic intrusion controlling the Gossey deposit.

Some 40 DD holes totalling 8,934 m and 200 RC holes totalling 37,053 m were drilled in this drill campaign.

A follow-up drill campaign was also completed over the targets defined during the previous campaign and on new geochemical anomalies revealed by AC drilling earlier in the year. This campaign consisted of 89 RC holes totalling 13,065 m and 27 DD holes totalling 6,367 m.

10.2.2 2017 AND 2018 DRILL CAMPAIGNS

During the second quarter of 2017, an infill RC drill program on a 50 m x 50 m pattern was implemented at the Gossey deposit. A total of 15,000 m was proposed to upgrade the classification of the resource and infill the southern part of the deposit. This drill program was completed during the third quarter of 2017, and a total of 15,254 m (124 RC holes) were drilled.

Towards the end of 2017, an internal preliminary estimate of Mineral Resources was conducted by GMSI with no boundary imposed around the Gossey Village (Figure 10-6). Pit constrained Indicated Mineral Resources were estimated to total 10.3 Mt at an average grade of 0.84 g/t Au for a total of 285,000 ounces of gold. Pit constrained Inferred Mineral Resources were estimated at that time at 6.5 Mt at an average grade of 0.95 g/t Au for a total of 221,000 ounces of gold.

In 2018, a second infill drill program of 14,300 m commenced. The objectives of this Phase 2 drill campaign were to test for strike extensions of the deposit, test grade continuity to a vertical depth of approximately 100 m and convert Inferred Mineral Resources into Indicated Mineral Resources. In addition, drilling was intended to test for lateral and down-dip extensions of gold mineralization.

This program was completed during April 2018 and 14,284 m (191 RC holes) were drilled. The database compilation was subsequently completed and submitted to GMSI in mid-May 2018 for a Mineral Resource update.

Figure 10-7 presents significant intersections from the 2017 and 2018 drilling programs.

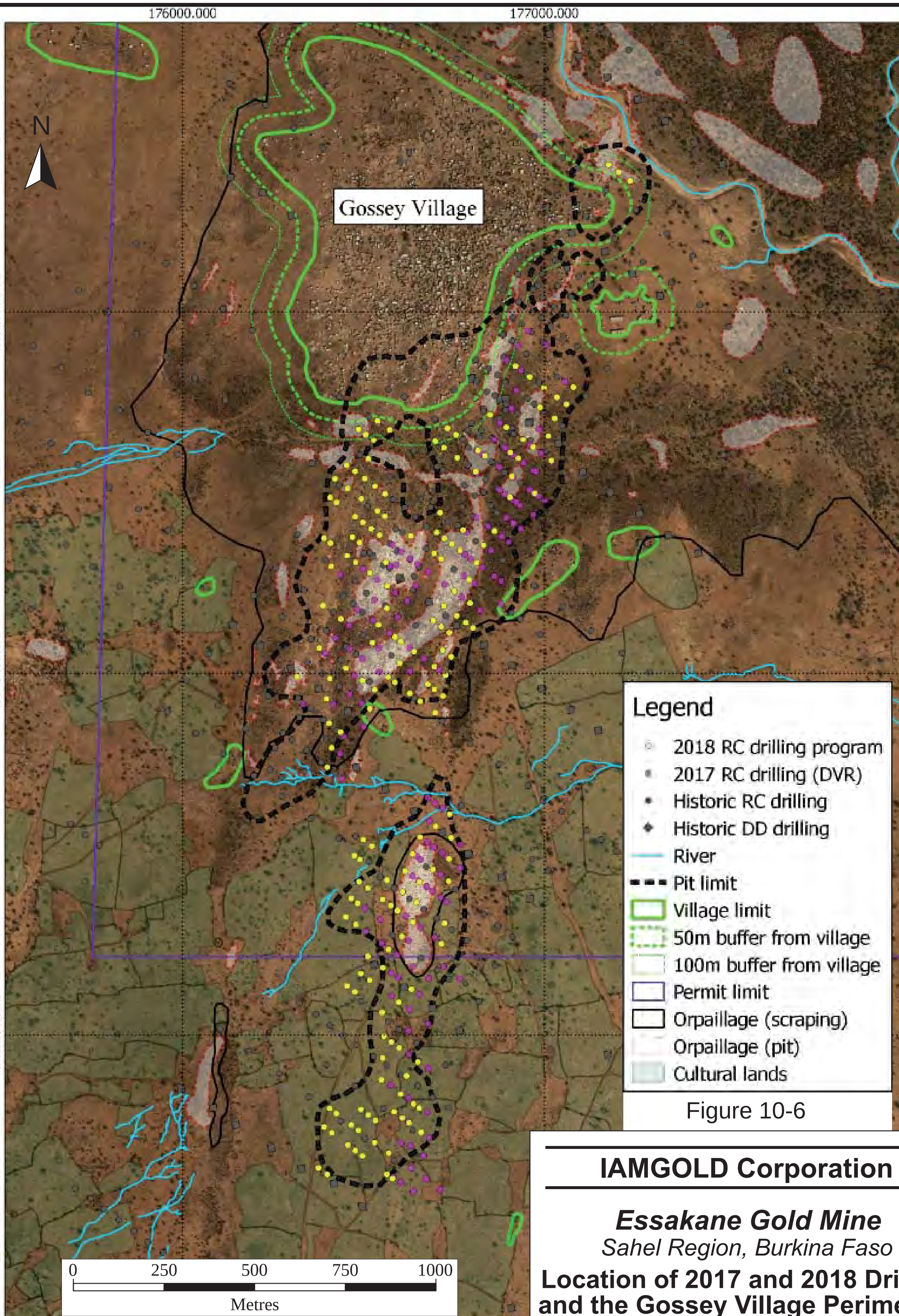


Figure 10-6

IAMGOLD Corporation

Essakane Gold Mine
Sahel Region, Burkina Faso

Location of 2017 and 2018 Drilling and the Gossey Village Perimeters

January 2020

Source: Sawadogo, 2018.

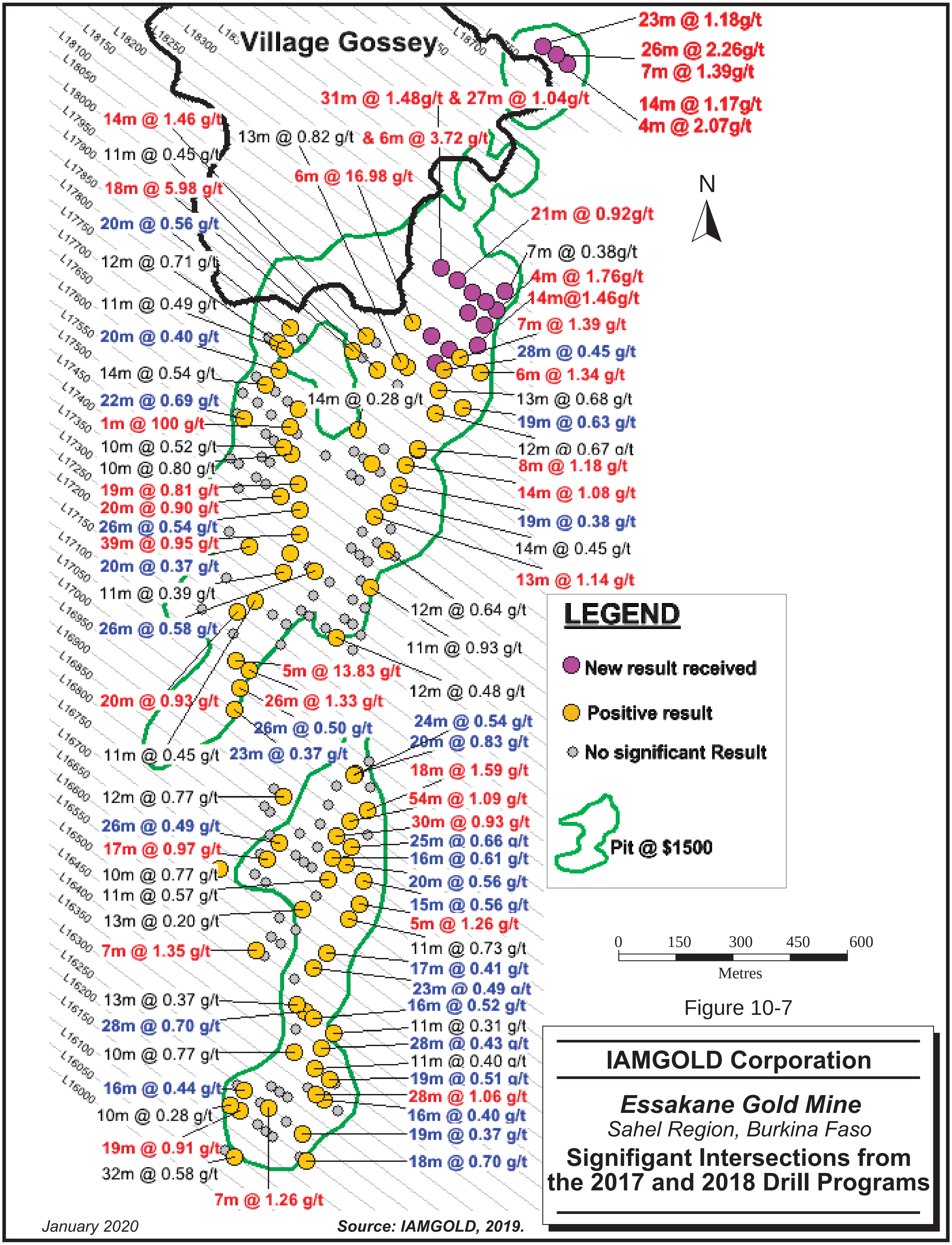

Village Gossey
23m @ 1.18g/t
26m @ 2.26g/t
7m @ 1.39g/t
14m @ 1.17g/t
4m @ 2.07g/t
31m @ 1.48g/t & 27m @ 1.04g/t
& 6m @ 3.72 g/t
13m @ 0.82 g/t
6m @ 16.98 g/t
14m @ 1.46 g/t
11m @ 0.45 g/t
18m @ 5.98 g/t
20m @ 0.56 g/t
12m @ 0.71 g/t
11m @ 0.49 g/t
20m @ 0.40 g/t
14m @ 0.54 g/t
22m @ 0.69 g/t
1m @ 100 g/t
10m @ 0.52 g/t
10m @ 0.80 g/t
19m @ 0.81 g/t
20m @ 0.90 g/t
26m @ 0.54 g/t
39m @ 0.95 g/t
20m @ 0.37 g/t
11m @ 0.39 g/t
26m @ 0.58 g/t
20m @ 0.93 g/t
11m @ 0.45 g/t
23m @ 0.37 g/t
12m @ 0.77 g/t
26m @ 0.49 g/t
17m @ 0.97 g/t
10m @ 0.77 g/t
11m @ 0.57 g/t
13m @ 0.20 g/t
7m @ 1.35 g/t
13m @ 0.37 g/t
28m @ 0.70 g/t
10m @ 0.77 g/t
16m @ 0.44 g/t
10m @ 0.28 g/t
19m @ 0.91 g/t
32m @ 0.58 g/t
7m @ 1.26 g/t
21m @ 0.92g/t
7m @ 0.38g/t
4m @ 1.76g/t
14m@1.46g/t
7m @ 1.39 g/t
28m @ 0.45 g/t
6m @ 1.34 g/t
14m @ 0.28 g/t
13m @ 0.68 g/t
19m @ 0.63 g/t
12m @ 0.67 g/t
8m @ 1.18 g/t
14m @ 1.08 g/t
19m @ 0.38 g/t
14m @ 0.45 g/t
13m @ 1.14 g/t
12m @ 0.64 g/t
11m @ 0.93 g/t
5m @ 13.83 g/t
12m @ 0.48 g/t
26m @ 1.33 g/t
26m @ 0.50 g/t
24m @ 0.54 g/t
20m @ 0.83 g/t
18m @ 1.59 g/t
54m @ 1.09 g/t
30m @ 0.93 g/t
25m @ 0.66 g/t
16m @ 0.61 g/t
20m @ 0.56 g/t
15m @ 0.56 g/t
5m @ 1.26 g/t
11m @ 0.73 g/t
17m @ 0.41 g/t
23m @ 0.49 g/t
16m @ 0.52 g/t
11m @ 0.31 g/t
28m @ 0.43 g/t
11m @ 0.40 g/t
19m @ 0.51 g/t
28m @ 1.06 g/t
16m @ 0.40 g/t
19m @ 0.37 g/t
18m @ 0.70 g/t
N
LEGEND
New result received
Positive result
No significant Result
Pit @ $1500
0 150 300 450 600
Metres
IAMGOLD Corporation
Essakane Gold Mine
Sahel Region, Burkina Faso
Signifigant Intersections from the 2017 and 2018 Drill Programs
January 2020
Source: IAMGOLD, 2019.

Figure 10-7

10.2.3 DRILLING PROCEDURES FOR THE 2017/2018 GOSSEY DRILL CAMPAIGN

Drilling in 2017 and 2018 comprised of RC drilling using a 140 mm (5 ½ in.) drill bit and a conventional cyclone set up. No sample splitting was undertaken at the drill site. All drill cuttings were collected every 0.5 m in a large plastic bag (800 mm x 500 mm). The cyclone was cleaned before the start of every drill hole and cleaned as required during drilling (if wet drilling conditions were encountered). The name of the drill hole and depth of the sample was written on each sample bag and recorded on the sample sheet (handwritten). The samples were sealed for transportation.

Once drilled, a cemented monument was installed on each drill collar with the hole identification and final depth written on it.

10.2.3.1 COLLAR SURVEYS

Once a drill hole was completed, the collar was initially surveyed using a handheld Garmin™ GPS, and subsequently resurveyed, at a later date, with a DGPS. The coordinate system used at Gossey is the UTM Zone 31, Datum WGS84.

10.2.3.2 DOWNHOLE SURVEYS

Downhole surveys were taken using the Reflex EZ-Shot at 5 m intervals. The data was subsequently filtered to remove any erroneous or suspicious data points.

10.2.3.3 SAMPLE STORAGE AND SECURITY

Samples were transported periodically from the drilling site to the Essakane mine site, located 12 km to the south-east of the Gossey deposit under the supervision of IAMGOLD geologists and field technicians. The samples were stored in the laydown of the exploration department, where sample preparation and splitting occur. The Essakane mine site is a highly secure facility, and only Essakane S.A. employees may enter the site.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 EMZ, FALAGOUNTOU WEST, AND WAFAKA DEPOSITS

11.1.1 SAMPLE PREPARATION AND ANALYSIS

The sample preparation protocol currently used by Essakane S.A. was developed by GF BVI in conjunction with Snowden Engineering Inc. (Snowden) in 2006 (Gignac, 2008). The aim of the protocol is to reduce grouping and segregation error (GSE) and fundamental sampling error (FSE) in a coarse gold environment.

Orezone Resources used cyanide saturated two kilogram bulk leach extractable gold (BLEG) analysis to improve assay reproducibility. A poor reproducibility was demonstrated with the fire assay method. In addition, fire assay of the BLEG residues showed an average leach of 97%. It was subsequently determined that rolling for an additional 24 hours with fresh cyanide resulted in higher BLEG solution grades. BLEG analyses were conducted by the following independent laboratories: SGS Tarkwa Ghana, SGS Essakane, and TransWorld Ghana (now Intertek Minerals Limited, Tarkwa Minerals Laboratory Branch). SGS' laboratories are accredited, however, IAMGOLD does not have any information regarding the accreditation status of the TransWorld Ghana laboratory.

In January 2006, GF BV replaced Orezone Resources' two kilogram BLEG bottle roll process with LeachWell rapid cyanide leach on one kilogram sub-samples (the LWL69M method).

Since the acquisition of Essakane S.A. by IAMGOLD in 2009, all assays have been carried out at the Essakane laboratory using the LeachWell method on one kilogram samples followed with fire assay of the tails when the grade is higher than 5 g/t Au.

The majority of drill holes are sampled at one metre intervals. Core is sawed in two, and one half is sent for assaying when the hole is either outside the MII pit shell or selected by the geologist. Otherwise the entire length is crushed and pulverized. The entire sample is crushed to 95% passing (P_{95}) two millimetres in a Terminator or Boyd crusher. It is then split in 12 parts in a rotary splitter and a 1.2 kg sub-sample is pulverized to P_{95} 105 µm with LM-5 or LM-2 mills. A 1,000 g sub-sample is assayed by LeachWell rapid cyanide leach over 12 hours with an atomic absorption spectroscopy (AAS) finish. Initially, 10% of assays that returned over 0.3 ppm Au had their solid residues re-assayed using fire assay. This percentage was raised to 25% in 2016. In addition, 5% of assays below 0.3 ppm Au had their solid residues re-assayed using fire assay.

It is noted that all Keegor mills have been replaced with LM-5 mills, however, they are still available during rush periods.

All crushing and pulverizing rejects are returned to and stored at the Resource Development facility where 20% are later selected for check assaying at a commercial laboratory in Ouagadougou using the same protocol. Check samples are selected based on the presence of arsenopyrite mineralization regardless of the original grade. It was found that choosing the check samples based on the Essakane laboratory assay results alone resulted in a selection bias (i.e., over a long term, check samples, on average, returned lower values than the Essakane laboratory's results). The sampling protocols for DD samples are shown in Table 11-1.

TABLE 11-1 DIAMOND DRILL SAMPLE PREPARATION AND ASSAYING PROTOCOL

Step	Description
1. Reception	• Dry (6h at 105ºC). • Weigh and note.
2. Crushing and Pulverization	• Crush entire sample in jaw crusher down to P_{95} 2 mm.
3. Sieving	• Test particle size at a frequency of 5% when prompted by the Laboratory Information Management System (LIMS).
4. Division	• Divide by RSD and combine enough pots for a 1 kg sub-sample. Pots must be opposite as much as possible. • Use the second set of alternating pots when a duplicate has been requested by the LIMS. • Time required to obtain final splits must not be less than two minutes. • Return rejects to the Resource Development storage facility.
5. Mill Washing with Quartz	• The LM2 and LM5 must be cleaned with blank quartz (or construction aggregate used by Camp Maintenance) after each pulverization.
6. LeachWell 1,000 g	• Assay 1,000 g sample (note exact weight): • Leaching period NaCN: 12 hours • AAS finish • Report weight of sample in grams and results in ppm
7. Fire Assay	• Fire assay 25% of solid residues that returned > 0.3 ppm Au in original assay and 5%of solid residues that returned <0.3 ppm Au

Since 2010, RC drilling has been carried out using 140 mm (5.5 in.) diameter holes with five metre sample intervals to a depth of 150 m or until the water table is intersected. The seven kilogram field split is dried and pulverized to P_{95} 500 µm in Keegor mills. Occasionally, when the sample is comprised of coarse particles, crushing is performed through a Terminator or Boyd Crusher prior to the pulverization stage. The sample is split in a rotary divider until two sub-samples weighing one kilogram each are obtained. One sub-sample is pulverized to P_{95} 500 µm and 1,000 g sample is assayed by LeachWell rapid cyanide leach. Similar to the DD samples, 10% solid residues are re-assayed using fire assay whenever the LeachWell result exceeds 0.3 ppm Au.

Approximately 20% of the crushed RC pulps are sent to ALS CHEMEX and SGS in Ouagadougou, for check assaying.

In 2014, revisions were made to the preparation protocols in order to address concerns raised by the Agoratek International Consultants Inc. (Agoratek) sampling consultant. The main concerns addressed were the mass of RC samples and the pulverization size. On the initial protocol the RC sample mass submitted to pulverization was 1.2 kg. Also pulp duplicate are sent to the external laboratory instead of coarse duplicate. The quantity of water and the rolling time have been revised as well.

The revisions included changing the pulverization size from P_{90} of 75 µm to P_{95} of 500 µm for RC samples (to avoid flattening of coarse gold) and matching preparation and assaying protocols of the primary (mine) laboratory and the check laboratory, particularly concerning the amount of water used in the LeachWell leaching stage and the time the bottles were rolled.

Sampling protocols for RC samples are shown in Table 11-2.

TABLE 11-2 REVERSE CIRCULATION SAMPLE PREPARATION AND ASSAYING PROTOCOL

Step	Description
1. Reception	• Dry (six hours at 105ºC). • Weight and note.
2. Crushing (occasionally) and Pulverization	• Pulverize entire 7 kg sample in Keegor mills to P_{95} 500 µm.
3. Sieve analysis	• Test particle size at a frequency of 5% when prompted by the LIMS.
4. Division	• Divide by RSD and combine enough pots for a 1 kg sub-sample. Pots must be opposite as much as possible. • Use the second set of alternating pots when a duplicate has been requested by the LIMS. • Time required to obtain final splits must not be less than two minutes. • Return rejects to the Resource Development storage facility.
5. Mill washing with quartz	• The LM2 and LM5 must be cleaned with blank quartz (or construction aggregate used by Camp Maintenance) after each pulverization.
6. LeachWell 1,000g	• Assay 1,000 g sample (note exact weight): • Leaching period NaCN: 12 hours • AAS finish • Report weight of sample in grams and results in ppm
7. Fire Assay	• Fire assay 25% of solid residues that returned > 0.3 ppm Au in original assay and 5% of solid residues that returned <0.3 ppm Au

11.1.2 SAMPLE SECURITY

Following the IAMGOLD acquisition of Orezone Resources and Essakane in 2009, all drill samples were collected under direct supervision of the Project staff from the drill rig and remained within the custody of the staff up to the moment the samples were delivered to the Essakane laboratory.

Samples, including duplicates, were delivered from the drill rig to a secure storage area within the fenced Essakane core facility. Then blanks and certified reference materials (CRM) were inserted. Chain of custody procedures consisted of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples were received by the laboratory. Sample security has relied upon the fact that the samples are always attended or locked in appropriate sample storage areas prior to dispatch to the sample preparation facility.

In the responsible QP's opinion, the sample preparation, analysis, and security procedures at Essakane are adequate for use in the estimation of Mineral Resources.

11.1.3 QUALITY ASSURANCE AND QUALITY CONTROL

Essakane S.A. is using a QA/QC system which involves insertion of CRMs supplied by Rocklabs Limited and locally sourced blanks.

The CRMs were selected based on the range of gold grades and type of material to be submitted to the laboratory (oxide or sulphide sample). A list of CRMs used in the assay program since 2010 is provided in Table 11-3.

TABLE 11-3 LIST OF CERTIFIED REFERENCE MATERIALS

OXIDE MATERIAL TYPE		SULPHIDE MATERIAL TYPE	
CRM	Assigned Value (g/t Au)	CRM	Assigned Value (g/t Au)
OXA131	0.077	SE44	0.606
OXA71	0.084	SE58	0.608
OXC109	0.201	SG84	1.026
OXC72	0.205	SG56	1.027
OXC129	0.205	SH82	1.333
OXC145	0.212	SH41	1.344
OXD108	0.414	SH69	1.346
OXD73	0.416	SH65	1.348
OXD87	0.417	SH55	1.375
OXD144	0.417	SI54	1.78
OXD107	0.452	SI64	1.78
OXE106	0.606	SJ63	2.632
OXF105	0.8	SJ53	2.637
OXF85	0.805	SJ80	2.656
OXF65	0.805	HISILK4	3.463
OXF125	0.806	HISILK2	3.474
OXH122	1.247	SK94	3.899
OXH97	1.278	SK62	4.075
OXi96	1.802	SK78	4.134
OXi67	1.817	SL77	5.181
OXJ95	2.337	SL51	5.909
OXJ68	2.342	SL61	5.931
OXK79	3.532	SL76	5.96
OXK119	3.604	SN60	8.595
		SN50	8.685
		SN74	8.891
		HISILP1	12.05
		SP59	18.12

Standards (100 g weight) are inserted at a rate of one standard per 20 samples. Results for every batch of CRMs, reported by the assay laboratory, are assessed by IAMGOLD's database manager prior to upload of any assay data into the SQL database. The average of the CRM results for each batch is reported to the laboratory manager in a qualitative way by e-mail (trends showing over or underestimation; evidence for poor instrumental drift corrections; differences occurring at operator shift changes, etc.). Records of these assessments are stored in the Essakane S.A. database.

Blanks consist of coarse granite sourced from the west of Burkina Faso (Table 11-4). They are inserted at a rate of one blank per 20 samples, mostly within the expected mineralized interval. Formerly, barren quartz was used as blank material. One kilogram bags of granite blank material are inserted into the sample stream and prepared in the same way as any other RC or DD sample.

TABLE 11-4 LIST OF LOCAL BLANKS

Material Type	Blank	Estimated Value (g/t Au)	Origin
GRANITE	GRT01	0.0005	WEST/BURKINA

The field duplicates insertion rate is one per 20 samples and 20% of pulps are selected for external laboratory checking.

The failure criteria are as follows:

- The standard is considered to have failed when it is outside ±3 standard deviations.
- Blanks are considered to have failed when the assay grade is greater than ten times the detection limit (D.L =0.001 g/t Au).
- Duplicate precision has been recommended after the construction of a ranked Half Absolute Relative (HARD) graph.

From the deviation (HARD) plot, it has been determined that a precision interval between ± 20% to ± 40% for 90% of the material varying from pulp to coarse reject (Long,1998) be targeted.

In the responsible QP's opinion, the QA/QC program as designed and implemented by Essakane S.A. is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.

Figures 11-1 through 11-5 illustrate the QA/QC graphs used by IAMGOLD team and the previous company.

FIGURE 11-1 STANDARD OXK119 PLOT

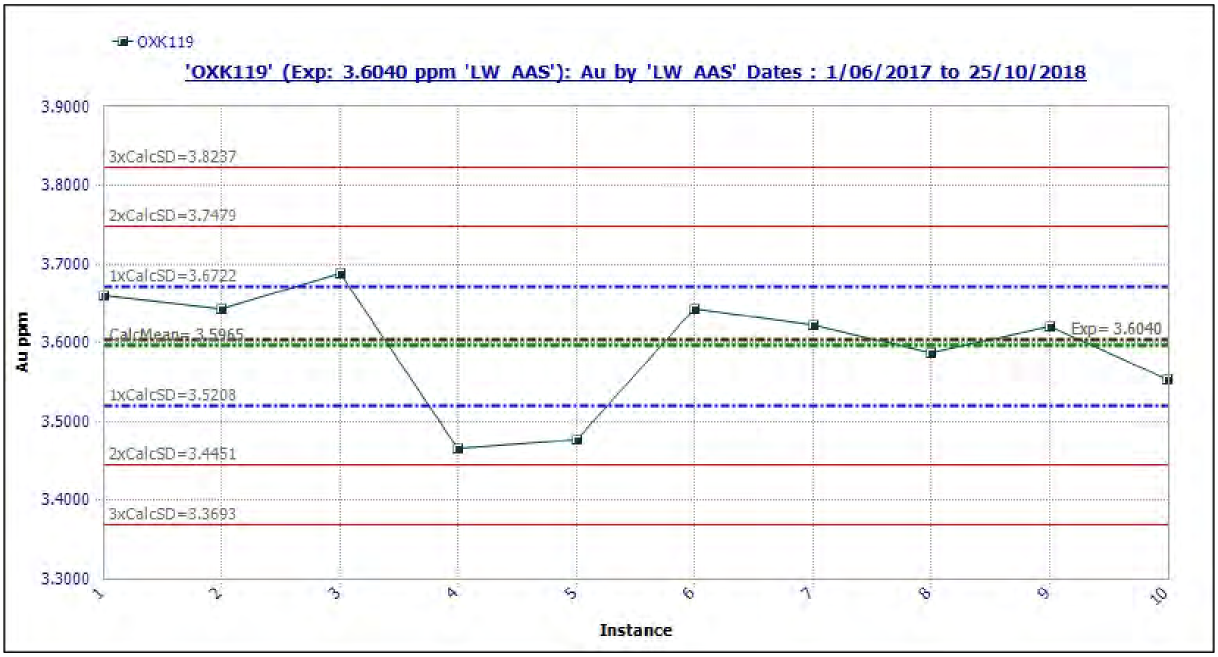


FIGURE 11-2 BLANK GRT01 PLOT

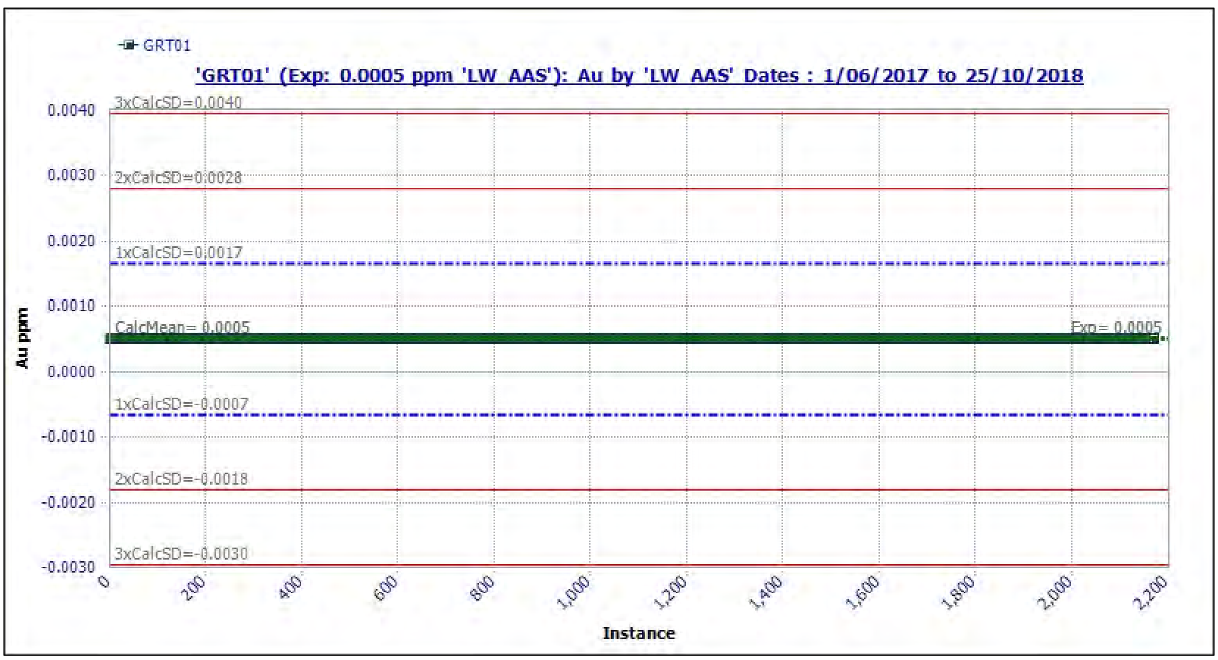


FIGURE 11-3 FIELD DUPLICATE VERSUS ORIGINAL SCATTERPLOT

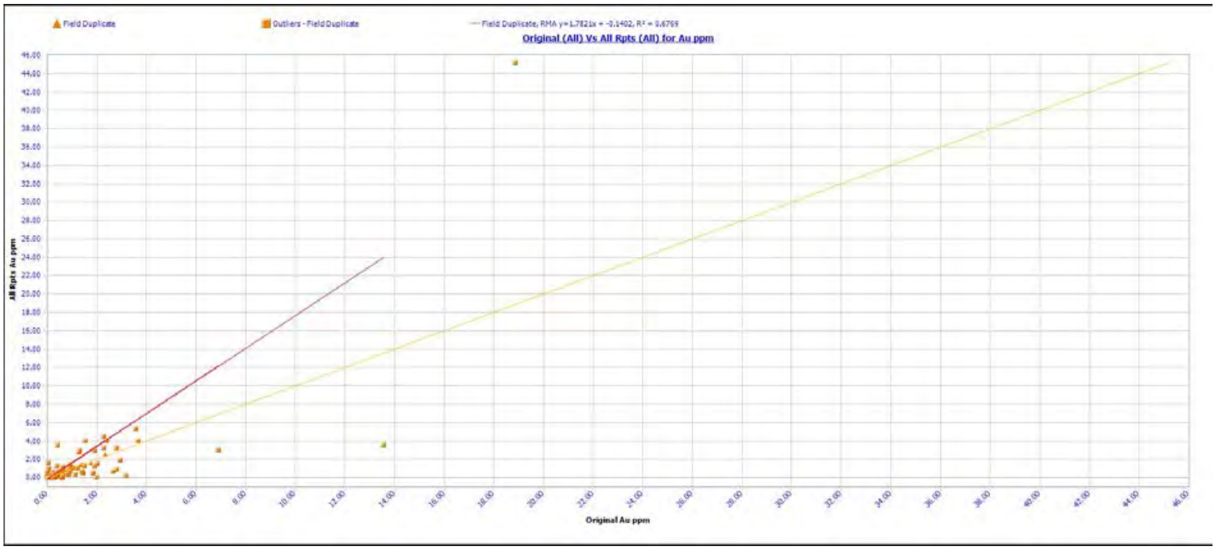


FIGURE 11-4 LOG-LOG DUPLICATE PLOT

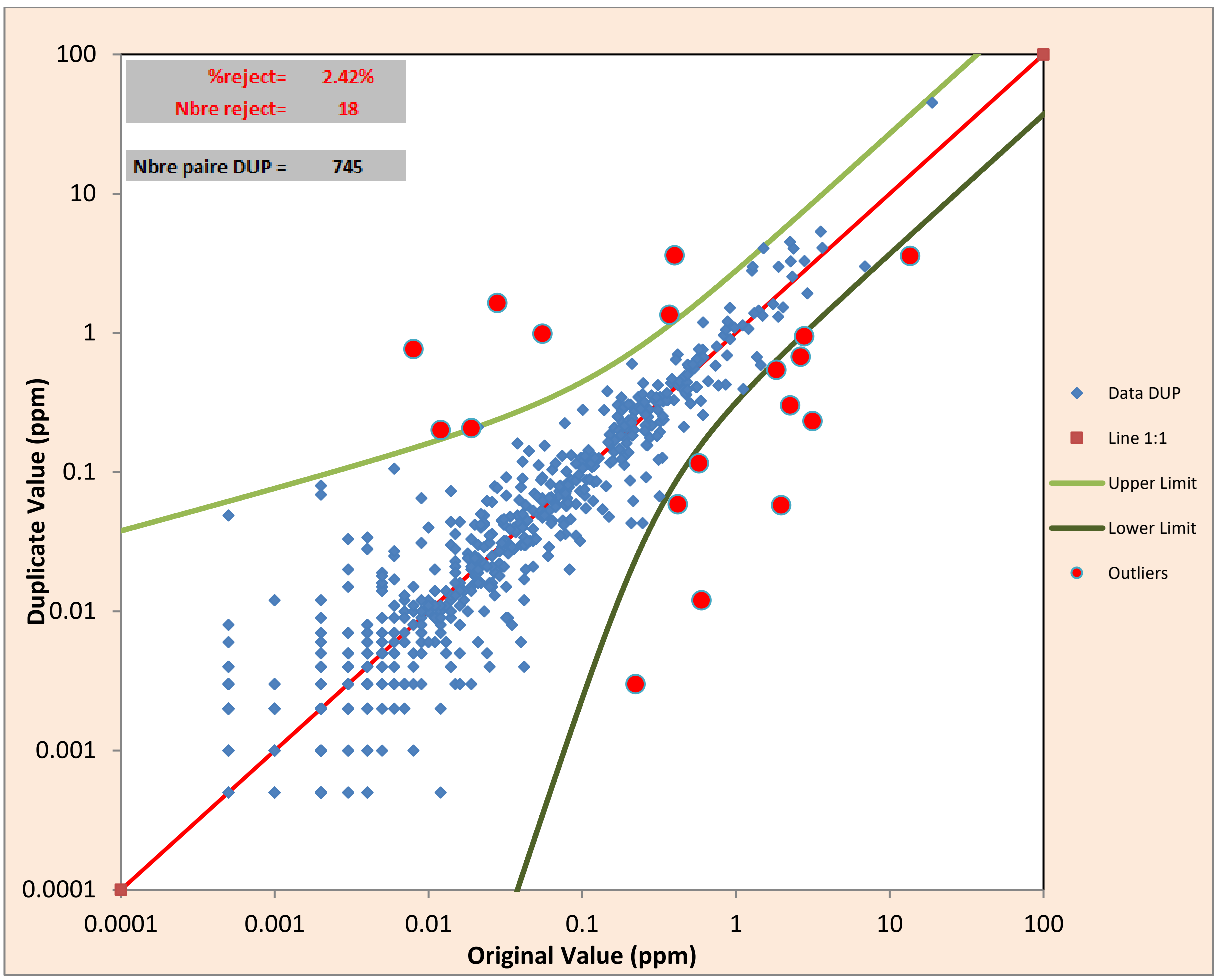
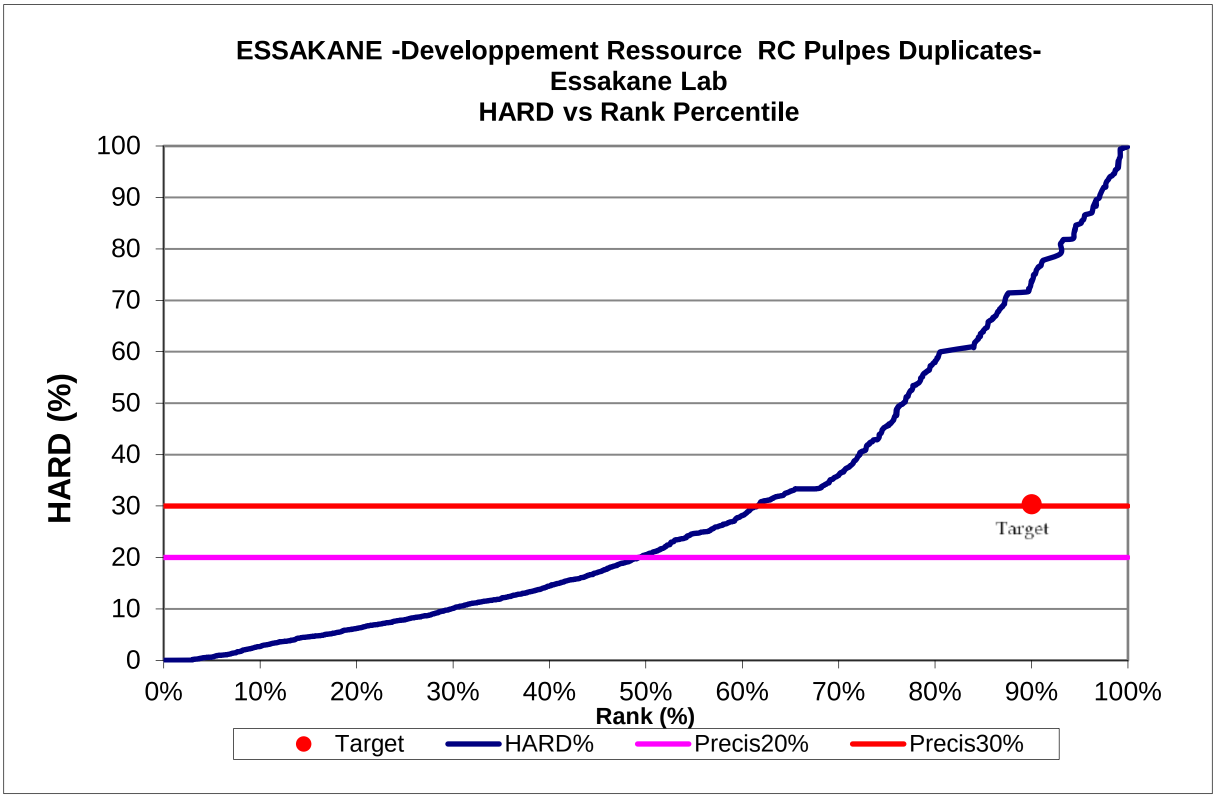


FIGURE 11-5 HARD PLOT VERSUS RANK PERCENTILE



11.2 GOSSEY DEPOSIT

The information contained within this section was sourced from the final 2017 Gossey drilling campaign report, and the March 2018 QA/QC report prepared by the Essakane mine resource development team. This document outlines the practices and workflows employed during the 2017 and 2018 Gossey drill programs designed for sample preparation, assaying, QA/QC, logging, and data collection.

11.2.1 REVERSE CIRCULATION PERCUSSION SAMPLING

Samples were collected every 0.5 m (around 10-20 kg each) and weighed at the drill rig. Once transported to the Essakane mine site, a one metre composite was formed by combining two 0.5 m samples. This was subsequently reduced in size through a 1-tier, 50:50 riffle splitter to produce a final split for the laboratory weighing approximately 5 kg, with a coarse reject preserved for archiving. The remaining material was discarded after a small portion was retained for the chip tray.

11.2.2 QA/QC PROTOCOL

The insertion of QA/QC samples into the sample stream has been completed in accordance with protocols developed for the Essakane Gold Mine. The purpose of submitting QA/QC samples is to ensure the integrity of the assay data, by drawing attention to any erroneous or suspicious laboratory results, and ensuring an auditable trail is available if any issues arise.

All samples were analyzed by the mine site laboratory, which was toured by GMSI in March 2018. Approximately 10–20% of the regular sample pulps were sent to an external laboratory (SGS Ouagadougou) as intra-laboratory umpire check assays.

A list of QA/QC samples used during the 2017 and 2018 Gossey drill campaigns are summarized in Table 11-5.

TABLE 11-5 LIST OF QA/QC SAMPLES – 2017 AND 2018 GOSSSEY DRILL CAMPAIGNS

Sample Type	Description	Grade (g/t Au)	Frequency of Insertion
FLDDUP	Field Duplicate	N/A	1 for every 20 regular samples
UMPIRE	Intra-lab duplicate (pulps sent to SGS Ouagadougou)	N/A	10% - 20% of all regular sample pulps > 0.3 g/t Au
BLANK	Blank quartz sand (uncertified)	0.0005	1 for every 20 regular samples, plus additional blanks before and after mineralised zones
STANDARDS	Various standards ranging from 0.07 g/t to 5.18 g/t from Rock Labs	0.07 g/t to 5.18 g/t	1 for every 20 regular samples

The QA/QC sample protocol results in a batch of 24 samples sent to the laboratory containing at least one blank, one standard, and one field duplicate. When a standard fails (result is greater than 3SD of the certified value), the ten samples before and after the failed sample (21 in total including the failed sample) are reanalyzed.

All standards for the 2017 and 2018 drilling campaigns at Gossey were sourced from Rock Labs. Twelve different CRMs (standards) were used ranging from very low-grade (0.077 g/t Au) to high-grade (5.96 g/t Au) and are shown in Table 11-6. GMSI notes that the Rock Labs CRMs quote an assigned value for the fire assay technique, however the Gossey samples were analyzed using the LeachWell technique. Due to the difference between the two analysis methods, some deviations in the overall mean values could have occurred.

Of the non-blank CRMs inserted into the sample stream, approximately 54% contained oxide matrices, and the remaining 46% contained sulphide matrices.

TABLE 11-6 LIST OF CERTIFIED REFERENCE MATERIALS USED DURING THE 2017 AND 2018 GOSSEY DRILL CAMPAIGNS

CRM	Assigned Value (g/t Au)	Standard Deviation	Sulphur (%)	Number Inserted	Mean of Analyses (g/t Au)
OXA131	0.077	0.007	-	129	0.079
OXC129	0.205	0.007	-	153	0.204
OCC145	0.212	0.007	-	113	0.207
OXD144	0.417	0.048	-	51	0.420
OXF125	0.806	0.02	-	62	0.800
OXH122	1.247	0.031	-	110	1.236
OXH97	1.278	0.03	-	24	1.280
OXJ95	2.337	0.057	-	80	2.391
SH82	1.333	0.027	2.8	71	1.288
SH69	1.346	0.026	2.7	139	1.308
SJ80	2.656	0.057	3.0	147	2.628
SK94	3.899	0.084	3.0	134	3.948
SL77	5.181	0.156	3.2	116	5.167
SL76	5.960	0.192	-	102	5.996
			Total	**1,431**	

11.2.3 DATA RECORDING

All relevant data was recorded on a sampling sheet and sample ID's were assigned accordingly to each one metre composite. The sample sheet includes the insertion of QA/QC samples. An example sheet is shown in Figure 11-6.

FIGURE 11-6 TYPICAL SAMPLE SHEET OUTLINING DATA COLLECTED DURING SAMPLE PREPARATION

IAMGOLD ESSAKANE SARL

PROJECT ESSAKANE
50 / 50 SPLITTER SAMPLING SHEET

HOLEID: KRC0729
Date: 16-Apr-17
Geo: TH
TECH: JW
Team Leader: KJB
Page: 1 of 4

						QC Sample				Geo	Weight	< 6kg No Split		6kg - 24kg 1st Split		> 24kg 2nd Split	
BHID	From	To	OK	SampleID	OK	Type	Parent_ID	Code	Jar ID	Notes	Start (l)	E1	R	E1	R1	R2	R3
KRC0729				1197493		BLK											
KRC0729	0	1															
KRC0729	1	2		1197494													
KRC0729	2	3		1197495													
KRC0729	3	4		1197496													
KRC0729	4	5		1197497													
KRC0729	5	6		1197498													
KRC0729	6	7		1197499													
KRC0729				1197500		STD											
KRC0729	7	8		1197501													
KRC0729	8	9		1197502													
KRC0729	9	10		1197503													
KRC0729	10	11		1197504													
KRC0729	11	12		1197505													
KRC0729	12	13		1197506													
KRC0729	13	14		1197507													
KRC0729	14	15		1197508													
KRC0729	15	16		1197509													
KRC0729	15	16		1197510		FD	1197509										
KRC0729	16	17		1197511													
KRC0729	17	18		1197512													
KRC0729	18	19		1197513													
KRC0729	19	20		1197514													
KRC0729	20	21		1197515													

Source: Allou, B., & Pratas, M., 2017

All sampling data is captured in Microsoft Excel and is transferred daily onto the Data Coordinator's computer. This data is subsequently imported into the central database (DataShed) either directly, or via the data input software (LogChief).

11.2.4 LABORATORY SAMPLE PREPARATION AND ANALYTICAL PROCEDURES

GMSI representatives toured the Essakane mine laboratory in March 2018 to oversee the sample preparation and assaying techniques applied to samples from the Gossey deposit. The laboratory was considered to meet the specifications required for reporting under the CIM guidelines (May 2014) and assay data produced are considered suitable for inclusion in the estimation of Mineral Resources.

The following procedure was observed for assaying RC samples for the Gossey deposit:

- Sample weighing and drying at 105°C
- Pulverisation of entire sample (5 – 7 Kg) in a Keegor pulveriser to P95 at 500 µm
- Splitting of pulverised sample using a rotary splitter to obtain a 1 kg split for assaying
- Analysis using the LeachWell techniques:
    - Two parts water to one-part sample (two litres of water)
    - Addition of one LeachWell tablet
    - Leach time of 12 hours, ensuring that the pH remains above 10.5
    - Decanting time of one hour, homogenization of the solution for seven minutes and final AAS finish of the solution
- 20% to 30% of the residues are analyzed by Fire Assay where the grade is > 0.3 g/ Au:
    - Residue is filter-pressed, washed, and dried at 105°C
    - Residue is rotary split to obtain a 50 g sub-sample
    - Typical fire assay route and AAS finish

11.2.5 RESULTS OF QA/QC PROGRAM

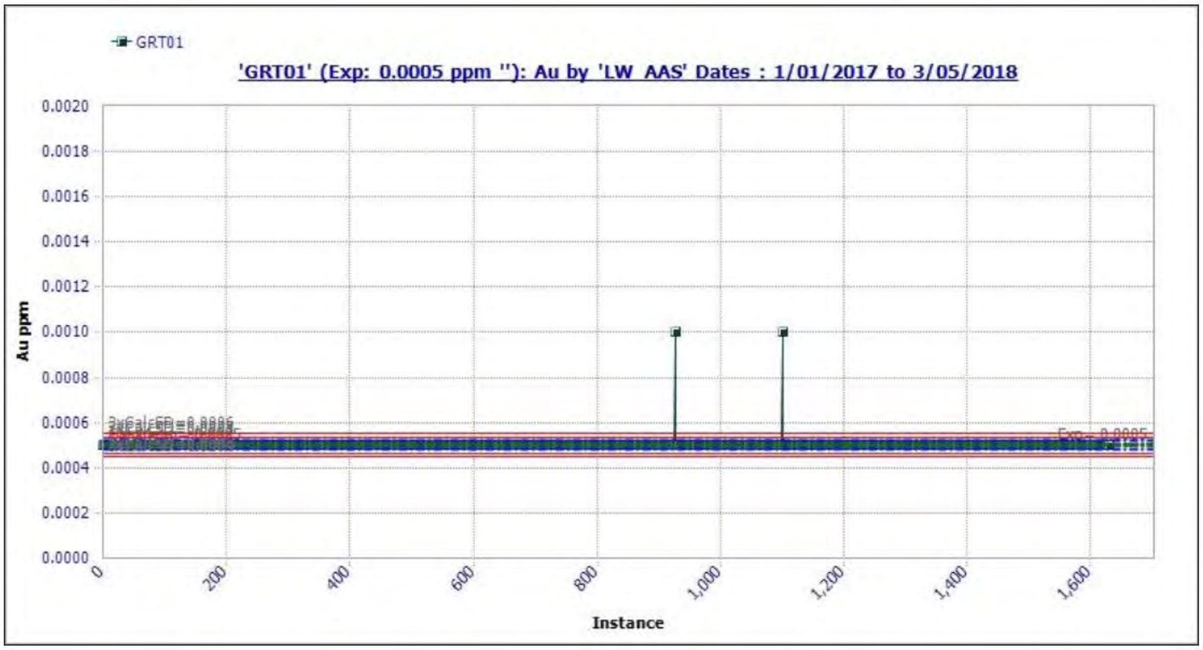
QA/QC sample analysis results for the 2017 and 2018 drill programs were provided to GMSI for review as part of the Mineral Resource for the Gossey deposit. No QA/QC data was made available to GMSI for the historical data.

11.2.5.1 BLANKS AND CRM RESULTS

Fifteen various CRMs (14 certified standards, one uncertified blank) were routinely inserted into the sample stream at a rate of two per batch of 24. In addition, quartz blanks used to wash the Keegor pulverisers were analyzed to detect any contamination. The various CRMs contain grades ranging from very low-grade (0.077 g/t Au) to high-grade (5.96 g/t Au) and represent either oxide or sulphide matrices. Essakane geologists consider any CRMs falling outside of three standard deviations as a failure, which resulted in reanalysis of the batch by the laboratory.

Blank performance is considered good, with both the Keegor blanks and the blanks inserted into each batch (2,134 in total) returning acceptable results, with only two instances where the detection limit (0.001 g/t Au) was achieved. The results of the batch-inserted blank analyses are shown in Figure 11-7.

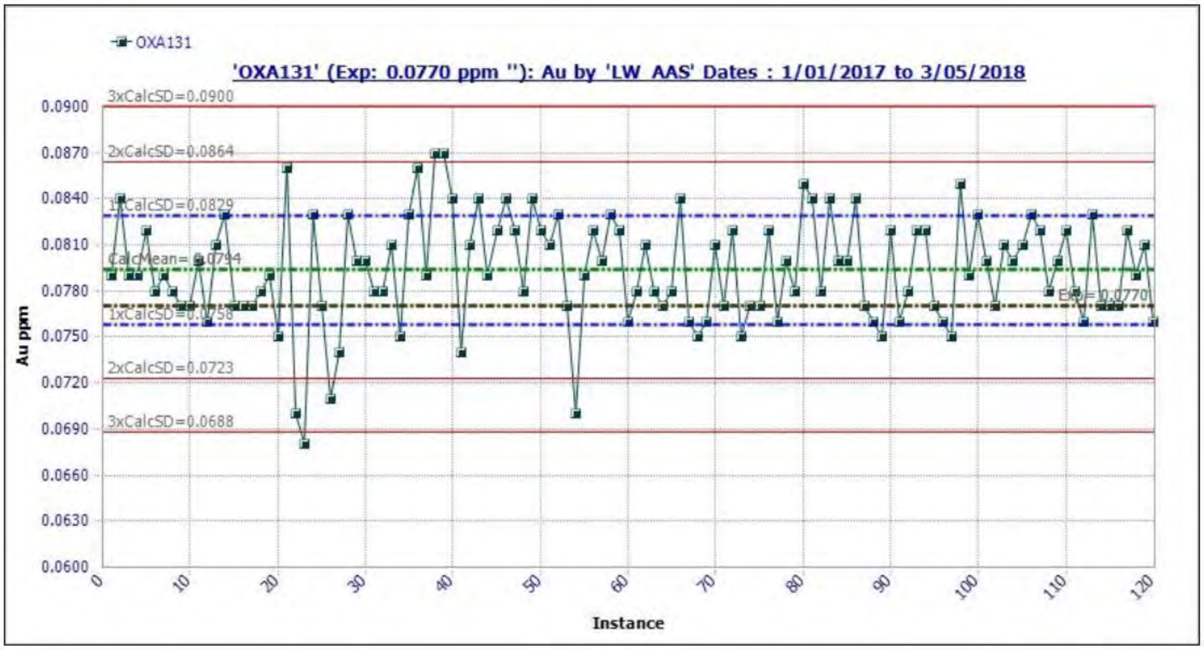
FIGURE 11-7 BATCH-INSERTED BLANK PERFORMANCE



Source: Zongo, B., Salle, A., 2018

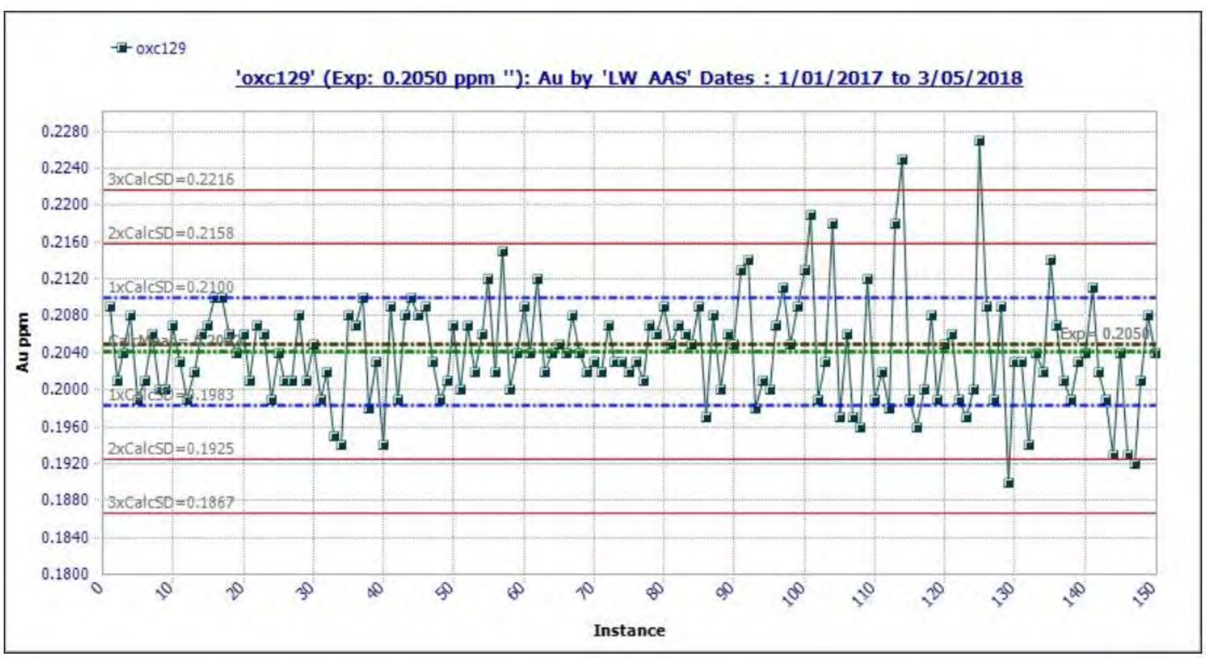
Overall performance of the oxide CRMs (OXA131, OXC129, OXC145, OXD144, OXF125, OXH122, and OXJ95) is considered acceptable, with the vast majority of samples falling within the three standard deviations supplied by the provider of the CRM. More variability was observed in the 2018 drilling for the low-grade (0.205 g/t Au) CRM OXC129; however, the results still centre around the expected certified value. The performance of each oxide CRM is shown in Figure 11-8 to Figure 11-14.

FIGURE 11-8 OXA131 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



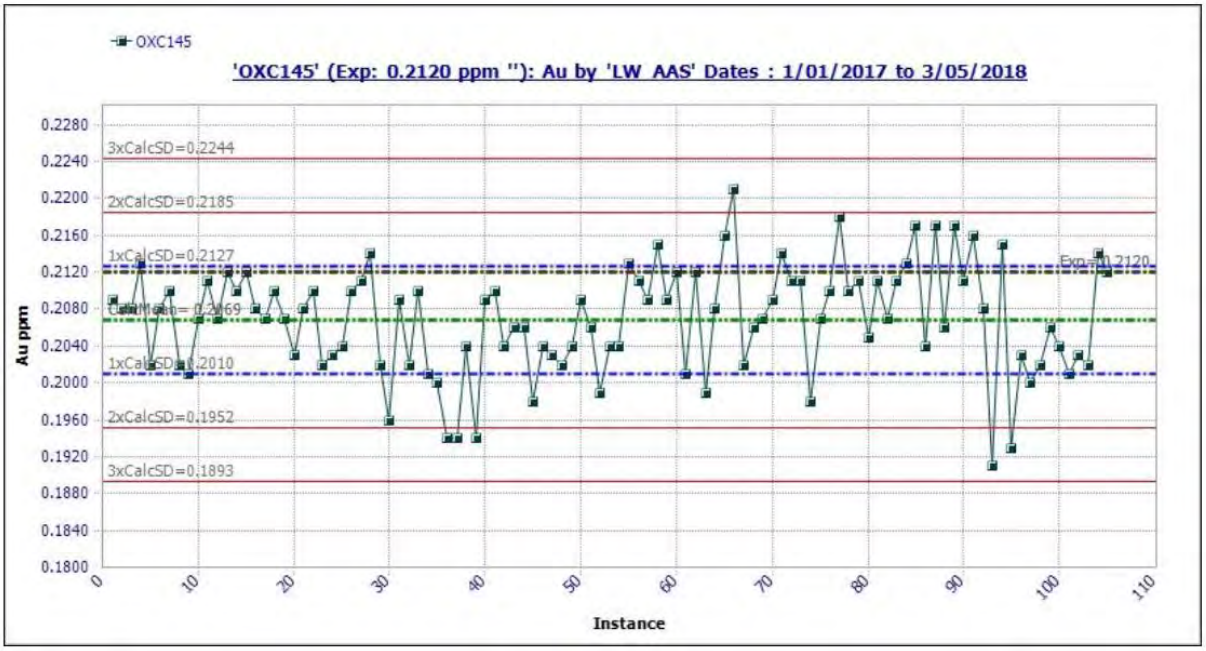
Source: Zongo, B., Salle, A., 2018

FIGURE 11-9 OXC129 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



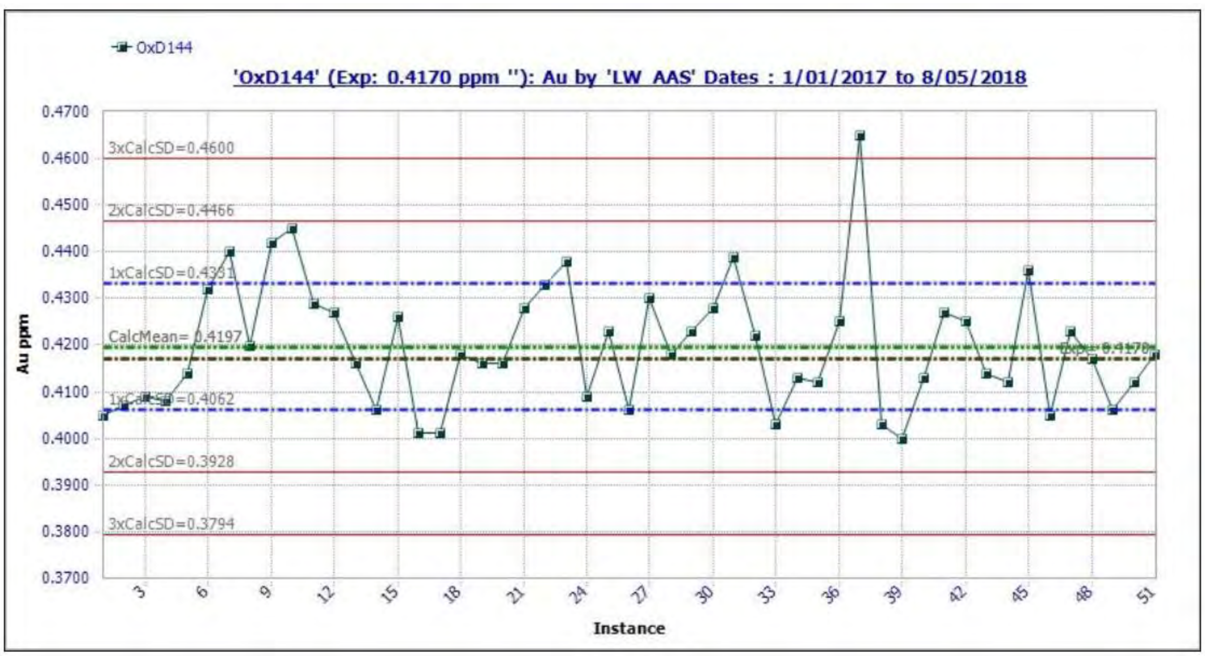
Source: Zongo, B., Salle, A., 2018

FIGURE 11-10 OXC145 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



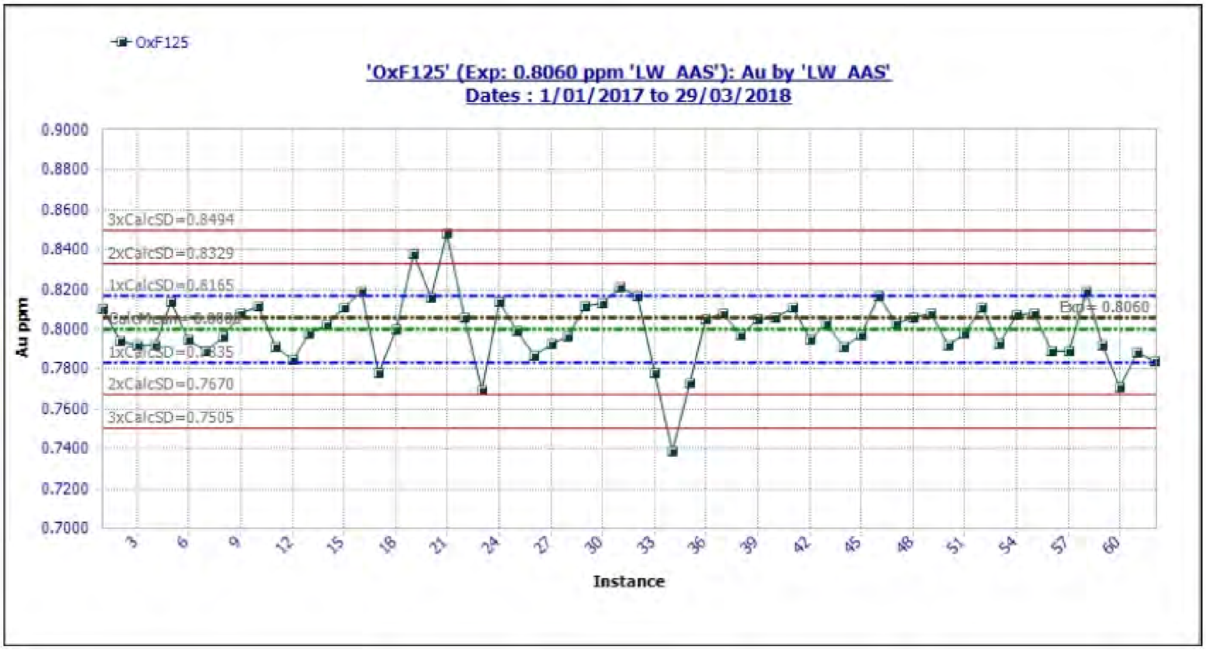
Source: Zongo, B., Salle, A., 2018

FIGURE 11-11 OXD144 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



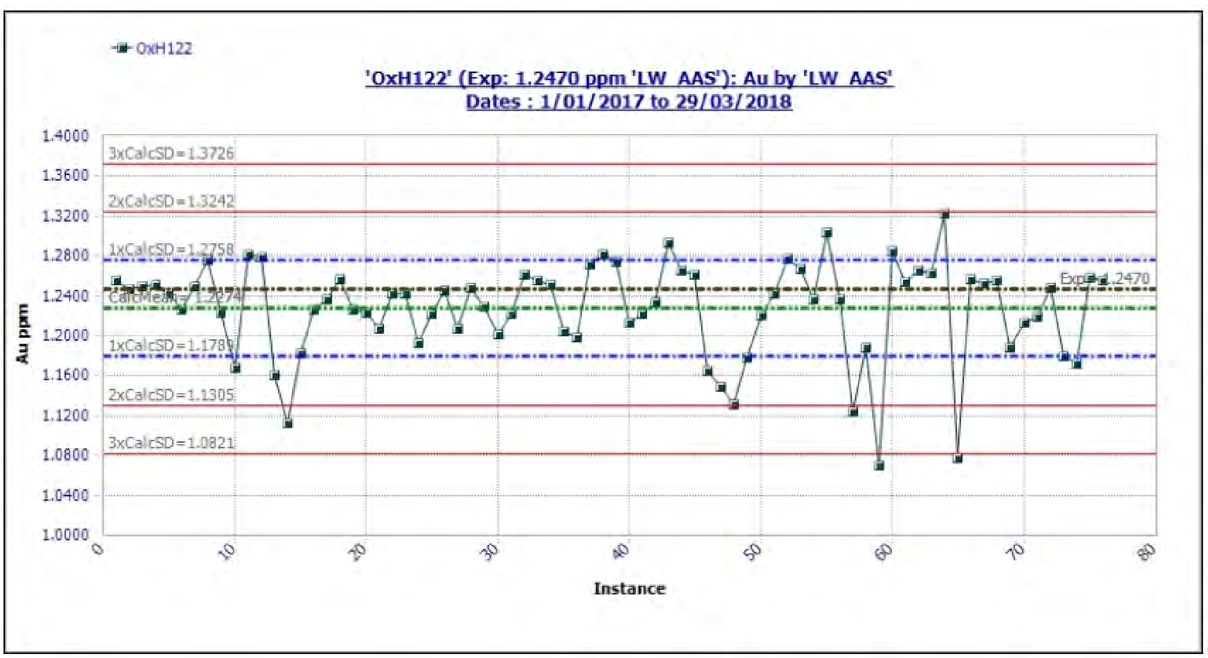
Source: Zongo, B., Salle, A., 2018

FIGURE 11-12 OXF125 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



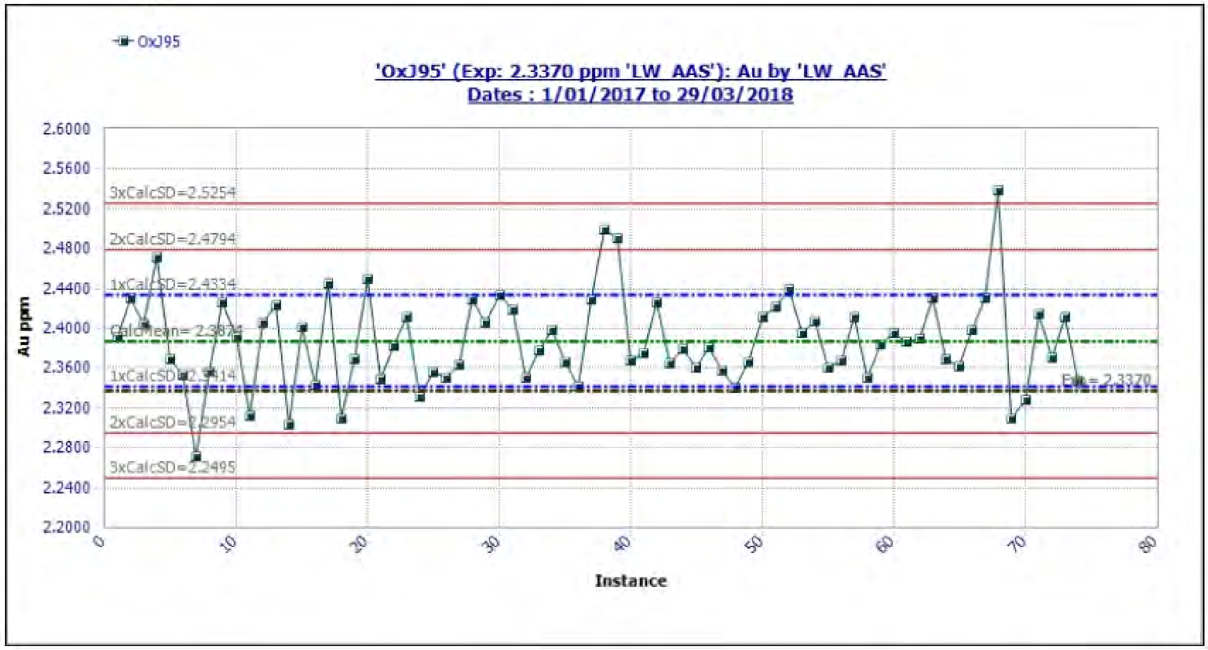
Source: Zongo, B., Salle, A., 2018

FIGURE 11-13 OXH122 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



Source: Zongo, B., Salle, A., 2018

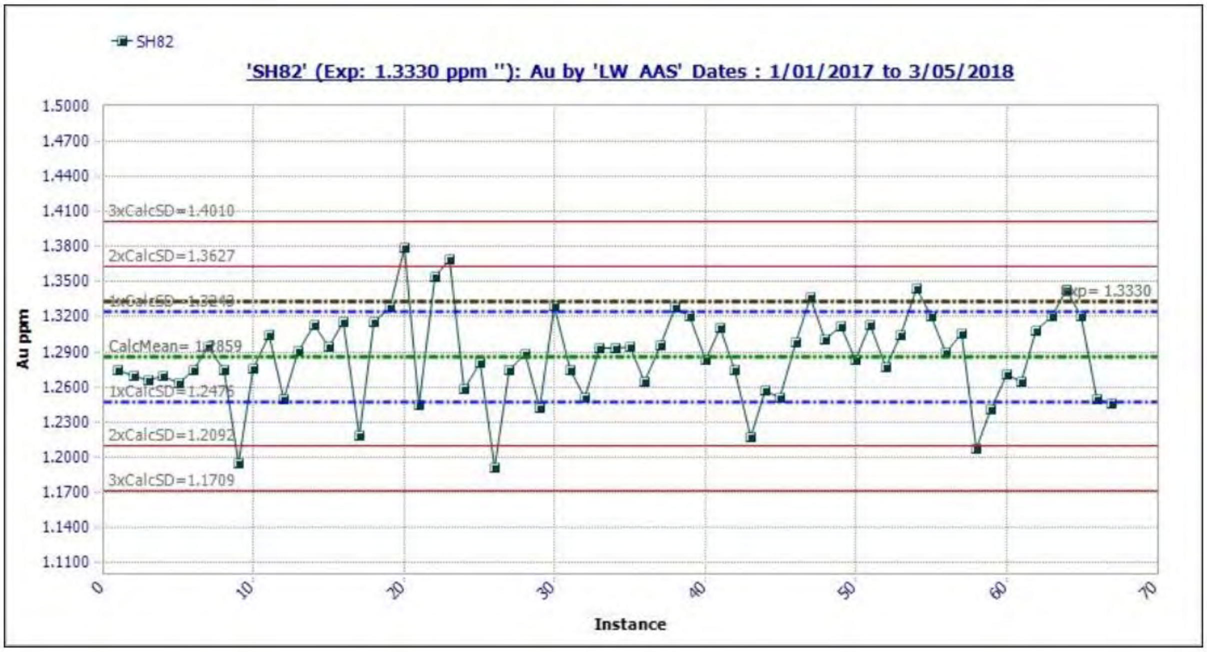
FIGURE 11-14 OXJ95 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



Source: Zongo, B., Salle, A., 2018

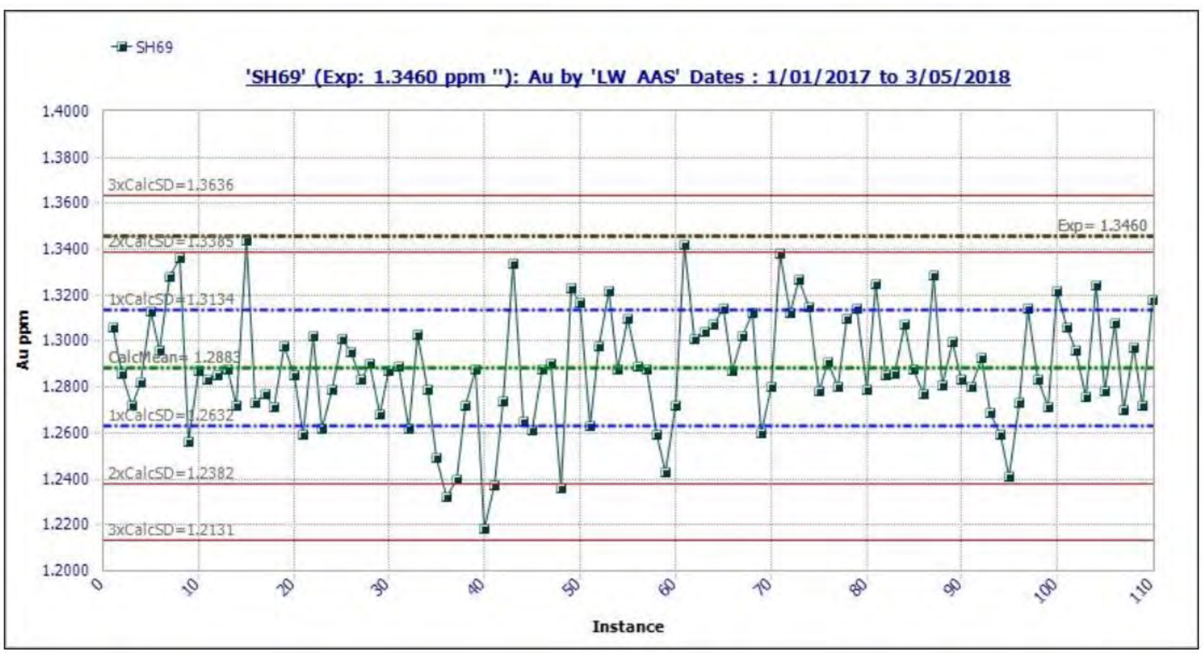
Overall performance of the sulphide CRMs (SH82, SH69, SL76, SI64, SJ80, SK94, and SL77) is considered acceptable, with the vast majority of samples falling within the three standard deviations supplied by the provider of the CRM. The performance of CRM SL77 (high-grade, 5.181 g/t Au) was relatively poor for the 2017 drill campaign; however, the results for this CRM during the 2018 drill campaign improved significantly. GMSI notes that no low-grade sulphide standard was available for analysis. Ideally, a low-grade sulphide standard in the range of 0.2 – 0.5 g/t Au should be used to ensure the effective analysis of low-grade material. The performance of each sulphide CRM is shown in Figure 11-15 to Figure 11-20.

FIGURE 11-15 SH82 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



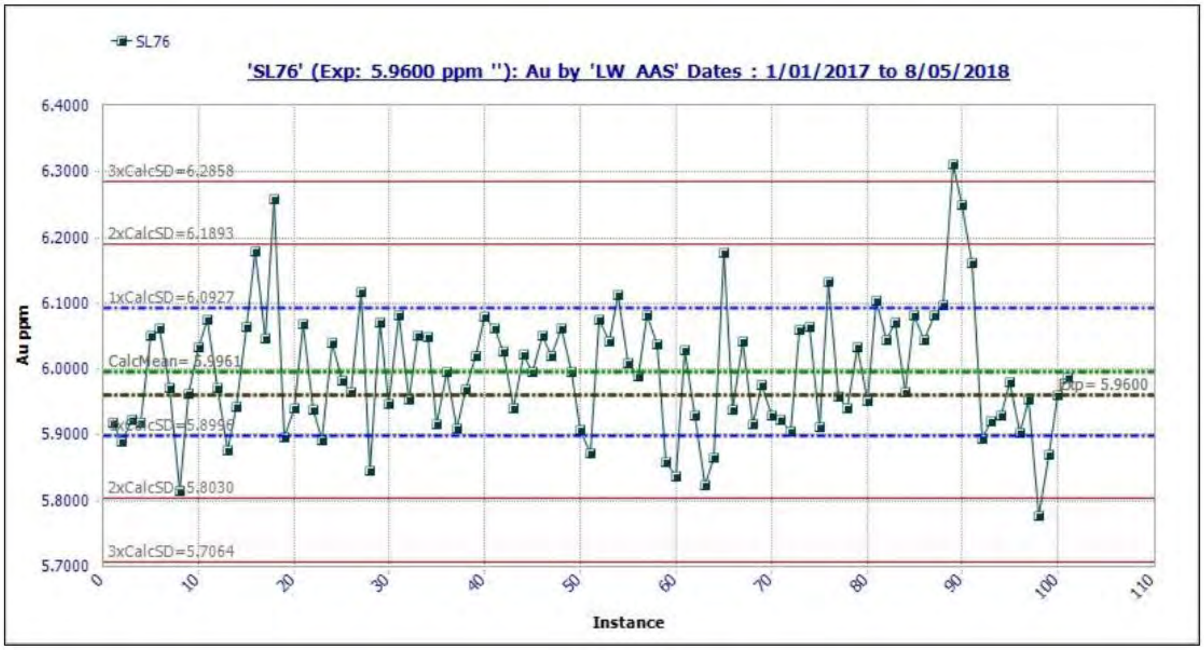
Source: Zongo, B., Salle, A., 2018

FIGURE 11-16 SH69 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



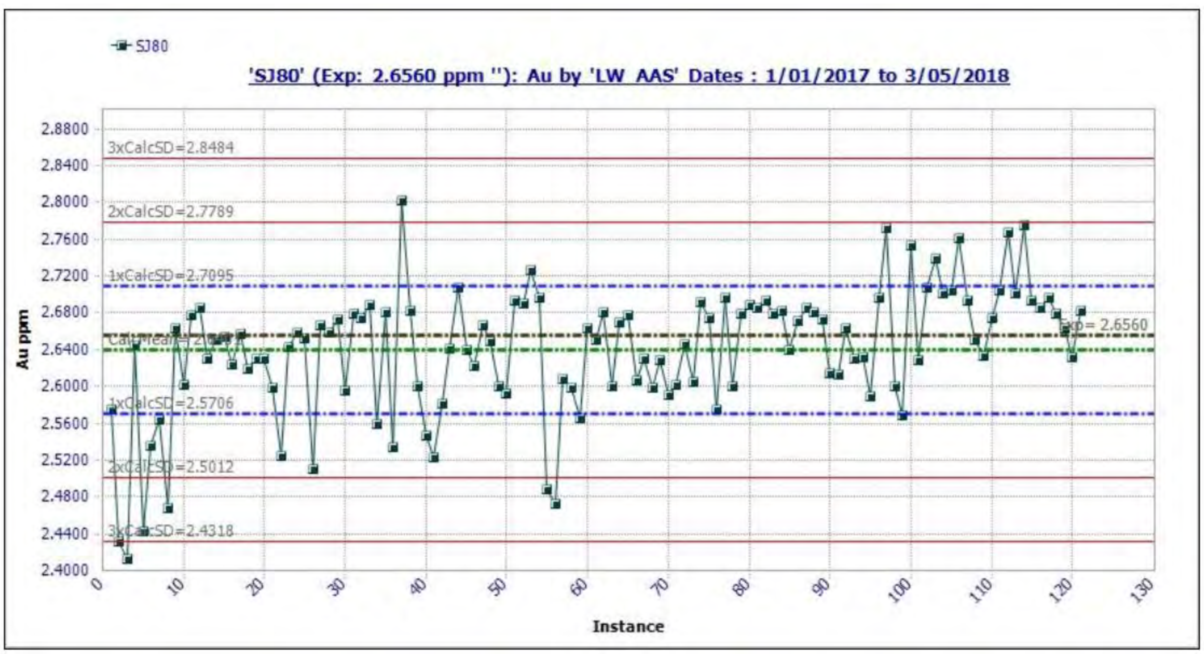
Source: Zongo, B., Salle, A., 2018

FIGURE 11-17 SL76 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



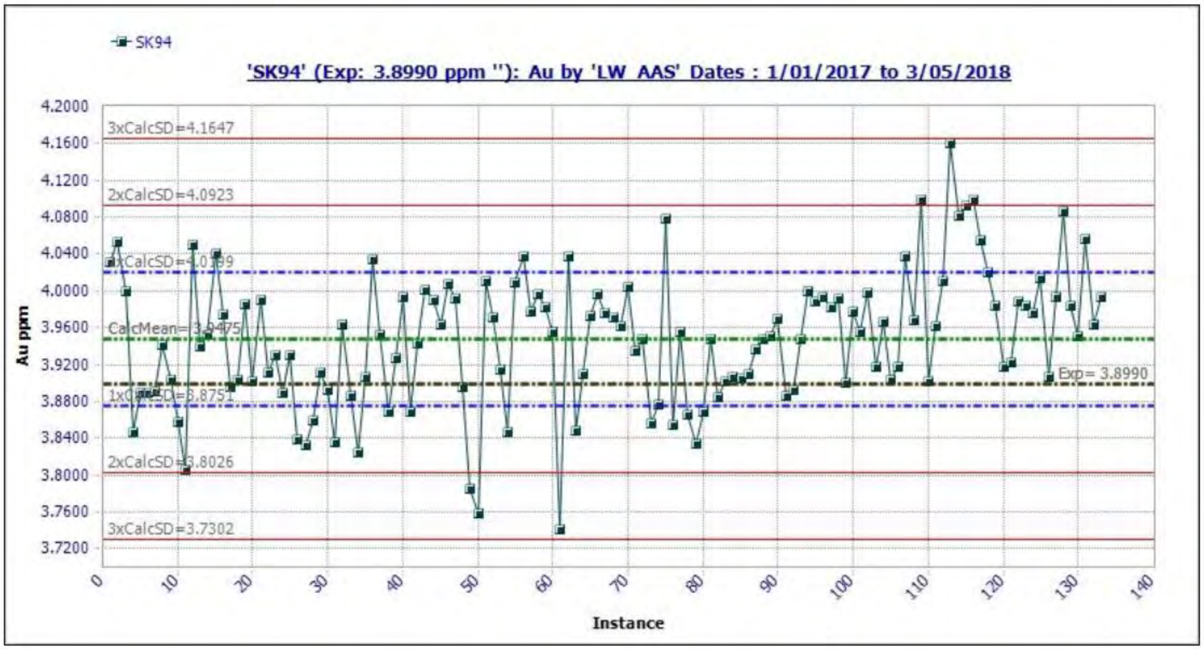
Source: Zongo, B., Salle, A., 2018

FIGURE 11-18 SJ80 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



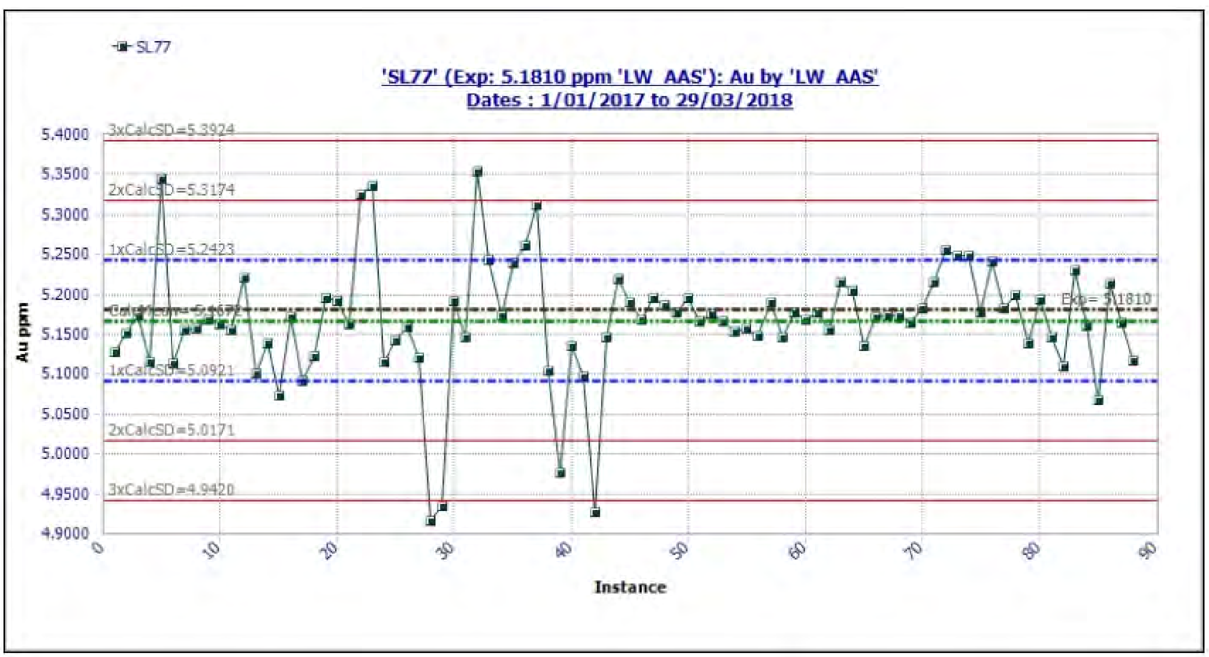
Source: Zongo, B., Salle, A., 2018

FIGURE 11-19 SK94 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



Source: Zongo, B., Salle, A., 2018

FIGURE 11-20 SL77 PERFORMANCE - 2017 AND 2018 DRILLING CAMPAIGNS AT GOSSEY



Source: Zongo, B., Salle, A., 2018

11.2.5.2 FIELD DUPLICATES

Field duplicates were taken at a frequency of 1 for every 20 regular samples. Field duplicates are analyzed at the Essakane laboratory and were produced from the coarse rejects (during initial sample splitting).

Figure 11-21 shows the relationship between the original result (X-Axis) versus the field duplicate result (Y Axis). Due to the high-nugget gold nature of the Gossey deposit, it is expected that a certain proportion of the field duplicates will fall outside of the accepted limits.

FIGURE 11-21 FIELD DUPLICATE PERFORMANCE SCATTER PLOT – 2017 AND 2018 DRILL PROGRAMS AT GOSSEY

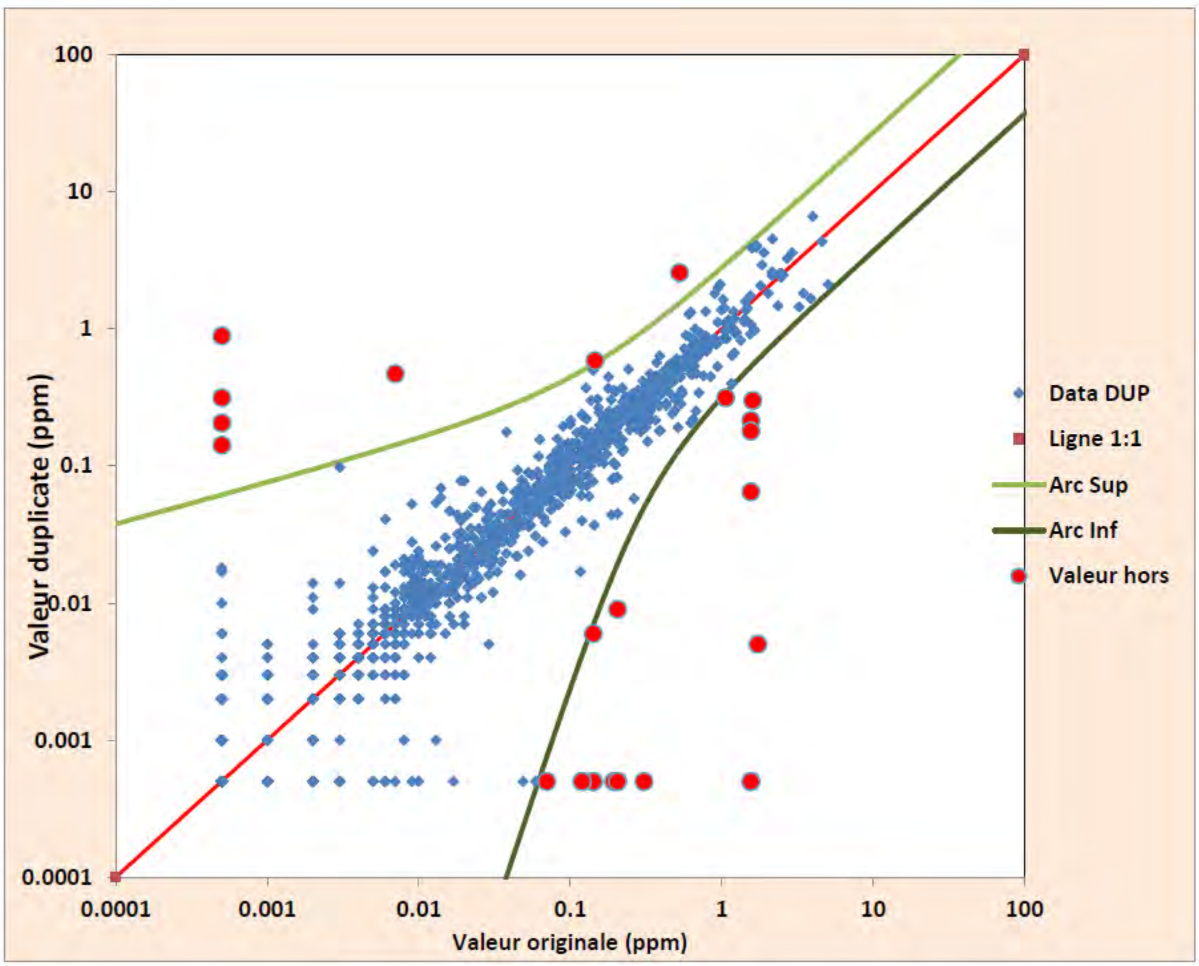
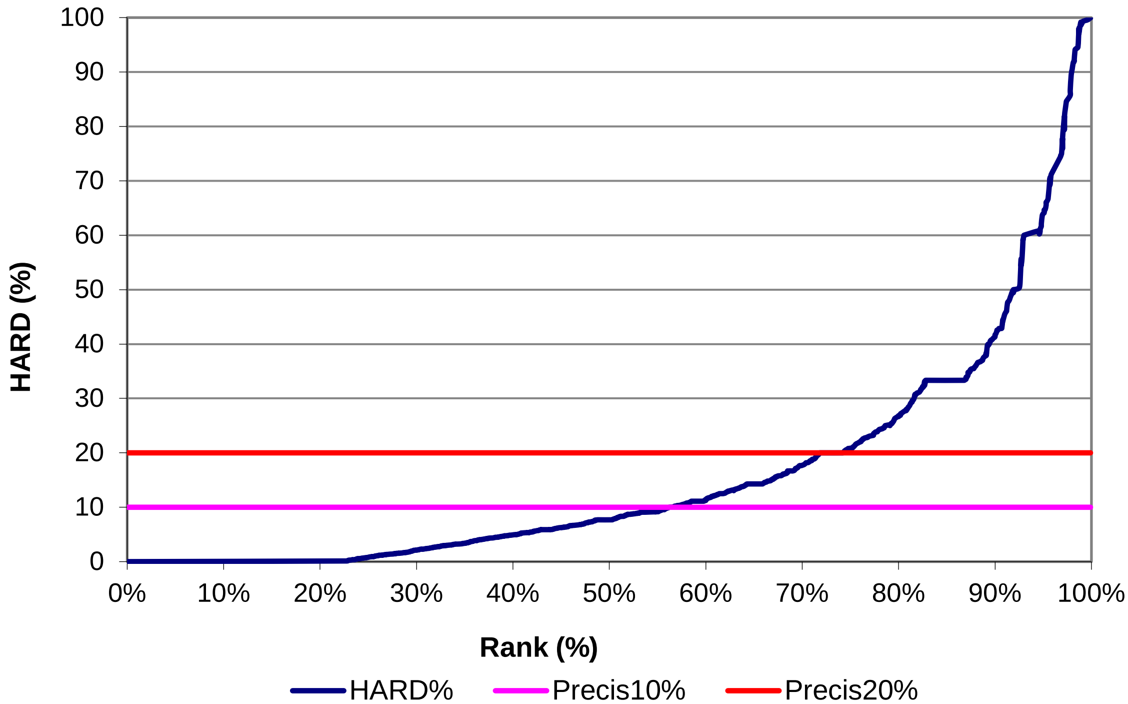


Figure 11-22 shows the HARD% (Half Absolute Relative Difference) plot for all samples for the 2017 and 2018 drill programs at Gossey.

FIGURE 11-22 HARD% PLOT OF FIELD DUPLICATES - 2017 AND 2018 DRILL PROGRAMS AT GOSSEY



11.2.5.3 INTRA-LAB CHECK ASSAYS (UMPIRE ASSAYS)

For the 2017 and 2018 drill campaigns, between 10% and 20% of the original sample pulps were sent to SGS Ouagadougou as umpire check assays. SGS Ouagadougou followed the same analytical procedure as the Essakane laboratory, ensuring that the results are comparable. Only pulps with an original assay result of greater than 0.3 g/t Au were sent to SGS Ouagadougou.

Figure 11-23 shows the relationship between the original result (X-Axis) versus the umpire pulp duplicate result (Y Axis). Figure 11-24 shows the HARD% plot for umpire pulp duplicates for all samples from the 2017 and 2018 drill programs at Gossey.

FIGURE 11-23 UMPIRE PULP DUPLICATES PERFORMANCE SCATTER PLOT – 2017 AND 2018 DRILL PROGRAMS AT GOSSEY

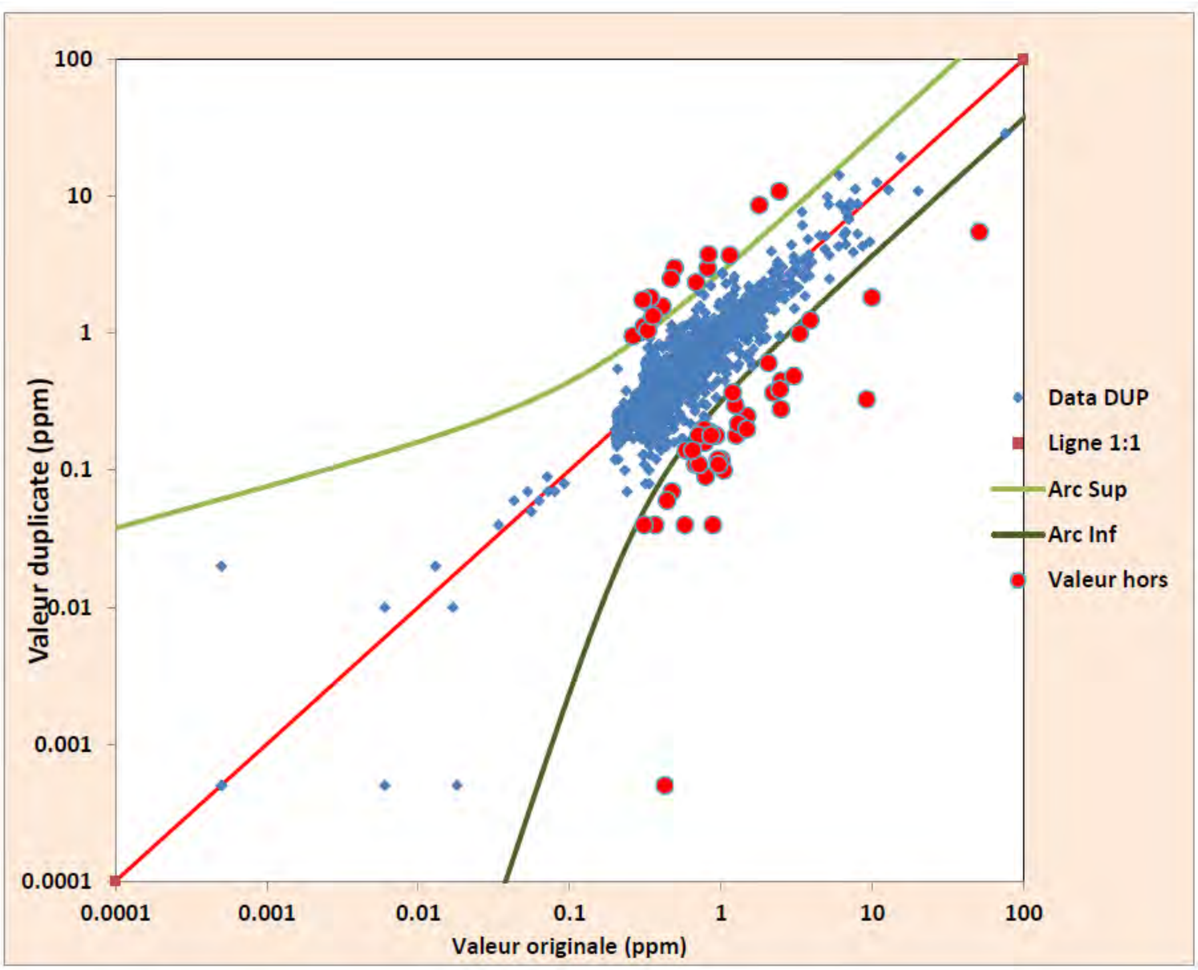
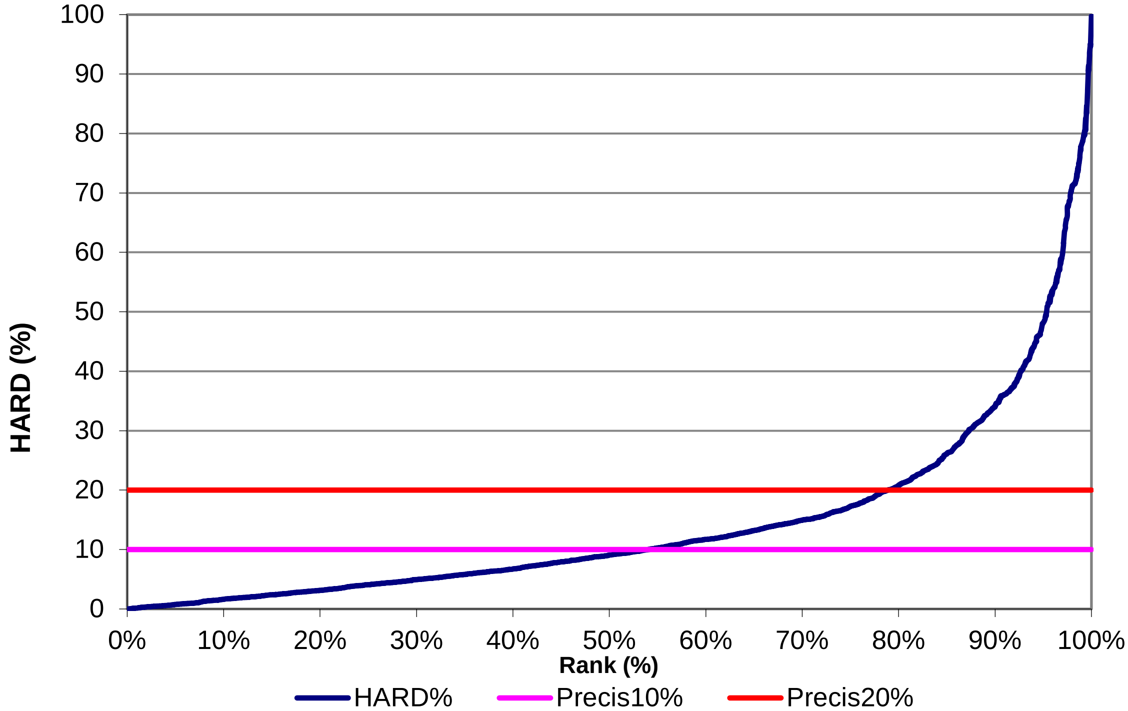


FIGURE 11-24 HARD% PLOT OF UMPIRE PULP DUPLICATES - 2017 AND 2018 DRILL PROGRAMS AT GOSSEY



Overall, the umpire pulp assays by SGS Ouagadougou are acceptable for the style of mineralization present at Gossey.

11.2.5.4 CONCLUSIONS

GMSI is satisfied that the QA/QC protocol in place for the 2017 and 2018 drill programs meets or exceeds industry standards, and the results from these studies confirm that the drilling database is of sufficient quality for use in Mineral Resource estimations.

12 DATA VERIFICATION

12.1 EMZ, FALAGOUNTOU WEST, AND WAFAKA DEPOSITS

Different procedures have been put in place to collect information depending on the exploration method used. In general, field collection of data is entered on paper forms at the drill site and is then transcribed into Excel worksheets at the exploration office (one worksheet per hole).

Since 2013, field data has been entered directly into a laptop using Maxwell GeoServices' LogChief geological database software and thereafter synchronized and transferred into the central database. This procedure is also followed for logging core and RC chips at the exploration office.

Data validation is carried out by the project or database geologist after all data entry for the hole has been completed. Another set of data validation (such as invalid from and to, out of range, or invalid type values) is run on the data once it has been imported into DataShed. A separate set of validation steps is followed for the assay data after it is imported into DataShed. All paper copies of logs and assay certificates in PDF and Excel format are archived for future reference.

Prior to any resource estimation work, 20% of the content of the database is validated. Holes are randomly selected, and the following fields are inspected for possible discrepancies: survey, assays, and lithology. Azimuth and dips are investigated for possible errors. The length fields of drill holes in the "Header" tab versus the final survey measurements are verified. A crosscheck of all samples of the selected drill holes is carried out between laboratory certificates and assay values in the GEOVIA GEMS database to make sure that all gold assay intervals match the laboratory certificates. Investigations are carried out on the lithological information as well.

The responsible QP is of the opinion that the database verification procedures for Essakane S.A. comply with industry standards and are adequate for the purposes of Mineral Resource estimation.

12.2 GOSSEY DEPOSIT

12.2.1 DATABASE

Drill hole information for the 2018 drill program at Gossey was provided to GMSI from the Essakane S.A. geologists (Resource Development Team) on May 10, 2018. The drilling database was provided as a single Microsoft Excel spreadsheet containing the various downhole tables (Collar, Survey, Assays, Lithology, Density, Hardness, and Alteration). GMSI imported the files into the original MS Access database used in the interim resource estimate using GEOVIA GEMS software. The drill hole information was then imported into the GEOVIA GEMS database.

A total of 1,106 drill holes were available for grade estimation. The database was reviewed and corrected, if necessary, prior to final formatting for resource evaluation. The following activities were performed during database validation:

- Validate total hole lengths and final sample depth data
- Verify for overlapping and missing intervals
- Check drill hole survey data for out of range or suspect down-hole deviations
- Visual check of spatial distribution of drill holes
- Validate lithology codes

A new weathering interpretation was provided to GMSI that standardizes the logging practices between the Resource Development team and the Exploration team. This ensures that hardness is recorded consistently and that laterite, saprolite, transition, and fresh rock is clearly interpreted between sections.

12.2.2 GMSI DATA VERIFICATION

GMSI was provided the assay certificates (both PDF and csv versions) for all Essakane S.A. drilling at Gossey since 2011. Forty-nine analysis certificates were compared with the drill database to ensure that assay data was appropriately imported into the database. A list of assay certificates that were checked is provided in Table 12-1, and spans drilling from 2011 up to 2018.

TABLE 12-1 LIST OF GOSSEY ASSAY CERTIFICATES - 2011 TO 2018

Assay-11-0582.CSV	Assay-11-0693.CSV
Assay-11-0719.CSV	Assay-11-0785.CSV
Assay-12-0631.CSV	Assay-12-0807.CSV
Assay-12-0878.CSV	Assay-12-0978.CSV
Assay-12-1060.CSV	Assay-12-1127.CSV
Assay-12-1194.CSV	Assay-12-1276.CSV
Assay-12-1416.CSV	Assay-12-1488.CSV
Assay-12-1621.CSV	Assay-12-1755.CSV
Assay-12-2064 _120705.CSV	Assay-12-2190 _120727.CSV
Assay-13-0044.CSV	Assay-13-0141 _133020.CSV
Assay-13-0220 _133033.CSV	Assay-13-0254.CSV
Assay-13-0390 _136016.CSV	Assay-13-0667 _133141.CSV
Assay-13-0748 _133165.CSV	Assay-13-0813 _133182.CSV
Assay-13-0993 _133232.CSV	Assay-13-1068 _133253.CSV
Assay-13-1123 _133267.CSV	Assay-13-1182.CSV
Assay-13-1211 _133301.CSV	Assay-17-0795 _MEX2410.CSV
Assay-17-0875 _MEX2425.CSV	Assay-17-0913 _G-MEX2434.CSV
Assay-17-0950 _G-MEX2447.CSV	Assay-17-1009 _G-MEX2457.CSV
Assay-17-1023 _G-MEX 2467.CSV	Assay-17-1049 _G MEX2477.CSV
Assay-17-1088 _GMEX2487.CSV	Assay-17-1115 _MEX2499.CSV
Assay-17-1153 _G-MEX2511.CSV	Assay-18-0251 _MEX2679.CSV
Assay-18-0300 _MEX2694.CSV	Assay-18-0342 _G-MEX2709.CSV
Assay-18-0376 _G-MEX2724.CSV	Assay-18-0413 _G-MEX2740.CSV
Assay-18-0434 _G-MEX2750.CSV	Assay-18-0463 _G-MEX2768.CSV

All assay results from the checked certificates agree with the stored database used for resource estimation.

12.2.3 DRILL HOLE COLLAR LOCATION

GMSI personnel visited numerous drill collars from the 2018 drill campaign during the site visit between March 27 and March 31, 2018. In addition, GMSI personnel reviewed the artisanal workings and the ongoing drilling to validate mineralization was present, and to review the drilling and sampling procedures.

Drill collars were identified by a concrete base with the name of the drill hole engraved onto it along with the end-of-hole depth. An example is shown in Figure 12-1.

FIGURE 12-1 GOSSEY DRILL HOLE COLLAR EXAMPLE – MGRC0267

12.2.4 QA/QC VALIDATION

GMSI reviewed the results of the QA/QC from the 2017 and 2018 drill campaigns (as discussed in Section 11) and found them to be within acceptable limits.

12.2.5 CONCLUSIONS

Overall, GMSI is comfortable with the data, analyses, QA/QC, and geological interpretation presented by IAMGOLD's Essakane S.A. geologist. It was performed in a professional manner using industry best practices. GMSI believes that the data is reliable for use in the estimation of Mineral Resources for the Gossey deposit presented in this Technical Report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Test work programs have been carried out on Essakane's major ore types by numerous international metallurgical laboratories since 1990. Test work undertaken between 1990 and 2007 was reported in a DFS document by GRD Minproc (Pty) Ltd. (2007). This was reviewed, summarized, and reported in the 2008 UFS document by GMSI, (2009). Since then, test work has been performed to refine the process design parameters for the 2014 plant expansion and to assess the amenability of Falagountou ore to the Essakane extraction method.

No metallurgical test work has been carried out on the Gossey deposit.

13.1 METALLURGICAL TEST WORK FROM 1990 TO 2011

It was determined, in the early stages of development of Essakane, that heap leaching would not be economically viable mainly due to quantities of cement required for agglomeration of the clay rich saprolite. Therefore, it was decided that a conventional crushing, milling, gravity concentration, and CIL gold plant was the optimal choice for Essakane.

Comminution test results on fresh ore dictated the design of the grinding circuit. Comminution parameters determined from the test work and used for design purposes are summarized in Table 13-1.

TABLE 13-1 COMMINUTION PARAMETER SUMMARY

Sample #	Ai	Cwi	Rwi	A*b	UCS	Lab	Rock Type	Sample Designation
1	0.046	19.7	16.9		203	SGS Jo	Fresh Argillite	EMZ8 Bbwiat106µm
2						SGS Jo	Fresh Argillite	EMZ8 Bbwiat150µm
3	0.322	23.4	16.3		205	SGS Jo	Fresh Argillite	EMZ9 Bbwiat106µm
4						SGS Jo	Fresh Argillite	EMZ9 Bbwiat150µm
5						SGS Jo	Oxide Saprolite	EMZ10 Bbwiat150µm
6	0.197	3.8			47	Philips	Saprock	3096 EDD 0140B (78-87 m)
7			8.8	62.3		SGS	Saprock	3096 EDD 0140B (78-87 m)
8						Philips	Saprolite/Arenite	3096 EDD 0154 (39-47 m)
9	0.036					Philips	Saprolite/Arenite	3096 EDD 0154 (65-72 m)
10	0.260	6.7			62	Philips	Fresh Arenite	3096 EDD 0154 (129-137 m)
11			13.7	34.7		SGS	Fresh Arenite	3096 EDD 0154 (129-137 m)
12	0.109	7.1			23	Philips	Fresh Argillite	3096 EDD 0154 (201-208 m)
13			17.5	29.3		SGS	Fresh Argillite	3096 EDD 0154 (201-208 m)
14	0.202	4.7				Philips	Fresh Arenite	3096 EDD 1671D (110-118 m)
15			11.9	48.1		SGS	Fresh Arenite	3096 EDD 1671D (110-118 m)
16	0.158	6.3			90	Philips	Fresh Argillite	3096 EDD 1671D (145-153 m)
17			16.0	31.3		SGS	Fresh Argillite	3096 EDD 1671D (145-153 m)

	Ore Types	Ai	Cwi	Rwi	A*b	UCS	From Samples	
1	oxide	0.036	10.0	8.8	80.0	47	5, 8-9	Oxide design (above 78 m)
2	transition	0.200	15.0	10.4	55.2	47	6-7, 14-15	Transition (78-118 m)
3	fresh avg.	0.183	21.6	15.4	35.9	204	1-4, 10-17	Fresh Average Design (110-210 m)
4	fresh hard	0.179	21.6	16.1	31.8	204	1-4, 10-13, 16-17	Fresh Hard Design (129-210 m)

Note:
1. Estimate data not used
2. SGS Jo - SGS Johannesburg

Extensive leaching tests were conducted on the various ore types. A common characteristic of Essakane ore is slow leaching kinetics if whole ore is subjected to cyanidation without removing the coarse gold particles in a gravity concentrate. While leaching is still on-going, leach extraction reaches a plateau after 50 hours if coarse gold is present in the ore feed, however, this is reduced to less than 20 hours if gravity gold is removed prior to the leaching stage.

Gravity concentration test work was included in the programs by SGS Johannesburg 2004 and 2005, KCA 2005 and 2006, and McClelland Laboratories 2006 and 2007. Gold recovered in the rougher concentrate varied from 40% to 90%, which is relatively high for gold deposits.

Gravity concentration was considered necessary for the Essakane CIL plant, even though this would place an additional burden on security. This rationale was based on the following factors:

- Due to the high nugget nature of coarse gold, gravity concentration would assist in reducing gold lock-up in the mills.
- Early removal of free gold particles would reduce the tendency for the particles to be flattened in the mill and to have impurities hammered into the gold surface with continued circulation via cyclone underflow.
- Coatings, which might inhibit cyanidation, can develop on gold particles undergoing prolonged recirculation in milling circuits.
- Larger gold particles, if not removed before entering the CIL circuit, may not have sufficient residence time to dissolve completely, thereby reducing overall recovery.
- The lower head grade in the CIL feed would reduce final solution losses.
- The ability to intensively cyanide leach certain gold-bearing heavier minerals such as pyrite or arsenopyrite can potentially increase overall gold recovery.

Optimization studies focusing on grind size and recovery versus operating costs concluded that the economical optimum grind size for fresh rock was P_{80} (80% passing) minus 125 µm. The presence of activated carbon during leaching showed improved leaching kinetics and recoveries. This observation led to the use of a CIL circuit as opposed to a Leach-Carbon in Pulp (CIP) circuit.

13.2 RECENT METALLURGICAL TEST WORK

As part of the plant expansion, additional metallurgical test work and ore characterization was carried out at SGS Lakefield Research Ltd (SGS Lakefield) during 2011. Comminution test work was done on fresh PQ drill core samples. The samples were found to be harder than those used for the initial plant design. Several gravity tests were conducted on the ore and confirmed a predicted gravity gold recovery of 45%. Leach tests were completed on the gravity tails and the run of mine ore. The results showed that a combined (gravity and leach) recovery of 92% should be expected with a 36 hour leach time. The estimated reagent consumptions are 0.4 kg/t for cyanide and 0.6 kg/t for lime after a planned leach time of 36 hours. Static settling tests included flocculant screening, feed percent solids optimization, and flocculant dosage optimization. A non-ionic flocculant was determined to be best suited for this operation with a feed dilution between 10% to 15% and a dosage rate of 40 g/t. Rheology test work was done on simulated underflow samples. All of the samples demonstrated Bingham plastic rheology behaviors. The samples at a higher pH gave similar or slightly higher shear stress values.

Metallurgical testing on drill core and samples from the Essakane CIL circuit was carried out by SGS Lakefield in June 2015 to further characterize the Essakane deposit, with an emphasis on fresh rock behavior. The metallurgical tests included gravity separation, CIL tests, preg-robbing validation tests, whole ore leach tests, intensive leach tests, and diagnostic leach tests, as well as investigations into the effects of grind size and the effects of surfactants on preg-robbing. The test program concluded that:

- The gravity component of the CIL plant is essential to maximize gold recovery and optimize the operation of the downstream CIL circuit. An average gravity recovery of 60% was achieved at laboratory level. This is similar to the average value of 59% obtained in previous studies performed by SGS Lakefield.
- The addition of carbon to the CIL circuit is needed to minimize the effects of preg-robbing carbonaceous material.
- Gold extraction increases with grind fineness, however, with the increased grind fineness, more carbonaceous preg robbing material is liberated and can prevent any observable increase in recovery.
- The use of surfactants or blinding agents at the supplier's recommended dosage did not improve gold recovery.
- Diagnostic leaching of CIL tails showed that only 10% of gold in the tailings is free milling, with the remainder being locked up in dolomite and labile sulphides or associated with sulphides, graphite, and silicates.

The June 2015 SGS Lakefield study indicated a risk for a lower recovery related to the amount of graphitic ore present in future mining zones, according to the life of mine (LOM). Essakane S.A. has initiated studies on the following initiatives to mitigate this issue:

- Oxygen addition to CIL: will reduce the preg-robbing effects of the ore, with a potential to decrease cyanide consumption, to increase recovery, and to increase leaching kinetics.
- Intensive Leach Process to treat gravity concentrate: will increase gold recovery from current shaking table.
- Optimization of the carbon profile in the CIL: will lead to better management of the gold inventory in the CIL and prevent preg-robbing.

Metallurgical testing on representative samples from the Falagountou deposit was completed in May 2014 by SGS Lakefield. The metallurgical tests included assaying, mineralogy, gravity separation, and CIL test work. The test program concluded that:

- Graphite content was low in all samples, as most carbon was associated with carbonate material.

- Sulphur grade was low in the saprolite and transition samples, and slightly higher in the fresh rock samples.
- The fresh rock samples were categorized as soft based on the Bond ball mill work index (BWI) and had excellent recoveries when treated in a gravity separation CIL circuit.

13.3 GEOMETALLURGY PROGRAM

The presence of sulphides (pyrite, arsenopyrite, and pyrrhotite) and graphite at a moderate to high intensity, combined with variable hardness of the rock as the pit exploitation evolves, impacts the plant performance, and makes planning forecasts challenging in terms of metal recovery and cost.

To reduce the impacts associated with the ore variability, a geometallurgical project was launched in 2016 to enhance ore management through a better understanding of the geology. All of the information will be incorporated in a geometallurgical block model by interpolation of different parameters in relation to the gold recovery in the CIL plant.

The geometallurgy program is constantly evolving and two new carbon and sulphur analyzers were purchased and installed in the assay laboratory and are used to analyze mill tails samples. Onsite testing of plant and grade control samples for graphitic carbon (Cg) and sulphur analysis are now carried out on a regular basis in the assay laboratory. Good correlations are observed between graphitic content and plant residues hence allowing for better control.

13.3.1 FUTURE WORK

The future geometallurgy program work includes:

- Continuing to populate and improve the block model with available information and correlations based on the previous work.
- Integrating new parameters in mine planning, if required. Monthly reconciliation will be used to improve the model.

Further development work is being planned for 2020. This includes additional drilling and laboratory testing to allow for better graphitic solid definition.

13.4 CARBON-IN-LEACH GOLD RECOVERIES

The average CIL gold recoveries used per rock type from the Falagountou and Essakane pits are summarized in Table 13-2, and are based on SGS Lakefield test work, as described previously.

TABLE 13-2 CIL GOLD RECOVERIES PER ROCK TYPE

Rock Type	Essakane Pit Recovery (%)	Falagountou Pit Recovery (%)
Saprolite	95.0	95.5
Transition	92.8	93.5
Hard Rock	91.9	92.0

13.5 HEAP LEACH METALLURGICAL TESTING

13.5.1 HEAP LEACH METALLURGICAL SUMMARY

Laboratory testing was conducted in three separate phases by KCA. The details of the first two phases are discussed in the “Technical Report on the Essakane Gold Mine Heap Leach Pre-Feasibility Study, Sahel Region, Burkina Faso,” dated, June 5, 2018.

The first phase of test work completed by KCA included head analysis, coarse bottle roll leach tests, percolation test work, compacted permeability tests, and column leach tests on two bulk grab samples from the EMZ deposit.

Based on the results from the first round of metallurgical testing at KCA, a second program was put together to have sufficient testing to be representative of the Argillite and Arenite rock types expected to be sent to the heap from the EMZ deposit. The second phase of test work completed by KCA included head analysis, bottle roll leach tests, comminution testing, HPGR testing, meteoric water motility testing, percolation test work, compacted permeability test work, and column leach tests on composites from core samples taken from 27 metallurgical drill holes.

A third program was put together to improve upon the operational representativity and add testing on the turbidite rock type. The third program tested material crushed to represent open and closed circuit HPGR products and extended the leach time on the column tests.

The third phase of test work completed by KCA included bottle roll leach tests, HPGR testing, compacted permeability test work, and column leach tests on selected composites from the second program, based on available material, and a turbidite composite.

13.5.2 KCA NOVEMBER 2017

This work was done on two bulk grab samples collected from the bottom of the EMZ pit, which should be representative of the deposit. The received material comprised two individual samples identified as PT6: Arenite and PT16: Argillite Rock/Graphite Faible.

Sample preparation was conducted to provide material for head analyses, head screen analyses, column leach test work, preliminary agglomeration test work, and compacted permeability test work.

Column leach tests were also conducted by KCA on the two samples from Essakane. The column leach tests were conducted at two different crush sizes to confirm the effects of crush size on recovery that were observed in the coarse bottle roll leach tests. The column tests confirmed the trend of higher recovery with finer crush size.

The column leach test results are presented in Table 13-3.

TABLE 13-3 COLUMN LEACH TEST RESULTS

KCA Sample No.	KCA Test No.	Description	Crush Size (mm)	Calculated Head (g/t Au)	Extracted (g/t Au)	Weighted Avg. Tail Screen (g/t Au)	Au Extracted (%)	Calculated Tail P_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Hydrated Lime (kg/t)
77401 H	77419	PT6: Arenite	19	0.678	0.399	0.279	60%	15.8	69	1.00	0.76
77401 I	77422	PT6: Arenite	8.0	0.620	0.429	0.191	71%	6.3	69	1.18	0.76
77402 H	77425	PT16 : Argillite Rock/Graphite Faible	19	0.822	0.493	0.329	62%	15.8	69	0.82	0.50
77402 I	77428	PT16 : Argillite Rock/Graphite Faible	8.0	0.648	0.436	0.212	67%	6.3	69	0.86	0.50
		Average	19	0.750	0.446	0.304	61%	15.8	69	0.91	0.63
		Average	8.0	0.634	0.433	0.202	69%	6.3	69	1.02	0.63

KCA Sample No.	KCA Test No.	Description	Crush Size (mm)	Calculated Head (g/t Ag)	Extracted (g/t Ag)	Weighted Avg. Tail Screen (g/t Ag)	Ag Extracted (%)	Calculated Tail P_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Hydrated Lime (kg/t)
77401 H	77419	PT6: Arenite	19	0.52	0.12	0.40	23%	15.8	69	1.00	0.76
77401 I	77422	PT6: Arenite	8.0	0.54	0.14	0.40	26%	6.3	69	1.18	0.76
77402 H	77425	PT16: Argillite Rock/Graphite Faible	19	0.52	0.09	0.43	18%	15.8	69	0.82	0.50
77402 I	77428	PT16: Argillite Rock/Graphite Faible	8.0	0.54	0.15	0.39	28%	6.3	69	0.86	0.50
		Average	19	0.52	0.11	0.42	21%	15.8	69	0.91	0.63
		Average	8.0	0.54	0.15	0.40	27%	6.3	69	1.02	0.63

13.5.3 KCA MAY 2018

Based on the results of the first round of test work, it was decided to complete a metallurgical testing program that would be representative of the resource to be mined for the heap leach. The trend of higher recovery with finer crush led to the decision to run column tests to compare fine conventional crushing with HPGR crushing.

The remaining material from the two samples utilized in the KCA testing program from November 2017 was also utilized for the next phase with HPGR test work along with additional material identified as PT6: Arenite and PT16: Argillite Rock/Graphite Faible. Core material from 27 drill holes, representative of the EMZ pit mineralization from the Project, were utilized for the test program. Portions of the received core material were selected and combined into 11 composite samples.

Sample preparation was conducted to provide material for head analyses, head screen analyses with assays by size fraction, bottle roll leach test work, column leach test work, preliminary agglomeration test work, and compacted permeability test work.

In addition to the above-mentioned test work, 54 samples were selected from these domains, prepared, and utilized for variability bottle roll leach test work.

Additionally, material was prepared and submitted to the University of British Columbia (UBC) for quantitative x-ray diffraction analyses (QXRD), Hazen Research, Inc. in Golden, Colorado for a Bond Crusher Impact (CWi) and Bond Abrasion work Index (Ai), SGS Lakefield for graphitic carbon, WETLABS for acid base accounting, and ThyssenKrupp Industrial Solutions in Germany for small-scale HPGR (ATWAL) abrasion testing.

The composite samples are generally in the correct range for the expected grades for the heap leach though a few composites (Arenite West 223, Upper Arenite East 243, Middle Arenite East 243, and Argillite East 343 Refractory) are below the cut-off grade for the heap. The silver assays are all near the detection limit for silver. The low grade and the nugget effect from the presence of coarse gold gives a high variance between the head assays.

The head material was assayed by quantitative methods for carbon, sulphur, and mercury. The mercury content ranged from 0.01 mg/kg to 0.40 mg/kg, with an average of 0.10 mg/kg.

The mercury content of the heap leach material should be similar to what has been treated in the CIL plant and the requirement for environmental controls should be the same as the existing CIL plant. The graphitic carbon assays range from below detection limit to 0.26%. The graphitic carbon is a known indicator for preg-robbing at the existing CIL plant.

In addition to the analyses on pulverized head material, single pass, and locked cycle HPGR crushed samples were utilized for head screen analyses. Portions of conventionally crushed and HPGR crushed material were utilized for head screen analyses with assays by size fraction.

13.5.3.1 HPGR CRUSHER TEST WORK

Material was crushed by KCA through a PILOTWAL (a pilot scale HPGR unit) to produce HPGR product for laboratory test work. A portion of the material was sent to ThyssenKrupp Industrial Solutions for ATWAL abrasion testing.

The average ATWAL specific abrasion for the four tests conducted was 20.1 g/t, considered moderately abrasive. The results of the abrasion testing are presented in Table 13-4.

TABLE 13-4 ATWAL ABRASION TEST RESULTS

Sample No.	Description	Moisture (%)	Feed Size (mm)	Specific Force (N/mm^2)	Wear Rate (g/t)
77433 A	PT6: Arenite	1	< 3.15	4.0	22.31
77433 A	PT6: Arenite	3	< 3.15	4.0	27.06
77435 A	PT16: Argillite Rock/Graphite Faible	1	< 3.15	4.0	14.87
77435 A	PT16: Argillite Rock/Graphite Faible	3	< 3.15	4.0	16.19

Typical ranges of wear rates on the ATWAL for other tested ores are given below:

Specific Wear Rates	Abrasiveness
> 40 g/t	high
30 to 40 g/t	moderate/high
20 to 30 g/t	moderate
10 to 20 g/t	low/moderate
< 10 g/t	low

The tested ore abrasiveness was classified as “moderate” for PT6: Arenite and “low/moderate” for PT16: Argillite Rock/Graphite Faible.

13.5.3.2 COMMINUTION TEST WORK

A portion of the head material from samples Arenite West 223 Core Material, Upper Arenite East 243, Middle Arenite East 243, Lower Arenite East 243, Argillite West 313, Argillite East 343, Argillite East 343 Refractory, Arenite West 413, Arenite East 443, Arenite 443 and 413 Refractory and Argillite 513, 543, 623, and 643 were submitted to Hazen Research, Inc in Golden, Colorado for comminution testing. Test work was completed to provide Bond Crusher Impact index (CWi) and a Bond Abrasion Work Index (A_i) for the composite samples.

The results are summarized in Table 13-5.

TABLE 13-5 SUMMARY OF COMMINUTION TESTING

KCA Sample No.	Description	A_i (g)	CW_i (kWh/t)
79136 A	Arenite West 223	0.5974	13.9
79137 A	Upper Arenite East 243	0.4801	15.1
79138 A	Middle Arenite East 243	0.5209	16.9
79139 A	Lower Arenite East 243	0.3888	19.5
79140 A	Argillite West 313	0.0786	20.1
79141 A	Argillite East 343	0.2103	18.3
79143 A	Arenite West 413	0.5315	19.3
79144 A	Arenite East 443	0.5236	18.0
79145 A	Arenite 443 and 413 Refractory	0.4139	18.4
79146 A	Argillite 513, 543, 623, 643	0.4414	13.6

The average A_i is 0.4187 g and the material would be considered abrasive.

The average CWi is 17.31 kWh/t for the material tested, which would be classified as hard to very hard.

13.5.3.3 PREG-ROBBING CYANIDE SHAKE TESTS

Preg-robbing cyanide shake tests were conducted utilizing portions of the pulverized head material from each composite. These tests provided preliminary indications of cyanide soluble gold loss from pregnant solutions.

The preg-robbing value can be positive or negative. A positive value indicates a level of preg-robbing while a negative value indicates that more gold was extracted from the sample with the addition of the gold spike than was extracted without a gold spike in place.

As a guide a preg-robbing value of less than 10% would be generally considered as non preg-robbing and a value of greater than 10% would be considered preg-robbing. Moderate preg-robbing would be from >10% to 20% and highly preg-robbing would be >20%. The graphitic carbon assay results for each sample are presented for comparison purposes and the results of individual preg-robbing tests are presented in Table 13-6.

TABLE 13-6 COMPOSITE PREG-ROBBING TEST RESULTS

					Leach Results				Preg Robbing Results			
KCA Sample No.	Description	Split	SGS Assay, Graphitic Carbon, (%)	Head Assay (g/t Au)	Final pH	Au (mg/L)	Extraction (g/t Au)	Est. Ext. Au (%)	Spike Au (mg/L)	Direct Au (mg/L)	Spiked Leach Au (mg/L)	Preg-robbing (%)
79136 B	Arenite West 223	A	<0.05	0.089	10.5	0.03	0.060	67%	1.00	0.03	1.01	2%
79136 B	Arenite West 223	B	<0.05	0.087	10.5	0.03	0.060	69%	1.00	0.03	1.01	2%
79137 C	Upper Arenite East 243	A	<0.05	0.195	10.5	0.05	0.100	51%	1.00	0.05	1.00	5%
79137 C	Upper Arenite East 243	B	<0.05	0.199	10.5	0.06	0.120	60%	1.00	0.06	1.01	5%
79138 C	Middle Arenite East 243	A	<0.05	0.178	10.4	0.04	0.080	45%	1.00	0.04	0.98	6%
79138 C	Middle Arenite East 243	B	<0.05	0.171	10.4	0.03	0.060	35%	1.00	0.03	0.98	5%
79139 C	Lower Arenite East 243	A	0.07	0.449	10.5	0.09	0.180	40%	1.00	0.09	1.01	8%
79139 C	Lower Arenite East 243	B	0.07	0.429	10.4	0.08	0.160	37%	1.00	0.08	1.00	8%
79140 C	Argillite West 313	A	0.26	0.309	10.4	0.06	0.120	39%	1.00	0.06	0.89	17%
79140 C	Argillite West 313	B	0.26	0.298	10.4	0.05	0.100	34%	1.00	0.05	0.88	17%
79141 C	Argillite East 343	A	0.18	0.552	10.4	0.06	0.120	22%	1.00	0.06	1.00	6%
79141 C	Argillite East 343	B	0.18	0.573	10.4	0.07	0.140	24%	1.00	0.07	1.00	7%
79142 A	Argillite East 343 Refractory	A	0.13	0.034	10.5	0.01	0.020	58%	1.00	0.01	0.87	14%
79142 A	Argillite East 343 Refractory	B	0.13	0.034	10.3	0.01	0.020	58%	1.00	0.01	0.85	16%
79143 B	Arenite West 413	A	0.19	0.552	10.5	0.09	0.180	33%	1.00	0.09	0.98	11%
79143 B	Arenite West 413	B	0.19	0.538	10.5	0.08	0.160	30%	1.00	0.08	0.97	11%
79144 C	Arenite East 443	A	0.16	0.456	10.3	0.22	0.440	96%	1.00	0.22	1.15	7%
79144 C	Arenite East 443	B	0.16	0.480	10.5	0.23	0.460	96%	1.00	0.23	1.15	8%
79145 B	Arenite 443 & 413 Refractory	A	0.14	0.346	10.4	0.06	0.120	35%	1.00	0.06	0.97	9%
79145 B	Arenite 443 & 413 Refractory	B	0.14	0.333	10.4	0.05	0.100	30%	1.00	0.05	0.96	9%
79146 B	Argillite 513, 543, 623, 643	A	0.12	0.312	10.3	0.08	0.160	51%	1.00	0.08	1.01	7%
79146 B	Argillite 513, 543, 623, 643	B	0.12	0.302	10.5	0.07	0.140	46%	1.00	0.07	1.01	6%

Three of the samples would be considered moderately preg-robbing while the rest would be considered not preg-robbing. The highest preg-robbing occurred with the highest graphitic carbon assay, however, there is not a direct correlation. Caution will need to be taken to minimize the amount of preg-robbing material stacked on the heap.

13.5.3.4 ACID-BASE ACCOUNTING

Acid-base accounting is a static test to determine the acid producing or acid neutralizing potential of a material. It is a general analysis for the elements of acid generation and does not indicate the potential rate at which generation or neutralization may occur.

It is generally accepted that a net neutralization potential (NNP) value greater than 20 is indicative of a non-acid producing material (acid neutralizing material), and that an NNP value less than -20 is an acid generating material.

Based upon the test work utilizing head composite material from Arenite West 223 Core Material, Upper Arenite East 243, Middle Arenite East 243, Lower Arenite East 243, Argillite West 313, Argillite East 343, Argillite East 343 Refractory, Arenite West 413, Arenite East 443, Arenite 443, Arenite 413 Refractory, and Argillite 513, 543, 623, 643 (KCA Test Nos. 79136 B, 79137 C, 79138 C, 79139 C, 79140 C, 79141 C, 79142 A, 79143 C, 79144 C, 79145 B, and 79146 B) showed a NNP value greater than 20, therefore indicating a low potential for acid producing potential for each sample.

13.5.3.5 BOTTLE ROLL TEST WORK

Standard, LeachWell, and variability bottle roll leach testing was conducted on a portion of the material.

For the standard bottle roll tests, gold extractions ranged from 15% to 93% (average of 78%) based on calculated heads which ranged from 0.107 g/t Au to 1.188 g/t Au. The sodium cyanide consumption ranged from 0.79 kg/t to 1.10 kg/t. The material utilized in leaching was blended with 1.25 kg/t to 1.75 kg/t hydrated lime.

The standard variability bottle roll tests compared to the standard composite tests are shown in Table 13-7. The variability bottle roll tests achieved lower gold recoveries than the composite samples by an average of 14.5%. Many of the variability bottle roll tests have head assays that do not compare well with the calculated head due to the low grades and nugget effect from coarse gold. The variability in the results may be due to the low grade and presence of coarse gold.

TABLE 13-7 VARIABILITY AND COMPOSITE BOTTLE ROLL COMPARISON

	Variability Bottle Roll Test				Composite Bottle Roll Test	
Composite Domain	Average Graphitic Carbon (%)	Average Preg-Robbing (%)	Average Calc. Head (g/t Au)	Average Au Extraction (%)	Average Calc. Head (g/t Au)	Average Au Extraction (%)
Arenite West 223	<0.05	4.250	0.262	74%	0.107	90%
Upper Arenite East 243	<0.05	3.600	0.152	66%	0.185	93%
Middle Arenite East 243	0.051	7.400	0.275	84%	0.307	87%
Lower Arenite East 243	<0.05	4.600	0.314	81%	0.577	90%
Argillite West 313	0.252	17.800	1.291	64%	0.208	45%
Argillite East 343	0.249	17.875	0.248	52%	1.188	89%
Arenite West 413	0.425	26.500	1.166	53%	0.305	80%
Arenite East 443	0.090	10.000	0.177	83%	0.409	93%
Arenite 443 & 413 Refractory	0.170	11.250	1.183	68%	0.810	90%
Argillite 513, 543, 623, 643	<0.05	6.250	0.358	76%	0.376	89%

13.5.3.6 COLUMN LEACH TEST WORK

The PT6: Arenite and PT16: Argillite Rock/Graphite Faible samples were at KCA from the first round of column testing and new column tests were conducted to check the effects of crushing finer and crushing with an HPGR. Each sample had a column with the following parameters:

- Conventionally crushed to P_{100} 4.75 mm and agglomerated with cement
- HPGR crushed to P_{100} 22.4 mm and agglomerated with cement

The rest of the composite samples were received at a later date and a series of column leach tests were conducted to obtain a large data set to compare conventional crushing to HPGR crushing. Each sample had a column with the following parameters:

- Conventionally crushed to P_{100} 8 mm and blended with hydrated lime
- HPGR crushed to P_{100} 19 mm and blended with hydrated lime

The column leach test results are presented in Table 13-8.

TABLE 13-8 SUMMARY OF COLUMN LEACH TEST RESULTS – GOLD

KCA Sample No.	KCA Test No.	Description	Crush Type	Calculated Head (g/t Au)	Extracted (% Au)	Calculated Tail P_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Hydrated Lime (kg/t)	Addition Cement (kg/t)
79118 A	79129	PT6: Arenite	Conventional	0.811	86%	--	86	0.82	--	4.02
79107 C	79123	PT6: Arenite	HPGR	0.871	83%	--	86	0.94	--	4.05
79119 A	79132	PT16: Argillite Rock/Graphite Faible	Conventional	0.731	78%	--	86	0.89	--	4.02
79108 C	79126	PT16: Argillite Rock/Graphite Faible	HPGR	0.688	76%	--	86	0.89	--	4.07
--	Average	Essakane	Conventional	0.771	82%	--	86	0.86	--	4.02
--	Average	Essakane	HPGR	0.780	80%	--	86	0.92	--	4.06
79136 B	79187	Arenite West 223	Conventional	0.204	80%	6.35	61	0.37	1.19	0.00
79160 C	80230	Arenite West 223	HPGR	0.147	56%	9.41	61	0.36	1.26	0.00
79137 B	79190	Upper Arenite East 243	Conventional	0.292	61%	6.73	61	0.30	1.26	0.00
79161 C	80233	Upper Arenite East 243	HPGR	0.263	56%	9.52	61	0.37	1.25	0.00
79138 C	80201	Middle Arenite East 243	Conventional	0.275	66%	5.70	61	0.39	1.22	0.00
79162 C	80236	Middle Arenite East 243	HPGR	0.307	72%	9.00	61	0.37	1.25	0.00
79139 C	80204	Lower Arenite East 243	Conventional	0.524	65%	6.25	61	0.28	1.26	0.00
79163 C	80239	Lower Arenite East 243	HPGR	0.761	62%	8.59	61	0.41	1.26	0.00
79140 C	80207	Argillite West 313	Conventional	0.649	50%	6.39	61	0.42	1.25	0.00
79164 C	80242	Argillite West 313	HPGR	0.465	51%	9.59	61	0.41	1.23	0.00
79141 C	80210	Argillite East 343	Conventional	0.687	61%	5.99	61	0.32	1.77	0.00
79165 C	80245	Argillite East 343	HPGR	1.156	47%	8.46	85	0.33	0.00	4.00

KCA Sample No.	KCA Test No.	Description	Crush Type	Calculated Head (g/t Au)	Extracted (% Au)	Calculated Tail P_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Hydrated Lime (kg/t)	Addition Cement (kg/t)
79142 A	80213	Argillite East 343 Refractory	Conventional	0.038	37%	5.79	61	0.34	1.51	0.00
79143 B	80216	Arenite West 413	Conventional	0.541	41%	5.92	61	0.35	1.25	0.00
79166 C	80248	Arenite West 413	HPGR	0.343	54%	9.71	61	0.38	1.25	0.00
79144 C	80219	Arenite East 443	Conventional	0.756	68%	5.96	61	0.32	1.25	0.00
79167 C	80251	Arenite East 443	HPGR	0.752	61%	9.97	85	0.64	1.26	0.00
79145 B	80222	Arenite 443 & 413 Refractory	Conventional	1.002	73%	6.27	61	0.36	1.50	0.00
79168 C	80254	Arenite 443 & 413 Refractory	HPGR	1.258	60%	8.19	85	0.66	1.51	0.00
79146 B	80225	Argillite 513, 543, 623, 643	Conventional	0.495	62%	6.08	61	0.26	1.50	0.00
79169 C	80257	Argillite 513, 543, 623, 643	HPGR	0.646	83%	8.29	62	0.26	1.50	0.00
--	Average	Overall - Composite	Conventional	0.497	60%	6.13	61	0.34	1.36	0.00
--	Average	Overall - Composite	HPGR	0.610	60%	9.07	68	0.42	1.18	0.40
--	Average	Overall - Arenite	Conventional	0.513	65%	6.17	61	0.34	1.28	0.00
--	Average	Overall - Arenite	HPGR	0.547	60%	9.20	68	0.46	1.29	0.00
--	Average	Overall - Argillite	Conventional	0.467	53%	6.06	61	0.34	1.51	0.00
--	Average	Overall - Argillite	HPGR	0.756	60%	8.78	69	0.33	0.91	1.33

Recovery by size fraction was reviewed by comparing the head screen analyses with the tails screen analyses. The finer crush sizes have better recoveries on average, but each size fraction has high variability. These results are summarized in Table 13-9.

A detoxification test was conducted on one of the HPGR column leach tests. The final two litres of barren solution were collected and submitted for Profile I analysis. The column was rinsed with Reno tap water until the weak acid dissociable (WAD) cyanide level in solution leaving the heap was less than 5.0 mg/L (20 days). The final wash solution was also submitted for a Profile I analysis. A portion of the leached material was submitted for Meteoric Water Mobility Testing (MWMT). The final wash solution from the columns exceeded drinking water standards for aluminum, antimony, and arsenic for all columns tested.

TABLE 13-9 HEAD VERSUS TAILS RECOVERY BY SIZE FRACTION

		Gold Recovery by Size Fraction, (%[1])					
KCA Test No	**Description\Passing, mm**	**8.0**	**6.3**	**4.00**	**1.70**	**0.600**	**0.212**
79187	Arenite West 223 - Conv	39%	25%	63%	60%	89%	92%
80230	Arenite West 223 - HPGR	0%	0%	24%	0%	52%	83%
79190	Upper Arenite East 243 - Conv Upper	0%	0%	19%	29%	51%	81%
80233	Arenite East 243 - HPGR Middle	0%	0%	58%	28%	60%	89%
80201	Arenite East 243 - Conv Middle Arenite	0%	0%	13%	55%	70%	88%
80236	East 243 - HPGR Lower Arenite East	37%	34%	41%	0%	73%	83%
80204	243 - Conv Lower Arenite East 243 -	45%	78%	59%	51%	82%	94%
80239	HPGR Argillite West 313 - Conv	21%	0%	75%	30%	69%	92%
80207	Argillite West 313 - HPGR	0%	0%	0%	14%	50%	75%
80242	Argillite East 343 - Conv	0%	0%	81%	35%	33%	81%
80210	Argillite East 343 - HPGR	54%	15%	80%	18%	66%	92%
80245	Argillite East 343 Refractory - Conv	0%	0%	0%	0%	20%	0%
80213	Arenite West 413 - Conv	0%	0%	0%	53%	67%	40%
80216	Arenite West 413 - HPGR	25%	0%	75%	0%	65%	86%
80248	Arenite East 443 - Conv	22%	42%	7%	28%	78%	83%
80219	Arenite East 443 - HPGR	0%	0%	0%	53%	65%	88%
80251	Arenite 443 & 413 Refractory - Conv	0%	0%	0%	0%	54%	59%
80222	Arenite 443 & 413 Refractory - HPGR	0%	89%	0%	67%	61%	84%
80254	Argillite 513, 543, 623, 643 - Conv	0%	0%	0%	0%	0%	35%
80225	Argillite 513, 543, 623, 643 - HPGR	10%	44%	0%	62%	70%	79%
80257		0%	24%	38%	31%	55%	90%
	Arenite Conv. Avg.	16%	27%	33%	45%	69%	87%
	Argillite Conv. Avg.	16%	15%	20%	37%	63%	71%
	Conv. Avg.	16%	23%	28%	42%	67%	82%
	Arenite HPGR Avg.	11%	11%	29%	12%	55%	75%
	Argillite HPGR Avg.	0%	8%	40%	22%	36%	57%
	HPGR Avg.	8%	10%	32%	15%	49%	70%
	Arenite Avg.	13%	19%	31%	29%	62%	81%
	Argillite Avg. Overall	9%	12%	28%	30%	52%	65%
	Avg.	12%	17%	30%	29%	59%	76%

Note:
1. Any negative recoveries were replaced with 0%

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Based on the results of the previous two rounds of test work it was decided to complete a metallurgical testing program that would be more representative of HPGR product achieved in the field and the resource to be mined for the heap leach.

The trend of higher recovery with finer crush led to the decision to run column tests to compare open circuit HPGR crushing with a closed circuit recirculating the HPGR edge material. The previous round of HPGR column leach tests utilized material that was run through the PILOTWAL, which produces a product that is comprised of approximately 23% edge material and 79% centre material. The material near the edge of the rolls in the HPGR is able to expand out the side of the roller and is therefore a coarser product than the material that is crushed near the centre of the rolls. A production unit has larger rolls than the PILOTWAL, so a smaller portion of edge material is created, estimated at 6% edge material. To represent a single pass through an open circuit production sized HPGR unit, testing was completed on HPGR crushed material that was blended at 6% edge material and 94% centre material. To represent a closed HPGR circuit with edge recycle, testing was also completed on centre crushed only material.

A majority of the column tests completed in the second phase of test work were still leaching when the tests were stopped. To find the ultimate recovery of each material type, the leach tests were run for longer periods in this third phase of testing.

The second phase of testing did not include the turbidite material, which makes up 6.5% of the material to be mined for the heap leach. A turbidite sample was added to the testing for this third phase.

The remaining material from the composite samples utilized in the KCA testing program from May 2018 was utilized for the next phase with HPGR test work. Six portions of previously crushed samples identified as Upper Arenite East 243, Middle Arenite East 243, Lower Arenite East 243, Argillite West 313, Argillite East 343, and Arenite East 443 were utilized for the current crushing program (Single Pass HPGR and Edge Recycle HPGR crushing). The material generated from these two crushing methods were utilized for metallurgical test work.

On July 30, 2018, the KCA laboratory facility in Reno, Nevada received two additional 55 gallon drums of bulk material from Essakane. The received material was from four core holes identified as turbidite. This material was utilized for the current crushing program (Single Pass HPGR and Edge Recycle HPGR crushing). The material generated from these two crushing methods was utilized for metallurgical test work.

The composite samples are summarized in Table 13-10.

TABLE 13-10 COMPOSITE SAMPLES

KCA Sample No.	Composite Sample Name	Composite Sample Short Name	Available Composite Weight (kg)
79137 B	Upper Arenite East 243	UARE243	157
79138 B	Middle Arenite East 243	MARE243	375
79139 B	Lower Arenite East 243	LARE243	169
79140 B	Argillite West 313	ARGW313	91
79141 B	Argillite East 343	ARGE343	175
79144 B	Arenite East 443	AREE443	170
82313 A	Turbidite	-	98

Sample preparation was conducted to provide material for head analyses, head screen analyses with assays by size fraction, bottle roll leach test work, column leach test work, preliminary agglomeration test work, and compacted permeability test work.

13.5.4.1 HEAD ANALYSIS AND LEACHWELL CYANIDE BOTTLE ROLL LEACH TEST WORK

Head analyses for Upper Arenite East 243, Middle Arenite East 243, Lower Arenite East 243, Argillite West 313, Argillite East 343, and Arenite East 443 were previously presented in this Technical Report.

The gold and silver assay results for these composite samples are presented in Table 13-11.

TABLE 13-11 COMPOSITE SAMPLE GOLD AND SILVER HEAD ANALYSIS

KCA Sample No.	Description	ALS Assay (g/t Au)	KCA Average Assay, (g/t Au)	KCA Average Assay, (g/t Ag)
79137 C	Upper Arenite East 243	0.228	0.197	0.41
79138 C	Middle Arenite East 243	0.119	0.175	0.31
79139 C	Lower Arenite East 243	0.383	0.439	0.41
79140 C	Argillite West 313	0.630	0.303	0.41
79141 C	Argillite East 343	0.376	0.562	0.31
79144 C	Arenite East 443	0.647	0.468	0.41

The silver assays are all near the detection limit for silver.

LeachWell cyanide bottle roll leach test work was completed on a portion of both the Single Pass HPGR Crushed and Edge Recycle HPGR crushed material (Upper Arenite East 243, Middle Arenite East 243, Lower Arenite East 243, Argillite West 313, Argillite East 343, Arenite East 443, and Turbidite) utilizing material ring and puck pulverized to a target size of P_{100} 0.150 mm.

The results of the LeachWell bottle roll tests are summarized in Tables 13-12 and 13-13.

TABLE 13-12 LEACHWELL CYANIDE BOTTLE ROLL LEACH TESTS - GOLD EXTRACTION

KCA Sample No.	KCA Test No.	Description	Finish	HPGR Crush Type	Target P_{100} Size (mm)	Head Average (g/t Au)	Average LeachWELL BRT (g/t Au)	Calc. Head (g/t Au)	Extracted (g/t Au)	Average Tails (g/t Au)	Au Extracted (%)	Leach Time (Hours)	Cons NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
81045 A	81057 A	Upper Arenite East 243	AAS ICP DIBK-AAS	Single Pass	0.15	0.197	0.270	0.254 0.261 0.294	0.240 0.247 0.280	0.014 0.014 0.014	95% 95% 95%	12	0.22	5.00
81046 A	81057 B	Upper Arenite East 243	AAS ICP DIBK-AAS	Edge Recycle	0.15	0.197	0.201	0.194 0.194 0.214	0.180 0.180 0.200	0.014 0.014 0.014	93% 93% 94%	12	0.14	5.00
81047 A	81057 C	Middle Arenite East 243	AAS ICP DIBK-AAS	Single Pass	0.15	0.175	0.231	0.222 0.208 0.262	0.200 0.186 0.240	0.022 0.022 0.022	90% 89% 92%	12	0.28	5.00
81048 A	81057 D	Middle Arenite East 243	AAS ICP DIBK-AAS	Edge Recycle	0.15	0.175	0.138	0.134 0.140 0.140	0.120 0.126 0.126	0.014 0.014 0.014	90% 90% 90%	12	0.22	5.00
81049 A	81057 E	Lower Arenite East 243	AAS ICP DIBK-AAS	Single Pass	0.15	0.439	1.264	1.261 1.271 1.261	1.240 1.250 1.240	0.021 0.021 0.021	98% 98% 98%	12	0.36	5.00
81050 A	81057 F	Lower Arenite East 243	AAS ICP DIBK-AAS	Edge Recycle	0.15	0.439	0.399	0.384 0.390 0.424	0.360 0.366 0.400	0.024 0.024 0.024	94% 94% 94%	12	0.28	5.00
81051 A	81058 A	Argillite West 313	AAS ICP DIBK-AAS	Single Pass	0.15	0.303	0.559	0.543 0.531 0.603	0.480 0.468 0.540	0.063 0.063 0.063	88% 88% 89%	12	0.28	5.00
81052 A	81058 B	Argillite West 313	AAS ICP DIBK-AAS	Edge Recycle	0.15	0.303	0.534	0.505 0.533 0.565	0.460 0.488 0.520	0.045 0.045 0.045	91% 92% 92%	12	0.46	5.00
81053 A	81058 C	Argillite East 343	AAS ICP DIBK-AAS	Single Pass	0.15	0.562	0.758	0.761 0.753 0.761	0.720 0.712 0.720	0.041 0.041 0.041	95% 95% 95%	12	0.40	5.00
81054 A	81058 D	Argillite East 343	AAS ICP DIBK-AAS	Edge Recycle	0.15	0.562	0.330	0.327 0.315 0.347	0.300 0.288 0.320	0.027 0.027 0.027	92% 91% 92%	12	0.34	5.00
81055 A	81058 E	Arenite East 443	AAS ICP DIBK-AAS	Single Pass	0.15	0.468	1.196	1.173 1.181 1.233	1.140 1.148 1.200	0.033 0.033 0.033	97% 97% 97%	12	0.32	5.00
81056 A	81058 F	Arenite East 443	AAS ICP DIBK-AAS	Edge Recycle	0.15	0.468	0.933	0.947 0.904 0.947	0.860 0.817 0.860	0.087 0.087 0.087	91% 90% 91%	12	0.26	5.00
82316 A	82318 A	Turbidite	AAS ICP DIBK-AAS	Single Pass	0.15	--	0.195	0.215 0.195 0.175	0.160 0.140 0.120	0.055 0.055 0.055	74% 72% 69%	12	0.04	5.00
82317 A	82318 B	Turbidite	AAS ICP DIBK-AAS	Edge Recycle	0.15	--	0.242	0.255 0.255 0.215	0.200 0.200 0.160	0.055 0.055 0.055	78% 78% 74%	12	0.36	5.00

IAMGOLD Corporation

TABLE 13-13 LEACHWELL CYANIDE BOTTLE ROLL LEACH TESTS - SILVER EXTRACTION

KCA Sample No.	KCA Test No.	Description	Finish	HPGR Crush Type	Target P_{100} Size (mm)	Head Average (g/t Ag)	Calculated Head (g/t Ag)	Extracted (g/t Ag)	Average Tails (g/t Ag)	Ag Extracted (%)	Leach Time (Hours)	Cons. NaCN (kg/t)	Additional $Ca(OH)_2$ (kg/t)
81045 A	81057 A	Upper Arenite East 243	AAS	Single Pass	0.15	0.41	0.31	0.10	0.21	33%	12	0.22	5.00
81046 A	81057 B	Upper Arenite East 243	AAS	Edge Recycle	0.15	0.41	0.31	0.10	0.21	33%	12	0.14	5.00
81047 A	81057 C	Middle Arenite East 243	AAS	Single Pass	0.15	0.31	0.39	0.08	0.31	21%	12	0.28	5.00
81048 A	81057 D	Middle Arenite East 243	AAS	Edge Recycle	0.15	0.31	0.41	0.10	0.31	24%	12	0.22	5.00
81049 A	81057 E	Lower Arenite East 243	AAS	Single Pass	0.15	0.41	0.37	0.16	0.21	44%	12	0.36	5.00
81050 A	81057 F	Lower Arenite East 243	AAS	Edge Recycle	0.15	0.41	0.31	0.10	0.21	33%	12	0.28	5.00
81051 A	81058 A	Argillite West 313	AAS	Single Pass	0.15	0.41	0.35	0.14	0.21	40%	12	0.28	5.00
81052 A	81058 B	Argillite West 313	AAS	Edge Recycle	0.15	0.41	0.43	0.12	0.31	28%	12	0.46	5.00
81053 A	81058 C	Argillite East 343	AAS	Single Pass	0.15	0.31	0.35	0.14	0.21	40%	12	0.40	5.00
81054 A	81058 D	Argillite East 343	AAS	Edge Recycle	0.15	0.31	0.31	0.10	0.21	33%	12	0.34	5.00
81055 A	81058 E	Arenite East 443	AAS	Single Pass	0.15	0.41	0.37	0.16	0.21	44%	12	0.32	5.00
81056 A	81058 F	Arenite East 443	AAS	Edge Recycle	0.15	0.41	0.24	0.14	0.10	58%	12	0.26	5.00
82316 A	82318 A	Turbidite	AAS	Single Pass	0.15	--	0.31	0.10	0.21	33%	12	0.04	5.00
82317 A	82318 B	Turbidite	AAS	Edge Recycle	0.15	--	0.37	0.16	0.21	44%	12	0.36	5.00

13.5.4.2 HPGR CRUSHER TEST WORK

Material was crushed by KCA through a PILOTWAL to determine the parameters required for the sizing of an industrial HPGR and to produce HPGR product for laboratory test work.

The PILOTWAL test results are presented in Table 13-14.

TABLE 13-14 SUMMARY OF HPGR THROUGHPUT DATA

KCA Sample No.	KCA Test No.	Description	HPGR Crush Type	Moisture Added (%)	Specific Energy Net (dry feed) (kWh/t)	Specific Throughput (dry feed) ($t*s/(m^3*h)$)	Net Power at Shaft (Kw)	Specific Force (N/mm^2)
81045 A	81070	Upper Arenite East 243	Single Pass	3.0	1.9	253.3	14.4	3.4
81046 A	81073	Upper Arenite East 243	Edge Recycle	3.0	1.9	254.1	13.9	3.4
81047 A	81076	Middle Arenite East 243	Single Pass	3.0	1.7	277.8	14.0	3.4
81048 A	81079	Middle Arenite East 243	Edge Recycle	3.0	1.7	277.2	14.0	3.4
81049 A	81082	Lower Arenite East 243	Single Pass	3.0	1.9	258.9	14.8	3.5
81050 A	81085	Lower Arenite East 243	Edge Recycle	3.0	1.8	270.5	14.6	3.5
81051 A	81088	Argillite West 313	Single Pass	3.0	2.3	259.6	17.1	3.7
81052 A	81091	Argillite West 313	Edge Recycle	3.0	2.0	280.8	16.2	3.6
81053 A	82301	Argillite East 343	Single Pass	3.0	2.0	270.8	16.0	3.6
81054 A	82304	Argillite East 343	Edge Recycle	3.0	1.9	281.6	16.0	3.6
81055 A	82307	Arenite East 443	Single Pass	3.0	2.0	265.4	15.3	3.5
81056 A	82310	Arenite East 443	Edge Recycle	3.0	1.8	268.8	14.4	3.4
82316 A	82319	Turbidite	Single Pass	3.0	1.9	251.9	14.0	3.2
82317 A	82322	Turbidite	Edge Recycle	3.0	1.9	252.0	14.1	3.2

The data presented in Table 13-4 shows an average specific throughput of 265 $(t \cdot s)/(m^3 \cdot h)$ and average specific energy of 1.8 kWh/t. The specific throughput does not take into account any recirculating load.

13.5.4.3 AGGLOMERATION AND COMPACTED PERMEABILITY TEST WORK

Preliminary agglomeration test work, as well as compacted permeability test work, was conducted on Argillite and Arenite composites of both the Single Pass HPGR Crushed and Edge Recycle HPGR crushed material.

For the test work, the material was agglomerated with various additions of cement. In the preliminary agglomeration testing, the agglomerated material was placed in a column with no compressive load and then tested for permeability. In the compaction testing, the agglomerated material was compacted in a column with a predetermined static load and then tested for permeability.

Compacted permeability test work was conducted on both the Argillite and Arenite crushed composite samples. Separate test samples were loaded into a column and subjected to loads equivalent to 20, 40, and 60 m of overall heap height (assuming a heap density equivalent to 1.8 t/m^3).

The permeability test work was completed including agglomeration with cement because the previous testing was showing samples begin to fail at the 60 m heap height and the crushed product in this round of test work has more fines.

The compacted permeability tests are summarized in Table 13-15.

TABLE 13-15 SUMMARY OF COMPACTED PERMEABILITY TESTS

KCA Sample No.	Sample Description	HPGR Crush Type	Test Phase	Cement Added (kg/t)	Effective Height (m)	Bulk Density (t/m^3)	Flow Rate (LpHr/m^2)	Flow Result Pass/Fail	Cum. Slump, (% Slump)	Slump Result Pass/Fail	Overall Pass/Fail
81063 A	Arenite Composite 1	Single Pass	Primary	4	20	1.97	2,899	Pass	0%	Pass	Pass
			Stage Load		40	2.06	532	Pass	5%	Pass	Pass
			Stage Load		60	2.13	234	Pass	8%	Pass	Pass
81064 A	Arenite Composite 2	Edge Recycle	Primary	4	20	1.95	3,319	Pass	3%	Pass	Pass
			Stage Load		40	2.05	1,894	Pass	7%	Pass	Pass
			Stage Load		60	2.10	1,130	Pass	10%	Pass	Pass
81065 A	Argillite Composite 1	Single Pass	Primary	4	20	1.97	7,177	Pass	1%	Pass	Pass
			Stage Load		40	2.01	2,434	Pass	3%	Pass	Pass
			Stage Load		60	2.09	835	Pass	7%	Pass	Pass
81066 A	Argillite Composite 2	Edge Recycle	Primary	4	20	1.92	7,506	Pass	2%	Pass	Pass
			Stage Load		40	2.04	793	Pass	7%	Pass	Pass
			Stage Load		60	2.09	410	Pass	9%	Pass	Pass

All of the samples passed at all three heap heights. A heap height of 50 m with 4 kg/t cement addition is recommended for this study.

13.5.4.4 COLUMN LEACH TEST WORK

The previous HPGR test work was on material that was run through the PILOTWAL at KCA. The product from the PILOTWAL contains approximately 23% edge material, which is coarser than a larger production unit. Due to the previously observed sensitivity to crush size, new columns were conducted to better emulate an open circuit production sized HPGR, with approximately 6% edge material, as well as operating an HPGR in closed circuit with edge recycle.

Each sample had a column with the following parameters:

- Single Pass: Single pass HPGR crushed to P_{100} 19 mm, blended 94% HPGR centre product with 6% HPGR edge product and agglomerated with cement;
- Edge Recycle: Single pass HPGR crushed to P_{100} 19 mm, took 100% centre product, and agglomerated with cement.

The column leach test results are presented in Tables 13-16 and 13-17.

TABLE 13-16 SUMMARY OF COLUMN LEACH TEST RESULTS - GOLD

KCA Sample No.	KCA Test No.	Description	HPGR Crush Type	Crush Size (mm)	Calculated Head P_{80} Size (mm)	Calculated Head (g/t Au)	Extracted (g/t Au)	Extracted (% Au)	Calculated Tail P_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Cement (kg/t)
81045 A	81070	Upper Arenite East 243	Single Pass	19	9.03	0.154	0.114	74%	8.71	93	0.47	3.98
81046 A	81073	Upper Arenite East 243	Edge Recycle	19	7.89	0.259	0.216	83%	7.84	93	0.58	4.00
81047 A	81076	Middle Arenite East 243	Single Pass	19	9.30	0.171	0.125	73%	8.20	93	0.58	3.99
81048 A	81079	Middle Arenite East 243	Edge Recycle	19	7.83	0.198	0.146	74%	7.20	93	0.49	3.99
81049 A	81082	Lower Arenite East 243	Single Pass	19	8.11	0.801	0.587	73%	7.83	130	0.75	3.98
81050 A	81085	Lower Arenite East 243	Edge Recycle	19	7.78	1.252	0.902	72%	6.96	130	0.90	3.97
81051 A	81088	Argillite West 313	Single Pass	19	9.70	0.595	0.302	51%	8.26	108	1.37	3.99
81052 A	81091	Argillite West 313	Edge Recycle	19	8.40	0.705	0.367	52%	7.04	108	1.01	4.01
81053 A	82301	Argillite East 343	Single Pass	19	8.17	1.143	0.868	76%	7.96	141	0.77	3.97
81054 A	82304	Argillite East 343	Edge Recycle	19	7.71	0.965	0.727	75%	7.20	141	0.92	3.99
81055 A	82307	Arenite East 443	Single Pass	19	9.19	1.675	1.267	76%	8.07	141	0.85	3.97
81056 A	82310	Arenite East 443	Edge Recycle	19	8.73	1.345	1.046	78%	7.80	141	0.84	3.98
82316 A	82319	Turbidite	Single Pass	19	6.93	0.243	0.090	37%	5.26	80	1.00	4.18
82317 A	82322	Turbidite	Edge Recycle	19	6.45	0.379	0.205	54%	4.43	80	0.99	4.15
Single Pass Average								66%			0.83	4.01
Edge Recycle Average								70%			0.82	4.01
Overall Average								68%			0.82	4.01

TABLE 13-17 SUMMARY OF COLUMN LEACH TEST RESULTS - SILVER

KCA Sample No.	KCA Test No.	Description	HPGR Crush Type	Crush Size (mm)	Calculated Head P_{80} Size (mm)	Calculated Head (g/t Ag)	Extracted (g/t Ag)	Extracted (% Ag)	Calculated Tail P_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Cement (kg/t)
81045 A	81070	Upper Arenite East 243	Single Pass	19	9.03	0.34	0.06	19%	8.71	93	0.47	3.98
81046 A	81073	Upper Arenite East 243	Edge Recycle	19	7.89	0.40	0.14	34%	7.84	93	0.58	4.00
81047 A	81076	Middle Arenite East 243	Single Pass	19	9.30	0.36	0.11	31%	8.20	93	0.58	3.99
81048 A	81079	Middle Arenite East 243	Edge Recycle	19	7.83	0.35	0.09	25%	7.20	93	0.49	3.99
81049 A	81082	Lower Arenite East 243	Single Pass	19	8.11	0.39	0.17	44%	7.83	130	0.75	3.98
81050 A	81085	Lower Arenite East 243	Edge Recycle	19	7.78	0.45	0.20	44%	6.96	130	0.90	3.97
81051 A	81088	Argillite West 313	Single Pass	19	9.70	0.45	0.18	40%	8.26	108	1.37	3.99
81052 A	81091	Argillite West 313	Edge Recycle	19	8.40	0.51	0.25	49%	7.04	108	1.01	4.01
81053 A	82301	Argillite East 343	Single Pass	19	8.17	0.55	0.30	55%	7.96	141	0.77	3.97
81054 A	82304	Argillite East 343	Edge Recycle	19	7.71	0.48	0.24	50%	7.20	141	0.92	3.99
81055 A	82307	Arenite East 443	Single Pass	19	9.19	0.62	0.30	48%	8.07	141	0.85	3.97
81056 A	82310	Arenite East 443	Edge Recycle	19	8.73	0.50	0.23	46%	7.80	141	0.84	3.98
82316 A	82319	Turbidite	Single Pass	19	6.93	0.43	0.10	23%	5.26	80	1.00	4.18
82317 A	82322	Turbidite	Edge Recycle	19	6.45	0.44	0.11	26%	4.43	80	0.99	4.15
Single Pass Average								37%			0.83	4.01
Edge Recycle Average								39%			0.82	4.01
Overall Average								38%			0.82	4.01

At the conclusion of leaching, drain down tests were conducted on each column. The 24 drain down results for the single pass columns varied from 26.3 L/t to 37.3 L/t with an average of 32.3 L/t. The 24 drain down results for the edge recycle columns varied from 18.8 L/t to 40.0 L/t with an average of 32.6 L/t. The drain down results are presented in Table 13-18.

TABLE 13-18 SUMMARY OF DRAIN DOWN TEST RESULTS

KCA Sample No.	KCA Test No.	Description	HPGR Crush Type	Sample Weight (kg)	Liters $H_2O/MT_{dry\ ore}$ 24 hours	48 hours	72 hours	96 hours
81045 A	81070	Upper Arenite East 243	Single Pass	40.25	37.3	40.5	42.0	43.0
81046 A	81073	Upper Arenite East 243	Edge Recycle	40.00	40.0	43.5	45.3	46.5
81047 A	81076	Middle Arenite East 243	Single Pass	40.15	31.4	35.1	37.1	39.1
81048 A	81079	Middle Arenite East 243	Edge Recycle	40.10	30.4	33.7	35.4	36.4
81049 A	81082	Lower Arenite East 243	Single Pass	40.19	37.3	39.6	41.1	42.1
81050 A	81085	Lower Arenite East 243	Edge Recycle	40.30	38.0	40.2	41.4	42.4
81051 A	81088	Argillite West 313	Single Pass	22.18	30.2	32.5	33.4	34.3
81052 A	81091	Argillite West 313	Edge Recycle	29.08	31.3	33.7	35.1	36.1
81053 A	82301	Argillite East 343	Single Pass	40.32	27.3	30.3	32.2	33.5
81054 A	82304	Argillite East 343	Edge Recycle	40.12	30.7	33.2	34.4	35.9
81055 A	82307	Arenite East 443	Single Pass	40.26	36.3	39.2	41.2	42.2
81056 A	82310	Arenite East 443	Edge Recycle	40.22	39.3	42.3	44.3	45.5
82316 A	82319	Turbidite	Single Pass	34.20	26.3	27.5	30.1	29.8
82317 A	82322	Turbidite	Edge Recycle	37.76	18.8	20.9	22.2	23.0
Single Pass Average					32.3	34.9	36.7	37.7
Edge Recycle Average					32.6	35.3	36.9	38.0
Overall Average					32.5	35.1	36.8	37.8

The retained moisture for the single pass columns ranged from 61.1 L/t to 193.6 L/t with an average of 87.9 L/t. The retained moisture for the HPGR columns ranged from 69.5 L/t to 199.7 L/t with an average of 92.8 L/t. The retained moisture is presented in Table 13-19.

TABLE 13-19 SUMMARY OF RETAINED MOISTURE

KCA Sample No.	KCA Test No.	Description	HPGR Crush Type	Days Leached	Calculated Head P_{80} Size (mm)	Retained Solution, (L/MT_{dryore})
81045 A	81070	Upper Arenite East 243	Single Pass	93	9.03	72.0
81046 A	81073	Upper Arenite East 243	Edge Recycle	93	7.89	82.5
81047 A	81076	Middle Arenite East 243	Single Pass	93	9.30	77.2
81048 A	81079	Middle Arenite East 243	Edge Recycle	93	7.83	74.8
81049 A	81082	Lower Arenite East 243	Single Pass	130	8.11	67.7
81050 A	81085	Lower Arenite East 243	Edge Recycle	130	7.78	69.5
81051 A	81088	Argillite West 313	Single Pass	108	9.70	73.0
81052 A	81091	Argillite West 313	Edge Recycle	108	8.40	78.7
81053 A	82301	Argillite East 343	Single Pass	141	8.17	70.4

KCA Sample No.	KCA Test No.	Description	HPGR Crush Type	Days Leached	Calculated Head P_{80} Size (mm)	Retained Solution, (L/MT_{dryore})
81054 A	82304	Argillite East 343	Edge Recycle	141	7.71	70.8
81055 A	82307	Arenite East 443	Single Pass	141	9.19	61.1
81056 A	82310	Arenite East 443	Edge Recycle	141	8.73	73.6
82316 A	82319	Turbidite	Single Pass	80	6.93	193.6
82317 A	82322	Turbidite	Edge Recycle	80	6.45	199.7
Single Pass Average						87.9
Edge Recycle Average						92.8
Overall Average						90.3

The height of material in each column was measured before and after leaching. This height was utilized to calculate the “slump” during leaching as well as to calculate the final apparent bulk density for the material in the column. The slump averaged 0.1% for both the single pass and edge recycle columns. The apparent bulk density averaged 1.63 and 1.60 t/m^3 for the single pass and edge recycle columns, respectively. A bulk density for stacked ore of 1.60 is recommended for this study. The slump results are presented in Table 13-20.

To determine the required leach time, KCA breaks leaching into solution limited and time limited portions. The beginning of the leach, where recovery is rapid, the solution application is limiting the rate of leaching and when the leach slows to a steady rate time for capillary action and gold dissolution is the limiting factor for leach rate. The recovery curve is first plotted against the tonnes of solution applied per tonne of ore in the laboratory to find where the curve transitions from a steep (rapid solution controlled leaching) curve to a flatter (slower time controlled leaching) curve. A heap requires more time than the laboratory for the solution driven portion of leaching due to the time it takes to add the same tonnes of solution per tonne of ore, which can be calculated. One laboratory day will equal one operational day for the remaining leach time. Based on the calculations described, a leach time of 180 days is recommended for the edge recycle HPGR crushed material for this study. These calculations are presented in Table 13-21.

TABLE 13-20 PERCENT SLUMP AND FINAL APPARENT BULK DENSITY

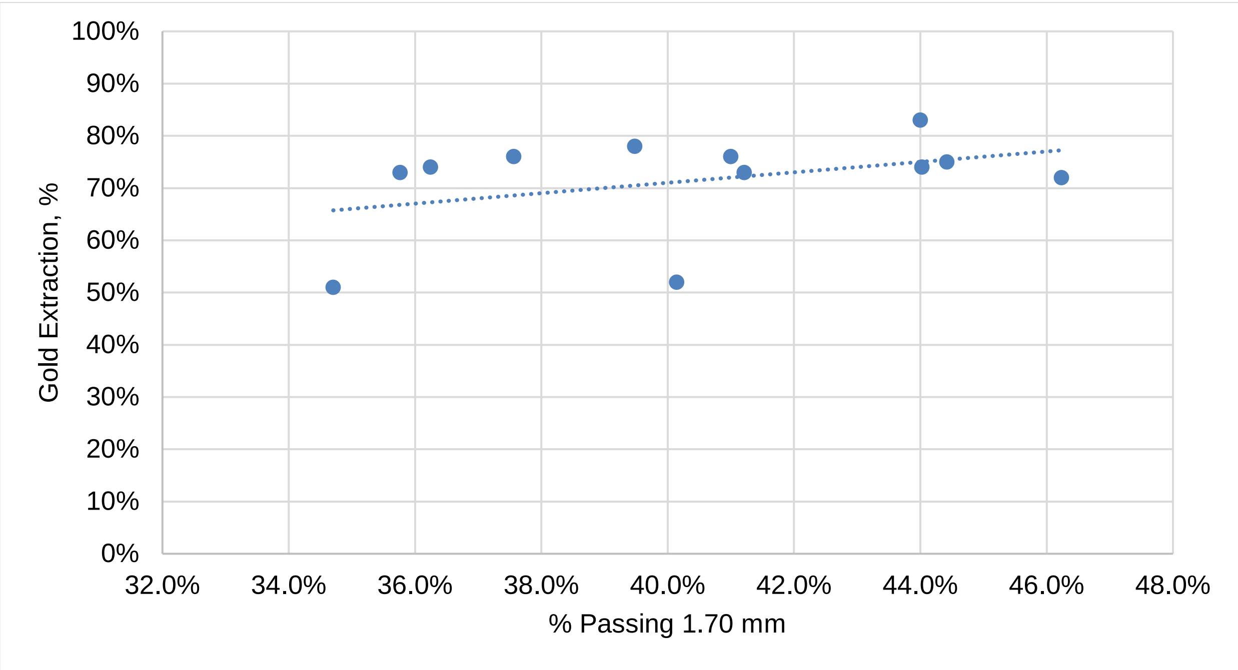
KCA Sample No.	KCA Test No.	Description	HPGR Crush Type	Calculated Head P_{80} Size (mm)	Initial Height (m)	Final Height (m)	Slump (%)	Final Apparent Bulk Density (dry t/m^3)
81045 A	81070	Upper Arenite East 243	Single Pass	9.03	3.007	3.007	0.0%	1.651
81046 A	81073	Upper Arenite East 243	Edge Recycle	7.89	3.080	3.080	0.0%	1.602
81047 A	81076	Middle Arenite East 243	Single Pass	9.30	2.978	2.978	0.0%	1.663
81048 A	81079	Middle Arenite East 243	Edge Recycle	7.83	3.048	3.048	0.0%	1.623
81049 A	81082	Lower Arenite East 243	Single Pass	8.11	3.042	3.042	0.0%	1.630
81050 A	81085	Lower Arenite East 243	Edge Recycle	7.78	3.042	3.042	0.0%	1.634
81051 A	81088	Argillite West 313	Single Pass	9.70	1.676	1.670	0.4%	1.638
81052 A	81091	Argillite West 313	Edge Recycle	8.40	2.102	2.092	0.5%	1.714
81053 A	82301	Argillite East 343	Single Pass	8.17	3.040	3.040	0.0%	1.636
81054 A	82304	Argillite East 343	Edge Recycle	7.71	3.067	3.058	0.3%	1.619
81055 A	82307	Arenite East 443	Single Pass	9.19	2.921	2.921	0.0%	1.700
81056 A	82310	Arenite East 443	Edge Recycle	8.73	3.010	3.010	0.0%	1.648
82316 A	82319	Turbidite	Single Pass	6.93	2.886	2.886	0.0%	1.462
82317 A	82322	Turbidite	Edge Recycle	6.45	3.369	3.369	0.0%	1.383
Single Pass Average				8.63			0.1%	1.63
Edge Recycle Average				7.83			0.1%	1.60
Overall Average				8.23			0.1%	1.61

TABLE 13-21 LEACH TIME CALCULATIONS

KCA Sample No.	KCA Test No.	Description	Crush Type	Transition, (ts/to)	Transition Au Recovery (%)	Transition Lab Days	Transition Field Days	Total Lab Leach Days	Total Field Leach Days	Ultimate Lab Au Recovery (%)	Field Deduction (%)	Estimated Field Au Recovery (%)
81045 A	81070	Upper Arenite East 243	Single Pass	0.56	62%	11	37	93	119	74%	2%	72%
81046 A	81073	Upper Arenite East 243	Edge Recycle	0.66	60%	15	44	93	122	83%	2%	81%
81047 A	81076	Middle Arenite East 243	Single Pass	0.64	62%	15	43	93	121	73%	2%	71%
81048 A	81079	Middle Arenite East 243	Edge Recycle	0.41	56%	8	27	93	112	74%	2%	72%
81049 A	81082	Lower Arenite East 243	Single Pass	1.19	58%	30	79	130	179	73%	2%	71%
81050 A	81085	Lower Arenite East 243	Edge Recycle	0.75	49%	17	50	130	163	72%	2%	70%
81051 A	81088	Argillite West 313	Single Pass	0.57	25%	6	38	108	140	51%	2%	49%
81052 A	81091	Argillite West 313	Edge Recycle	0.95	37%	16	63	108	155	52%	2%	50%
81053 A	82301	Argillite East 343	Single Pass	1.19	59%	30	79	141	190	76%	2%	74%
81054 A	82304	Argillite East 343	Edge Recycle	1.22	55%	32	81	141	190	75%	2%	73%
81055 A	82307	Arenite East 443	Single Pass	1.34	48%	35	89	141	195	76%	2%	74%
81056 A	82310	Arenite East 443	Edge Recycle	1.1	49%	28	73	141	186	78%	2%	76%
82316 A	82319	Turbidite	Single Pass	0.7	24%	15	47	80	112	37%	2%	35%
82317 A	82322	Turbidite	Edge Recycle	0.7	27%	16	47	80	111	54%	2%	52%
HPGR Field Single Pass		Arenite Average		0.93	58%	23	62	114	154	74%	2%	72%
		Argillite Average		0.88	42%	18	59	125	165	63%	2%	61%
		Turbidite Average		0.70	24%	15	47	80	112	37%	2%	35%
		Overall Average		0.88	48%	20	59	112	151	66%	2%	64%
HPGR Field Edge Recycle		Arenite Average		0.73	54%	17	49	114	146	77%	2%	75%
		Argillite Average		1.09	46%	24	72	125	173	64%	2%	62%
		Turbidite Average		0.70	27%	16	47	80	111	54%	2%	52%
		Overall Average		0.83	48%	19	55	112	149	70%	2%	68%

Due to the nature of HPGR crushing, using the conventional P_{80} is not the best way to compare the crushed product. As shown in Figure 13-1, the largest difference is in the percent passing 1.7 mm. The gold recoveries for the single pass and edge recycle column leach tests were plotted against the percent passing 1.7 mm showing a trend of higher recoveries with a higher generation of fines.

FIGURE 13-1 GOLD RECOVERY VERSUS PERCENT PASSING 1.7 MM



13.6 HEAP LEACH GOLD RECOVERY

The four crushing options were compared and, on average, the HPGR operated in closed circuit with the edge recycle had the highest gold recovery. The last series of tests also showed that the extended leach time gave better recoveries. The tests that were run for longer leach times had carbon samples taken at 60 days, which compares to the total leach time for the majority of the shorter leach tests. This allowed for a direct comparison between the different crushing tests. This comparison is presented in Table 13-22.

Based on these results and the trend of higher gold recovery with a higher percent passing 1.7 mm, the flowsheet recommended for this study is crushing with an HPGR in closed circuit with edge recycle. The field leach time, based on the edge recycle column tests, is recommended at 180 days. The extended leach time increases the size of the carbon columns and will cause a lower grade pregnant solution. To minimize these effects, it is recommended that solution be applied at 10 L/h/m^2 for the solution-controlled portion of leaching, estimated at 60 days based on the leach time presented in Table 13-21, and be reduced to 5 L/h/m^2 for the remainder of the leach time. If the drip emitters do not allow for the lower flow rate, intermittent leaching could also be used.

TABLE 13-22 CRUSH TYPE COMPARISON

KCA Sample No.	KCA Test No.	Description	Crush Type	Crush Size (mm)	Calculated Head (g/t Au)	Total Lab Leach Days	60 Day Au Recovery (%)	Ultimate Au Recovery (%)
79137 C	79190	Upper Arenite East 243	Conventional	8	0.273	61	61.3%	61.3%
79161 C	80233	Upper Arenite East 243	HPGR Lab Single Pass	19	0.258	61	56.3%	56.3%
81045 A	81070	Upper Arenite East 243	HPGR Field Single Pass	19	0.144	93	70.3%	74.0%
81046 A	81073	Upper Arenite East 243	HPGR Field Edge Recycle	19	0.238	93	76.1%	83.4%
79138 C	80201	Middle Arenite East 243	Conventional	8	0.259	61	66.2%	66.2%
79162 C	80236	Middle Arenite East 243	HPGR Lab Single Pass	19	0.295	61	71.6%	71.6%
81047 A	81076	Middle Arenite East 243	HPGR Field Single Pass	19	0.176	93	68.5%	73.0%
81048 A	81079	Middle Arenite East 243	HPGR Field Edge Recycle	19	0.175	93	69.1%	73.7%
79139 C	80204	Lower Arenite East 243	Conventional	8	0.512	61	65.4%	65.4%
79163 C	80239	Lower Arenite East 243	HPGR Lab Single Pass	19	0.736	61	61.5%	61.5%
81049 A	81082	Lower Arenite East 243	HPGR Field Single Pass	19	0.770	130	65.3%	73.3%
81050 A	81085	Lower Arenite East 243	HPGR Field Edge Recycle	19	1.185	130	59.9%	72.1%
79140 C	80207	Argillite West 313	Conventional	8	0.642	61	49.7%	49.7%
79164 C	80242	Argillite West 313	HPGR Lab Single Pass	19	0.457	61	50.6%	50.6%
81051 A	81088	Argillite West 313	HPGR Field Single Pass	19	0.550	108	45.6%	50.7%
81052 A	81091	Argillite West 313	HPGR Field Edge Recycle	19	0.678	108	46.7%	52.0%
79141 C	80210	Argillite East 343	Conventional	8	0.647	61	61.4%	61.4%
79165 C	80245	Argillite East 343	HPGR Lab Single Pass	19	1.136	85	47.1%*	47.1%
81053 A	82301	Argillite East 343	HPGR Field Single Pass	19	1.158	141	67.3%	76.0%
81054 A	82304	Argillite East 343	HPGR Field Edge Recycle	19	0.827	141	65.5%	75.3%
79144 C	80219	Arenite East 443	Conventional	8	0.717	61	67.7%	67.7%
79167 C	80251	Arenite East 443	HPGR Lab Single Pass	19	0.719	85	60.5%*	60.5%
81055 A	82307	Arenite East 443	HPGR Field Single Pass	19	1.602	141	62.1%	75.6%
81056 A	82310	Arenite East 443	HPGR Field Edge Recycle	19	1.332	141	66.6%	77.8%
Conventional			Arenite Average			61	65%	65%
			Argillite Average			61	56%	56%
			Overall Average			61	62%	62%
HPGR Lab Single Pass			Arenite Average			67	62%	62%
			Argillite Average			73	49%	49%
			Overall Average			69	58%	58%
HPGR Field Single Pass			Arenite Average			114	67%	74%
			Argillite Average			125	56%	63%
			Overall Average			118	63%	70%
HPGR Field Edge Recycle			Arenite Average			110	68%	76%
			Argillite Average			125	56%	64%
			Overall Average			118	64%	72%

*Recoveries at 85 days

The leach cycle, heap recovery and reagent consumptions used in this study are based on the HPGR edge recycle column leach tests completed in the April 2019 program and, for material types not tested in 2019, HPGR column leach tests completed in the May 2018 KCA program.

KCA typically discounts the gold recovery in column tests by 2-3% when estimating field recoveries; 2% was used for Essakane due to the high number of column tests completed. The silver was discounted by 3% for estimating field recoveries. A summary of the column tests utilized for recovery calculations and the discounted field recoveries are presented in Table 13-23.

TABLE 13-23 COLUMN TEST FIELD RECOVERY DISCOUNTS

KCA Test No	Rock Type	Lithology	Crush Type	Column Au Recovery (%)	Field Au Rec. (%)	Column Ag Rec. (%)	Field Ag Rec. (%)
80230	Arenite West	223	HPGR Lab Single Pass	56%	54%	15%	12%
81070	Upper Arenite East	243	HPGR Field Edge Recycle	83%	81%	34%	31%
81076	Middle Arenite East	243	HPGR Field Edge Recycle	74%	72%	25%	22%
81082	Lower Arenite East	243	HPGR Field Edge Recycle	72%	70%	44%	41%
81088	Argillite West	313	HPGR Field Edge Recycle	52%	50%	49%	46%
82301	Argillite East	343	HPGR Field Edge Recycle	75%	73%	50%	47%
80248	Arenite West	413	HPGR Lab Single Pass	54%	52%	20%	17%
82307	Arenite East	443	HPGR Field Edge Recycle	78%	76%	46%	43%
80257	Argillite	513, 543, 623, 643	HPGR Lab Single Pass	83%	81%	21%	18%
82322	Turbidite		HPGR Field Edge Recycle	54%	52%	26%	23%
	Arenite Average			69%	67%	31%	28%
	Argillite Average			70%	68%	40%	37%
	Arenite 243 Average			76%	74%	34%	31%
	Overall Average			68%	66%	33%	30%

The mine plan was separated into the different rock and lithologies tested. Not all rock types and lithologies were represented directly by column tests and average recoveries of rock types were applied to these, specifically Argillite 123. All of the column leach tests were completed on material from the EMZ pit.

The field recovery was calculated based on the rock types and grades presented in the mine plan. Any change in grades or rock type would change the overall estimated gold and silver recoveries. There is some risk of lower recoveries according to the variability testing shown in Table 13-7, where the variability bottle roll tests achieved lower gold recoveries than the composite samples by an average of 14.5%. The estimated field recovery for gold is 67% and for silver it is 34%. The gold and silver recovery results are presented in Table 13-24.

TABLE 13-24 ESTIMATED HEAP RECOVERY

Rock Type	Lithology	Tonnage (000 t)	Estimated Au Grade (g/t Au)	Lithology Used	Average Au Recovery (%)	Average Ag Recovery (%)
Argillite	10203	43	0.33	Argillite Average	68%	41%
Argillite	1023	12	0.32			
Argillite	3103	5,996	0.43	Argillite West 313	50%	46%
Argillite	313	112	0.41			
Argillite	3403	6,094	0.43	Argillite East 343	73%	47%
Argillite	343	3,232	0.42			
Argillite	5103	626	0.46	Argillite 513, 543, 623, 643	81%	31%
Argillite	5403	1,095	0.45			
Argillite	543	38	0.46			
Argillite	6203	55	0.41			
Argillite	6403	42	0.35			
Arenite	4103	3,396	0.43	Arenite 413	52%	17%
Arenite	4403	7,667	0.43	Arenite 443	76%	43%
Arenite	443	348	0.45			
Arenite	2203	6,344	0.42	Arenite West 223	54%	12%
Arenite	223	304	0.43			
Arenite	2403	4,816	0.44	Arenite 243 Average	74%	31%
Arenite	243	10,279	0.43			
Turbidite	10303	795	0.40	Turbidite	52%	23%
Turbidite	1033	272	0.39			
Turbidite	10403	1,687	0.44			
Turbidite	1043	786	0.42			
Argillite Total		**17,346**	**0.43**		**66%**	**45%**
Arenite Total		**33,153**	**0.43**		**68%**	**29%**
Turbidite Total		**3,539**	**0.423**		**52%**	**23%**
Total		**54,039**	**0.429**		**67%**	**34%**

With mostly clean non-reactive ores, cyanide consumption in production heaps is typically 25% to 33% of 2 m tall laboratory column test consumptions. The HPGR edge recycle column leach tests had an average consumption of 0.82 kg NaCN per tonne of ore. These column tests were conducted in 3 m tall columns and a cyanide consumption factor of 0.33% was used. The estimated field cyanide consumption is 0.27 kg NaCN per tonne of ore.

All of the edge recycle tests were conducted with 4 kg of Portland Type II cement per tonne of ore and this addition rate should be used in the field.

13.7 CONCLUSION FOR OVERALL ORES

The results of the metallurgical test programs indicate that the ore types tested are amenable to standard heap leaching methods. The available test results are sufficient to support this FS.

A summary of the design parameters and considerations for heap leaching closed circuit HPGR crushed ore based on these test programs is as follows:

- HPGR closed circuit product size: P_{100} 9 mm (P_{80} 7 mm)
- HPGR average throughput: 265 (t·s)/(m^3·h)
- HPGR specific energy: 1.8 kWh/t
- ATWAL specific abrasion: 20.1 g/t, considered moderately abrasive
- CWi: 17.31 kWh/t, classified as hard to very hard
- A_i: 0.4187 g, considered abrasive
- Agglomeration is required for stacking up to 50 m with 4 kg cement per tonne of ore.
- Leach Time: 180 days
- Gold Recovery: 67%
- Silver Recovery: 34%
- Sodium Cyanide Consumption: 0.27 kg/t
- Some of the material is preg-robbing and caution will need to be taken to minimize the amount of preg-robbing material stacked on the heap
- There is low potential for the material to be acid producing

14 MINERAL RESOURCE ESTIMATE

14.1 EMZ, FALAGOUNTOU WEST, AND WAFAKA DEPOSITS

The resource estimation methodologies, results, and validations for the EMZ, Falagountou West, and Wafaka deposits are presented in this section.

The Mineral Resource estimates were prepared in accordance with CIM (2014) definitions and are reported in accordance with the NI 43-101 guidelines. Classification, or assigning a level of confidence to Mineral Resources, has been undertaken with strict adherence to CIM (2014) definitions. In the opinion of the responsible QP, the resource evaluation reported herein is a reasonable representation of the Mineral Resources delineated at Essakane as of August 31, 2019.

The Essakane Mineral Resource estimate at August 31, 2019 for the Project is summarized in Table 14-1 and is reported on a 100% basis. The Mineral Resource estimate is inclusive of Mineral Reserves.

The 0.25 g/t Au heap leach cut-off grade for fresh rock at EMZ is lower than the fresh rock cut-off grade at Falagountou as the Falagountou material is not considered for treatment at the HLF.

Indicated Mineral Resources at the EMZ, Falagountou West, and Wafaka deposits are currently estimated to total 155 Mt at an average grade of 0.98 g/t Au for a total of 4,878,000 ounces of gold. Inferred Mineral Resources are estimated to be 13 Mt at an average grade of 1.10 g/t Au for a total of 454,000 ounces of gold. IAMGOLD's attributable Indicated Mineral Resources are 140 Mt totalling 4,390,000 ounces of gold and Inferred Mineral Resources are 12 Mt totalling 409,000 ounces of gold.

The responsible QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the EMZ, Falagountou West, and Wafaka Mineral Resource estimates.

TABLE 14-1 ESSAKANE MINERAL RESOURCE ESTIMATE – AUGUST 31, 2019

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Measured	-	-	-
Indicated	154,854	0.98	4,878
Total Measured + Indicated	**154,854**	**0.98**	**4,878**
Inferred	12,823	1.10	454

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade which varies between 0.25 and 0.55 g/t Au depending on material type and pit.
3. Mineral Resources are estimated using an average long-term gold price of US$1,500/oz.
4. A minimum mining width of 10 m was used for Falagountou and 10 m for EMZ.
5. Bulk density is estimated by Ordinary Kriging (OK) by weathering type.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Numbers may not add due to rounding.

14.1.1 EMZ DEPOSIT

14.1.1.1 DATA

The EMZ deposit resource estimation used the results of three types of holes: 1) DD, 2) RC, and 3) RCD, for a total of 3,333 holes and 558,905 m drilled.

Table 14-2 details the series of holes by type and year of drilling. Note that because AC and RAB sampling is more subject to segregation bias, their assay results have not been used in the resource estimate.

TABLE 14-2 EMZ RESOURCE DATABASE SUMMARY

Hole Type	Year	Series	No. of Holes	Metres Drilled
DDH	2018	MEDD0610-MEDD064	38	10,088.5
		MEDD0483-MEDD0609	127	26,308.5
		MGMT0006 - MGMT0007	2	300
	2017	MGDD0017	1	380
	2016	MGMT001-MGMT005	5	710
	2015	MEDD0464-MEDD0482	19	5,508.5
	2014	MEDD0403-MEDD0463	60	19,084
		MEDD0327-MEDD0402	75	22,958.85
	2013	EDD0248-EDD0292	43	7,939
		EDD0376-EDD0379	4	900
		HSDD0001-HSDD0010	10	1,547
	2012	MEDD0235-MEDD0326	92	44,054.95
	2011	EDD0345-EDD0375	31	5,521.23
		MEDD0105-MEDD0234	129	52,323.65
		EDD0305-EDD0344	40	11,819.37
	2010	MEDD0001-MEDD0104	100	26,159.75
	2009	EDD0295-EDD0304	10	2,208.5
	Before 2009	EDD0001-EDD0247	204	37,315.88
RC	2018	MERC0558-MERC0641	69	11,228
	2017	MERC0489-MERC0557	68	4,910
	2016	MERC0384-MERC0488	105	13,654
	2015	MERC0307-MERC0383	52	8,109
	2014	MERC0274-MERC0306	31	4.123
	2013	MERC0146-MERC0273	128	18,199
		MERC0101-MERC0145	35	4,492
	2012	ERC2003-ERC2078	75	10,006
		ERC1904-ERC2002	89	10,536
	2011	ERC1824-ERC1918	89	13,041
	2010	MERC0001-MERC00099	96	12,437
	2009	ERC1786-ERC1823	38	4,388
	Before 2009	ERC0001-ERC2005	1,100	101,944
RCD	2018	MERC0635D	1	240
		MERC0628D & MERC0633D	2	579.5
		MERC0621D & MERC0626D	2	535.5
		MERC0593D & MERC0594D	2	743
		MERC0585D-MERC0590D	6	2,606
	2017	MERC0567D & MERC0569D	2	800
	2017	MERC0535D	1	128
	2015	MERC0324D-MERC0377D	25	5,843
	2014	MERC0297D	1	401
	2010	MERC0048D-MERC0050D	3	1,032
	Before 2009	ERC0120D-ERC1692D	323	57,922
		Total	**3,333**	**558,905**

The current resource estimate includes a series of new holes. A total of 114 holes (DD, RC, and RCD) for 12,536 m were drilled since the June 5, 2018 resource estimate and were added to the Essakane resource database. Figure 14-1 shows the location of all the drill holes available in the database for the current resource estimate (on the left side).

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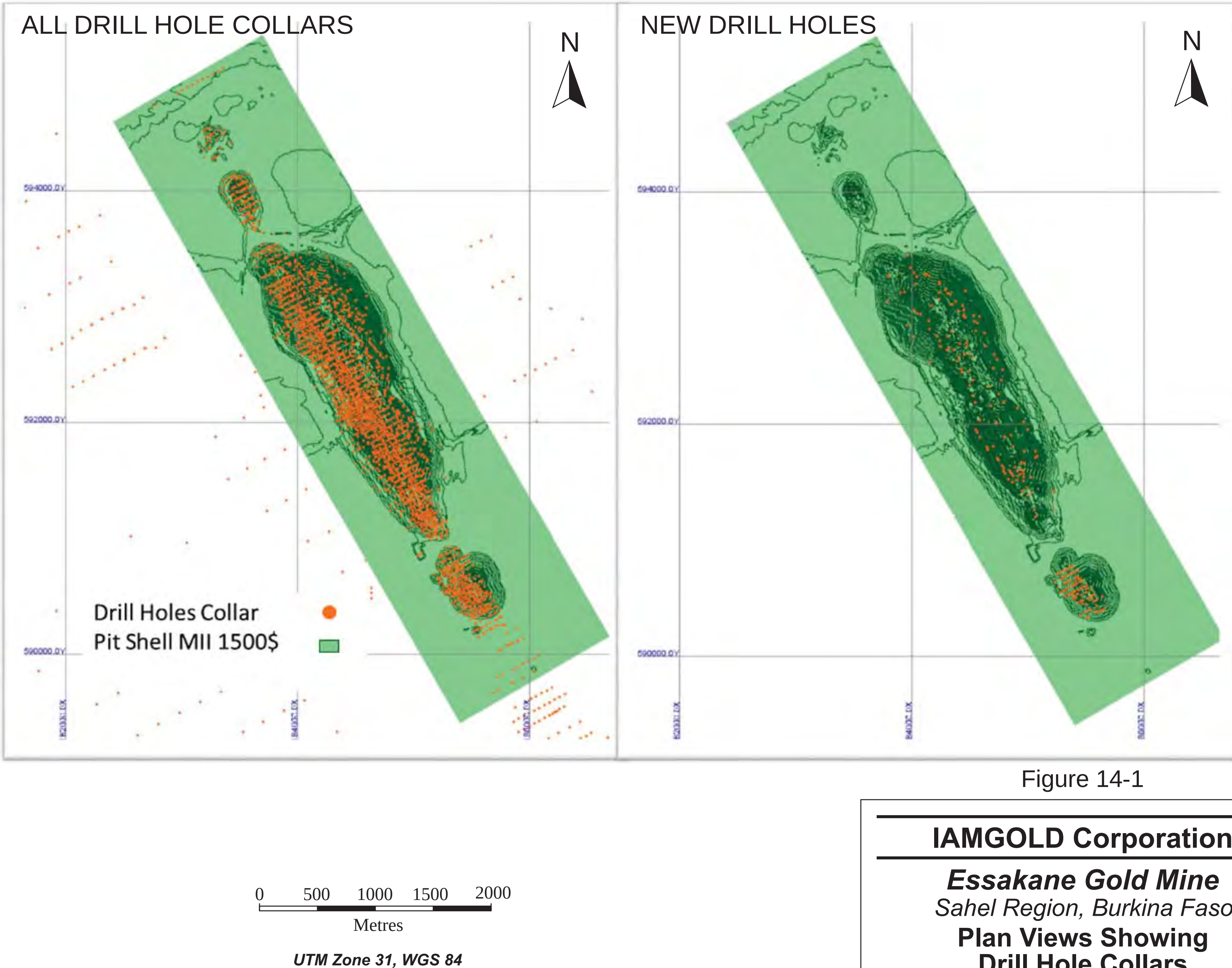


Figure 14-1

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Plan Views Showing Drill Hole Collars and New Drill Holes

January 2020

Source: IAMGOLD, 2019.

14.1.1.2 ASSAYS

The December 2018 assay database, used in the current resource update, consists of 396,996 records including 347,922 assay results above gold limit detection with an average sample length of 1.16 m, representing 460,031 assayed metres. Some 70% of the sampled intervals are one metre long while 28% are 1.5 m in length. The remaining 2% of the sampled intervals range from 0.2 m to 7.5 m.

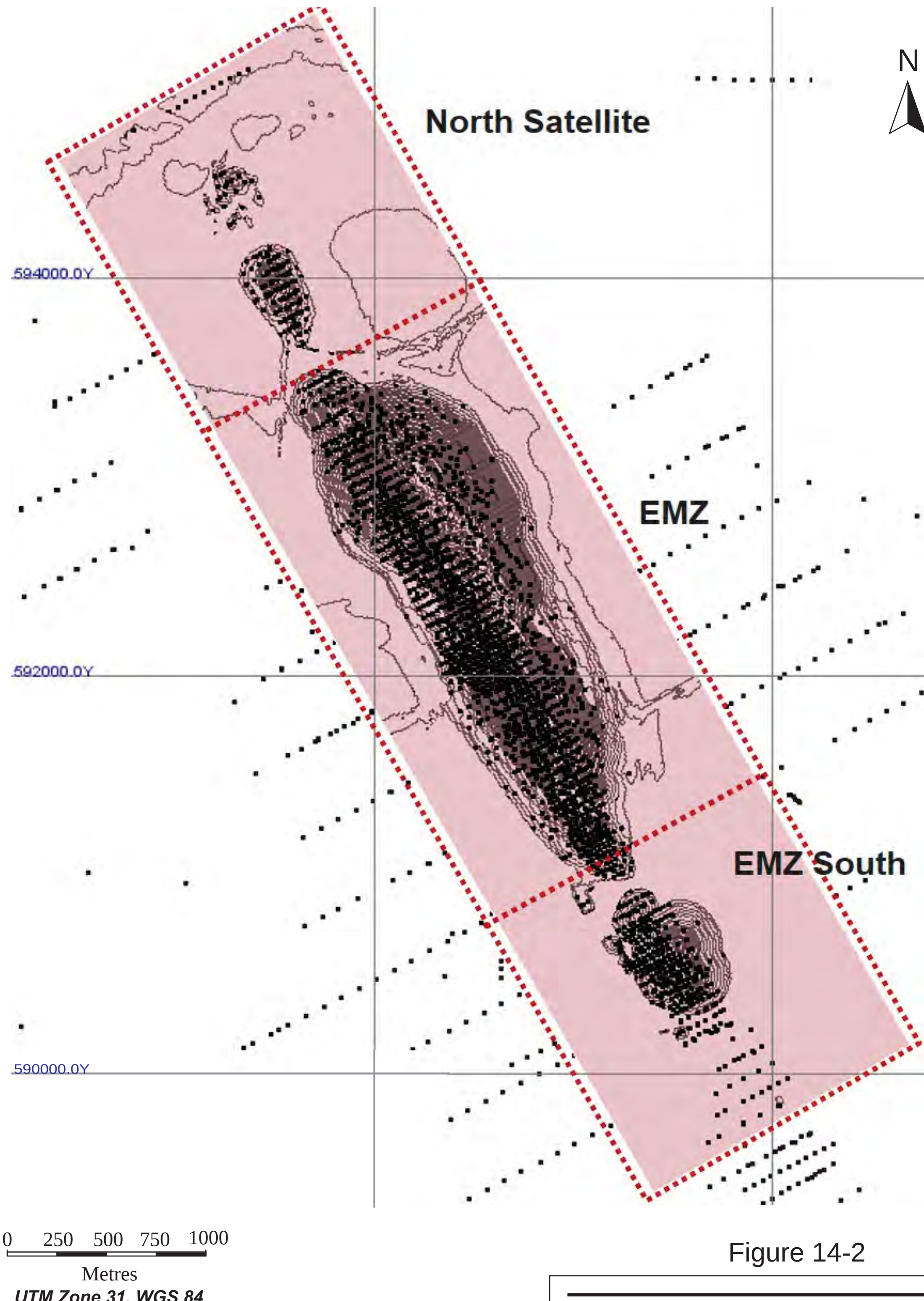
Gold grades vary from 0.0005 g/t Au to 430.0 g/t Au with an average grade of 0.44 g/t Au.

Note that in the EMZ deposit, a total of 187 holes are not assayed and this includes abandoned holes, holes invalidated by the responsible QP due to failed QA/QC protocols, unsampled holes when the property changed hands, and holes excluded for other reasons. Even though the assay results from these holes have not been retained for estimation purposes, some valuable information such as lithological, structural, or density data was used for modelling.

14.1.1.3 DRILL HOLE SPACING

The drill hole spacing is variable depending on which area of the Project the drilling is carried out. Three areas of resource development were defined within the EMZ deposit (Figure 14-2):

- North Satellite
- EMZ
- EMZ South



Legend:

- Drill Hole Collar
- Area Contour
- US$ 1,500/oz Whittle Pit Shell

Figure 14-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Plan View Showing the Three EMZ Resource Areas

January 2020

Source: IAMGOLD, 2019.

In the North Satellite area, the drill spacing is generally 50 m by 50 m and locally 25 m. The EMZ area is more densely drilled with a 25 m by 25 m spacing on the eastern limb of the fold and a wider spacing of 50 m by 50 m on the western limb. The EMZ South area is currently drilled on a 50 m by 50 m grid and locally 25 m spacing.

The current drill spacing in the EMZ deposit is judged adequate to develop a reasonable model of the mineralization distribution and to quantify its volume and quality with a good level of confidence in all three areas of the Project.

14.1.1.4 MODELLING

The modelling work was performed by Essakane S.A. personnel. The last update of the wireframes was performed at the end of October 2018. New drilling information showed a good correspondence with the actual model. The modelling was carried out using GEOVIA GEMS version 6.8.

Table 14-3 lists the surfaces and solids that were available or created for the use of the current resource estimate. The source and/or procedure of creation of the weathering, lithostructural, and topography elements are discussed in detail in the following sub-sections.

TABLE 14-3 SURFACES AND SOLIDS USED FOR THE EMZ MINERAL RESOURCE ESTIMATE

Count	Domain	Description	Order of Precedence	Triangulation Name
1		Topography		Topobaseclip/Mai2017/DVR_2017
2	1	Saprolite		DVR_Solide/Sap/Sept2017/DVR_2017
3	2	Transition	1	DVR_Solide/Trans/Sept2017/DVR_2017
4	3	Fresh Rock	2	DVR_Solide/RoC/Sept2017/DVR_2017
5	220	Main Arenite W Flank, middle thrust	8	DVR_Solide/223/ Oct2018/DVR_18
6	240	Main Arenite, E Flank	3	DVR_Solide/243/ Oct2018/DVR_18
7	310	Footwall Argillite W Flank, lower thrust	9	DVR_Solide/313 /Oct2018/DVR_18
8	340	Footwall Argillite E Flank, upper thrust (N)	4	DVR_Solide/342/ Oct2018/DVR_18
9	410	Lower Arenite	10	DVR_Solide/413/ Oct2018/DVR_18
10	440	Lower Arenite	5	DVR_Solide/443/ Oct2018/DVR_18
11	510	Deep Argillite	11	DVR_Solide/513/ Oct2018/DVR_18
12	540	Deep Argillite	6	DVR_Solide/543/ Oct2018/DVR_18
13	620	Upper Argillite W Flank	12	DVR_Solide/620/ Oct2018/DVR_18
14	640	Upper Argillite E Flank Turbidites W Flank	7	DVR_Solide/643/ Oct2018/DVR_18
15	1010	Turbidites E Flank	13	DVR_Solide/1013/ Oct2018/DVR_18
16	1020	Turbidites W Flank	14	DVR_Solide/1023/ Oct2018/DVR_18
17	1030	Turbidites E Flank	15	DVR_Solide/1033/ Oct2018/DVR_18
18	1040	Nose	16	DVR_Solide/1043/ Oct2018/DVR_18
19	10			DVR_Solide/Nose/ Oct2018/DVR_18

Weathering Profile Modelling

Two surfaces of weathering were used in this resource estimate; the saprolite and the transition surfaces. They represent the bottom limit of the corresponding weathering zone.

The surfaces previously used for the February 2018 resource estimate have been updated with new hole information based initially on the density measurements, where available, and then by placing the limits midway between density values showing a change in the weathering zone. Where no density measurements were available or where interpretations were conflicting, the hardness information from the drill log, defined by the Brown Index, was used for weathering modelling. The hardness codes are categorized into Saprolite (S1, S2, S3, and S4), Transition (S5, S6, R0, and R1), and Fresh Rock (R2, R3, R4, R5, and R6) as presented in Table 14-4. An example of the modelled surfaces is shown in Figure 14-3.

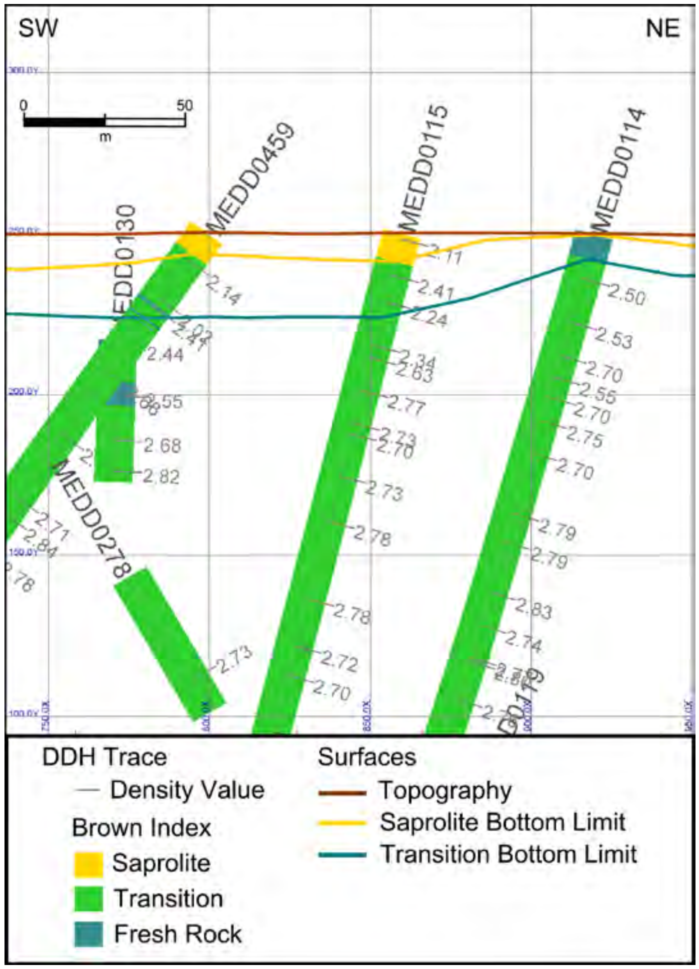
TABLE 14-4 BROWN INDEX OF SOIL AND ROCK STRENGTH

Essakane S.A.	Grade	Description	Identification	Approximate Range of UCS (MPa)
Saprolite Upper Saprolite (WSU)	S1	Very Soft	Easily penetrated several inches by fist.	< 0.025
	S2	Soft	Easily penetrated several inches by thumb.	0.025 – 0.05
	S3	Firm	Can be penetrated several inches by thumb with moderate effort.	0.05 – 0.10
	S4	Stiff	Readily indented by thumb but penetrated only with great effort.	0.10 – 0.25
Transition Lower Saprolite (WSL) Saprock (WSR1), (WSR2)	S5	Very Stiff	Readily indented by thumb nail.	0.25 – 0.50
	S6	Hard	Indented with difficulty by thumb nail.	> 0.50
	R0	Extremely weak rock	Indented by thumb nail.	0.50 – 1.0
	R1	Very weak rock	Crumbles under firm blow with point of geological hammer.	1.0 – 5.0
Rock Saprock (WSR2) Fresh Rock	R2	Weak rock	Can be peeled by a pocketknife.	5.0 – 25
	R3	Medium strong rock	Can be peeled by a pocketknife with difficulty; shallow indentations made by firm blow with point of geological hammer.	25 – 50
	R4	Strong rock	Cannot be scraped or peeled with a pocketknife, specimen can be fractured with a single firm blow of geological hammer.	50 – 100
	R5	Very strong rock	Specimen requires more than one blow of geological hammer to fracture it.	100 – 250
	R6	Extremely strong rock	Specimen requires many blows of geological hammer to fracture it. Specimen can only be chipped with geological hammer.	> 250

Note:

1. UCS: Uniaxial Compressive Strength

FIGURE 14-3 SECTION 52275N – EMZ WEATHERING SURFACES



Lithostructural Modelling

The geological wireframes modelled for the EMZ deposit included the structural and lithological elements available in the database.

The lithological model comprised units of arenite and argillite. Each unit was digitized as an individual layer juxtaposed one above the other. The units were divided into four parts, relating to the anticlinal fold axis, and identified as West or East flank units and according to their positions in the folds such as at the nose or the flank (geometric association). These units, as illustrated on Figures 14-4 and 14-5, determined the main lithostructural domains.

FIGURE 14-4 ISOMETRIC VIEW – EMZ LITHOLOGICAL MODEL

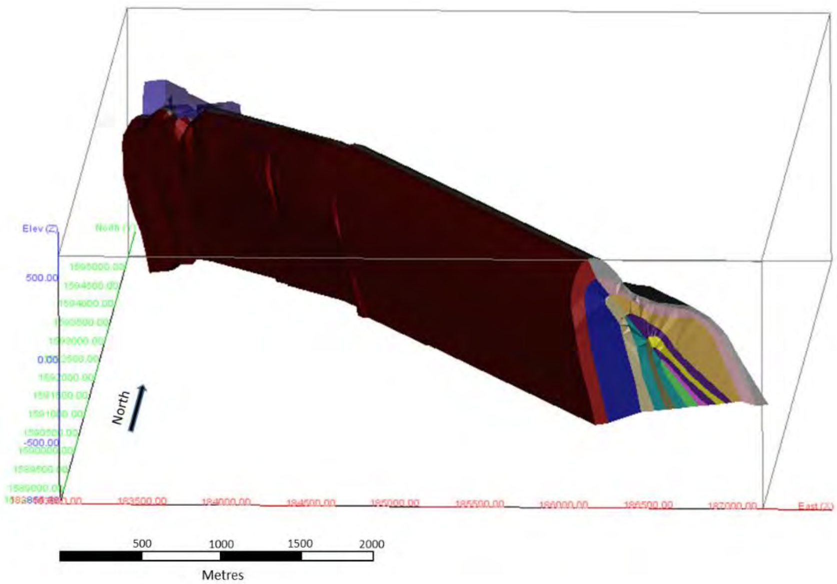
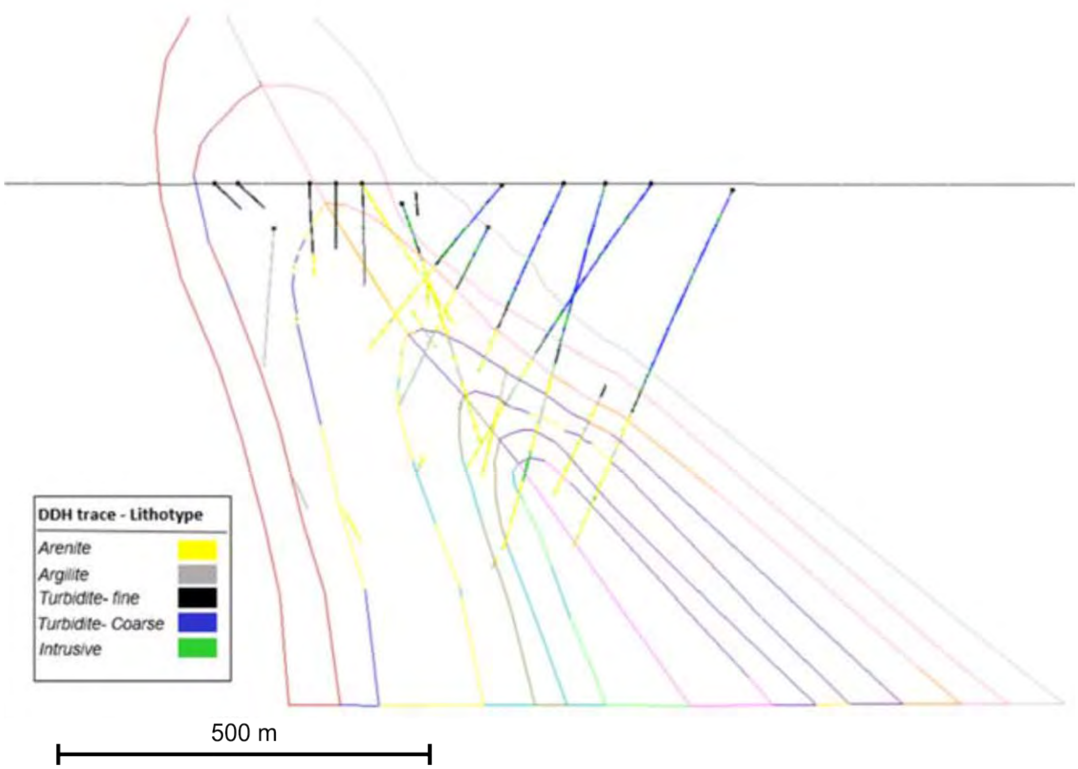


FIGURE 14-5 SECTION 51825N – EMZ LITHOLOGICAL MODEL
Looking Northwest



Following the EMZ deposit lithostructural model, the North Satellite area was modelled as the continuation of the northern extension of the EMZ deposit, an anticlinal folded sedimentary sequence gently plunging to the north. The upper unit contains the mineralization in the north. For the 2018 Mineral Resource, all of Essakane was modelled using the same sequence of domains.

Surface Topography

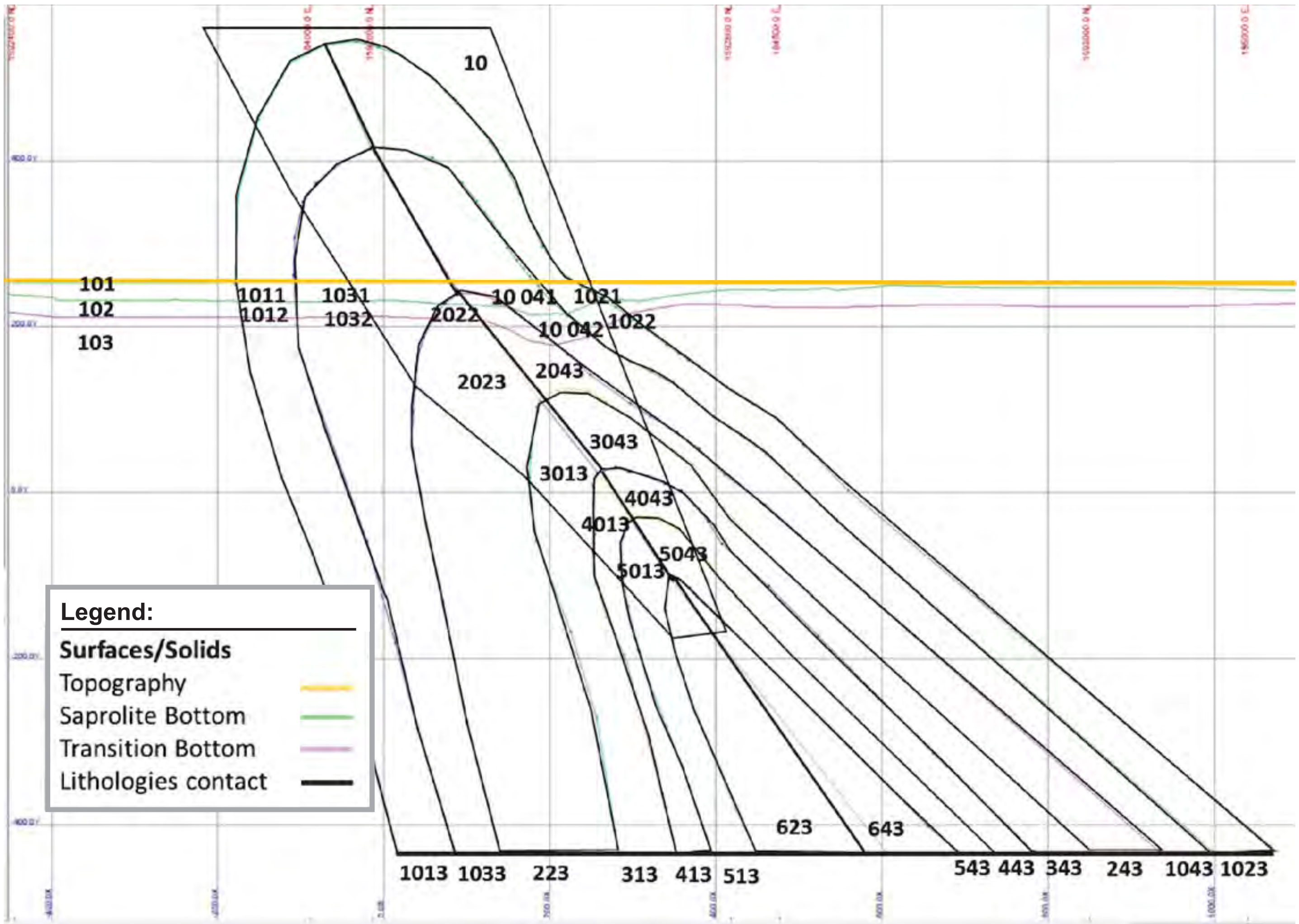
The surface (pre-mining) named “Topo_2009” was used to code all blocks above it as “Air” in the block model.

14.1.1.5 STATISTICAL ANALYSIS

Statistics of the Uncapped Assays

The analysis was done by domain.

The lithological and weathering codes were extracted from drill hole and solid intersections and later combined in the assay database to build the domain codes as illustrated in Figure 14-6.



Section Looking North-West

0 100 200 300 400
Metres
UTM Zone 31, WGS 84

Figure 14-6

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Essakane Gold Mine
Sahel Region, Burkina Faso

EMZ Section 51550N – Example of Domain Coding

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation

Essakane S.A. observed, through their statistical analysis, typically, the data displayed extreme skewness and high coefficients of variation separated by uninformed grade ranges. The assay outliers were examined on both log-probability plots and histograms. Grade capping was applied to restrict the influence of outliers in the composites used for grade interpolation. Capping levels are determined using rock type and position in the flank (east or west). Table 14-5 shows the selected capped value and the metal loss.

TABLE 14-5 STATISTICS OF THE EMZ ASSAYS GROUPED BY LITHOLOGY

Lithology	Capped Value (g/t Au)	Total Data	Capped Data	Metal Loss (%)
Arénite Est	65	103,852	118	6.2%
Arénite Ouest	40	28,980	37	7.6%
Argilite Est	35	57,433	84	10.4%
Argilite Ouest	40	53,464	31	4.8%
Turbidite Est	30	82,388	31	5.6%
Turbidite Ouest	20	30,992	27	7.6%
Total		**357,109**	**328**	**6.8%**

A grade restriction was applied on rock codes in the second pass. The grade restriction was based on the study carried out by Géovariance, a geostatistical consulting company, using the probability of transition for each domain. This restriction is then applied during the interpolation in the second pass.

Compositing

The drill hole database coded within each interpreted domain was composited to achieve a uniform sample support. Considering the current bench heights of the mining operation (5 m to 10 m benches), the variance of the assay population, and the drill hole spacing, it was decided to composite the data with a regular 5 m run length (down hole) within the limits of each interpreted domain using the capped value of the assay samples. Composites of less than one metre were excluded from the composite database.

Statistics of the 5 m Composites

Descriptive statistics of the 5 m composites were generated and grouped by domain. The gold grade datasets for the various estimation domains are characterized by a generally high coefficient of variation, which is common for this type of gold deposit. Table 14-6 shows the 5 m composite statistics by individual domains.

TABLE 14-6 STATISTICS OF THE EMZ 5 M COMPOSITES BY DOMAIN

Domain Code	No.	Mean (g/t Au)	Max (g/t Au)	STD (1) (g/t Au)	CoV (2)
221	149	0.15	5.27	0.57	3.80
222	196	0.09	2.76	0.25	2.86
223	4145	0.08	19.12	0.43	5.60
241	3249	1.18	85.50	2.92	2.48
242	2371	1.08	70.57	2.65	2.45
243	8757	0.97	117.08	3.41	3.53
312	15	0.05	0.31	0.09	1.90
313	3434	0.23	32.31	1.27	5.60
341	77	0.10	3.99	0.45	4.69
342	210	0.13	2.85	0.29	2.27
343	3555	0.38	75.35	2.26	5.99
413	1148	0.20	15.14	0.66	3.39
443	403	0.37	15.39	0.98	2.61
513	360	0.21	9.41	0.78	3.66
543	145	0.25	6.37	0.67	2.68
623	407	0.12	4.21	0.41	3.33
643	238	0.26	3.54	0.50	1.91
1011	66	0.04	0.41	0.06	1.78
1012	72	0.02	0.26	0.04	1.89
1013	487	0.06	10.71	0.51	8.27
1021	1149	0.07	5.16	0.23	3.19
1022	923	0.06	9.67	0.40	6.66
1023	4130	0.03	4.02	0.14	4.24
1031	227	0.09	3.27	0.28	3.26
1032	273	0.06	1.27	0.13	2.25
1033	2128	0.10	34.65	0.83	8.02
1041	2390	0.32	19.21	0.97	3.07
1042	1134	0.29	57.50	1.88	6.39
1043	5024	0.12	24.39	0.74	6.21
2201	660	0.44	17.00	1.01	2.29
2202	681	0.42	10.54	0.87	2.09
2203	4003	0.42	28.07	1.40	3.33
2401	2095	0.57	42.94	1.86	3.28
2402	904	0.77	33.21	2.14	2.78
2403	2850	0.84	79.63	2.86	3.41
3101	146	0.15	9.62	0.80	5.35

Domain Code	No.	Mean (g/t Au)	Max (g/t Au)	STD (1) (g/t Au)	CoV (2)
3102	198	0.13	3.53	0.32	2.52
3103	3309	0.59	45.73	2.20	3.72
3401	902	0.25	10.26	0.72	2.84
3402	1077	0.29	33.00	1.16	4.05
3403	7181	0.60	39.74	1.73	2.89
4103	1850	0.48	32.18	1.64	3.40
4403	3078	0.65	52.81	2.43	3.73
5103	804	0.82	50.08	3.26	3.98
5403	851	0.75	32.59	2.59	3.48
6203	377	0.48	30.11	1.90	3.96
6403	236	0.54	35.17	2.50	4.59
10101	28	0.02	0.10	0.02	1.25
10102	95	0.08	0.82	0.18	2.22
10103	539	0.07	1.61	0.19	2.54
10201	76	0.63	33.96	3.90	6.23
10202	440	0.28	14.53	0.96	3.44
10203	753	0.18	17.19	0.77	4.37
10301	586	0.28	9.55	0.63	2.23
10302	601	0.29	11.40	0.92	3.20
10303	2348	0.31	44.70	1.27	4.16
10401	256	0.31	6.34	0.69	2.19
10402	650	0.27	11.01	0.72	2.72
10403	2774	0.26	34.02	0.96	3.72

Notes:

1. STD - Standard Deviation
2. CoV - Coefficient of Variation

Density Data

The density database contained 29,651 measurements taken from DD and RCD holes. Some outliers were removed from the GEOVIA GEMS density database. The excluded values, listed in Table 14-7, were generally too low with regards to neighbouring values or the weathering profile.

TABLE 14-7 EXCLUDED EMZ DENSITY MEASUREMENTS

HOLE-ID	Depth (m)	Density (g/cm^3)
EDD0206	246.55	3.53
EDD0203	282.55	3.75
EDD0191	97.30	3.50
EDD0193	61.00	1.60
EDD0186	139.40	3.68
EDD0113	12.50	1.69
EDD0113	29.00	1.69
EDD0129	207.50	3.39
ERC1686D	77.25	2.89
EDD0067	39.30	2.82
ERC1635D	59.20	1.25
EDD0104	249.10	3.63
EDD0046	55.35	1.41
EDD0043	66.80	1.83
ERC0867D	87.00	3.36
EDD0050	61.10	3.24
EDD0051	26.10	1.59
EDD0065	151.50	2.88
EDD0069	0.00	1.78
EDD0069	21.45	2.51
EDD0151	0.00	1.07
EDD0070	42.05	2.77
EDD0070	45.60	2.76
EDD0070	141.90	2.89
ERC1201D	226.90	1.96
EDD0219	161.70	3.70
EDD0214	232.30	2.15
MEDD0014	5.55	1.60
MEDD0065	29.70	1.78
MEDD0087	14.02	2.79
MEDD0095	42.80	3.85
MEDD0143	40.30	2.81
ERC2070	18.00	2.95
MEDD0450	28.85	1.21
MEDD0463	16.00	1.20
MERC0327D	112.00	1.00
MERC0327D	124.00	1.00
MERC0327D	146.00	1.00
MERC0327D	175.00	1.00
MERC0327D	202.00	1.00
MERC0327D	229.00	1.00
MERC0327D	244.00	1.00

From the density database, a total of 18,706 measurements, including values between 1.0 t/m^3 and 3.58 t/m^3 within the resource domains, were extracted for statistical studies. The statistics of the density measurements are presented in Table 14-8. The median value of each domain was used as the default value in the block model, except for the domains that had less than 30 density measurements.

TABLE 14-8 STATISTICS OF THE EMZ DENSITY MEASUREMENTS BY DOMAIN

Weathering	Domain Code	Number	Min (g/cm^3)	Max (g/cm^3)	Mean (g/cm^3)	Median (g/cm^3)	Density Used in Block Model (g/cm^3)
Saprolite	221	37	1.66	2.64	2.18	2.13	2.13
	241	428	1.12	2.71	1.82	1.81	1.81
	341	5	1.69	1.86	1.77	1.73	-
	1011	36	1.42	2.64	2.11	2.02	1.73
	1021	89	1.07	2.46	1.84	1.86	2.02
	1031	13	1.00	2.11	1.87	1.92	1.86
	1041	100	1.02	2.28	1.84	1.87	1.92
	2201	10	1.76	2.23	1.93	1.85	1.87
	2401	283	1.37	2.81	1.94	1.85	1.85
	3101	1	2.12	2.12	2.12	2.12	1.85
	3401	43	1.62	2.57	2.01	1.95	1.95
	10101	2	1.81	2.00	1.91	1.81	1.81
	10201	10	1.28	2.11	1.87	1.93	1.93
	10301	59	1.65	2.36	2.02	2.04	2.04
	10401	33	1.55	2.17	1.90	1.91	1.91
	All	**1,149**	**1.00**	**2.81**	**1.89**	**1.86**	**1.90**
Transition	222	29	2.11	2.75	2.51	2.51	2.3
	242	573	1.37	3.03	2.30	2.35	2.3
	312	6	2.31	2.76	2.55	2.54	2.3
	342	24	2.05	2.77	2.40	2.40	2.3
	1012	35	2.09	2.72	2.47	2.52	2.5
	1022	198	1.48	2.96	2.16	2.13	2.1
	1032	15	1.90	2.61	2.25	2.22	2.3
	1042	157	1.59	2.77	2.17	2.16	2.2
	2202	36	1.77	2.77	2.37	2.41	2.5
	2402	158	1.38	3.07	2.30	2.33	2.4
	3102	10	2.04	2.68	2.27	2.24	2.3
	3402	113	1.69	2.80	2.33	2.38	2.4
	10102	18	1.76	2.53	2.03	1.88	2.3
	10202	41	1.59	2.22	1.88	1.85	1.9
	10302	79	1.80	2.77	2.19	2.11	2.1
	10402	142	1.39	2.55	2.04	2.03	2.1

Weathering	Domain Code	Number	Min (g/cm^3)	Max (g/cm^3)	Mean (g/cm^3)	Median (g/cm^3)	Density Used in Block Model (g/cm^3)
	All	**1,634**	**1.37**	**3.07**	**2.24**	**2.25**	**2.3**
Fresh Rock	243	3,834	1.62	3.48	2.72	2.74	2.7
	343	1,089	2.21	3.28	2.78	2.79	2.7
	413	181	2.70	3.01	2.77	2.78	2.8
	443	72	2.67	2.87	2.77	2.76	2.8
	513	60	2.70	2.91	2.79	2.80	2.8
	543	31	2.63	2.90	2.80	2.81	2.8
	623	73	2.61	2.96	2.79	2.78	2.8
	643	46	2.65	2.90	2.79	2.79	2.8
	1013	95	2.27	2.90	2.73	2.75	2.8
	1023	1,091	1.73	3.16	2.64	2.75	2.8
	1033	242	2.24	2.97	2.76	2.79	2.8
	1043	1,174	1.57	3.41	2.69	2.76	2.7
	2203	897	2.02	3.33	2.73	2.73	2.8
	2403	1,215	2.12	3.07	2.72	2.73	2.8
	3103	877	2.23	3.36	2.79	2.80	2.7
	3403	2,504	1.75	3.47	2.76	2.79	2.7
	4103	358	2.56	3.11	2.78	2.77	2.8
	4403	763	2.12	3.16	2.78	2.77	2.8
	5103	131	2.56	3.07	2.81	2.82	2.8
	5403	171	2.48	2.94	2.80	2.81	2.8
	6203	73	2.51	2.89	2.78	2.79	2.8
	6403	61	2.31	2.88	2.74	2.74	2.8
	10103	1	2.84	2.84	2.84	2.84	2.8
	10203	55	1.90	2.99	2.58	2.68	2.8
	10303	221	1.74	3.00	2.62	2.76	2.8
	10403	608	1.77	3.58	2.55	2.61	2.7
	All	**15,923**	**1.57**	**3.58**	**2.73**	**2.76**	**2.8**

14.1.1.6 VARIOGRAPHY

Mapping has highlighted at least three vein sets at the EMZ deposit. All vein orientations are mineralized and carry gold. Gold occurs as free particles within the veins and it is also inter-grown with arsenopyrite, either on vein margins, or in the host rocks. Disseminated arsenopyrite and gold mineralization rapidly decrease away from the veins. Direction in a small scale can be hard to identify. Variogram directions were mainly chosen in order to follow the global trend inside the lithology. For the east flank, the majority of the variograms are oriented using a dip of -45º towards 060N and for the west flank the variograms are oriented using a dip of -70º towards 060N. The direction was adjusted mainly in the nose of the flank. Variograms were computed in GEOVIA and done on the 5 m composites.

Variograms were modelled for each rock code (61). Grade control drilling was used in the calculation of the variograms. This allowed a better definition of the first structure of the variogram. Grade control data were put on the same support and the comparison between the two sets of data did not show a bias.

Down hole variograms were used to confirm the nugget effect values. Variogram maps were produced to establish the main continuity direction. The models were fit in GEOVIA mostly with two spherical structures.

The components of the modelled variograms are summarized in Table 14-9. Generally, the nugget effect is approximately 20% of the total variance. The range is approximately 10 m which is considered to be small.

TABLE 14-9 SEMI-VARIOGRAM PROFILES USED FOR THE EMZ DOMAINS

Variogram	Model Type	Anisotropy			1st Structure				2nd Structure				% Nugget
		Az	Dip	Az	Sill	x	y	z	Sill	x	y	z	
221_DVR	Spherical	330	10	150	0.3	11.7	10.6	10.3	0.6	23.6	21.3	20.7	37%
222_DVR	Spherical	150	10	330	0.1	12.2	12.2	12.2	0.0	29.5	29.5	29.5	54%
223_DVR	Spherical	60	-70	150	0.1	8.0	4.6	4.9	0.1	46.4	26.8	28.6	7%
241_DVR	Spherical	150	10	60	3.9	14.3	13.8	13.7	-	-	-	-	58%
242_DVR	Spherical	60	-45	150	6.7	10.4	9.5	10.4	1.5	44.5	40.6	44.5	15%
243_DVR	Spherical	60	-45	150	5.4	11.3	11.3	7.9	1.6	43.5	43.5	30.2	17%
312_DVR	Spherical	60	-70	150	0.0	10.1	7.3	6.4	0.0	18.6	13.5	11.7	3%
313_DVR	Spherical	60	-70	150	0.5	10.0	8.7	3.4	0.1	46.9	41.2	15.9	12%
341_DVR	Spherical	60	-45	150	0.0	11.1	9.3	5.6	0.0	27.8	23.5	14.2	23%
342_DVR	Spherical	60	-45	150	1.2	7.9	7.3	4.8	0.8	14.9	13.8	9.0	20%
343_DVR	Spherical	60	-45	150	1.6	10.8	10.8	10.8	0.7	48.4	48.4	48.4	21%
413_DVR	Spherical	60	-70	150	0.2	5.9	5.9	5.5	0.2	20.5	20.5	18.9	7%
443_DVR	Spherical	60	-45	150	0.4	7.2	7.1	7.2	0.2	34.2	33.7	34.2	25%
513_DVR	Spherical	60	-70	150	0.8	9.1	9.1	3.1	0.5	31.2	31.2	10.7	0%
543_DVR	Spherical	60	-45	150	0.1	13.4	9.7	4.0	0.1	45.3	33.0	13.5	33%
623_DVR	Spherical	60	-70	240	0.0	8.2	6.5	6.3	0.0	38.0	30.2	29.2	20%
643_DVR	Spherical	60	-45	150	0.1	7.4	4.3	7.4	0.1	29.5	17.3	29.5	16%
1011_DVR	Spherical	60	-70	150	0.0	8.7	2.6	8.7	0.1	30.6	9.4	29.3	6%
1012_DVR	Spherical	60	-70	150	0.0	10.2	10.2	8.8	0.0	22.7	22.7	19.6	0%

Variogram	Model Type	Anisotropy			1st Structure				2nd Structure				% Nugget
		Az	Dip	Az	Sill	x	y	z	Sill	x	y	z	
1013_DVR	Spherical	60	-70	150	0.0	8.5	8.5	4.1	0.0	21.9	21.9	10.6	0%
1021_DVR	Spherical	150	0	0	0.0	12.8	11.6	11.1					0%
1022_DVR	Spherical	60	-45	150	0.0	10.2	9.8	8.3	0.0	33.5	32.0	27.1	0%
1023_DVR	Spherical	60	-45	150	0.0	11.5	10.1	11.5	0.0	27.0	23.6	27.0	12%
1031_DVR	Spherical	60	-70	240	0.0	8.2	8.2	5.8	0.0	16.4	16.4	11.6	0%
1032_DVR	Spherical	60	-70	150	0.4	12.3	12.3	9.8	0.4	22.3	22.3	17.7	0%
1033_DVR	Spherical	60	-70	150	0.1	12.2	12.2	9.2	0.1	45.3	45.3	34.3	6%
1041_DVR	Spherical	150	0	240	0.1	8.4	6.9	4.7	0.3	43.8	36.1	24.6	39%
1042_DVR	Spherical	60	-45	150	0.3	11.0	8.5	9.3	0.1	31.2	24.2	26.3	39%
1043_DVR	Spherical	60	-45	150	0.4	7.9	7.9	7.4	0.4	38.9	38.9	36.7	20%
2201_DVR	Spherical	0	0	90	0.5	10.1	7.7	7.2	0.5	22.2	16.9	15.7	1%
2202_DVR	Spherical	150	0	0	0.6	11.0	10.4	9.6	0.2	25.7	24.4	22.4	19%
2203_DVR	Spherical	60	-70	150	0.4	13.2	7.3	8.8	0.1	24.8	13.6	16.6	3%
2401_DVR	Spherical	240	-45	330	1.0	9.9	4.9	4.6	0.6	29.5	14.5	13.9	27%
2402_DVR	Spherical	157	7	253	2.4	14.0	7.4	10.4					16%
2403_DVR	Spherical	150	0	0	2.0	17.6	9.0	13.5	2.0	17.6	9.0	13.5	13%
3101_DVR	Spherical	60	-70	18.237	0.0	10.7	10.7	10.7					0%
3102_DVR	Spherical	60	-70	240	0.0	12.4	10.3	4.0					26%
3103_DVR	Spherical	60	-70	150	1.3	11.8	5.5	5.0	1.5	33.7	15.7	14.3	0%
3401_DVR	Spherical	0	0	270	0.1	13.6	10.0	13.6	0.0	34.1	25.3	34.1	3%
3402_DVR	Spherical	0	0	270	0.1	7.2	3.2	7.2	0.2	15.3	6.8	15.3	8%
3403_DVR	Spherical	60	-45	150	1.9	10.4	7.3	5.4	1.1	46.4	32.6	24.0	14%
4103_DVR	Spherical	60	-70	28.433	0.9	9.2	6.3	9.2	0.4	56.7	39.2	56.7	20%
4403_DVR	Spherical	150	0	240	1.4	6.8	6.8	3.2	1.6	40.6	40.6	18.8	34%
5103_DVR	Spherical	60	-70	150	1.2	7.8	4.0	6.5	1.6	27.0	13.9	22.4	15%
5403_DVR	Spherical	90	0	0	1.6	16.5	8.9	13.4					33%
6000_DVR	Spherical	60	-50	150	0.6	8.6	8.6	4.5	0.4	46.3	46.3	23.9	3%
10101_DVR	Spherical	0	0	0	0.8	18.2	11.2	10.3					15%
10102_DVR	Spherical	114	-58	159.39	0.0	11.4	6.3	5.9	0.0	31.2	17.2	16.1	14%
10103_DVR	Spherical	119	-54	161.17	0.0	12.0	10.2	6.8	0.0	26.0	22.1	14.9	14%
10201_DVR	Spherical	60	-45	150	0.0	13.3	13.3	8.5	0.0	30.3	30.3	19.4	0%
10202_DVR	Spherical	150	0	60	0.0	6.7	4.1	3.8	0.0				0%
10203_DVR	Spherical	90	0	0	0.0	12.4	5.6	8.8	0.0	25.5	11.5	18.1	22%
10301_DVR	Spherical	60	-80	150	0.1	9.5	8.1	7.5	0.4	20.7	17.7	16.4	36%
10302_DVR	Exponential	60	-45	240	0.4	16.2	16.2	13.0	0.7	35.4	35.4	28.5	3%
10303_DVR	Spherical	60	-75	150	0.3	7.5	7.5	6.8	0.3	15.8	15.8	14.2	0%
10401_DVR	Spherical	60	-45	150	0.1	12.7	12.7	8.3	0.4	50.1	50.1	32.5	54%
10402_DVR	Spherical	60	-45	240	0.1	10.7	5.2	8.0	0.2	20.4	9.9	15.2	6%
10403_DVR	Spherical	60	-45	150	0.4	11.3	7.8	11.3	0.5	58.6	40.3	58.6	28%

14.1.1.7 BLOCK MODELLING

Block Model Parameters

A single block model was constructed for the EMZ deposit, including South EMZ, EMZ, and North Satellite areas. The block model covers an area large enough to manage the open pit developments and WRDs. The block model was developed using GEOVIA GEMS version 6.8.

The choice of block dimensions (10 m x 10 m x 10 m) is based on the existing drilling pattern (25 m x 25 m or 25 m x 50 m in some areas), mine planning considerations (10 m benches), current material selectivity, and the characteristics of the assay population. Table 14-10 presents the location and dimension settings of the block model.

TABLE 14-10 EMZ BLOCK MODEL PARAMETERS

Block Model Name	Orientation	Origin [1] (m)	Number of Columns, Rows, Levels	Block Size (m)	Rotation [2] (Degrees)
	East	185,400	150	10	
RES19Offic	North	1,589,400	600	10	30
	Elevation	270	48	10	

Notes:
1. In GEOVIA GEMS, the origin point stands at the southwest corner and highest level of the block model
2. For a positive value, the direction of rotation is counterclockwise around the elevation axis (Z)

A series of block model attributes were created during the block modelling estimation and incorporated into the block model project. The attributes containing the final results are presented in Table 14-11.

TABLE 14-11 EMZ FINAL BLOCK MODEL ATTRIBUTES

Attribute Name	Description
Rock Type	Rock codes of combined lithology and weathering (Refer to Table 14-12)
Weathering	Weathering rock codes: (1) Saprolite, (2) Transition, (3) Fresh Rock
Density	Density assigned to the block
Au_LIX	Interpolated gold grades
Categ	Resource Category: (1) Measured, (2) Indicated, (3) Inferred
Elevation	Surface Elevation Grids

Rock Type Models

The Weathering attribute was coded from the Saprolite, Transition, and Fresh Rock wireframes, and constituted a simple rock type attribute to be used in cases where a more detailed rock description is not required. A block was coded with a weathering rock code if 50% of its volume was located inside the weathering wireframe.

The Weathering attribute was used as a background code for the Rock Type attribute. Then the wireframe constraints (weathering and lithostructural domains), presented previously, were used to codify the Rock Type attribute. A block was assigned a domain rock code if its volume was a least 33.3% inside this domain. In the situation where a block is located in multiple domains, in the fold hinge for example where many domains meet, the highest percentage of volume (above the limit) prevails, unless precedence applies.

An order of priority, defined as precedence in GEOVIA GEMS, was set to all domains. Domains were modelled as juxtaposed (no overlaps) wireframes. The rock codes attributed from the lithostructural domains were adjusted afterward with the corresponding weathering code. The adjustments were made on the last digit of the code as follows: 1 for Saprolite, 2 for Transition, and 3 for Fresh Rock. The domain 220, for example, yielded rock codes 221 for Saprolite, 222 for Transition, and 223 for Fresh Rock. Details of the rock codes present in the Rock Type attribute are listed in Table 14-12.

In both Weathering and Rock Type attributes, the blocks which were located 99.9% above the pre-mining topography surface were defined as "Air" and coded 0.

TABLE 14-12 ROCK CODES FOUND IN THE EMZ ROCK TYPE ATTRIBUTE

Domain Code	Description	Precedence (1)	Rock codes used in model		
			Saprolite	**Transition**	**Fresh Rock**
100	Host	Background	101	102	103
220	Arenite	8	221	222	223
240	Arenite	3	241	242	243
310	Argillite	9	-	-	313
340	Argillite	4	-	-	343
410	Arenite	10	-	412	413
440	Arenite	5	-	442	443
510	Argillite	11	-	-	513
540	Argillite	6	-	-	543

Domain Code	Description	Precedence [1]	Rock codes used in model Saprolite	Transition	Fresh Rock
620	Argillite	12	621	622	623
640	Argillite	7	-	642	643
1010	Argillite	13	1011	1012	1013
1020	Argillite	14	1021	1022	1023
1030	Turbidite	15	1031	1032	1033
1040	Turbidite	16	1041	1042	1043
1000	Nose Intersection with Host		1001	1002	1003
2200	Nose Intersection with Arenite		2201	2202	2203
2400	Nose Intersection with Arenite		2401	2402	2403
3100	Nose Intersection with Argillite		3101	3102	3103
3400	Nose Intersection with Argillite		3401	3402	3403
4100	Nose Intersection with Arenite		-	-	4103
4400	Nose Intersection with Arenite		-	-	4403
5100	Nose Intersection with Argillite		-	-	5103
5400	Nose Intersection with Argillite		-	-	5403
6200	Nose Intersection with Argillite		-	-	6203
6400	Nose Intersection with Argillite		-	-	6403
10100	Nose Intersection with Argillite		10101	10102	10103
10200	Nose Intersection with Argillite		10201	10202	10203
10300	Nose Intersection with Turbidite		10301	10302	10303
10400	Nose Intersection with Turbidite		10401	10402	10403

Notes:

[1] Precedence are priority levels attributed to domain such that the smallest precedence often corresponds to the youngest lithology/domain

Density Model

Default values determined from the median values, as presented previously, were first set into each domain. Table 14-13 lists the background densities used in the block model.

TABLE 14-13 DEFAULT DENSITY VALUES USED IN THE EMZ BLOCK MODEL

Saprolite		Transition		Fresh Rock	
Rock Code	Density	Rock Code	Density	Rock Code	Density
101	1.9	102	2.3	103	2.8
221	2.1	222	2.3	223	2.7
241	1.9	242	2.3	243	2.7
311	1.9	312	2.3	313	2.8
341	1.8	342	2.3	343	2.8
411	NE	412	NE	413	2.8

Saprolite		Transition		Fresh Rock	
Rock Code	Density	Rock Code	Density	Rock Code	Density
441	NE	442	NE	443	2.8
511	NE	512	NE	513	2.8
541	NE	542	NE	543	2.8
621	NE	622	NE	623	2.8
641	NE	642	NE	643	2.8
1011	2.0	1012	2.5	1013	2.8
1021	1.9	1022	2.1	1023	2.7
1031	2.0	1032	2.3	1033	2.8
1041	1.9	1042	2.2	1043	2.8
2201	2.1	2202	2.5	2203	2.7
2401	1.8	2402	2.4	2403	2.7
3101	2.0	3102	2.3	3103	2.8
3401	1.9	3402	2.4	3403	2.8
4101	NE	4102	NE	4103	2.8
4401	NE	4402	NE	4403	2.8
5101	NE	5102	NE	5103	2.8
5401	NE	5402	NE	5403	2.8
6201	NE	6202	NE	6203	2.8
6401	NE	6402	NE	6403	2.8
10101	2.0	10102	2.3	10103	2.8
10201	2.0	10202	1.9	10203	2.7
10301	2.0	10302	2.1	10303	2.8
10401	1.9	10402	2.1	10403	2.6

A density interpolation was carried out using an Ordinary Kriging (OK) interpolator in combination with flat search ellipses with dimensions of 100 m x 100 m x 50 m (X, Y, Z). The details of the density interpolation are described in Tables 14-14 and 14-15. The results, where estimated, overwrote the background density values previously entered.

TABLE 14-14 INTERPOLATION DETAILS FOR EMZ DENSITY ESTIMATION

Block Model Parameters	Description
Data Source	Density Measurements > 1 and < 3.6 t/m^3 from DD and RCD Hole Types
Interpolation Method	Ordinary Kriging
Minimum/Maximum Sample	4/30
Maximum Sample per Hole	No maximum defined
Boundary Type	Soft and Hard Boundaries
Number of Iterations	1
Search Ellipses (X, Y, Z)	100 m x 100 m x 50 m
High Grade Transition Limit	Not used

TABLE 14-15 SOFT AND HARD BOUNDARIES USED FOR THE EMZ DENSITY INTERPOLATION

Target Rock Code	Limit Target Rock Codes			
221	221	241	2201	2401
222	222	242	2202	2402
223	223	243	2203	2403
241	241	221	2201	2401
242	242	222	2202	2402
243	243	223	2203	2403
311	311	341	3101	3401
312	312	342	3102	3402
313	313	343	3103	3403
341	341	311	3101	3401
342	342	312	3102	3402
343	343	313	3103	3403
413	413	443	4103	4403
443	443	413	4103	4403
513	513	543	5103	5403
543	543	513	5103	5403
623	623	643	6203	6403
643	643	623	6203	6403
1011	1011	1021	10101	10201
1012	1012	1022	10102	10202
1013	1013	1023	10103	10203
1021	1021	1011	10101	10201
1022	1022	1012	10102	10202
1023	1023	1013	10103	10203
1031	1031	1041	10301	10401
1032	1032	1042	10302	10402
1033	1033	1043	10303	10403

Target Rock Code	Limit Target Rock Codes			
1041	1041	1031	10301	10401
1042	1042	1032	10302	10402
1043	1043	1033	10303	10403
2201	221	241	2201	2401
2202	222	242	2202	2402
2203	223	243	2203	2403
2401	241	221	2201	2401
2402	242	222	2202	2402
2403	243	223	2203	2403
3101	311	341	3101	3401
3102	312	342	3102	3402
3103	313	343	3103	3403
3401	341	311	3101	3401
3402	342	312	3102	3402
3403	343	313	3103	3403
4103	413	443	4103	4403
4403	443	413	4103	4403
5103	513	543	5103	5403
5403	543	513	5103	5403
6203	623	643	6203	6403
6403	643	623	6203	6403
10101	1011	1021	10101	10201
10102	1012	1022	10102	10202
10103	1013	1023	10103	10203
10201	1021	1011	10101	10201
10202	1022	1012	10102	10202
10203	1023	1013	10103	10203
10301	1031	1041	10301	10401
10302	1032	1042	10302	10402
10303	1033	1043	10303	10403
10401	1041	1031	10301	10401
10402	1042	1032	10302	10402
10403	1043	1033	10303	10403

Grade Estimation Methodology

Grade estimation for the EMZ deposit was done using OK and 5 m composites tagged by domain codes. The blocks are interpolated by domains (Target Rock Code) from composites coded within this domain only (hard boundary) or with other specified domains (soft boundary). The nature of the boundaries (soft or hard) between domains is detailed by restrictive rock codes presented in Table 14-16 and is largely derived from the statistical relation between composites' domain populations.

Estimation is carried out in three passes, however, only the first two passes are used for mineral reporting. The passes are anisotropic according to the main orientation of the lithology. The east flank is oriented at an azimuth of 60º with a dip of -45º, and the west flank is dipping at -70º. The first pass is 40 m along the main azimuth and the dip 20 m perpendicular to the bedding. The second pass respects the same orientation using 60 m and 30 m. The neighborhood is defined by a minimum of five samples and a maximum of 22 samples. A high grade restriction is applied in the second pass.

TABLE 14-16 LIST OF EMZ ROCK CODES TREATED BY THE INTERPOLATION PROFILES AND ASSOCIATED VARIOGRAPHY PROFILES

Interpolation Profile	Target Rock Code	Semi Variogram	Limit Target Rock Code						
18DUARG	623	623_DVR	623	6203	6403				
	543	543_DVR	543	5403	5103				
	513	513_DVR	513	5103	5403				
	643	643_DVR	643	6403	6203				
	413	413_DVR	413	4103	4403				
	443	443_DVR	443	4403	4103				
	6203	6000_DVR	6203	6403	623	643			
	5403	5403_DVR	5403	5103	513	543			
	5103	5103_DVR	5103	5403	513	543			
	4103	4103_DVR	4103	4403	413	443			
	4403	4403_DVR	4403	4103	443	413			
	6403	6000_DVR	6403	6203	643	623			
18LARG	311	312_DVR	311	3101	312	3102			
	312	312_DVR	312	311	3101	3102	313	3103	
	313	313_DVR	313	3103	312	3102			
	341	341_DVR	341	342	3401	3402			
	342	342_DVR	342	343	341	3401	3402		
	343	343_DVR	343	342	3402	3403			
	3101	3101_DVR	3101	3401	311	3102	312	3402	
	3102	3102_DVR	3102	311	312	313	3101	3103	3401
	3103	3103_DVR	3103	312	313	3102	3402	3403	
	3401	3401_DVR	3401	341	342	3402	3101	3102	
	3402	3402_DVR	3402	341	342	343	3101	3102	3103
	3403	3403_DVR	3403	342	343	3102	3103	3402	
18MAREN	221	221_DVR	221	222	2201	2203			
	222	222_DVR	222	221	223	2201	2202	2203	
	223	223_DVR	223	222	2202	2203			
	241	241_DVR	241	242	2401	2402			
	242	242_DVR	242	241	243	2401	2402	2403	

Interpolation Profile	Target Rock Code	Semi Variogram	Limit Target Rock Code						
	243	243_DVR	243	242	2402	2403			
	2201	2201_DVR	2201	221	222	2202	2401	2402	
	2202	2202_DVR	2202	221	222	223	2201	2203	2401
	2203	2203_DVR	2203	222	223	2202	2402	2403	
	2401	2401_DVR	2401	241	242	2201	2202	2402	
	2402	2402_DVR	2402	241	242	243	2201	2202	2203
	2403	2403_DVR	2403	242	243	2202	2203	2402	
18NORD	1011	1011_DVR	1011	10101	1012	10102			
	1012	1012_DVR	1012	1011	1013	10101	10102	10103	
	1013	1013_DVR	1013	1012	10103	10102			
	1021	1021_DVR	1021	1022	10201	10202			
	1022	1022_DVR	1022	1021	1023	10201	10202	10203	
	1023	1023_DVR	1023	1022	10202	10203			
	10101	10101DVR	10101	1011	1012	10102	10201	10202	
	10102	10102DVR	10102	1011	1012	1013	10101	10103	
	10103	10103DVR	10103	1012	1013	10102	10202	10203	
	10201	10201DVR	10201	1021	1022	10101	10102	10202	
	10202	10202DVR	10202	1021	1022	1023	10101	10102	
	10203	10203DVR	10203	1022	1023	10102	10103	10202	
	1031	1031_DVR	1031	1032	10301	10302			
	1032	1032_DVR	1032	1031	1033	10301	10302	10303	
	1033	1033_DVR	1033	1032	10302	10303			
	1041	1041_DVR	1041	1042	10401	10402			
	1042	1042_DVR	1042	1041	1043	10401	10402	10403	
	1043	1043_DVR	1043	1042	10402	10403			
	10301	10301DVR	10301	1031	1032	10302	10401	10402	
	10302	10302DVR	10302	1031	1032	1033	10301	10303	
	10303	10303DVR	10303	1033	1032	10302	10402	10403	
	10401	10401DVR	10401	1041	1042	10301	10302	10402	
	10402	10402DVR	10402	1041	1042	1043	10301	10302	
	10403	10403DVR	10403	1042	1043	10302	10303	10402	

14.1.1.8 CLASSIFICATION AND RESOURCE REPORTING

The CIM (2014) definitions provide standards for the classification of Mineral Resource and Mineral Reserve estimates into various categories. The category to which a resource or reserve estimate is assigned depends on the level of confidence in the geological information available on the mineral deposit, the quality and quantity of data available, the level of detail of the technical and economic information which has been generated about the deposit, and the interpretation of that information. Under CIM (2014) definitions:

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 limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. The estimate is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings, and drill holes.

An "*Indicated Mineral Resource*" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with sufficient confidence to allow appropriate the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

A "*Measured Mineral Resource*" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

In addition, the resource classification is based on the robustness of the various data sources available, including:

- Quality and reliability of drilling and sampling data
- Distance between sample points (drilling density)
- Confidence in the geological interpretation
- Continuity of the geological structures and the continuity of the grade within these structures
- Variogram models and their related ranges (first and second structures)
- Statistics of the data population
- Quality of assay data
- Tonnage factor

Based on these criteria, the resources have been classified according to a data search used to estimate each block. The main classification criteria are based on an estimation pass described below. In addition to these passes, a wireframe has been constructed in the west flank and at depth of the deposit to capture zones with lower geological confidence. This wireframe corresponds more or less to the turbidites of the west flank. The wireframe was also prolongate at depth to minimize the extrapolation of grade of the deepest drill hole in section. All blocks inside the wireframe were assigned to a lower classification. Additionally, all resource blocks must show reasonable prospects for eventual economic extraction. In the case of the EMZ deposit, the resource blocks were contained within a pit shell based on the mining costs, the metallurgical parameters, and the financial parameters used for the latest LOM plan.

Measured Mineral Resources were previously defined by blocks located within 10 m of at least three holes, including grade control RC (GC) holes. However, the volume defined is so close to current work faces that they are fully depleted by the effective date of this Technical Report. Consequently, no Measured Mineral Resources were defined.

Indicated Mineral Resources encompassed all blocks estimated in the first estimation pass using composites from a minimum of three different drill holes within domains of soft and hard boundaries. Isolated inferred blocks inside the indicated zone were changed to indicated.

Inferred Mineral Resources corresponded to the blocks estimated in the second pass for which composites from a minimum of one drill hole were interpolated within domains of soft and hard boundaries.

14.1.1.9 BLOCK MODEL VALIDATION

Multiple validations were completed on the EMZ deposit block model. The process included visual checks, statistical validation, comparison of estimations issued from different interpolation methods, swath plots, and comparisons with previously validated models. An external audit was also carried by the Amec Foster Wheeler's (now Wood) Vancouver office in 2017.

Visual Validation

The visual checks consisted of visualizing slices of the block model (section and plan views) with domain wireframes, composites, and drill hole information. The data source was visually compared with the different model attributes (rock type, density, and gold grades) throughout the deposit.

It was found that due to the juxtaposed wireframes modelling and the choice of updating rock type blocks using a minimum percentage of 33.3% inside wireframes, some blocks inside the fold structure were left blank. The unfilled blocks that were expected to receive a rock code were corrected by removing, for these blocks only, the minimum re-assign percentage limit. This way, the blank blocks were coded with the highest proportion of the wireframe in which they were located.

Also, it was observed that the method used to assign rock codes to blocks favoured the outer bed of the fold structure. More blocks are being coded on the exterior shell of the fold. The minimum percent of 33.3% cannot truly apply to the blocks within the interior beds of the folded structure as they will all generally be coded during the process.

The visual verification of the grade attribute outlined that the wireframes occasionally obstruct the distribution of the grade. The lithostructural wireframes are representative of the geological model but might not be the most accurate mineralization model system locally. The actual folded shell structure appeared to adequately constrain the mineralization. However, within the shell, the modelled bedding sometimes divides what seems to be a continuous mineralized interval in a way that is not necessarily concordant with the boundary's restrictions. The grade distribution was locally influenced by this aspect of the modelling. More specifically the mineralization between the west and the east flank is strongly influenced by the position of the split which is well defined in the log. The geometric definition of the nose is also interpretative, the exact location of such geological feature is well represented for scale of the pit but is not well represented locally. Grade distribution is highly affected by the hard boundaries in the nose between the layers and may vary as the mining advances.

Visually, the models (Rock Type, Density, and Au Grade) were found to be globally representative of the known geological and structural controls of mineralization for the EMZ deposit.

Statistical Validation

A statistical comparison between composites used in the interpolation and interpolated block grades was performed to evaluate if samples used in the estimation are well represented in the block model. Table 14-17 summarizes the comparison of statistics between the mean grade of the declustered composites (using Nearest Neighbour) with block mean value by rock type. A successful grade interpolation protocol will result in block grade estimates that demonstrate a minimum amount of bias. Comparison was done with the weight average of the nose with the flank of the same unit.

Overall, there is no significant bias between the grade of the composites and the estimated grade. This statistical analysis demonstrates that the block model provides a reasonable estimate of the Mineral Resources of the EMZ deposit. The zone which showed the largest difference has a very low impact on the Mineral Resource.

TABLE 14-17 COMPARISON OF EMZ KRIGED BLOCKS AND MEAN COMPOSITE GRADES

	RES OFF (OK)						Nearest Neighbor					
	Flank			Nose		Flank & Nose	Flank		Nose		Flank & Nose	Difference
Domain	No.	Mean	Domain	No.	Mean	Mean Pond	No.	Mean	No.	Mean	Mean Pond	Flank & Nose
221	450	0.148	2201	2,381	0.426	0.382	450	0.215	2,381	0.429	0.395	0.97
222	528	0.088	2202	2,049	0.404	0.339	528	0.073	2,049	0.456	0.378	0.90
223	8,825	0.104	2203	15,489	0.375	0.277	8,825	0.092	15,489	0.374	0.272	1.02
241	9,345	0.925	2401	6,531	0.487	0.745	9,345	0.937	6,531	0.458	0.740	1.01
242	6,291	0.867	2402	2,596	0.613	0.793	6,291	0.817	2,596	0.614	0.758	1.05
243	29,000	0.902	2403	10,400	0.761	0.865	29,000	0.888	10,400	0.762	0.855	1.01
312	43	0.068	3102	664	0.124	0.121	43	0.023	664	0.107	0.102	1.18
313	8,632	0.278	3103	13,032	0.546	0.439	8,632	0.282	13,032	0.571	0.456	0.96
341	285	0.096	3401	3,040	0.239	0.227	285	0.105	3,040	0.237	0.226	1.00
342	666	0.101	3402	3,341	0.234	0.212	666	0.076	3,341	0.223	0.199	1.07
343	17,946	0.328	3403	24,674	0.52	0.439	17,946	0.332	24,674	0.51	0.435	1.01
413	1,339	0.322	4103	8,556	0.472	0.452	1,339	0.311	8,556	0.523	0.494	0.91
443	1,414	0.49	4403	21,501	0.567	0.562	1,414	0.513	21,501	0.581	0.577	0.97
513	62	0.756	5103	2,445	1.204	1.193	62	0.570	2,445	1.237	1.221	0.98
543	120	0.236	5403	3,849	0.59	0.579	120	0.206	3,849	0.529	0.519	1.12
623	331	0.315	6203	930	0.678	0.583	331	0.245	930	0.638	0.535	1.09
643	149	0.448	6403	1,031	0.545	0.533	149	0.164	1,031	0.629	0.570	0.93
1011	15	0.054	10101	120	0.045	0.046	15	0.089	120	0.037	0.043	1.08
1012	58	0.056	10102	228	0.083	0.078	58	0.071	228	0.058	0.061	1.28
1013	66	0.014	10103	1,116	0.08	0.076	66	0.022	1,116	0.085	0.081	0.94
1021	6,883	0.073	10201	653	0.124	0.077	6,883	0.075	653	0.145	0.081	0.96
1022	4,981	0.052	10202	1,684	0.239	0.099	4,981	0.044	1,684	0.25	0.096	1.03
1023	24,497	0.031	10203	3,155	0.195	0.050	24,497	0.029	3,155	0.177	0.046	1.08
1031	265	0.158	10301	2,563	0.257	0.248	265	0.192	2,563	0.244	0.239	1.04
1032	578	0.097	10302	2,336	0.228	0.202	578	0.094	2,336	0.259	0.226	0.89
1033	4,082	0.115	10303	9,617	0.274	0.227	4,082	0.104	9,617	0.27	0.221	1.03
1041	7,418	0.199	10401	1,486	0.227	0.204	7,418	0.206	1,486	0.193	0.204	1.00
1042	3,893	0.145	10402	2,105	0.224	0.173	3,893	0.133	2,105	0.209	0.160	1.08
1043	24,244	0.078	10403	12,200	0.206	0.121	24,244	0.082	12,200	0.211	0.125	0.97
All	**322,918**	**0.403**				**0.403**	**322,918**	**0.402**		**2594**	**0.402**	**1.00**

Validation Using Different Interpolation Methods

The validation of the block model was also done using Inverse Distance Squared (ID^2) and Inverse Distance Cubed (ID^3) interpolation as well as multiple neighbourhood strategy to compare with the OK estimate. The same set of composites, search ellipse, and settings were used and only the interpolation technique differed. The results were compared visually. Gold content in the different models is similar.

Swath Plots

Swath plots were generated to assess the correlation between composites used in the interpolation and the total gold content estimated in the blocks. Swath plots were produced in northing and easting vertical sections, and elevation. This validation method works as a visual means to identify possible bias in the interpolation (OK in blue). Swath plots were produced for all of the interpolated blocks, by classification and by rock type. This was done to check the consistency of the interpolation by rock type. Swath plots were constructed using the long-range block model compared to declustered five metre composites (in orange) (using Nearest Neighbour interpolation).

The following swath plots are generated with Measured, Indicated, and Inferred Mineral Resources within a US$1,500/oz Au pit shell. Figures 14-7, 14-8, and 14-9 illustrate swath plots for interpolated blocks by easting, northing, and elevation. The figures show a reasonable to a good correlation between blocks and composites.

FIGURE 14-7 SWATH PLOT FOR EASTING

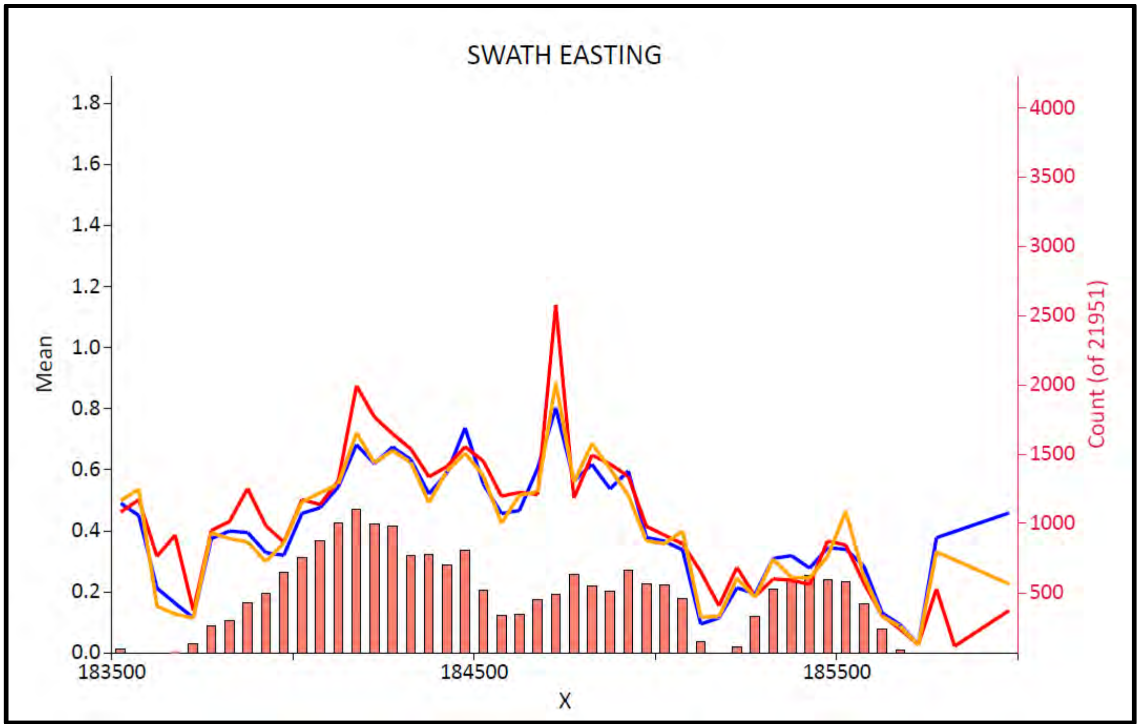


FIGURE 14-8 SWATH PLOT FOR NORTHING

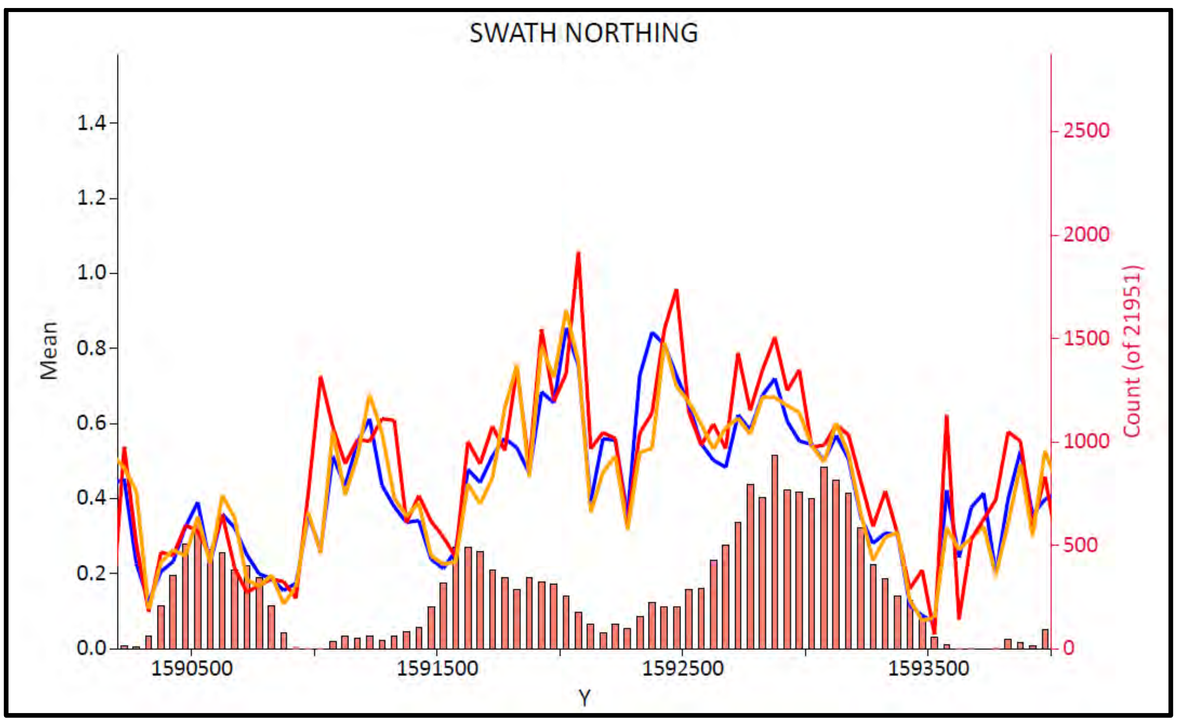
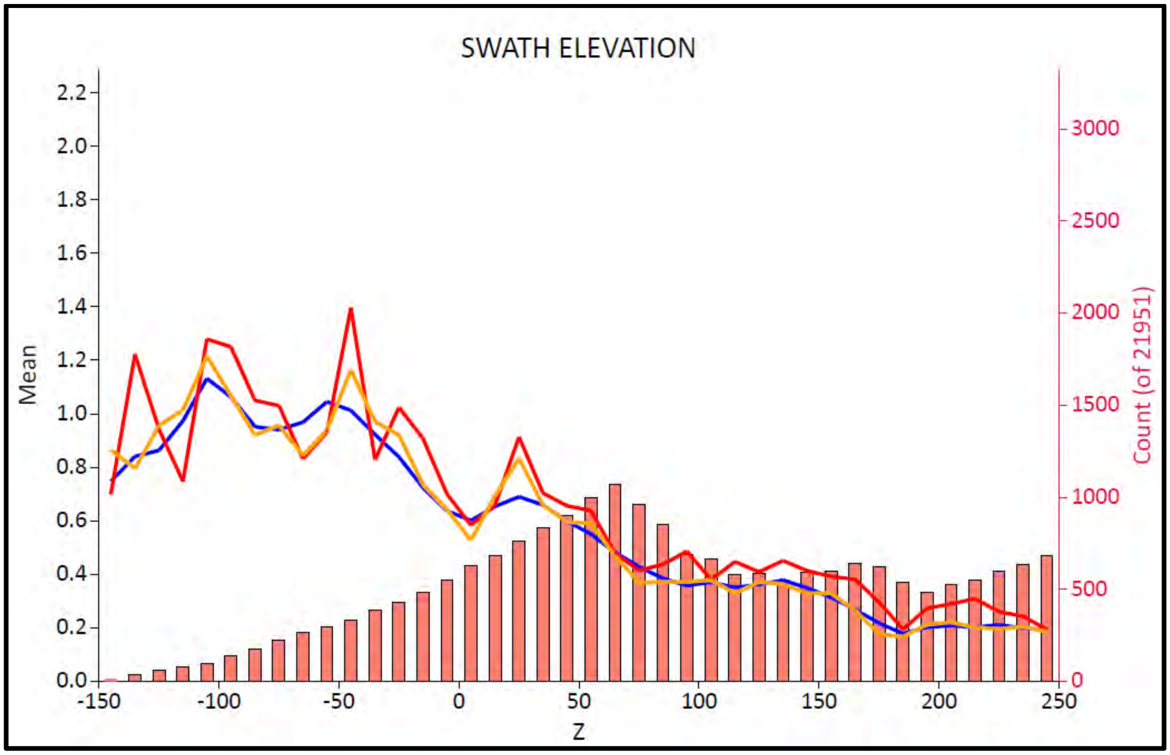


FIGURE 14-9 SWATH PLOT ELEVATION



14.1.1.10 RECONCILIATION

Essakane S.A. uses a reconciliation methodology developed by Parker (2012). Reconciliation factors F1, F2, and F3 apply for tonnage, grade, and ounces. Essakane S.A. has added another factor called F1.5. The reconciliation factors are described below:

- F1: Ratio between the short range model and the long term model
- F1.5: Ratio between the dispatch system (Wenco) and short-range model
- F2: Ratio between the CIL plant feed and Wenco
- F3 = F1 * F1.5 * F2, which gives the ratio of the reserve to CIL plant production

To be considered acceptable, the reconciliation ratio needs to be with 10% (between 0.9 and 1.1).

Reconciliation data was used in order to calibrate the model. Block models resulting from the different interpolation strategies were tested against reconciliation data from January 2016 to October 2018. The calibration was carried out mainly by changing the neighbourhood used for the interpolation and adjusting the geological wireframes.

When comparing the end of year (EOY) 2018 block model directly to the Wenco result with the previous model, an improvement was observed for predictability of tonnage and grade. From the last model the tonnage factor went from 0.94 to 0.89, and the grade factor went from 1.14 to 1.06. Multiple tests were carried out on different interpolation strategies. The selection of interpolation strategy used for the EOY block model is justified with the improvement of the reconciliation factor using this model.

14.1.2 FALAGOUNTOU DEPOSITS

14.1.2.1 DATA

The current Mineral Resource estimates on the Falagountou West and Wafaka deposits are updates of the GMSI March 2017 models. New drilling for the Falagountou deposits is shown in Figure 14-10. The database includes geotechnical and lithology logging information as well as gold assay and density sample results of the holes drilled on the Falagountou deposits. The current Falagountou deposits resource estimates are derived exclusively from this database. The responsible QP reviewed the information stored in the database and found it to be in good standing.

Since the previous estimate, 52 holes have been added to the database. The GEOVIA GEMS project consists of 1,243 holes for a total of 135,625 m of different types of drilling covering the Falagountou deposits and exploration areas around these deposits. Table 14-18 details the series of holes by type and year of drilling. Note that because AC and RAB sampling is more subject to segregation bias, their results are not used in the estimate process. They were used to guide geological and ore zone modelling, however, no vertex snapping was done on these types of holes.

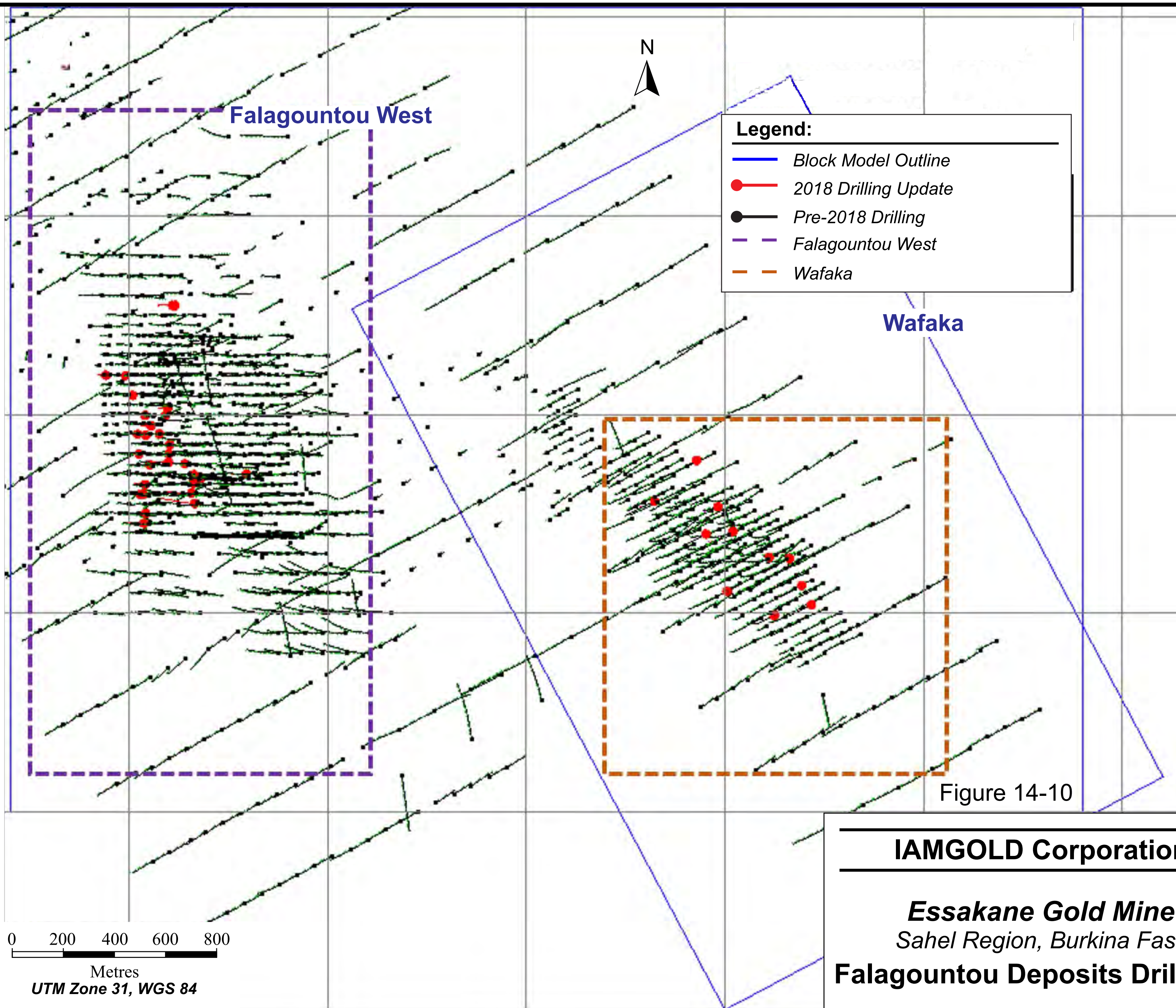


Figure 14-10

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Falagountou Deposits Drill Plan

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation

TABLE 14-18 FALAGOUNTOU WEST AND WAFAKA RESOURCE DATABASE

Hole Type	Year	Series	Number of Holes	Metres Drilled
DDH	2018	MFDD0116 - MFDD126	11	1,741
	2017	MFDD0113 - MFDD115	3	515
	2015	MFDD0103 - MFDD112	10	1,727
	2014	MFDD0013 - MFDD0102	90	13,652
	2012	MFDD0001 - MFDD0012	12	3,440
	2011	FDD0043 - FDD0068	26	5,780
	2008	FDD0025 - FDD0042	18	2368
	2006	FDD0016 - FDD0024	9	1,384
	2004	FDD0012 - FDD0015	4	819
RC	2018	MFRC0552- MFRC0581	30	2,180
	2016	MFRC0383 - MFRC0551	169	16,632
	2015	MFRC0245 - MFRC0382	139	14,457
	2014	MFRC0131 - MFRC0244	116	10,135
	2013	MFRC0042 - MFRC0130	89	11,239
	2012	MFRC0001 - MFRC0041	41	3,769
		FRC1908 - FRC0251	144	20,396
	2011	FRC1817 - FRC1907	91	12,497
	2010	FRC1772 - FRC1816	45	6,742
	2008	FRC1734 - FRC1771	38	2,822
	2006	FRC1704 - FRC1730	27	3,405
		FRC1732 - FRC1733	2	325
	2004	FRC0568 - FRC0571	4	300
		FRC0635 - FRC0670	36	3,228
		FRC0780 - FRC0800	21	1,493
		FRC0901 - FRC0946	46	3,861
	2003	FRC0428 - FRC0429	2	215
		FRC0467 - FRC0469	3	208
		FRC0478 - FRC0490	13	825
	1995	FRC0001 - FRC0003	3	139
RCD	2006	FRC1731D	1	225
Total			**1,243**	**135,625**

14.1.2.2 DRILL HOLE SPACING

The drill hole spacing covering the Falagountou West deposit illustrates a pattern, on vertical sections, of between 15 m to 50 m. Drilling is mostly located along east-west sections with 25 m spacings in the centre of the deposit, and up to 50 m spacings on the northern and southern fringes where the 2015 drilling update took place. The vertical sections are perpendicular to mineralized horizons with a strike oriented along an approximate north-south axis.

Drill hole spacing on the Wafaka deposit presents a sparser grid spacing compared to the Falagountou West deposit. Drill holes are spaced from 50 m to 75 m on sections. The 25 m to 200 m spaced sections are oriented at 060° and are perpendicular to mineralization, which strikes along a 150°- 330° axis.

Drill hole spacings on the Falagountou West and Wafaka deposits are judged as adequate to develop a reasonable model of the mineralization distribution and to quantify its volume and quality with an acceptable level of confidence. A cross section example of both areas is presented in Figure 14-11.

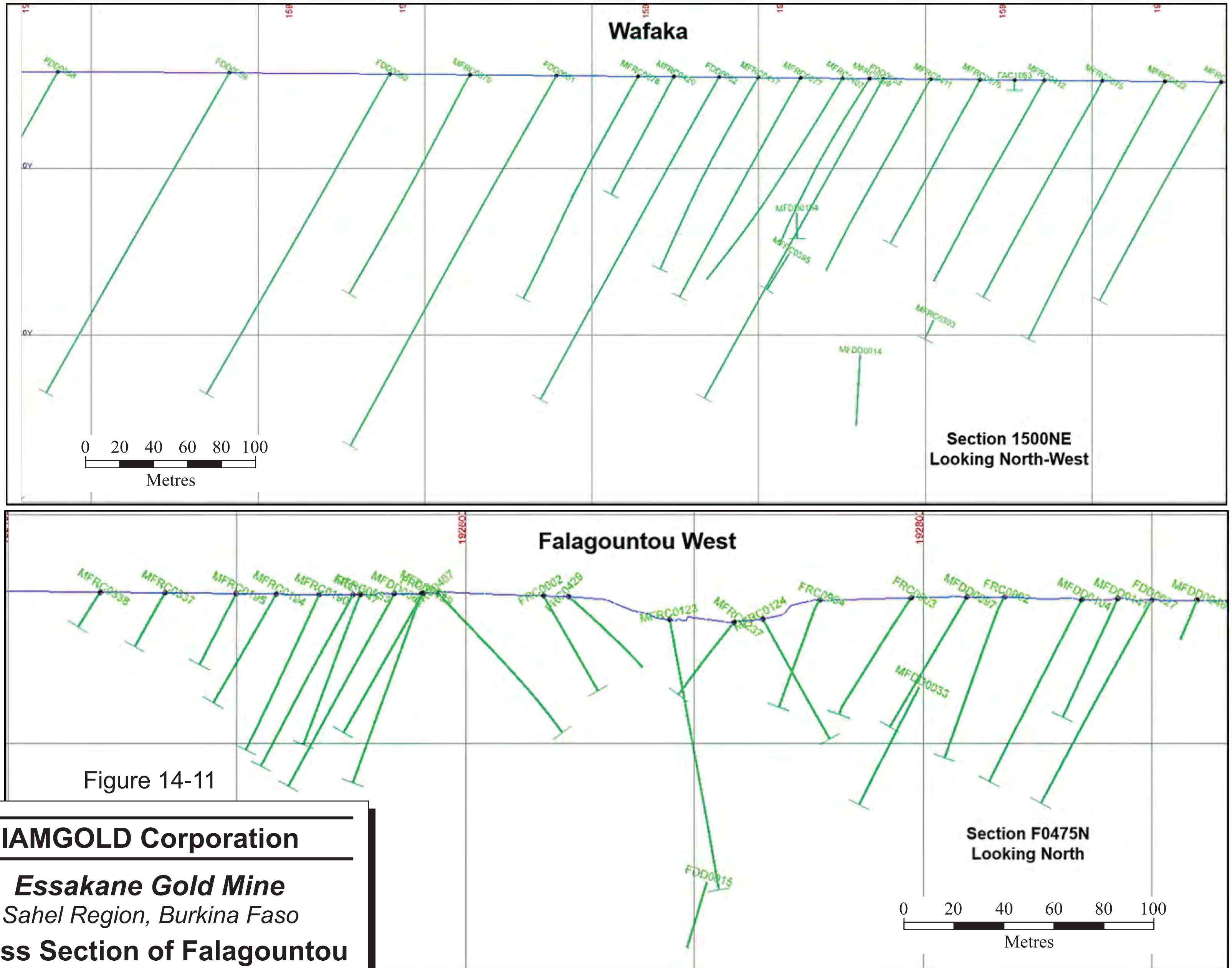


Figure 14-11

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Cross Section of Falagountou West and Wafaka Deposits

January 2020

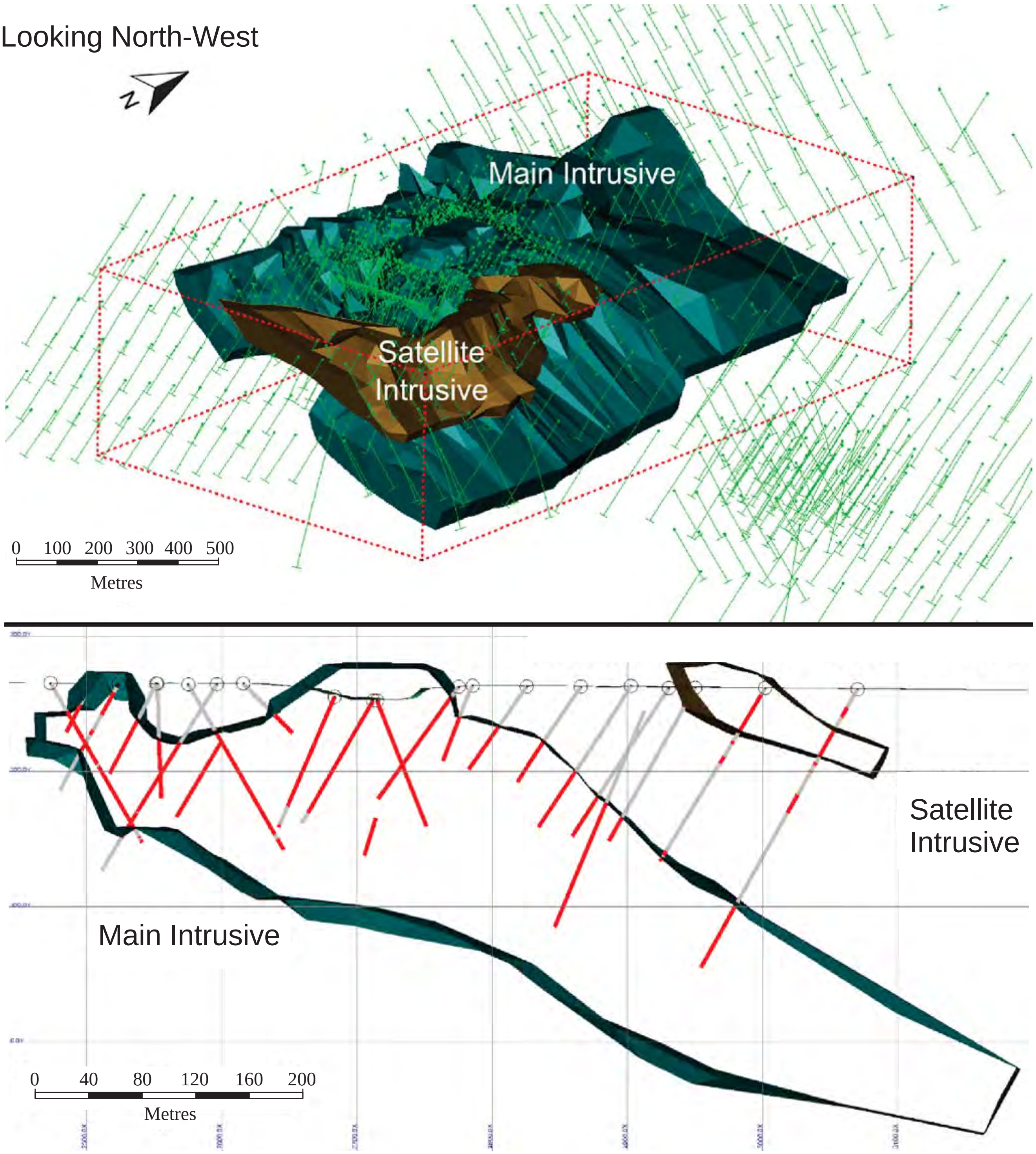
Source: IAMGOLD, 2019.

14.1.2.3 MODELLING

Numerous 2D and 3D modelling elements such as lithology, weathering, and mineralization solids were generated for the purpose of the current resource estimate using GEOVIA GEMS version 6.8.

Lithology Solids

For the Falagountou West deposit, two intrusive solids were designed from the lithological information found in the database. On each section, the intrusive rock contour was drawn and from this series of contour lines, the Main intrusive and Satellite intrusive solids were generated (Figure 14-12). The folded contact between intrusive and sedimentary rocks guided the shape of the mineralization zones. The 2018 drilling update allowed the rock model to be refined, as well as to extend the solid modelling. Rock codes utilized in the geological model are presented in Table 14-19.



Section F0450N - Looking North

UTM Zone 31, WGS 84

Figure 14-12

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

**Intrusive Solid -
Falagountou West Deposit**

January 2020

Source: IAMGOLD, 2019.

TABLE 14-19 FALAGOUNTOU WEST ROCK CODE DESCRIPTION

Rock Code	Description
5	Air
9	Overburden
1000	Sediments
2000	Main Intrusive
2100	Satellite Intrusive

Weathering Wireframes

For each DD, RC, and RCD hole in the database encompassed inside the Falagountou West and Wafaka deposits, points were created to mark the beginning and the end of the following weathering layers: regolith, saprolite, transition, and fresh rock. The weathering intervals were defined using the following information, in order of priority: density measurements, then hardness observations. The saprolite layer was limited to density measurements below 2.00 t/m^3, while the transition layers were limited to density values between 2.00 t/m^3 and 2.55 t/m^3. Where no density measurements were available, as with RC holes, the weathering contacts were determined from the hardness information. Brown’s rock strength classification, used to categorize the association between hardness and weathering, was presented previously in Table 14-4. Based on the relationships between density measurements and hardness information, GMSI reclassified the S5-hardness in saprolite rather than transition.

The points defining the regolith bottom limit were created from the lithology information. A point was placed at the collar of the hole if the regolith interval was lacking. Regolith intervals were later merged with the saprolite layer.

Where necessary, the intervals of weathering were divided into sub layers to accommodate the density variation through the weathering type. The saprolite was kept as one single bed (Saprolite 1) as the layer is relatively thin and density values have little variation. The transition intervals were divided into three equal length sub layers: Transition 1, Transition 2, and Transition 3. The fresh rock was separated into two beds: Rock 1 and Rock 2. The limit between Rock 1 and Rock 2 was established at 20 m below the bottom limit of the Transition 3 (or contact between Transition and Rock) sublayer.

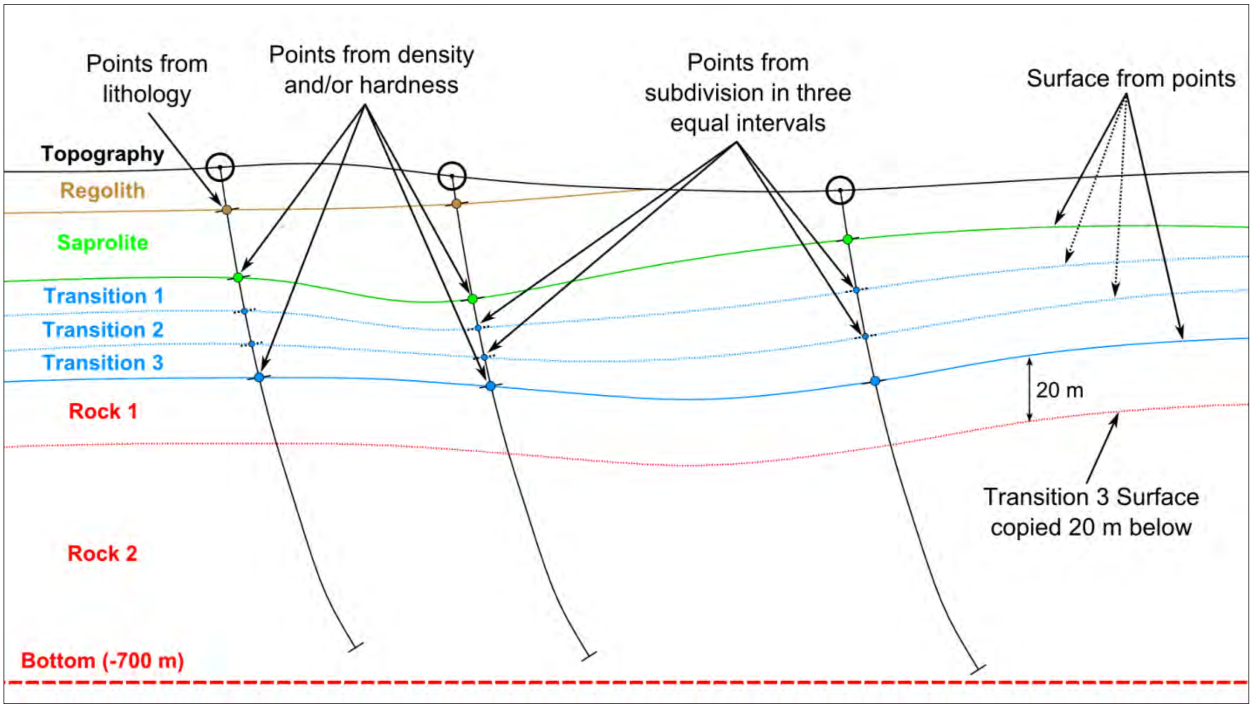
The weathering surfaces were created from their specific set of points except for Rock 1 surface, which was generated by copying the Transition 3 surface down 20 m and the Rock 2 bottom surface which is a flat surface located below the deepest hole.

As a last step, solids were constructed from the surfaces. Table 14-20 and Figure 14-13 summarize the procedure for the construction of the weathering solids.

TABLE 14-20 SUMMARY OF THE MANIPULATION EXECUTED TO CREATE THE FALAGOUNTOU AND WAFAKA WEATHERING SOLIDS

Weathering Layer	Density Measurements Limits (t/m^3)	Code	Hardness Category	Creation Method
Regolith	-	9	-	Points from Lithology → Surface → Solid
Saprolite	0 to 2.00	1	S1, S2, S3, S4, S5	Points from Density and/or Hardness → Surface → Solid
Transition	2.00 to 2.55	2	S6, R0, R1	Points resulting from Transition interval division → Surface → Solid
				Points from Density and/or Hardness → Surface → Solid
Rock	> 2.55	3	R2, R3, R4, R5, R6	Transition 3 Surface copied down 20 m → Solid
				Flat Surface at Elevation -700 m → Solid

FIGURE 14-13 ILLUSTRATION OF WEATHERING SOLIDS CREATION TECHNIQUES



Mineralization Zones

The mineralization zones were designed based on the structure of the geology and on the gold assay results.

More specifically, in the Falagountou West deposit, the folded contact between the sedimentary and intrusive rocks served as a guideline for drawing the mineralization units (Figure 14-14). Gold assay grades above 0.25 g/t Au were included in the zones with some intersects with lower grade included in order to maintain a better continuity between zones. The minimum thickness of the zones was modelled at approximately three metres. Only the DD holes and the RC holes were used for the modelling of the zones. The zones were drawn on each section, smoothed, revised for consistency through sections, and linked together by tie lines to create solids. Eight mineralized zone were modelled for Falagountou West. On average, the zones are 6 m to 27 m thick (Table 14-21). All previous mineralization zones have been revised and updated in order to be used in the current resource estimate.

In the Wafaka deposit, the mineralization envelopes were designed to follow the direction of the mineralization mainly observed in the grade control data using Leapfrog. Grade shells were constructed using the structural trend with 2.5 m composites around a value of 0.08 g/t Au. This low value was chosen in order to achieve continuity through the Wafaka deposit. Through observation using exploration and grade control data, the structural trend used was a dip of 20º to the west. The average thickness of the zones is approximately 10 m (Table 14-22). Mineralized zones were identified with a unique code 10 which is displayed in Figure 14-15. All rock outside the grade shell was identified with code 100.

In both the Falagountou West and Wafaka deposits, the mineralization 3D envelopes were used as hard boundaries to constrain the interpolation of the gold grades.

TABLE 14-21 ROCK CODES AND AVERAGE THICKNESS - FALAGOUNTOU WEST DEPOSIT

Zone	Average Thickness (m)
10	7.7
20	27.7
30	12.9
40	7.9
50	8.2
60	6.8
70	6.6
80	6.6

FIGURE 14-14 MINERALIZATION ZONES - FALAGOUNTOU WEST DEPOSIT

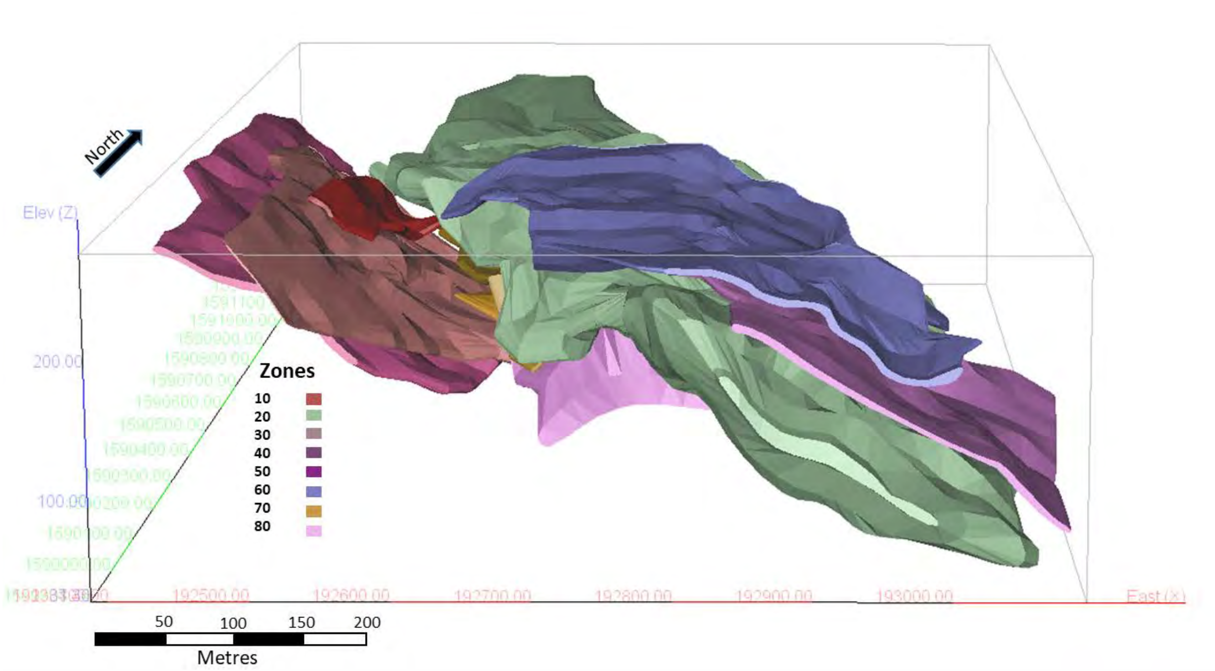
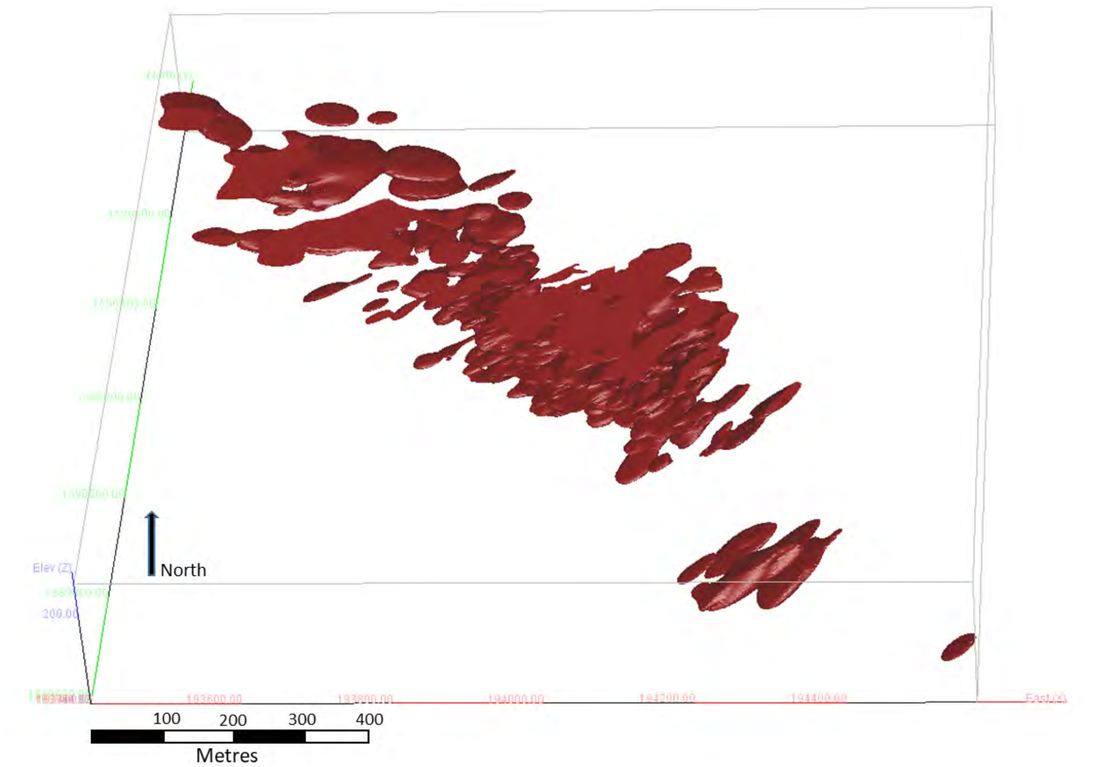


TABLE 14-22 ROCK CODES AND AVERAGE THICKNESS - WAFAKA DEPOSIT

Zone	Average Thickness (m)
10	9.67

FIGURE 14-15 MINERALIZATION ZONES - WAFAKA DEPOSIT



Topography Surface

The surface named "Topo Clip 27Aout12" covers most of the block model area apart from a small portion in the northeast, where no drilling is present. Blocks from the model were updated using this surface.

14.1.2.4 STATISTICAL ANALYSIS

Statistics of the Raw-Assays

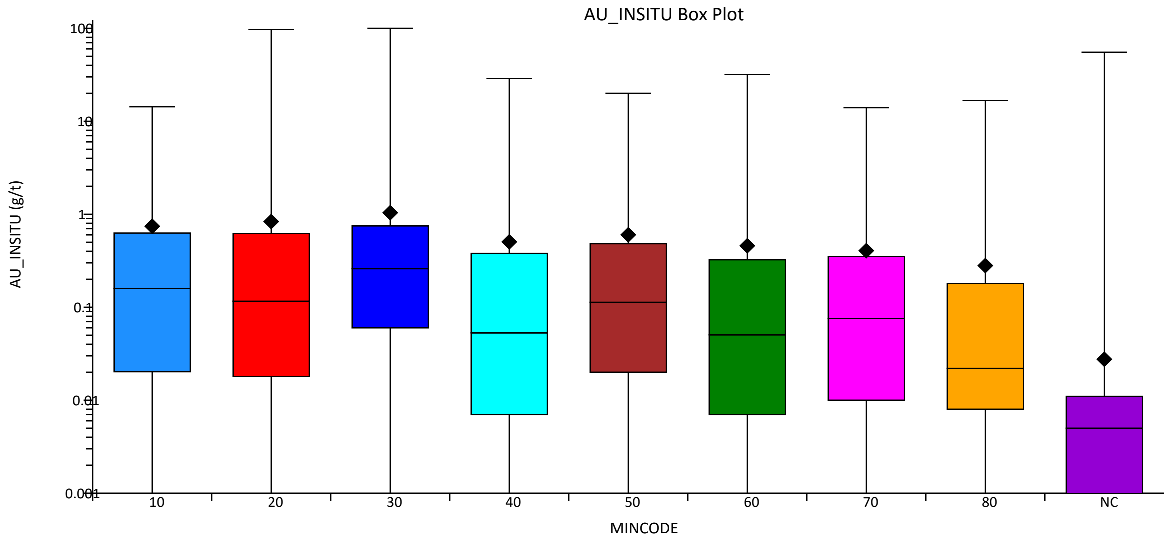
Statistics of the raw gold assays were computed using X10, a statistical program. Statistics were studied for assays grouped by mineralization domains for the Falagountou West deposit (Table 14-23) and for the Wafaka deposit (at the end of Table 14-23).

BOX-and-Whisker plots showed that the gold mineralization is constant through the eight zones of the Falagountou West deposit (Figure 14-16). The two main zones (20 at the east and 50 at the west) have a mean grade above the cut-off grade (COG) used during modelling.

TABLE 14-23 STATISTICS OF GOLD ASSAYS BY MINERALIZED ZONE - FALAGOUNTOU WEST AND WAFAKA DEPOSITS

Zone	No. of Assays	Gold Raw Assays (g/t Au) Min	Max	Mean	Standard Deviation	CoV
Falagountou West						
12	104	0.004	9.507	0.612	1.37	2.25
13	132	0.001	9.830	0.633	1.36	2.15
19	10	0.041	14.326	2.208	4.41	2.00
21	281	0.003	30.827	1.051	2.70	2.57
22	1,782	0.001	37.005	0.569	1.54	2.70
23	8,765	0.001	97.190	0.881	3.15	3.58
29	56	0.010	14.300	0.912	2.00	2.19
31	16	0.002	0.900	0.192	0.25	1.28
32	318	0.001	18.345	0.879	1.97	2.24
33	1,465	0.001	100.000	1.086	3.85	3.54
39	22	0.007	12.597	0.937	2.65	2.83
41	2	0.257	0.414	0.336	0.11	0.33
42	108	0.002	13.900	0.624	1.63	2.62
43	843	0.001	28.800	0.491	1.84	3.74
51	3	0.011	0.050	0.030	0.02	0.65
52	147	0.001	7.411	0.491	0.99	2.02
53	981	0.001	20.000	0.622	1.71	2.75
59	7	0.155	1.390	0.403	0.44	1.10
61	88	0.005	2.910	0.406	0.61	1.50
62	346	0.001	31.848	0.550	2.36	4.29
63	615	0.001	21.374	0.420	1.39	3.30
69	91	0.001	3.700	0.454	0.73	1.60
73	168	0.001	14.000	0.406	1.25	3.08
83	1,007	0.001	16.700	0.280	1.02	3.65
Wafaka						
11	2,701	0.001	100	0.629	3.508	5.58
12	1,363	0.001	67.649	0.721	3.287	4.56
13	6,002	0.001	100	0.575	2.770	4.81
19	251	0.001	25.879	0.58	2.194	3.78
101	7,631	0.001	23.906	0.031	0.399	13.06
102	5,581	0.001	15.711	0.024	0.258	10.81
103	20,310	0.001	27.459	0.026	0.329	12.56
109	634	0.001	2.746	0.066	0.213	3.21

FIGURE 14-16 BOX AND WHISKER PLOT FOR FALAGOUNTOU WEST DEPOSIT



Capping was mainly determined using the break (gap) in the distribution on the probability plot graph. There are various capping limits depending on the mineralized zone for the Falagountou West and Wafaka. Table 14-24 tabulates the capping values used on the raw-assays. Capping values for the Falagountou West deposit range from 10 to 25 g/t Au and all domains at the Wafaka deposit are capped at 15 and 30 g/t Au.

TABLE 14-24 FALAGOUNTOU WEST AND WAFAKA GOLD CAPPING VALUES

Zone	No. of Assays	Capping (g/t Au)	No. of Assays Capped	% of Assays Capped	Metal Capped
Falagountou West					
10	250	10	1	0.40%	2.26%
20	10,951	25	27	0.25%	5.20%
30	1,833	25	7	0.38%	5.30%
40	959	15	3	0.31%	4.10%
50	1,143	12	4	0.35%	3.40%
60	1,166	15	3	0.26%	5.20%
70	168	N/A	0	0.00%	0.00%
80	1,007	10	3	0.30%	3.30%
All Zones	**17,477**	**15**	**84**	**0.48%**	**8.70%**
Wafaka					
10	19,004	30	28	-	-
100	51,075	15	15	-	-
All Zones	**70,079**	**-**	**43**		

Compositing

The capped raw assays were composited into 2.5 m run lengths (down hole) within each domain coded in the drill hole database. Each composite was coded using the domain's code from the corresponding domain, as well as the appropriate weathering profile code. Composites measuring less than 0.5 m in length were removed from the database (e.g., composites created at the end of a domain).

Statistics of the Composites

A statistical analysis was undertaken to describe the characteristics of the gold grades within each of the mineralized zones and to assess the need for limiting the influence of very high grade assays during interpolation. The statistics of the 2.5 m composites, within the mineralized domains of the Falagountou West deposit, are summarized in Table 14-25. Statistics of composites for the Wafaka deposit were grouped in a single population and are summarized at the bottom of Table 14-25.

TABLE 14-25 STATISTICS OF COMPOSITES BY MINERALIZED ZONE - FALAGOUNTOU WEST AND WAFAKA DEPOSITS

Zone	No. of Composites	Gold Composites (g/t Au)					CoV
		Min	Max	Mean	Median	Standard Deviation	
Falagountou West							
12	44	0.01	4.374	0.687	0.313	0.965	1.41
13	54	0.001	5.998	0.657	0.240	1.030	1.57
19	5	0.062	5.341	2.194	2.189	2.091	1.09
21	130	0.015	9.770	1.105	0.356	1.797	1.67
22	760	0.001	13.013	0.552	0.193	0.996	1.94
23	3,820	0.001	19.174	0.844	0.224	1.715	2.58
29	31	0.010	14.300	1.209	0.514	2.613	2.16
31	7	0.003	0.506	0.190	0.129	0.186	0.98
32	129	0.002	8.073	0.874	0.330	1.456	1.67
33	629	0.001	15.532	0.998	0.380	1.889	2.24
39	10	0.004	5.621	0.840	0.278	1.700	2.02
41	1	0.276	0.276	0.276	0.276	-	-
42	46	0.018	10.094	0.671	0.242	1.520	2.27
43	377	0.001	10.000	0.506	0.139	1.205	2.78
51	1	0.033	0.033	0.033	0.033	-	-
52	64	0.002	3.120	0.470	0.176	0.656	1.39
53	417	0.001	9.302	0.577	0.189	1.070	1.94
59	3	0.194	1.116	0.534	0.291	0.507	0.95
61	43	0.011	1.821	0.447	0.300	0.470	1.05
62	147	0.001	9.275	0.461	0.097	1.118	2.99
63	253	0.001	12.932	0.449	0.095	1.179	3.04
69	48	0.003	1.928	0.430	0.283	0.458	1.06
73	79	0.001	8.402	0.455	0.130	1.118	2.46
83	7,530	0.001	19.174	0.733	0.208	1.527	2.53
All Zones							
Wafaka							
11	1,042	0.001	40.09	0.64	0.14	2.32	3.63
12	529	0.001	17.28	0.68	0.18	1.72	2.52
13	2,350	-	40.45	0.58	0.16	1.85	3.16
19	97	0.001	7.93	0.43	0.13	0.95	2.19
101	2,985	-	9.65	0.03	0	0.26	8.41
102	2,190	-	7.48	0.02	0	0.19	7.77
103	8,183	-	12.4	0.03	0	0.25	9.1
All Zones	**17,625**	**-**	**40.45**	**0.16**	**0.01**	**0.99**	**6.15**

Density Data

The database includes specific gravity measurements for 5,210 samples. Table 14-26 summarizes the basic statistics used to establish the density model. To avoid the influence of outliers, the median value was judged to be a good representation of background values for these weathering horizons.

TABLE 14-26 STATISTICS OF FALAGOUNTOU WEST AND WAFAKA SPECIFIC GRAVITY SAMPLES

Weathering Profile	Code	No. of Measurements	Median (t/m^3)	Average (t/m^3)
Saprolite	1	115	1.87	1.87
Trans-3	2	365	2.38	2.35
Rock-2	3	4,730	2.74	2.74

14.1.2.5 VARIOGRAPHY

Grade variography was generated in preparation for the estimation of gold grades with OK and to assess the spatial dependence of samples. The variography was based on the 2.5 m down-hole composite for all data and for all mineralization zones in the Falagountou West deposit and in two groups of samples for the Wafaka deposit. Variography was done using GEOVIA GEMS.

For the Falagountou West deposit, considering the geometry of the mineralized zones, two main orientations were used for variography modelling. In paired zones (10-30-50) associated with the west intrusion a dip of -40º to the east was used. A shallower dipping angle was used for the main intrusive to the west (-20º) of the paired zones (20-40-60). The nugget of the variogram was based on the downhole variogram. Results from the main mineralized zone from the east (20) were applied to zone associates with the eastern intrusive, and the result from the main west zone (30) was applied to all of the other zones associated with the west intrusive. The behaviour of the mineralization was considered the same. Model results are summarized in Table 14-27.

For the Wafaka deposit, a series of correlograms was generated from the capped gold grades every 30º of azimuth and at 30º dip increments. The optimal anisotropy directions were determined through the variogram map in GEOVIA GEMS. The minimum number of composite pairs required for variography was 10. The variography model included a nugget effect and two spherical structures. The global variogram model results are summarized in Table 14-27.

TABLE 14-27 VARIOGRAM MODELS FOR GOLD CAPPED COMPOSITES - FALAGOUNTOU WEST AND WAFAKA DEPOSITS

Deposit	Domain	Nugget	Ranges of Influence (m) 1st Structure Sill	Maximum Range (m)	2nd Structure Sill	Maximum Range (m)	Rotation AZ /DIP/AZ
Falagountou West	FW_20	0.824	1.825	50	N/A	N/A	90/-20/180
	FW_30	0.358	2.439	50	N/A	N/A	90/-40/180
Wafaka							
	WAF_10	0.943	0.422	11	0.630	30	145/0/0
	WAF_100	0.053	0.062	8	0.141	23	6/-50.32/275

14.1.2.6 BLOCK MODELLING

Two block models were constructed containing both the Falagountou West and Wafaka deposits. The block models cover an area large enough to manage pit optimizations and associated pit slopes. The block models were built using GEOVIA GEMS version 6.8.

Block Model Parameters

The drilling pattern, thickness of the zones, and the open pit mine planning considerations guided the choice of block dimensions. The block model parameters are summarized in Table 14-28.

TABLE 14-28 FALAGOUNTOU WEST AND WAFAKA BLOCK MODELS SETTINGS

Axis	Origin and Rotation (m)	Block Size (m)	No. of Blocks	
Falagountou West				
X	192,200	10	270	Columns
Y	1,589,500	15	135	Rows
Z	350	5	70	Level
Rotation	0			
Wafaka				
X	194,000	10	125	Columns
Y	1,589,000	10	200	Rows
Z	350	5	70	Level
Rotation	28			

Additionally, a series of attributes, required during the block model development, were incorporated into the block model project. Table 14-29 presents the list of attributes found in the block model projects “FAL19Offic_W” and “WAF19Offic” in the standard folder.

TABLE 14-29 LIST OF ATTRIBUTES FOUND IN THE FALAGOUNTOU WEST AND WAFAKA BLOCK MODELS

Folder Name	Model Name	Description
	Weathering	Weathering profile coding (saprolite, transition, rock)
	Rock Type	Geological coding (intrusive, sedimentary, overburden)
Standard	Density	Specific gravity
	AU	Ordinary Kriging gold grades (g/t)
	CATEG	Resource categorization

Rock Type Model

The rock type model, or domain coding, relied on the rock coding associated with the multiple wireframes for the Falagountou West and Wafaka deposits which were designed based on the structure of the geology and on the gold assay results and acted as hard boundaries.

Density Model

In conjunction with the weathering profiles discussed earlier and density statistics, the density was populated in the block model in a two-step manner.

First, all specific gravity samples were assigned a weathering code depending on their 3D location within the weathering solids. Secondly, blocks in the model were assigned a background value related to their weathering profile and equal to the median density value, as tabulated in Table 14-30. Since all new drilling occurred in the Falagountou West deposit, background values from the March 2015 resource estimate were kept the same for the Wafaka deposit; only the weathering surfaces underwent minor changes. Once all blocks in the model were set a background value, an interpolation profile was set up to estimate the block densities in the vicinity of the density measurements. The results were stored in a separate attribute.

TABLE 14-30 BACKGROUND DENSITY VALUES USED IN THE FALAGOUNTOU AND WAFAKA MODELS

Weathering Profile	Code	Density (t/m^3) Falagountou West	Wafaka
Saprolite	1	1.87	1.86
Trans-2	2	2.29	2.25
Rock-2	3	2.74	2.74

Grade Estimation Methodology

The final interpolation method selected for the Falagountou West and Wafaka deposits is OK. Nearest Neighbour (NN or ID^{20}) methods were also tested to compare with the OK method. The OK method was judged to be the most suitable to replicate composite grades throughout the Falagountou West and Wafaka deposits.

Grade estimates were generated using the 2.5 m composites. Mineralized domains were considered as hard boundaries through each interpolation step. A block being interpolated used only composites from within its corresponding domain. GEOVIA GEMS version 6.8 software was used for the estimate.

The sample search approach used to estimate the blocks for the Falagountou West and Wafaka deposits is summarized below:

Falagountou West

- **First Pass:** A minimum of seven and a maximum of 30 composites within the search ellipse ranges. A maximum of three composites per hole could be used for any block estimate.

- **Second Pass:** A minimum of five and a maximum of 30 composites within the search ellipse ranges. A maximum of three composites per hole could be used for any block estimate. Only blocks which were not estimated during the first pass could be estimated during the second pass.
- **Third Pass:** A minimum of one and a maximum of 30 composites within the search ellipse ranges. A maximum of three composites per hole could be used for any block estimate. Only blocks which were not estimated during the first and second pass could be estimated during the third pass.

Wafaka

- **First Pass:** A minimum of seven and a maximum of 20 composites within the search ellipse ranges. A maximum of three composites per hole could be used for any block estimate.
- **Second Pass:** A minimum of five and a maximum of 20 composites within the search ellipse ranges. A maximum of three composites per hole could be used for any block estimate. Only blocks which were not estimated during the first pass could be estimated during the second pass.
- **Third Pass:** A minimum of one and a maximum of 20 composites within the search ellipse ranges. A maximum of three composites per hole could be used for any block estimate. Only blocks which were not estimated during the first and second pass could be estimated during the third pass.

The various profiles of interpolation and search ellipses for gold composites utilized in the estimation of the resources are tabulated in Tables 14-31 to 14-33 for the Falagountou West deposit and in Tables 14-31, 14-32, and 14-34 for the Wafaka deposit.

TABLE 14-31 FALAGOUNTOU WEST AND WAFAKA INTERPOLATION PROFILE SETTINGS

Deposit	Profile Name	Pass	Sample			Target Rock Code	Ellipses Name
			Min	Max	Max per Hole		
WEST	FW_18_1	1	7	30	3	See List of Rock Codes	See Table Naming of Search Ellipse Profiles (Table 14-32)
	FW_18_2	2	5	30	3		
	FW_18_3	3	1	30	3		
WAFAKA	WAF1_LT	1	7	20	3	10 & 100	See Table Naming of Search Ellipse Profiles (Table 14-32)
	WAF2_LT	2	3	20	3		
	WAF3_LT	3	1	20	3		

TABLE 14-32 SEARCH ELLIPSE NAMES - FALAGOUNTOU WEST AND WAFAKA DEPOSITS

Rock Codes	Ellipse Profile		
	Pass 1	Pass 2	Pass 3
Falagountou West			
10	FW_30	FW_30_2	FW_30_3
20	FW_20	FW_20_2	FW_20_3
30	FW_30	FW_30_2	FW_30_3
40	FW_20	FW_20_2	FW_20_3
50	FW_30	FW_30_2	FW_30_3
60	FW_20	FW_20_2	FW_20_3
70	FW_30	FW_30_2	FW_30_3
80	FW_20	FW_20_2	FW_20_3
Wafaka			
10	WAF_10_1	WAF_10_2	WAF_10_3
100	W_100_1	W_100_2	W_100_3

TABLE 14-33 SEARCH ELLIPSOID SETTINGS - FALAGOUNTOU WEST DEPOSIT

Ellipse Profile Name [(2)]	Pass	Rotation			Anisotropy Range (m)		
		Az	Dip	Az	X	Y	Z
FW_20	1	90	-20	180	50	30	20
FW_30	1	90	-40	180	50	30	20
FW_20_2	2	90	-15	180	75	45	35
FW_30_2	2	90	-40	180	75	45	35
FW_20_3	3	90	-15	180	100	60	40
FW_30_3	3	90	-40	180	100	60	40

TABLE 14-34 SEARCH ELLIPSOID SETTINGS - WAFAKA DEPOSIT

Ellipse Profile Name [(2)]	Pass	Rotation			Anisotropy Range (m)		
		Az	Dip	Az	X	Y	Z
WAF_1-_1	1	145	0	N/A	30	15	20
W_100_1	1	5	-50	275	40	40	40
WAF_10_2	2	145	0	N/A	60	30	40
W_100_2	2	5	-50	275	60	60	60
WAF_10_3	3	145	0	N/A	120	60	40
W_1--_3	3	90	-50	275	120	120	120

14.1.2.7 CLASSIFICATION AND RESOURCE REPORTING

The Mineral Resource estimate was classified in accordance with the CIM (2014) definitions (see "Classification and Resource Reporting" under "14.1.1 EMZ Deposit").

In addition, the classification of interpolated blocks was undertaken by considering the following criteria:

- Quality and reliability of drilling and sampling data
- Distance between sample points (drilling density)
- Confidence in the geological interpretation
- Continuity of the geologic structures and the continuity of the grade within these structures
- Variogram models and their related ranges (first and second structures)
- Statistics of the data population
- Quality of assay data
- Tonnage factor

The resources were classified according to the above mentioned criteria which also directed the choice of the search parameters for each interpolation pass during the block estimation. No Measured Mineral Resources are estimated in the Falagountou West and Wafaka deposits. Indicated Mineral Resources are the blocks estimated from the first and second passes. Inferred Mineral Resources are the blocks estimated from the third pass. In addition, for Wafaka, blocks that were estimated with more than eight composites within the third pass were included in the Indicated category.

Figure 14-17 shows how the Mineral Resource categories are distributed in the Falagountou West deposit. Indicated Mineral Resources are essentially concentrated in the centre of the mineralization domains and form the bulk of the unconfined mineralization, where the drill hole density is the highest. Inferred Mineral Resources are peripheral to Indicated Mineral Resources and are mainly limited to the eastern extremity of the mineralization domains at depth, where drilling density is the lowest.

FIGURE 14-17 RESOURCE CATEGORIES - FALAGOUNTOU WEST DEPOSIT

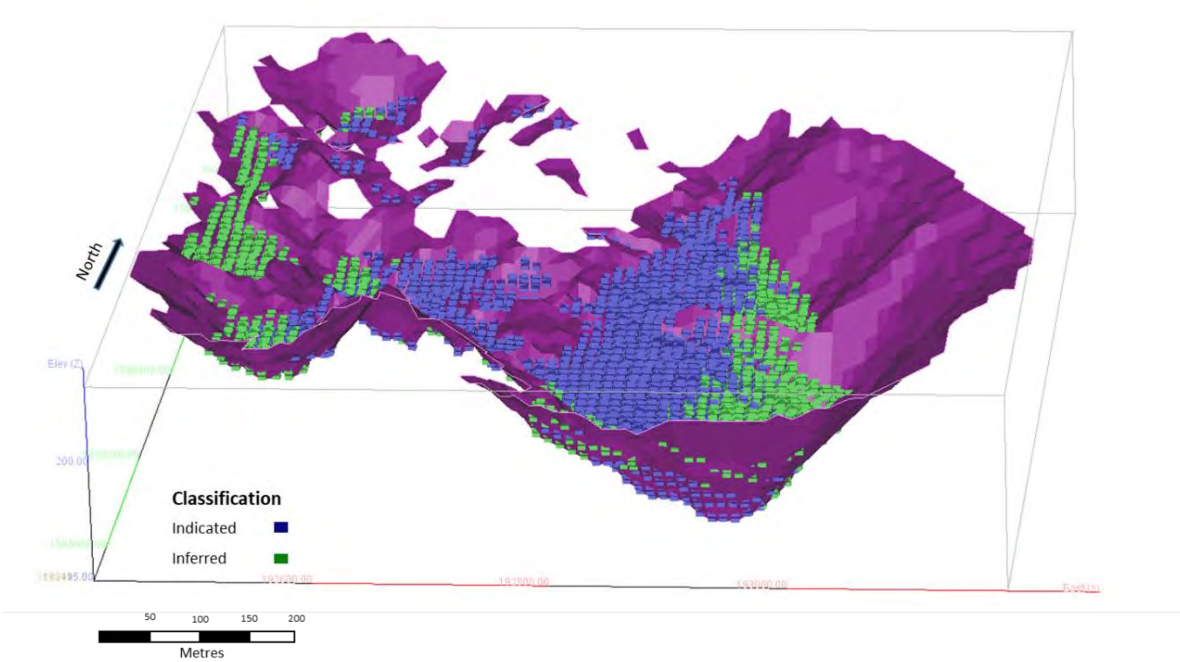
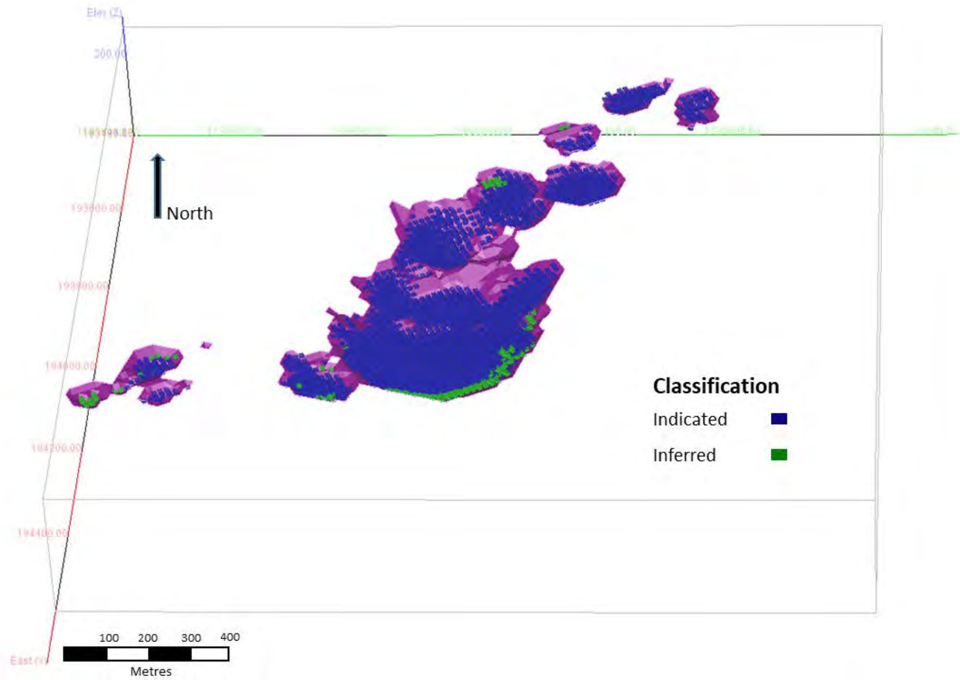


Figure 14-18 shows how the Mineral Resource categories are distributed in the Wafaka deposit (plan view). Indicated Mineral Resources make up the great majority of interpolated blocks. Inferred Mineral Resources are mostly limited to the edges of the mineralization domains, as well as to the northwestern satellite zone near surface.

FIGURE 14-18 RESOURCE CATEGORIES – WAFAKA DEPOSIT



14.1.2.8 BLOCK MODEL VALIDATION

Validation was completed on both the Falagountou West and Wafaka deposits block models. The validation process included visual checks, statistical validation of the model, comparison with models built from other interpolation methods, and swath plots.

Visual Validation

The visual checks consisted of visualizing slices of the block model, mineralized zones, and drill hole database. The slicing was performed vertically on 25 m intervals and horizontally on 5 m intervals. The data source was visually compared with the different model attributes (rock type, weathering type, mineralization zone, density, and gold grades) throughout the strike length of the deposit. The mineralized domains, the weathering profile layers, and the rock types are well represented in their proper attribute model. The OK based Mineral Resource estimate was found to be a good representation of the drill hole composites.

Statistical Validation

A statistical analysis between composites used in the interpolation and interpolated block grades was performed to evaluate if samples used in the estimation were well represented in the block model. Statistics were compiled for both the Falagountou West and Wafaka deposits. Tables 14-35 and 14-36 summarize statistics between declustered composite (using Nearest Neighbour) grades used in the interpolation process and grades of blocks interpolated for the Falagountou West and Wafaka deposits, in that order.

TABLE 14-35 AVERAGE COMPOSITE VERSUS BLOCK GRADES PER ESTIMATION DOMAIN - FALAGOUNTOU WEST DEPOSIT

Zone	No. of Blocks	Mean (NN) Composite Grade (g/t Au)	Mean Block Grade (g/t Au)
12	46	0.69	0.63
13	58	0.53	0.53
21	198	1.23	1.12
22	1,076	0.66	0.63
23	11,339	0.69	0.71
31	15	0.32	0.36
32	222	0.98	0.79
33	1,188	0.87	0.96
43	1,175	0.41	0.45
52	51	0.56	0.73
53	1,099	0.55	0.55
61	76	0.37	0.41
62	264	0.35	0.53
63	740	0.43	0.48
73	281	0.29	0.36
83	815	0.32	0.28
Total	**18,643**	**0.64**	**0.66**

TABLE 14-36 AVERAGE COMPOSITE VERSUS BLOCK GRADES PER ESTIMATION DOMAIN - WAFAKA DEPOSIT

Zone	No. of Blocks	Mean (NN) Composite (g/t Au)	Mean Block Grade (g/t Au)
11	2,818	0.494	0.525
12	1,629	0.599	0.553
13	10,658	0.597	0.542
19	579	0.535	0.551
101	52,031	0.020	0.023
102	45,900	0.017	0.019
103	365,483	0.020	0.021
109	6,860	0.037	0.040
Total	**485,958**	**0.038**	**0.038**

14.1.3 CONSTRAINED MINERAL RESOURCES

To establish a Mineral Resource estimate, an open pit development scenario is the most suitable due to the geology/geometry, tonnage, and grade of both the EMZ and Falagountou deposits. The deposit models were imported into Whittle to determine optimal pit shells based on the Lerchs-Grossmann algorithm. The method works on a block model of the orebody, and progressively constructs lists of related blocks that should, or should not, be mined. The method uses the values of the blocks to define a pit outline that has the highest possible economic value, subject to the required pit slopes defined as structure arcs in the software.

For resource reporting, all blocks classified as Indicated and Inferred were utilized in the pit optimization process.

This analysis requires several input parameters such as slope constraints, gold prices, process recoveries, and operating costs. A cut-off grade for each weathering type of mineralized rocks (saprolite, transition, and fresh rock) was determined in this process.

14.1.3.1 OPTIMIZATION PARAMETERS

Conceptual mining parameters used to calculate block values in Whittle for the EMZ and Falagountou deposits are presented in Section 15.

14.1.3.2 PIT CONSTRAINED MINERAL RESOURCES

The compilation of the Mineral Resource was carried out with the projected Mined Surfaces for August 31, 2019 for the EMZ and Falagountou West deposits.

The EMZ deposit pit constrained Indicated Mineral Resource is estimated to be 149.4 Mt at an average grade of 0.97 g/t Au, for a total of 4,662,000 ounces of gold, including 361,000 ounces of gold stored in stockpiles. The EMZ deposit pit constrained Inferred Mineral Resource is estimated to be 10.8 Mt at an average grade of 1.05 g/t Au, for a total of 365,000 ounces of gold.

Gold grade distribution and resource categorization for the EMZ deposit are illustrated in Figures 14-19 and 14-20, respectively.

FIGURE 14-19 ISOMETRIC VIEW OF EMZ DEPOSIT GOLD GRADE DISTRIBUTION INSIDE US$1,500/OZ AU WHITTLE PIT SHELL

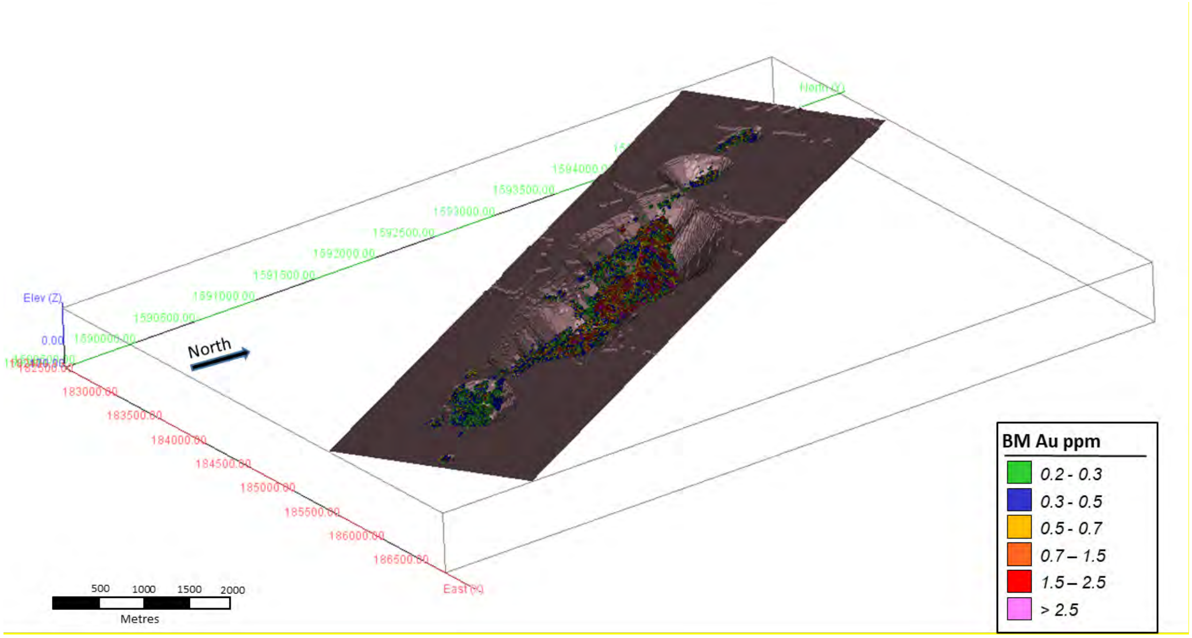
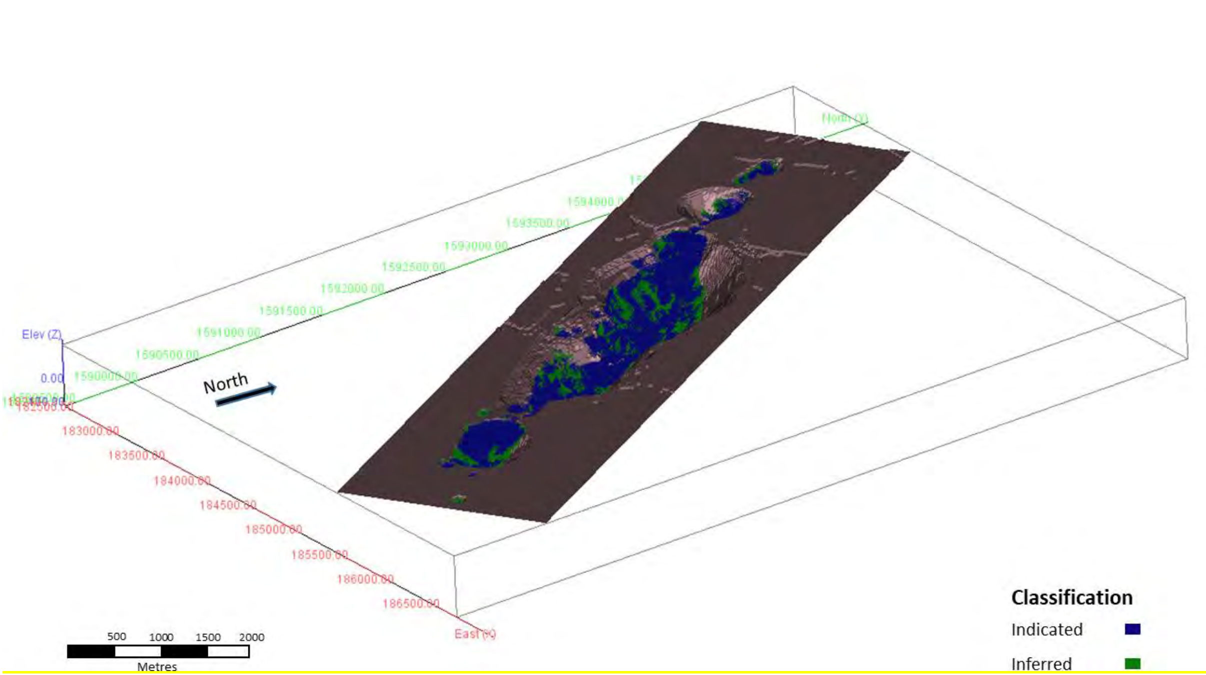


FIGURE 14-20 ISOMETRIC VIEW OF EMZ DEPOSIT RESOURCE CLASSIFICATION INSIDE US$1,500/OZ AU WHITTLE PIT SHELL



The Falagountou West deposit pit constrained Indicated Mineral Resource is estimated to be 4.28 Mt at an average grade of 1.29 g/t Au for a total of 177,000 ounces of gold. The Falagountou West deposit pit constrained Inferred Mineral Resource is estimated to be 1.98

Mt at an average grade of 1.38 g/t Au for a total of 88,000 ounces of gold. The Wafaka deposit pit constrained Indicated Mineral Resource is estimated to be 1.13 Mt at an average grade of 1.07 g/t Au for a total of 39,000 ounces of gold. The Wafaka deposit pit constrained Inferred Mineral Resource is estimated to be 64 thousand tonnes (kt) at an average grade of 0.60 g/t Au for a total of 1,200 ounces of gold. Saprolite, transition, and fresh rock weathering profiles are combined in this resource estimate statement. Gold grade distribution and resource categorization are illustrated in Figures 14-21 and 14-22 for the Falagountou West and Wafaka deposits, respectively.

Total Indicated Mineral Resources at Essakane are currently estimated to be 155 Mt at an average grade of 0.98 g/t Au for a total of 4,878,000 ounces of gold, while Inferred Mineral Resources are estimated to be 13 Mt at an average grade of 1.10 g/t Au for a total of 454,000 ounces of gold. IAMGOLD's attributable Mineral Resources are 139.3 Mt totalling 4,390,000 ounces of gold in Indicated Mineral Resources and 12 Mt totalling 409,000 ounces of gold in Inferred Mineral Resources.

Table 14-37 shows the projected stockpile status as of August 31, 2019.

TABLE 14-37 STOCKPILE STATUS AS OF AUGUST 31, 2019

Material Type	Stockpiles	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Saprolite	-	298	0.90	8.26
Transition	Stock LG (Falagountou)	84	0.77	2.1
Fresh Rock	Stock LG	8,130	0.54	141
	Stock Marginal	5,424	0.47	82
	Stock Heap Leach	2,625	0.33	28
	Stock HG (Falagountou)	642	0.76	16
	Stock LG	3,656	0.63	74
	Run of Mine (ROM) Pad	39	1.08	1
	Primary Crusher	284	0.93	9
	Total (Fresh Rock)	12,670	0.51	210
Total Stockpiles		**21,283**	**0.53**	**361**

Notes:
1. LG – low grade
2. HG – high grade

Details of the resource estimate are given in Table 14-38. Resources are tabulated by deposit (EMZ and Falagountou), resource category (Indicated and Inferred), and weathering material type (saprolite, transition, and fresh rock).

FIGURE 14-21 PIT CONSTRAINED MINERAL RESOURCES: GOLD GRADES - FALAGOUNTOU WEST DEPOSIT

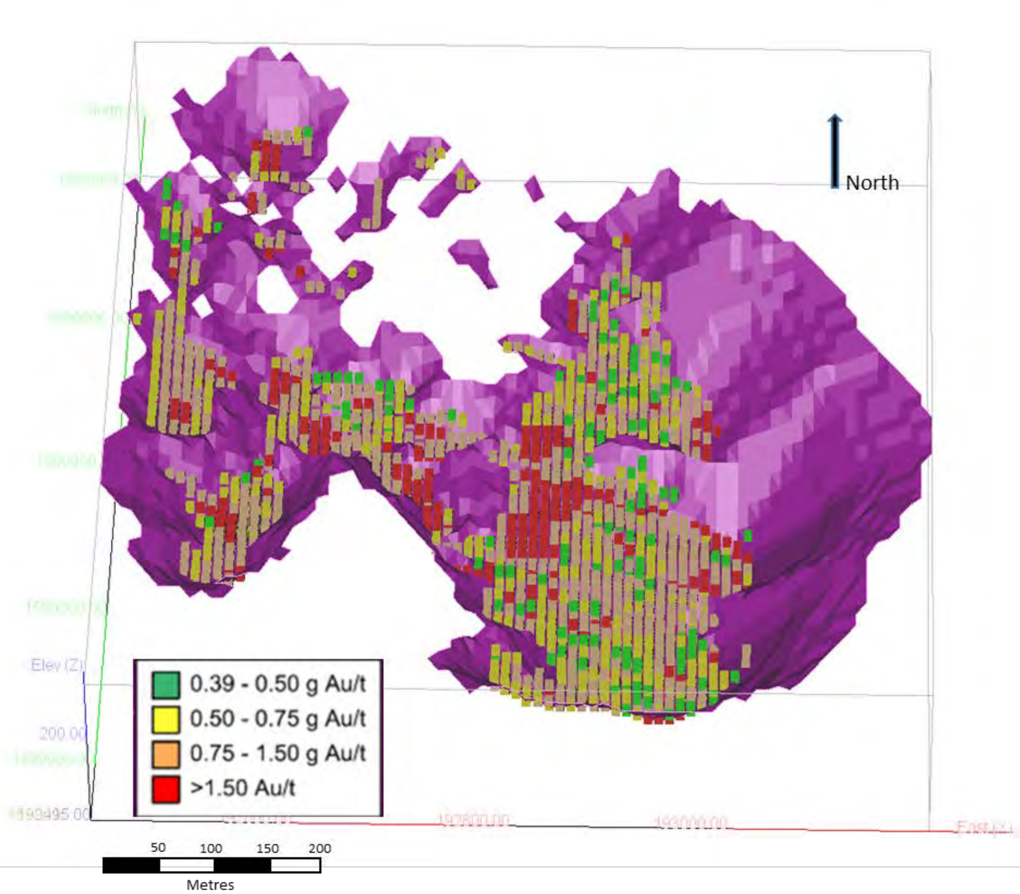


FIGURE 14-22 PIT CONSTRAINED MINERAL RESOURCES: GOLD GRADES WAFAKA DEPOSIT

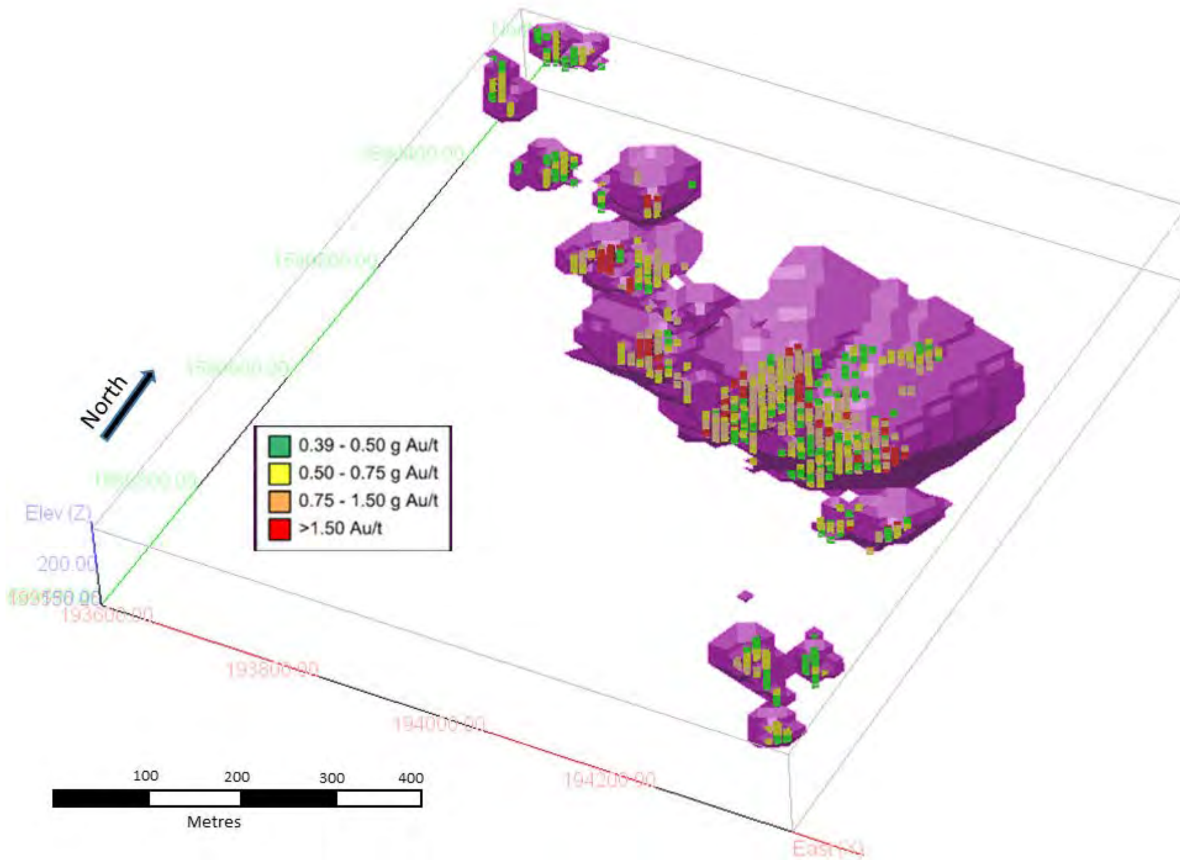


TABLE 14-38 ESSAKANE CONSOLIDATED MINERAL RESOURCES - AUGUST 31, 2019

Material Type	**Saprolite**			**Transition**			**Fresh Rock**			**Total (All Materials)**		
Resource Category	**Tonnage (000 t)**	**Grade (g/t Au)**	**Contained Metal (000 oz Au)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Contained Metal (000 oz Au)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Contained Metal (000 oz Au)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Contained Metal (000 oz Au)**
Essakane Main Zone (EMZ)												
Cut-off grade		**0.32 g/t Au**			**0.40 g/t Au**			**0.25 g/t Au**				
Measured	-	-	-	-	-	-	-	-	-	-	-	-
Indicated	2,062	0.54	35	1,036	0.68	23	125,163	1.05	4,243	128,261	1.04	4,301
Total Measured & Indicated	**2,062**	**0.54**	**35**	**1,036**	**0.68**	**23**	**125,163**	**1.05**	**4,243**	**128,261**	**1.04**	**4,301**
Stockpiles	298	86	8	8,214	0.54	143	12,670	0.51	210	21,183	0.53	361
Total EMZ M&I Resources	**2,360**	**0.58**	**44**	**9,250**	**0.56**	**165**	**137,833**	**1.00**	**4,453**	**149,444**	**0.97**	**4,662**
EMZ Inferred	193	0.53	3	41	0.69	0.9	10,540	1.06	361	10,774	1.05	365
Falagountou West and Wafaka												
Cut-off Grade		**0.35 g/t Au**			**0.44 g/t Au**			**0.55 g/t Au**				
Measured	-	-	-	-	-	-	-	-	-	-	-	-
Indicated	278	0.76	7	227	1.03	8	4,906	1.28	202	5,410	1.24	216
Total Measured & Indicated	**278**	**0.76**	**7**	**227**	**1.03**	**8**	**4,906**	**1.28**	**202**	**5,410**	**1.24**	**216**
Total Falagountou M&I Resources	**278**	**0.76**	**1**	**227**	**1.03**	**8**	**4,906**	**1.28**	**202**	**5,410**	**1.24**	216
Falagountou Inferred	58	0.57	1	6	0.90	0	1,985	1.38	88	2,049	1.36	89
Consolidated Essakane Resources (EMZ & Falagountou)												
M&I Resources	2,368	0.60	51	9,477	0.57	173	142,740	1.01	4,654	154,854	0.98	4,878
Inferred Resources	251	0.54	4	47	0.72	1	12,525	1.11	449	12,823	1.10	454
Attributable M&I Resources (90%)	2,374	0.60	45	8,529	0.57	156	128,466	1.01	4,189	139,369	0.98	4,390
Attributable Inferred Resources (90%)	226	0.54	4	42	0.72	1	11,273	1.11	404	11,541	1.10	409

Notes:
1. EMZ and Falagountou: Inside US$1,500/oz Au Whittle pit shells optimized on Measured, Indicated, and Inferred Mineral Resources.
2. Includes Proven and Probable Mineral Reserves.
3. M&I: Measured & Indicated.

14.1.3.3 PIT CONSTRAINED MINERAL RESOURCE SENSITIVITY TO CUT-OFF GRADE

EMZ Deposit

The sensitivity analysis presents the pit constrained Mineral Resources combining saprolite, transition, and fresh rock material estimated in the EMZ deposit block model at a series of cut-off grades, varying between 0.10 g/t Au and 2.00 g/t Au. The cut-off grade of 0.40 g/t Au was replaced by the material cut-offs used to estimate the Official Mineral Resource as follows: 0.32 g/t Au for saprolite, 0.40 g/t Au for transition, and 0.25 g/t Au for fresh rock. The Mineral Resources, as detailed in Table 14-39, are constrained below the mining surface as of August 31, 2019 and inside the US$1,500/oz Whittle pit shell optimized on Indicated and Inferred Mineral Resources.

TABLE 14-39 MINERAL RESOURCE SENSITIVITY TO SELECTED CUT-OFF GRADES – EMZ DEPOSIT

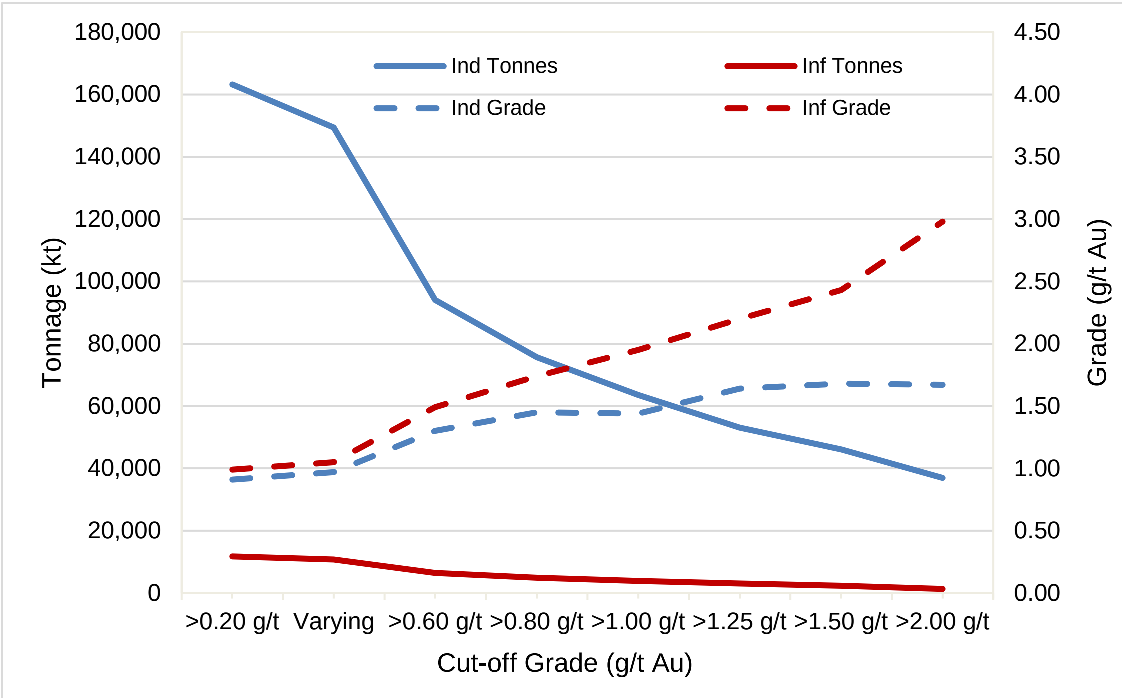
Cut-Off Grades (g/t Au)	Indicated			Inferred		
	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
>2.00	36,945	1.67	1,983	1,313	2.98	126
>1.50	46,055	1.68	2,489	2,353	2.43	184
>1.25	53,039	1.64	2,796	3,014	2.20	213
>1.00	63,513	1.44	3,173	3,925	1.95	243
>0.80	75,691	1.45	3,523	4,876	1.74	273
>0.60	94,005	1.30	3,931	6,416	1.49	307
Variable (2)	149,444	0.97	4,662	10,774	1.05	365
>0.20	163,211	0.91	4,767	11,744	0.99	372

Notes:
1. Mineral Resources are constrained below the mining surface as of August 31, 2019 and inside the US$1,500/oz Au Whittle pit shell optimized on Indicated and Inferred Mineral Resources.
2. August 31, 2019 Mineral Resource Cut-off Grades: Saprolite 0.32 g/t Au; Transition 0.40 g/t Au; and Fresh Rock 0.25 g/t Au.

Figure 14-23 shows grade-tonnage curves for Indicated and Inferred Mineral Resources versus cut-off grade for the EMZ deposit. A decrease of the cut-off grade from 0.5 g/t Au to 0.3 g/t Au results in an increase of 15% of the Indicated ounces; the impact is more significant on the Inferred Mineral Resource, which increases by 25%. An increase of the cut-off grade from 0.5 g/t Au to 0.8 g/t Au will result in a decrease of 22% in Indicated ounces. The EMZ grade curve does not show a significant degree of sensitivity to cut-off grades below 1.50 g/t Au in terms of gold grades, as the curves have a linear progression. The tonnage curves of the Indicated and Inferred Mineral Resources are not sensitive to cut-off grades. The Inferred Mineral Resources, contained within the US$1,500/oz Au Whittle pit shell optimized for

Indicated and Inferred Mineral Resources, are present in lesser tonnage amounts than the Indicated Mineral Resources, as represented by the corresponding tonnage curve.

FIGURE 14-23 INDICATED AND INFERRED MINERAL RESOURCE GRADE - TONNAGE CURVES – EMZ DEPOSIT



Falagountou Deposits

Table 14-40 summarizes the sensitivity of the pit constrained Mineral Resources of the Falagountou deposits (Falagountou West and Wafaka combined) for a series of selected gold cut-off grades. The sensitivity analysis uses gold cut-off grades between 0.20 g/t Au and 2.0 g/t Au. The sensitivity table and graph are both a compilation of the saprolite, transition, and fresh rock weathering profiles.

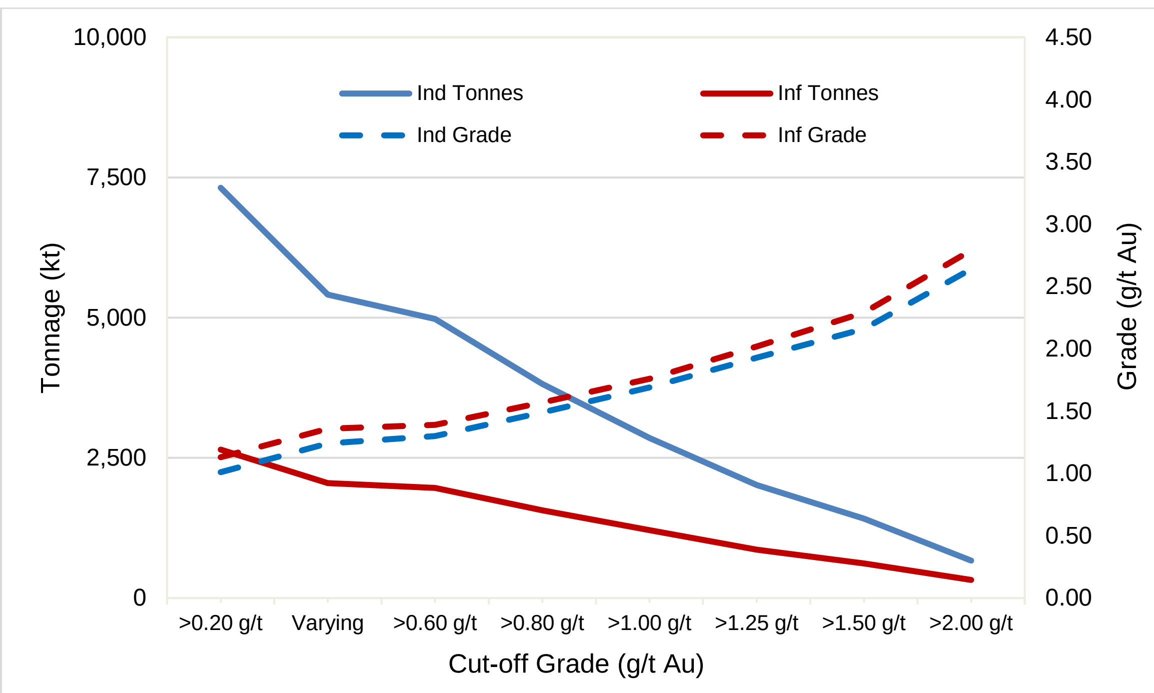
Figure 14-24 shows grade-tonnage curves for Indicated and Inferred Mineral Resources versus cut-off grade for the Falagountou West and Wafaka deposits.

TABLE 14-40 INDICATED MINERAL RESOURCE SENSITIVITY - FALAGOUNTOU WEST AND WAFAKA DEPOSITS COMBINED

Cut-off Grade	Indicated			Inferred		
(g/t Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
>2.00	668	2.64	57	323	2.79	29
>1.50	1,415	2.16	98	615	2.29	45
>1.25	2,013	1.93	125	860	2.02	56
>1.00	2,856	1.69	155	1,209	1.76	69
>0.80	3,817	1.49	182	1,567	1.57	79
>0.60	4,979	1.30	209	1,963	1.39	88
Varying	5,410	1.24	216	2,049	1.36	89
>0.20	7,316	1.01	238	2,647	1.13	96

Notes:
1. Mineral Resources are constrained below the mining surface as of August 31, 2019 and inside the US$1,500/oz Au Whittle pit shell optimized on Indicated and Inferred Mineral Resources (Falagountou West and Wafaka deposits combined).
2. August 31, 2019 Mineral Resource Cut-off Grades: Saprolite: 0.35 g/t Au, Transition: 0.44 g/t Au, and Fresh Rock: 0.55 g/t Au.

FIGURE 14-24 GRADE - TONNAGE CURVES OF PIT CONSTRAINED INDICATED MINERAL RESOURCE – FALAGOUNTOU WEST AND WAFATA COMBINED



14.1.4 SENSITIVITY TO GOLD PRICE

14.1.4.1 ESSAKANE (EMZ AND FALAGOUNTOU DEPOSITS)

Sensitivity to gold price for the EMZ and Falagountou deposits, combined, is illustrated in Figure 14-25 for gold prices varying between US$1,000/oz and US$1,700/oz. Cut-off grades were adjusted accordingly as tabulated in Table 14-41, per deposit.

FIGURE 14-25 EMZ AND FALAGOUNTOU DEPOSITS INDICATED AND INFERRED MINERAL RESOURCE SENSITIVITY TO GOLD PRICES (SAP+TRANS+ROCK)

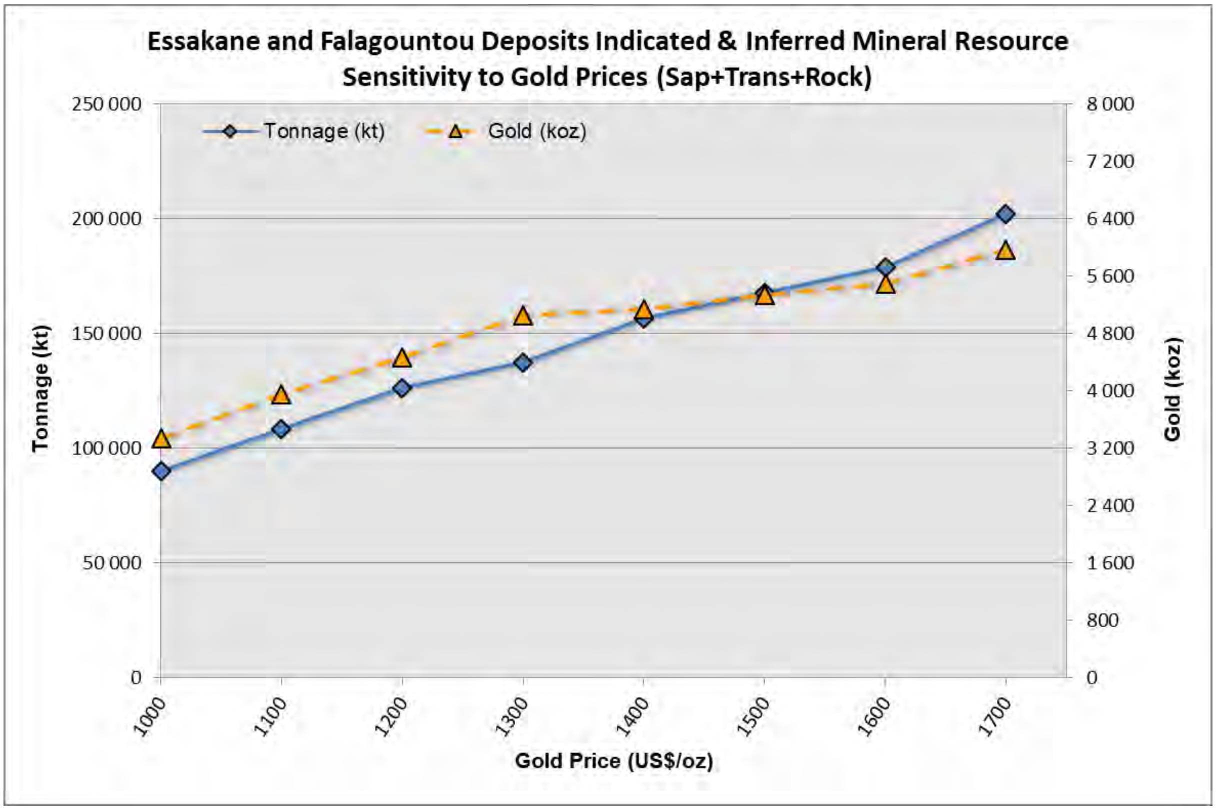


TABLE 14-41 EMZ AND FALAGOUNTOU DEPOSITS CUT-OFF GRADES FOR VARYING GOLD PRICES

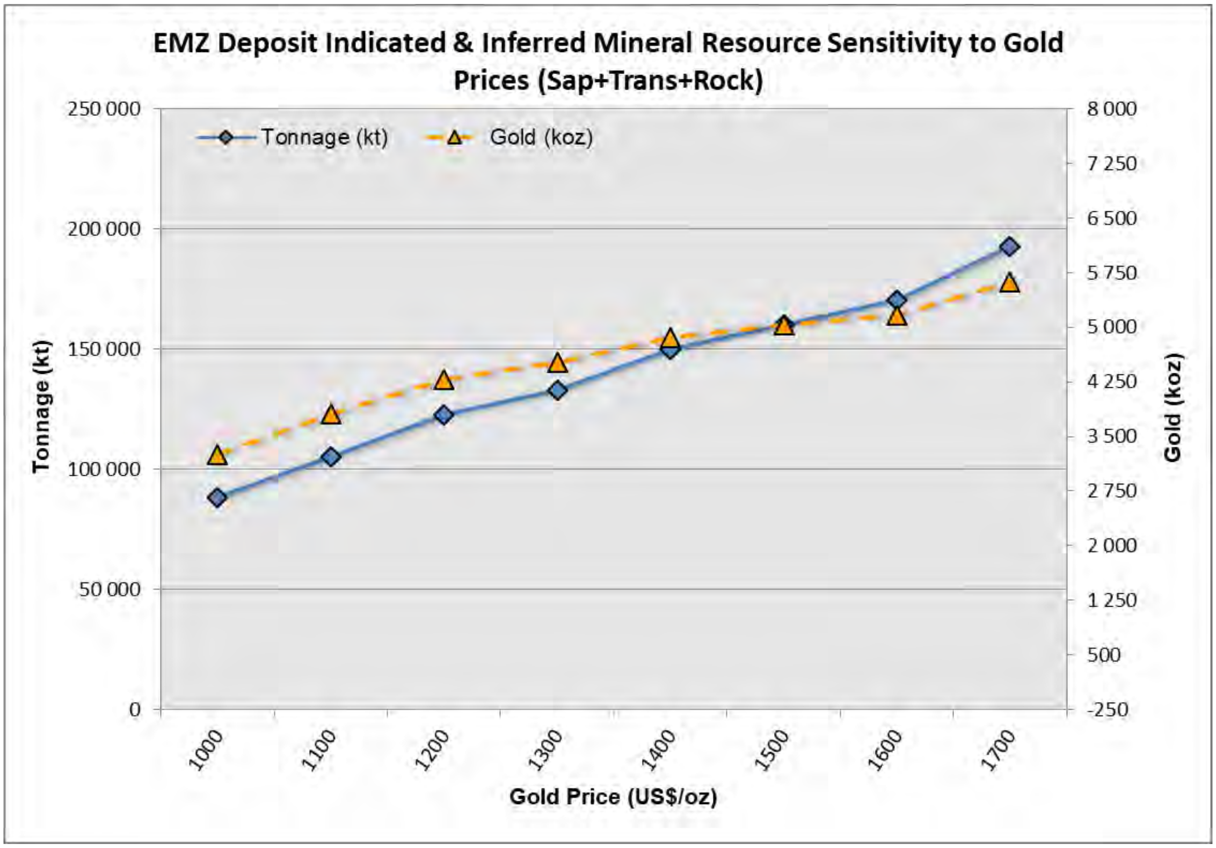
Gold Price (US$/oz)	EMZ Deposit Cut-off Grades (g/t Au)			Falagountou Deposits Cut-off Grades (g/t Au)		
	Saprolite	Transition	Fresh Rock	Saprolite	Transition	Fresh Rock
1,000	0.50	0.62	0.37	0.54	0.68	0.83
1,100	0.45	0.56	0.34	0.49	0.61	0.75
1,200	0.41	0.51	0.31	0.44	0.56	0.69
1,300	0.37	0.47	0.29	0.41	0.51	0.63
1,400	0.34	0.43	0.27	0.38	0.47	0.59
1,500	0.32	0.40	0.25	0.35	0.44	0.55
1,600	0.30	0.37	0.23	0.33	0.41	0.52
1,700	0.28	0.35	0.22	0.30	0.38	0.49

14.1.4.2 EMZ DEPOSIT

The sensitivity of the pit constrained Mineral Resources combining saprolite, transition, and fresh rock materials, estimated in the EMZ deposit block model at varying gold prices, is illustrated in Figure 14-26. All of the Mineral Resources are constrained below the mining surface as of August 31, 2019 and within the corresponding gold price Whittle pit shell and cut-off grades. The stockpiles are also included in the sensitivity analysis.

A 25% increase in gold price going from US$1,200/oz to US$1,500/oz would result in an ore tonnage increase of +34% and a lower increase of +23% in ounces of gold. The higher sensitivity of tonnage compared to ounces, in a changing gold price environment, reflects the low grade nature of the deposit.

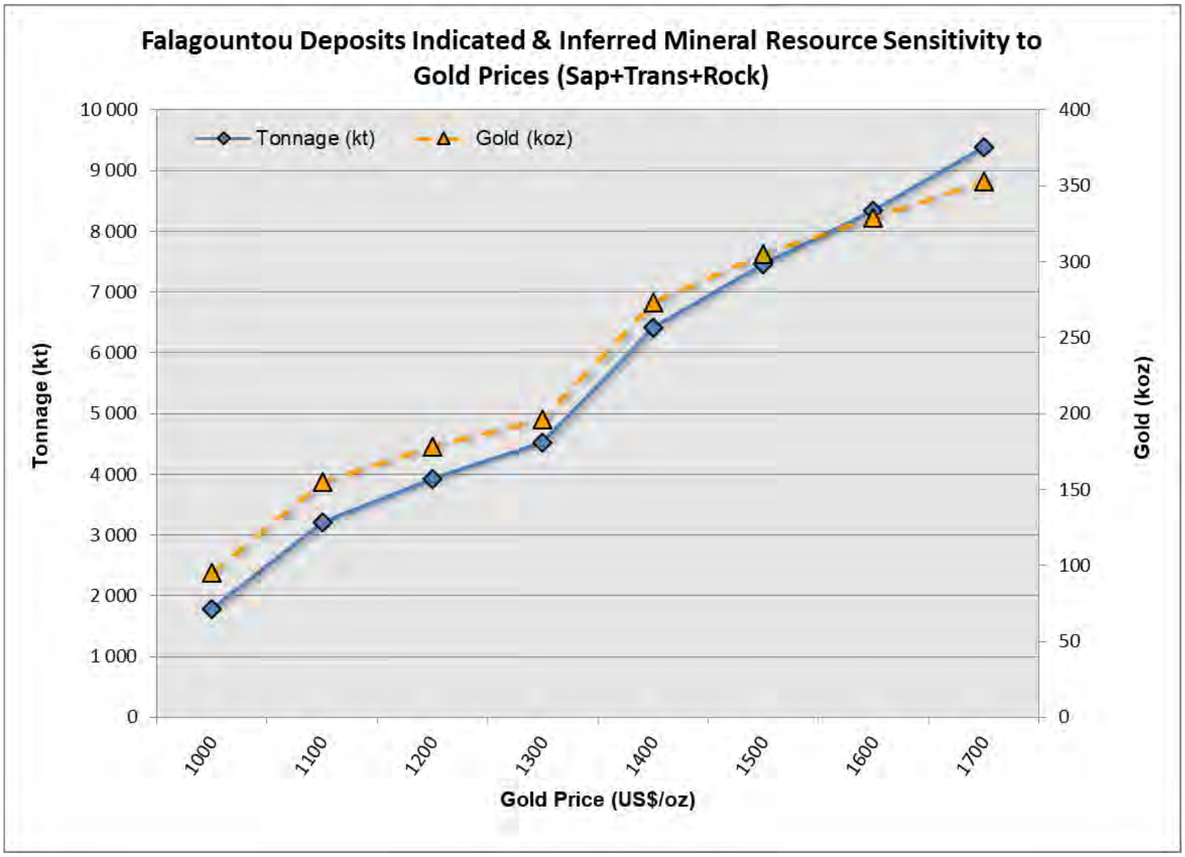
FIGURE 14-26 EMZ DEPOSIT INDICATED AND INFERRED MINERAL RESOURCES SENSITIVITY TO GOLD PRICE



14.1.4.3 FALAGOUNTOU DEPOSITS

Sensitivity to gold price for the Falagountou deposits is illustrated in Figure 14-27 for gold prices varying between US$1,000/oz to US$1,700/oz. Mineral Resources were estimated inside Whittle pit shells drawn from fixed optimization parameters as described in Section 15, however, with varying gold prices. Cut-off grades were adjusted accordingly, as tabulated in Table 14-41. The Falagountou West and Wafaka deposits tonnage augmentation progressively decreases, except when passing from a US$1,300/oz to US$1,400/oz gold price assumption. When comparing US$1,200/oz Au and US$1,500/oz Au Whittle pit shells, increases of 48% in tonnage and 41% in gold content and a decrease of 11% in grade are anticipated.

FIGURE 14-27 MINERAL RESOURCE SENSITIVITY TO GOLD PRICE - FALAGOUNTOU DEPOSITS



14.1.5 COMPARISON TO PREVIOUS MODELS

14.1.5.1 EMZ DEPOSIT

Table 14-42 compares the EMZ deposit Indicated and Inferred Mineral Resource estimates of August 31, 2019 to those of June 5, 2018. As of June 5, 2018, the EMZ deposit Indicated Mineral Resources were estimated to total 4,338,000 ounces of gold and increased by 292,000 ounces of gold to a total estimate of 4,662,000 ounces of gold as of August 31, 2019. Inferred Mineral Resources lost a total of 109,000 ounces of gold from 474,000 ounces of gold as of June 5, 2018 to 365,000 ounces of gold as of August 31, 2019.

The variations expressed as differences in tonnage, grade, and ounces in Table 14-42 are influenced by multiple factors, such as various changes in the block model, update of optimization parameters and costs, mining depletion, and stockpile variations.

The block model modifications included the addition of drill hole results which led to an update of the modelling. The actual model links together the EMZ north, main, and south by using the same lithostructural domain as the previous model. This is a change compared to the previous model, which consisted of three unlinked models. To do so, an addition of the upper turbidites was necessary. In addition to the update of the geological model, a new wireframe has been used to separate data inside the nose of the flank in order to complete different variography in accordance to the position of the data inside the fold.

Overall, the estimation strategy did not change from the previous model, however, variography was redone within the updated wireframes. The boundary strategies between layers has been adjusted with the new layers. In order to reduce smoothing of the estimation, the maximum composites used for the estimation was adjusted from 30 to 22. Other block modelling elements remained unchanged from the previous model including capping limits on assays, compositing style, and search ellipse ranges.

The differences in the EMZ deposit Mineral Resources also account for losses due to the planned 2018 mining depletion, gains coming from the revision of the costs and optimization parameters, and a positive stockpile variation. The main difference is due to the change of cut-off used for the heap leach project.

TABLE 14-42 COMPARISON OF EMZ DEPOSIT MINERAL RESOURCES AS OF AUGUST 31, 2019 TO MINERAL RESOURCES AS OF JUNE 5, 2018

Material Type		Saprolite			Transition			Fresh Rock			Total (All Materials)		
Resource Category		Tonnage (000 t)	Grade (g/t Au)	Gold (000 oz)	Tonnage (000 t)	Grade (g/t Au)	Gold (000 oz)	Tonnage (000 t)	Grade (g/t Au)	Gold (000 oz)	Tonnage (000 t)	Grade (g/t Au)	Gold (000 oz)
Resources June 5, 2018	**Cut-off grade**		**0.33 g/t Au**			**0.43 g/t Au**			**0.30 g/t Au**				
	M & I	1,345	0.47	20	917	0.62	18	128,770	0.96	3,970	131,031	0.95	4,009
	Stockpiles	94	0.24	1	8,792	0.54	152	9,168	0.60	177	18,054	0.57	330
	Total	**1,439**	**0.44**	**21**	**9,708**	**0.5**	**171**	**137,938**	**0.94**	**4,147**	**149,085**	**0.91**	**4,339**
	EMZ Inferred	460	0.54	8	195	0.63	4	18,296	0.78	462	18,952	0.78	474
Resources August 31, 2019	**Cut-off grade**		**0.32 g/t Au**			**0.40 g/t Au**			**0.25 g/t Au**				
	M & I	2,062	0.54	35	1,036	0.68	23	125,163	1.05	4,243	128,261	1.04	4,301
	Stockpiles	298	0.86	8	8,214	0.54	143	12,670	0.51	210	21,183	0.53	361
	Total	**2,360**	**0.58**	**44**	**9,250**	**0.56**	**165**	**137,833**	**1.00**	**4,453**	**149,444**	**0.97**	**4,662**
	EMZ Inferred	193	0.53	3	41	0.69	1	10,540	1.06	361	10,774	1.05	365
Difference	M & I	717	0.68	16	119	1.09	4	(3,606)	(2.35)	272	(2,770)	(3.28)	292
	Stockpiles	205	1.15	8	(577)	0	(9)	3,502	0.29	33	3,129	0.31	31
	Total	**922**	**0.78**	**23**	**(458)**	**0.35**	**(5)**	**(105)**	**(90.73)**	**305**	**359**	**28.01**	**323**
	EMZ Inferred	(267)	0.55	(5)	(154)	0.62	(3)	(7,756)	0.41	(101)	(8,177)	0.41	(109)

14.1.5.2 FALAGOUNTOU WEST AND WAFAKA DEPOSITS

The August 31, 2019 Falagountou West and Wafaka deposits Mineral Resource estimate is a major update from the June 5, 2018 model, since the mineralization zones were completely remodelled by the Essakane geology team in 2018. The Mineral Resource variations for the Falagountou deposits are summarized in Table 14-43.

The most notable difference for Falagountou West is due to a change of interpretation of the mineralization. Overall, the geological interpretation is the same, however, wider mineralized zones were modelled. Grade shells for the Wafaka deposit were redone using Leapfrog. The grade shells were mainly based on grade control data. In both cases, the effect was notable by decreasing overall grade and increasing tonnage. The loss of 168,000 ounces of gold for both Falagountou West and Wafaka is mainly due to the change to the Wafaka model as the change to the Falagountou West model only had a small impact on the quantity of gold ounces.

TABLE 14-43 COMPARISON OF FALAGOUNTOU WEST AND WAFAKA DEPOSITS MINERAL RESOURCES AS OF AUGUST 31, 2019 TO MINERAL RESOURCES AS OF JUNE 5, 2018

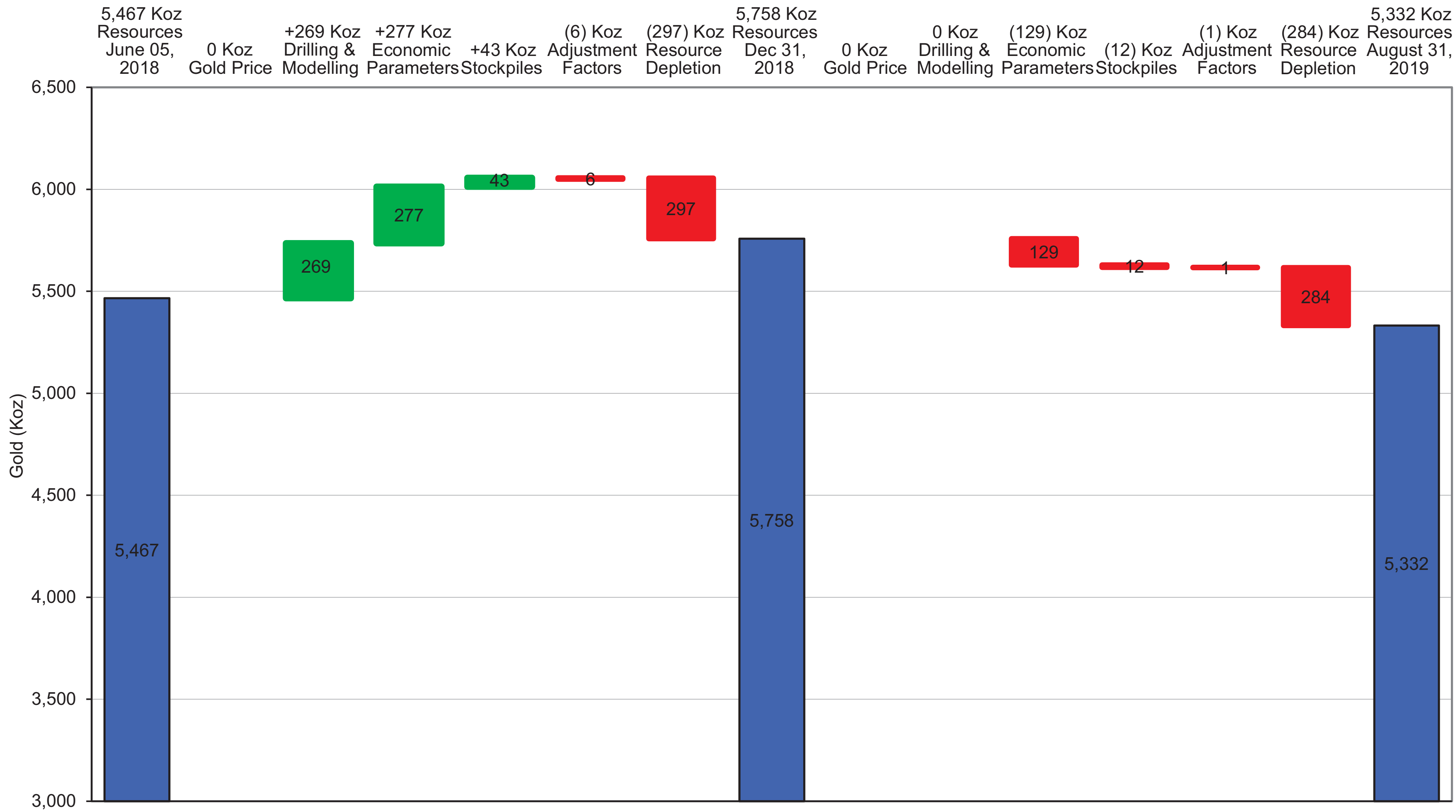
Material Type		**Saprolite**			**Transition**			**Fresh Rock**			**Total (All Materials)**		
Resource Category		**Tonnage (000 t)**	**Grade (g/t Au)**	**Gold (000 oz)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Gold (000 oz)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Gold (000 oz)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Gold (000 oz)**
June 5, 2018 (West+Wafaka)	**Cut-off grade**		**0.36 g/t Au**			**0.46 g/t Au**			**0.52 g/t Au**				
	M&I	1,209	1.05	41	665	1.18	25	8,851	1.66	473	10,725	1.56	539
	Inferred	308	1.08	11	83	1.51	4	1,401	2.23	100	1,792	2.00	115
August 31, 2019 (West+Wafaka)	**Cut-off grade**		**0.35 g/t Au**			**0.44 g/t Au**			**0.55 g/t Au**				
	M&I	278	0.76	7	227	1.03	8	4,906	1.28	202	5,410	1.24	216
	Inferred	58	0.57	1	6	0.90	0	1,985	1.38	88	2,049	1.36	89
Difference	M&I	(931)	1.13	(34)	(438)	1.25	(18)	(3,945)	2.14	(272)	(5,315)	1.89	(323)
	Inferred	(250)	1.19	(10)	(77)	1.56	(4)	584	0.65	(12)	257	3.20	(26)

14.1.5.3 ESSAKANE RESOURCE VARIATION THROUGH AUGUST 31, 2019

Factors leading to yearly changes of the Mineral Resources for the EMZ and Falagountou (West and Wafaka combined) deposits are presented in Figure 14-28 as a waterfall graph. Most of the changes to the Mineral Resources can be summarized by the following:

- **Changes to costs**, and consequently to optimization parameters, cut-off grades, and Whittle pit shells. New parameters can be viewed in Section 15.
- **Changes to model**: while the modelling update at the EMZ deposit yielded a gain of resources, the remodelling and new drilling at the Essakane deposit are responsible for the addition

For reference, separate waterfall graphs for the EMZ and Falagountou (West and Wafaka combined) deposits are presented in Figures 14-29 and 14-30.



IAMGOLD Corporation

Figure 14-28

IAMGOLD Corporation

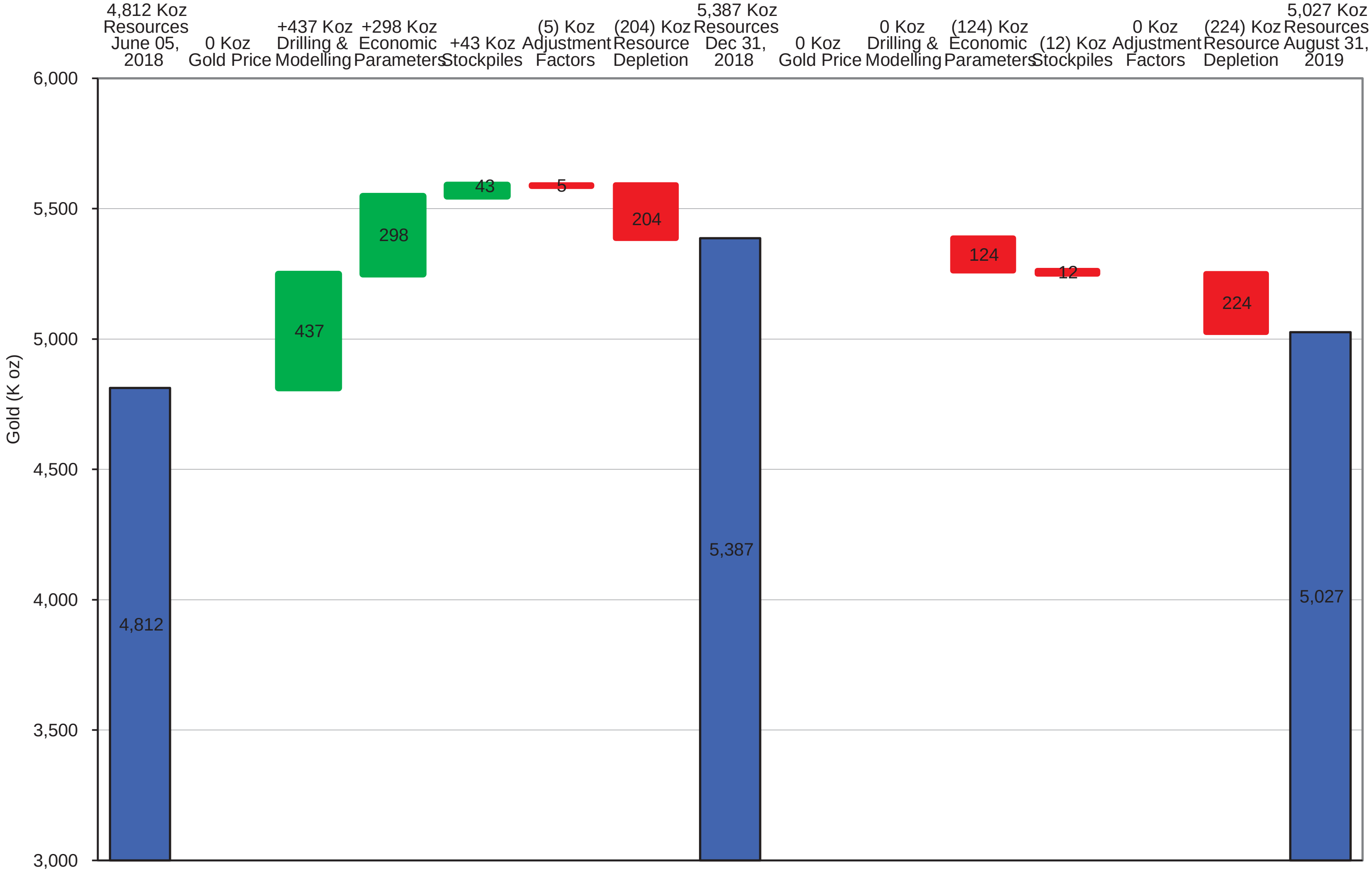
Essakane Gold Mine

Sahel Region, Burkina Faso

Graph Showing Evolution of the EMZ and Falagountou Mineral Resource from June 5, 2018 to August 31, 2019

January 2020

Source: IAMGOLD, 2019.



IAMGOLD Corporation

Figure 14-29

IAMGOLD Corporation

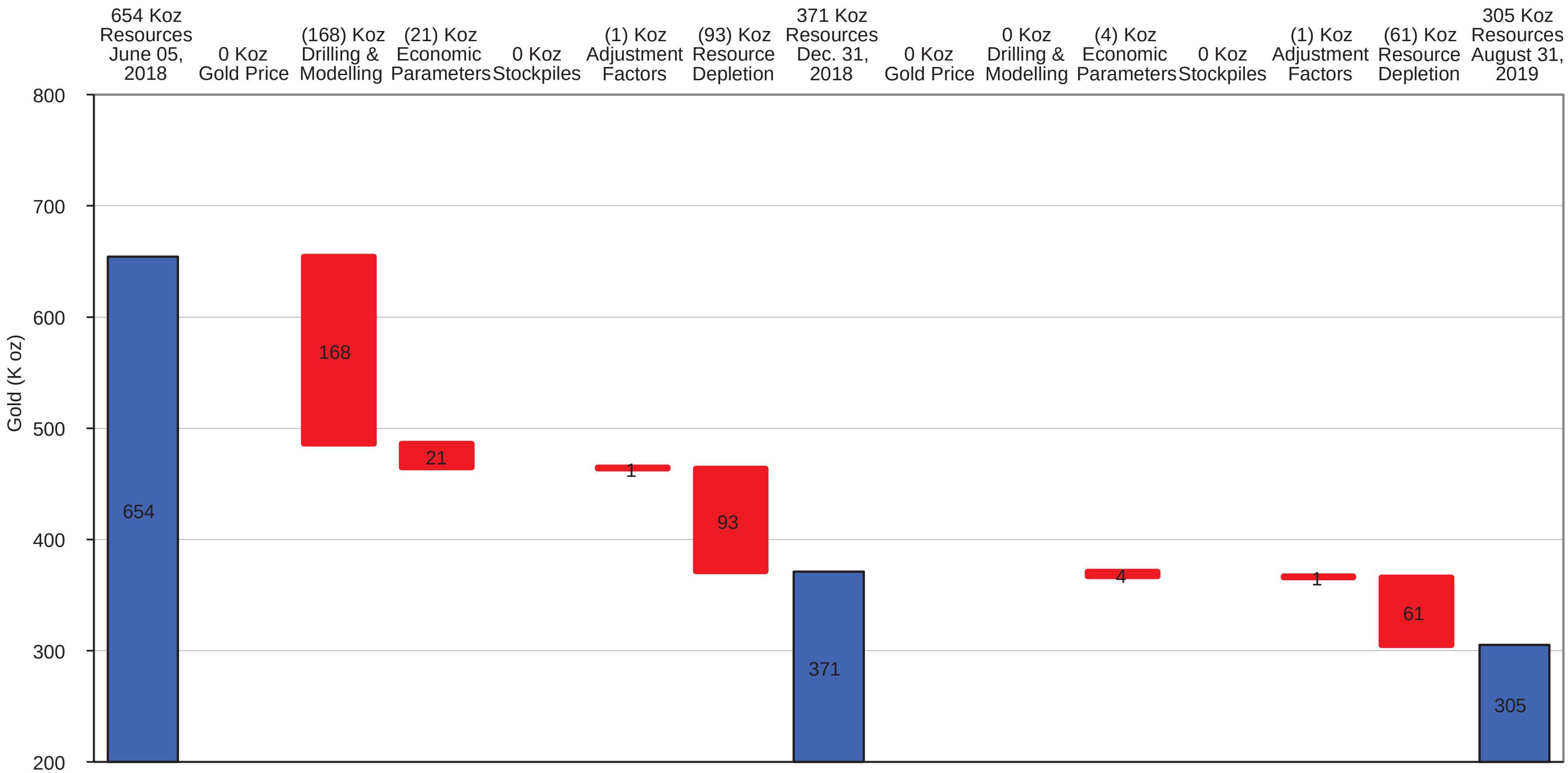
Essakane Gold Mine

Sahel Region, Burkina Faso

Graph Showing Evolution of the EMZ Mineral Resource from June 5, 2018 to August 31, 2019

January 2020

Source: IAMGOLD, 2019.



IAMGOLD Corporation

Figure 14-30

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Graph Showing Evolution of the Falagountou Mineral Resource from June 5, 2018 to August 31, 2019

January 2020

Source: IAMGOLD, 2019.

14.2 GOSSEY DEPOSIT

The resource estimation methodologies, results, and validations for the Gossey deposit are presented in this section.

The Gossey Mineral Resource estimate was prepared in accordance with CIM (2014) definitions and is reported in accordance with NI 43-101 guidelines. Classification, or assigning a level of confidence to Mineral Resources, has been undertaken with strict adherence to CIM (2014) definitions. The Gossey Mineral Resource estimate was prepared by GMSI. In the opinion of the responsible QP, the resource evaluation reported herein is a reasonable representation of the Mineral Resources delineated at Gossey as of May 25, 2018.

The Gossey Mineral Resource estimate at May 25, 2018 is summarized in Table 14-44 and is reported on a 100% basis. The Gossey deposit does not contain a Mineral Reserve.

TABLE 14-44 GOSSEY DEPOSIT PIT CONSTRAINED MINERAL RESOURCE SUMMARY – MAY 25, 2018

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Measured	-	-	-
Indicated	10,454	0.87	291
Total Measured + Indicated	**10,454**	**0.87**	**291**
Inferred	2,939	0.91	85

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade which varies between 0.33 and 0.47 g/t Au depending on material type.
3. Mineral Resources are estimated using an average long-term gold price of US$1,500/oz.
4. Bulk density is estimated by Inverse Distance Squared (ID^2) by weathering type.
5. Mineral Resources are inclusive of Mineral Reserves.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

The Gossey pit constrained Indicated Mineral Resource is estimated to be 10.4 Mt at an average grade of 0.87 g/t Au for a total of 291,000 ounces of gold. The Gossey pit constrained Inferred Mineral Resource is estimated to be 2.9 Mt at an average grade of 0.91 g/t Au for a total of 85,000 ounces of gold. IAMGOLD's attributable Mineral Resources for Gossey are 9.4 Mt totalling 262,000 ounces of gold in Indicated Mineral Resources and 2.6 Mt totalling 77,000 ounces of gold in Inferred Mineral Resources.

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

14.2.1 DATA

The drill hole database was handed over to GMSI on May 10, 2018 and consisted of a single Excel spreadsheet containing down-hole drilling files (collar, survey, assays, lithology, and weathering). All information was subsequently imported into GEOVIA GEMS project and a Leapfrog project provided by Essakane S.A. during the site visit on March 27 – 30th, 2018.

14.2.1.1 DRILL HOLES

The GEOVIA GEMS Gossey project holds 1,016 exploration drill holes of different types covering the Gossey deposit and its surroundings. The mineralization modelling and resource estimation reported herein focus on three mineralized bodies defined along a semi-continuous trend, where 97,959 m were drilled and assayed between 2003 and 2018. Three types of drill holes were used for the estimation; DD, RC, and RCD.

Note that because AC, RAB, and Trench (TR) sampling are more subject to segregation bias, their results are not used in the estimation process. They were used as guide for geological and weathering modelling only.

Approximately 14% of the drilled metres used for the Gossey Mineral Resource estimate come from DD holes, with the remainder from RC holes (86%). Table 14-45 details the drill holes found in the Gossey deposit. The drilling pattern of 50 m (between sections) by 25 m (on-section) in the three main portions of the deposit is presented in Figure 14-31.

TABLE 14-45 SUMMARY OF GOSSEY DRILL HOLES AS OF MAY 2018

Hole Type	Number of Holes	Metres Drilled
DD	60	13,652
RC	673	81,257
Total	**733**	**94,909**

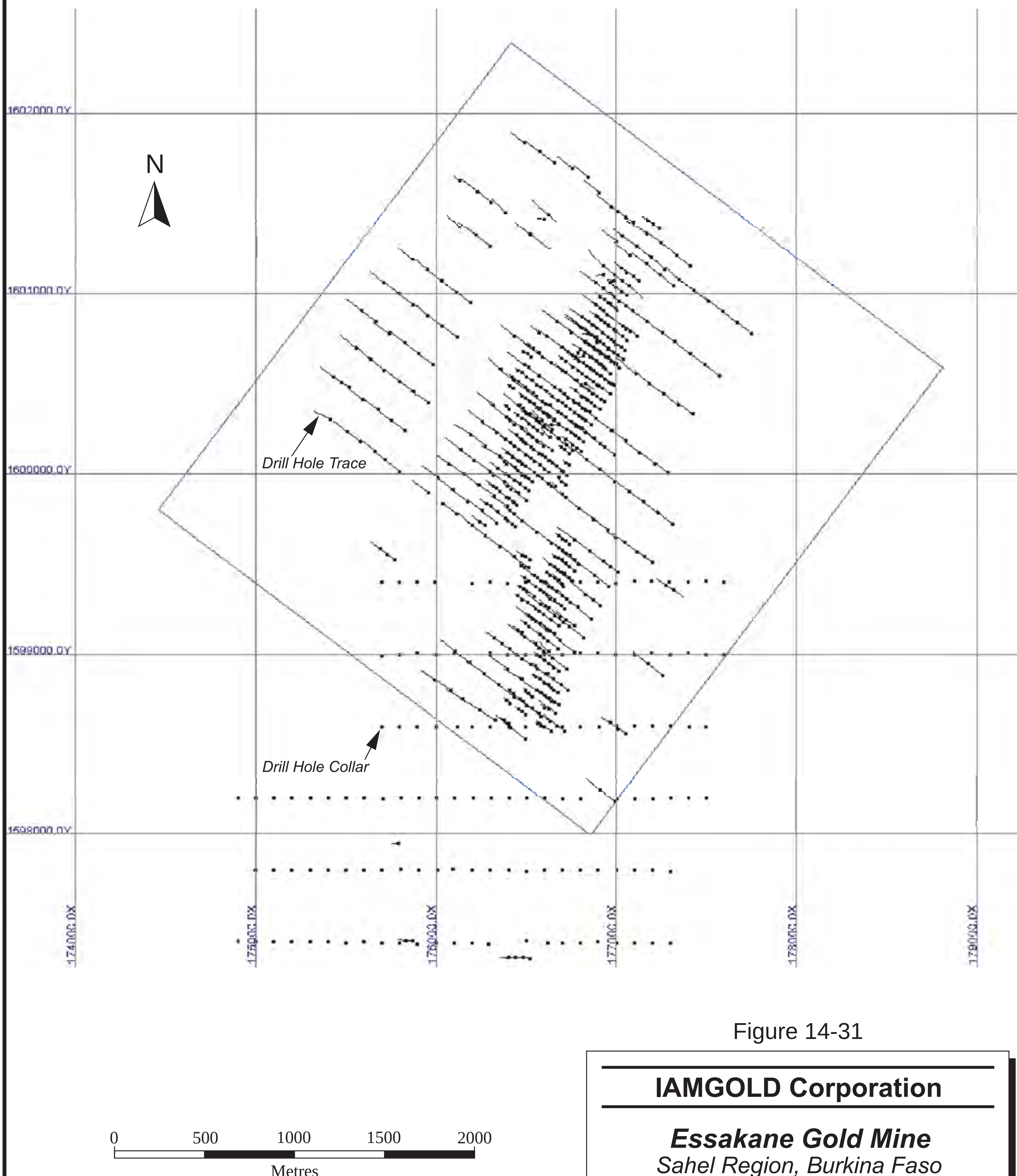


Figure 14-31

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Gossey Deposit Drill Hole Plan - May 2018

January 2020

Source: IAMGOLD, 2019.

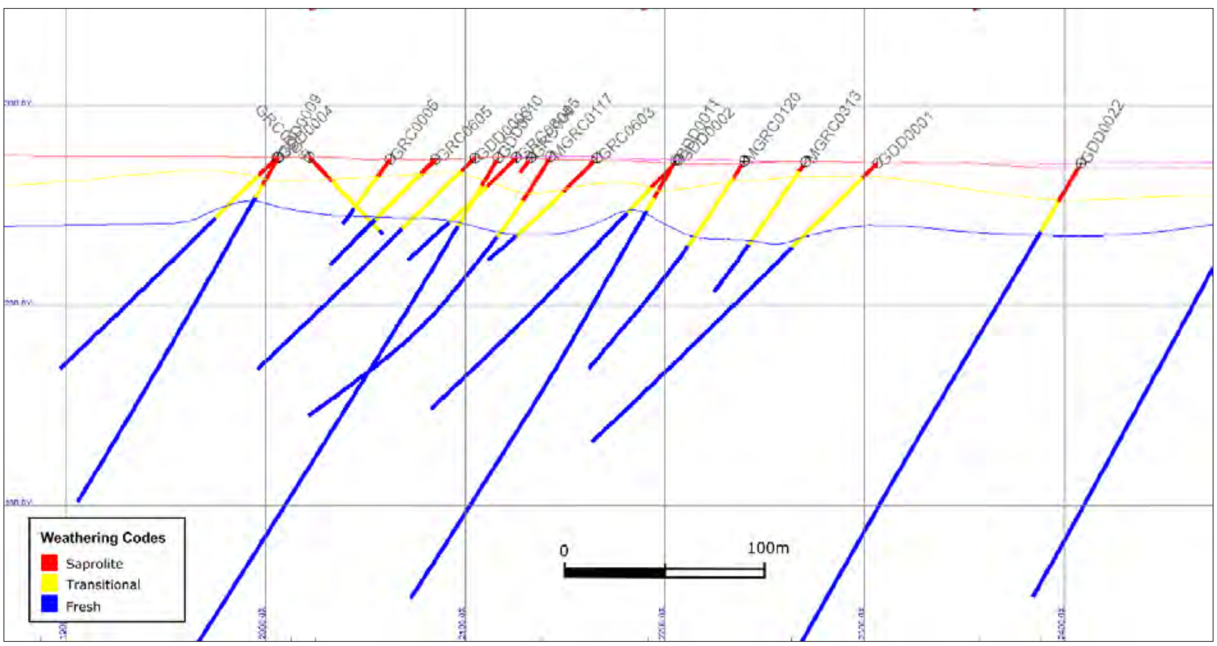
14.2.2 MODELLING

Weathering and mineralization solids were generated for the Gossey resource estimate using Leapfrog Geo (version 4.0) and were based on the drilling database provided.

14.2.2.1 WEATHERING WIREFRAMES

Two surfaces were generated based on new data provided in May 2018; 1) base of saprolite, and 2) base of transition. Each surface represents a lower contact, delimiting layers of weathering. Weathering codes were determined from hardness tests and visual assessment to produce a consistent input for modelling. These wireframes are used for the density model. Figure 14-32 illustrates the provided surfaces in relation with the drill hole information.

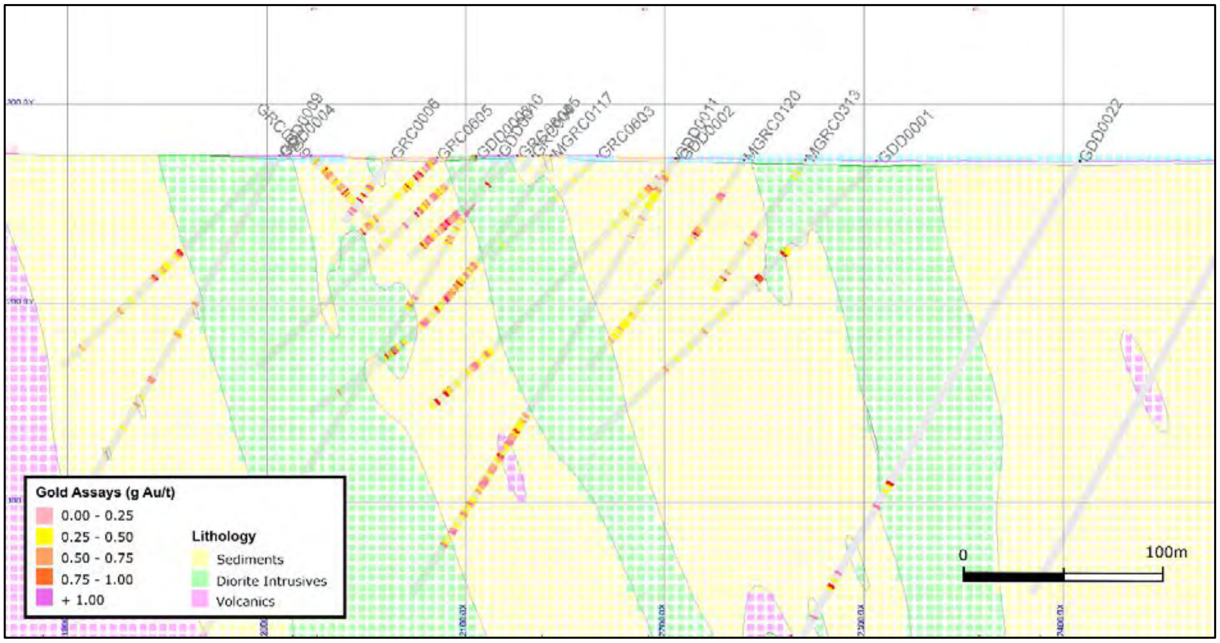
FIGURE 14-32 GOSSEY DEPOSIT WEATHERING PROFILE MODEL - SECTION 18100 LOOKING NE



14.2.2.2 MINERALIZATION ZONES

Due to the strong lithological control on gold mineralization at Gossey (strongest mineralization is found within the sediments close to contacts with the intrusive rocks), GMSI built a lithology model based on the provided grouped logging codes (Figure 14-33). In addition, grade shells were built using arsenic assays for comparative purposes, as there is also a strong link demonstrated between gold and arsenic. At a cut-off of 400 ppm arsenic, the key mineralizing fluid pathways become clear in the datasets and define a clear envelope along the mineralized trend at Gossey. In addition, at a 250 ppm arsenic cut-off, a continuous halo constrains the majority of mineralized intercepts.

FIGURE 14-33 GOSSEY DEPOSIT LITHOLOGY INTERPRETATION AND GOLD ASSAYS – SECTION 18100



14.2.2.3 TOPOGRAPHY SURFACE

A Light Detection and Ranging (LiDAR) topography surface for Gossey was provided to GMSI, which was subsequently resized as a surface elevation grid (SEG) in GEOVIA GEMS.

14.2.3 STATISTICAL ANALYSIS

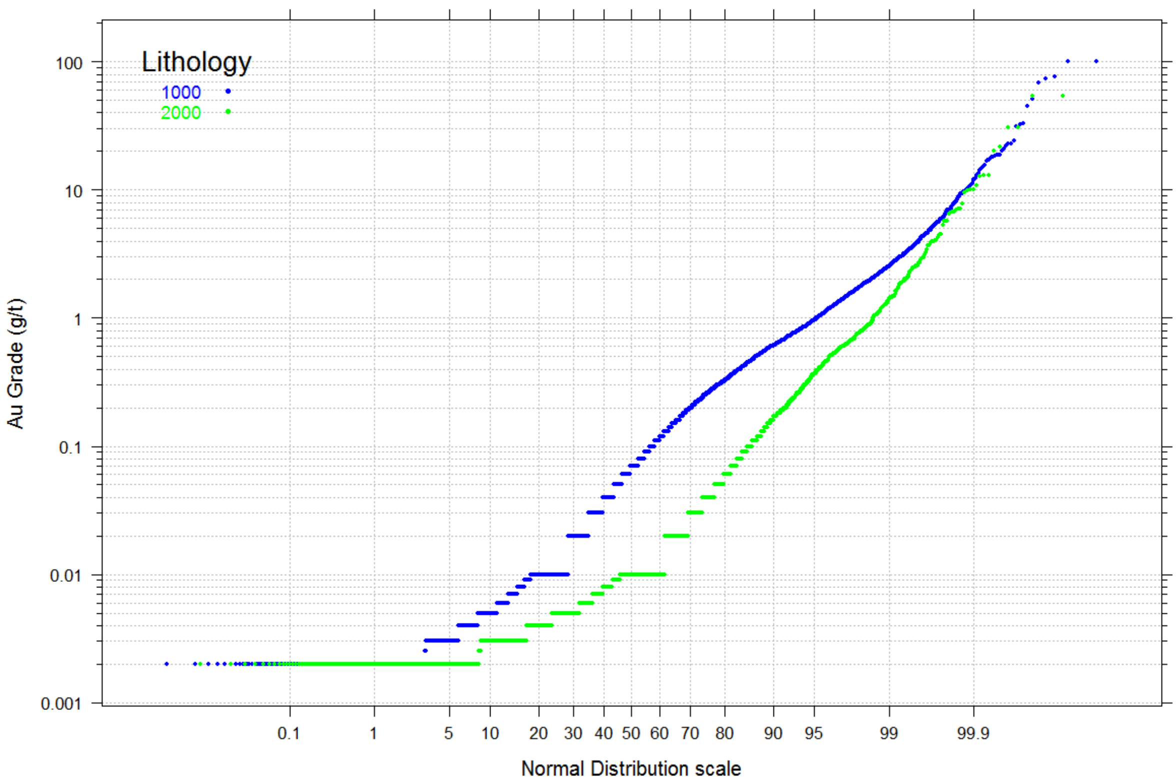
14.2.3.1 STATISTICS OF THE RAW-ASSAYS

Length-weighted statistics of the raw Au assays were computed using R statistical software. Statistics were studied for assays grouped by lithology domains and are presented in Table 14-46. A specific grade capping was applied to the raw assays for each domain based on probability plots of the raw assays (Figure 14-34).

TABLE 14-46 GOSSEY DEPOSIT BASIC STATISTICS OF RAW ASSAYS BY MINERALIZED DOMAIN

Domain	Description	No. of Assays	Gold Raw Assays (g/t Au)						Capping (g/t Au)
			Min	Max	Mean (uncapped)	Median	Standard Deviation	CoV	
1000	Arenites	52,926	0.0005	100.00	0.193	0.0200	1.098	5.70	20
2000	Diorite	27,528	0.0005	54.26	0.042	0.0005	0.627	14.92	8
3000	Argillite	2,400	0.0005	19.00	0.050	0.0005	0.452	9.07	2
4000	Volcanics	5,571	0.0005	18.54	0.050	0.0005	0.360	7.22	2
6000	Regolith	1,129	0.0005	10.20	0.199	0.0700	0.481	2.41	3

FIGURE 14-34 GOSSEY DEPOSIT PROBABILITY PLOTS OF RAW ASSAYS (UNCAPPED) GROUPED BY THE TWO KEY LITHOLOGIES



Note: Arenite (1000) and Diorite (2000)

Due to the strong lithological control on mineralization, no further domaining was considered.

14.2.3.2 COMPOSITING

The capped raw-assays were composited into 2.5 m run lengths (down-hole) within each of the mineralization domains. The determination of composite length was based on assay average length (1.0 m), the thickness of mineralization domains, and the likely selectivity during mining. Composites were subdivided by domain, and all composite lengths were used in the estimation of the blocks.

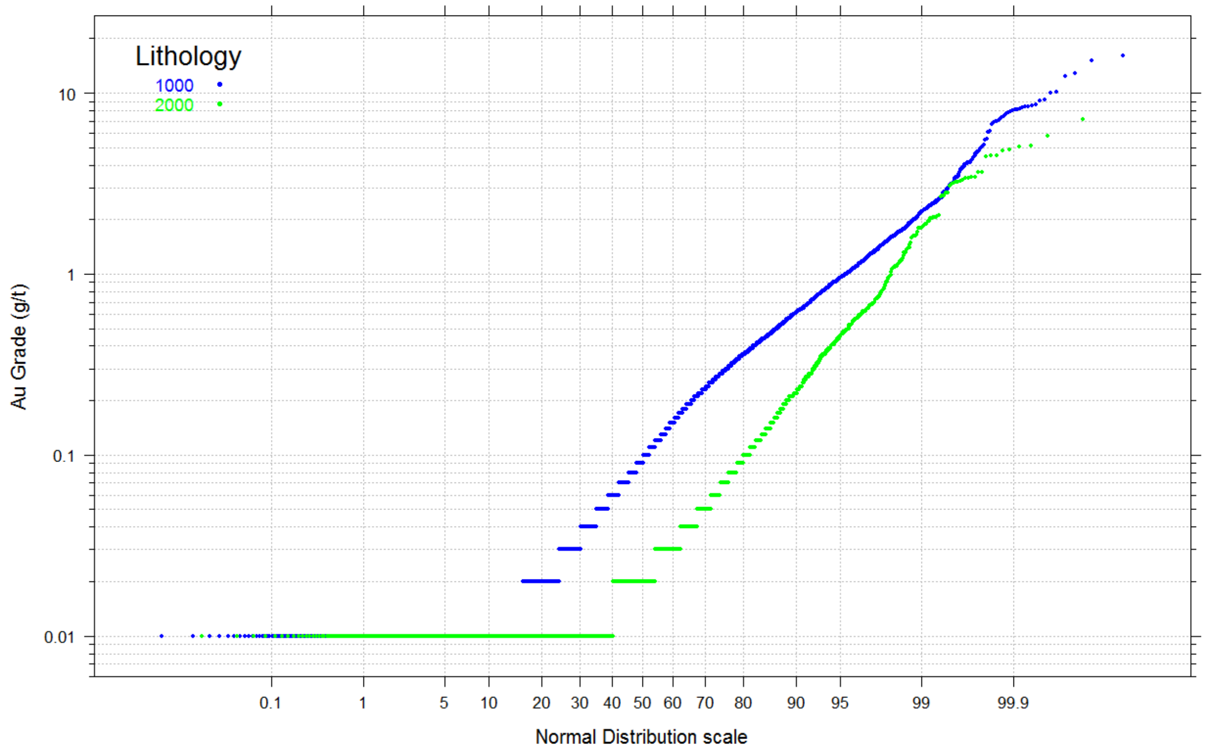
Descriptive statistics of the 2.5 m composites generated within the mineralization domains are summarized in Table 14-47.

TABLE 14-47 GOSSEY DEPOSIT BASIC STATISTICS OF COMPOSITES BY LITHOLOGY

Domain	No. of Composites	Gold (g/t Au)					
		Min	Max	Mean (capped)	Median	Standard Deviation	CoV
1000	22,822	0.0005	16.00	0.182	0.0300	0.489	2.69
2000	12,162	0.0005	7.12	0.038	0.0005	0.225	5.92
3000	1,061	0.0005	1.64	0.041	0.0005	0.154	3.72
4000	2,596	0.0005	1.61	0.044	0.0005	0.119	2.68
6000	834	0.0005	8.00	0.179	0.0800	0.406	2.27

Figure 14-35 shows the probability distributions of the 2.5 m composites for the two key lithologies (Arenite and Diorite).

FIGURE 14-35 GOSSEY DEPOSIT PROBABILITY PLOT OF 2.5 M COMPOSITES GROUPED BY THE TWO KEY LITHOLOGIES (ARENITE (1000) DIORITE (2000))



14.2.3.3 BULK DENSITY DATA

The database includes specific gravity measurement from 13,318 samples, of which 69% are derived from drill core, with the remaining 31% derived from RC drilling. Table 14-48 summarizes the basic statistics obtained from these samples. Samples were subdivided by weathering profile and were analysed on this basis.

TABLE 14-48 GOSSEY DEPOSIT STATISTICS OF SPECIFIC GRAVITY SAMPLES

Weathering Profile	Weathering Code	Average Bulk Density (g/cm^3)	Number of Observations
Regolith	6000	2.29	58
Saprolite	1	2.29	986
Transition	2	2.51	1,516
Fresh Rock	3	2.72	10,816

14.2.3.4 VARIOGRAPHY

Mineral Resource classification at Gossey was based predominantly on a drill spacing of 50 m between sections and 25 m on-section (Figures 14-31 and 14-36), which is similar to surrounding satellite deposits such as the Falagountou deposits. Variography studies undertaken at the surrounding satellite deposits have shown a very high nugget variance, and variogram ranges are difficult to interpret. GMSI also considered other factors such as the quality of the geological and structural interpretation and the quality of the sampling and assay data to assign resource categories.

14.2.4 BLOCK MODELLING

A single block model was constructed containing the Gossey deposit. The block model was rotated 37 degrees clockwise from north around the origin to align with drill sections. The block model was created in the GEOVIA GEMS version 6.8.1 database environment.

14.2.4.1 BLOCK MODEL PARAMETERS

The drilling pattern, the thickness, the continuity of mineralization domains, and the open pit mine planning considerations were factors in the choice of block dimensions. The block model parameters are summarized in Table 14-49.

TABLE 14-49 GOSSEY DEPOSIT BLOCK MODEL SETTINGS

Axis	Origin and Rotation (m)	Block Size (m)	No. of Blocks	
X	174,460	5	600	Columns
Y	1,599,800	10	325	Rows
Z	300	5	60	Levels
Rotation	37.00			

Note:
1. Block Model Project “May 18, 2018_10 m”

Additionally, a series of attributes required during the block modelling development were incorporated into the block model project. Table 14-50 presents the list of attributes found in the block model project “May 18_10 m” in the Standard folder.

TABLE 14-50 LIST OF ATTRIBUTES FOUND IN THE GOSSEY DEPOSIT BLOCK MODEL "MAY 18_10 M"

Folder Name	Model Name	Description
	LF_Weathering	Weathering profile coding (ovb, sap., trans., rock)
	LF_LITHO	Domain coding (shown in Table 14-46)
	DENS_FINAL	Specific gravity (g/cm^3)
Standard	LIT_AuCAP	Inverse Distance Cubed gold grades (g/t) – Capped Assays – Used for final resource figures
	LIT_CATEG	Resource classification (2 = Indicated, 3 = Inferred, 4 = Blue Sky)
	LIT_As	Inverse Distance Cubed arsenic (ppm)
	LIT_Pass	Estimation pass

14.2.4.2 ROCK TYPE MODEL

The rock type model, or domain coding, was built from the lithology solids presented in Section 14.2.2 and acted as hard boundaries. Table 14-51 lists all the rock coding associated to the domains developed from the solids and used in the block model. No block percentages were recorded in the block model.

TABLE 14-51 GOSSEY DEPOSIT ROCK CODES

Code	Lithology
1000	Arenites
2000	Diorite
3000	Argillite
4000	Volcanics
6000	Regolith

Note:
1. File "LF_LITHO Attributes"

14.2.4.3 DENSITY MODEL

The assignment of bulk density was undertaken using a single estimation pass within each of the four weathering profiles (regolith, saprolite, transition, and fresh rock). The density estimation pass is described below:

- A horizontally orientated search ellipse with the dimensions 25 m (X) x 25 m (Y) x 10 m (Z)
- Minimum of two density readings to estimate the block
- Inverse Distance Squared (ID2) interpolation method
- Hard boundaries between the weathering domains.

Blocks which were not estimated during the initial estimation pass were assigned the average bulk density for each weathering domain according to Table 14-48.

14.2.4.4 GRADE ESTIMATION METHODOLOGY

The interpolation technique selected for the estimation of gold grades at the Gossey deposit is the ID^3 method. No other interpolation methods were tested at this stage of the project. The ID^3 method was judged suitable to replicate composite grades throughout the deposit.

Grade estimates were produced using the 2.5 m composites. Mineralized domains were considered as hard boundaries through each interpolation step. Each domain was estimated using only composites pertaining to the domain in question. GEOVIA GEMS version 6.8.1 software was used for the estimate.

Four interpolation passes were used iteratively to estimate the blocks for Gossey deposit. The sample selection methodology for each pass is summarized below:

- **First Pass:** A minimum of nine and a maximum of 20 composites within the selected search ranges. A maximum of three composites per hole could be used for any block estimate.
- **Second Pass:** A minimum of six and a maximum of 20 composites within the selected search ranges. A maximum of three composites per hole could be used for any block estimate. Only blocks which were not estimated during the first pass could be estimated during the second pass.
- **Third Pass:** A minimum of three and a maximum of 20 composites within the selected search ranges. A maximum of three composites per hole could be used for any block estimate. Only blocks which were not estimated during the first and second passes could be estimated in the third pass.
- **Fourth Pass:** A minimum of one and a maximum of 20 composites within the selected search ranges. A maximum of three composites per hole could be used for any block estimate. Only blocks which were not estimated during the first and second passes could be estimated in the third pass

Search ellipse ranges were determined considering the drill hole spacing and the desired level of extrapolation of gold grade (for Mineral Resource categorization purposes). The search ellipse distances progressively increase from 25 m x 25 m in the first interpolation pass towards longer distances of up to 100 m by 100 m in the fourth pass, while the thickness of the search ellipse remains constant at 10 m throughout the interpolation process.

No restrictions were placed on the search ellipse to limit the influence of high-grade composites.

The various profiles of interpolation and search rectangles for gold composites utilized in the estimation of the Gossey resource are tabulated in Table 14-52 and Table 14-53.

TABLE 14-52 GOSSEY DEPOSIT INTERPOLATION PROFILE SETTINGS FOR RESOURCE ESTIMATION

Profile Name*	Pass	Sample			Target Rock Code	Ellipse Name
		Min	Max	Max per Hole		
LIT_AU_1	1	9	20	3	1000 2000 3000 4000 6000	PASS_1
LIT_AU_2	2	6	20	3		PASS_2
LIT_AU_3	3	3	20	3		PASS_3
LIT_AU_4	4	1	20	3		PASS_4

TABLE 14-53 GOSSEY DEPOSIT SAMPLE SEARCH RECTANGLES SETTINGS FOR RESOURCE ESTIMATION

Pass	Rotation			Anisotropy Range (m)		
	Z	Y	Z	X	Y	Z
1	20°	-25°	0°	25	25	10
2				50	50	10
3				75	75	10
4				100	100	10

The grades interpolated were saved in attributes “LIT_AuCAP” (capped Au grade estimate). The Mineral Resources discussed in the next sections refer to the capped gold grades.

14.2.5 CLASSIFICATION AND RESOURCE REPORTING

The Gossey Mineral Resource estimate was classified in accordance with the CIM (2014) definitions (see “Classification and Resource Reporting” under “14.1.1 EMZ Deposit”).

In addition, the classification of interpolated blocks was undertaken by considering the following criteria:

- Quality and reliability of drilling and sampling data
- Distance between sample points (drilling density)
- Confidence in the geological interpretation
- Continuity of the geologic structures and the continuity of the grade within these structures

- Statistics of the data population
- Quality of assay data

The Gossey Mineral Resources were classified according to the above-mentioned criteria which also directed the choice of the search parameters for each interpolation pass during the block estimation.

No Measured Mineral Resources are estimated in the Gossey deposit.

Indicated Mineral Resources are the blocks estimated from the first and second passes.

Inferred Mineral Resources are the blocks estimated from the third pass.

Figure 14-36 shows the distribution of the resource categories in the Gossey deposit for the ID^3 interpolation method. Indicated Mineral Resources are essentially concentrated in the southern, central, and northern part of the deposit where the drill hole density is the highest. Inferred Mineral Resources are peripheral to Indicated Mineral Resources, however, they are mainly located in the extensions at depth of the deposit where drill spacing is wider and confidence in grade continuity is low.

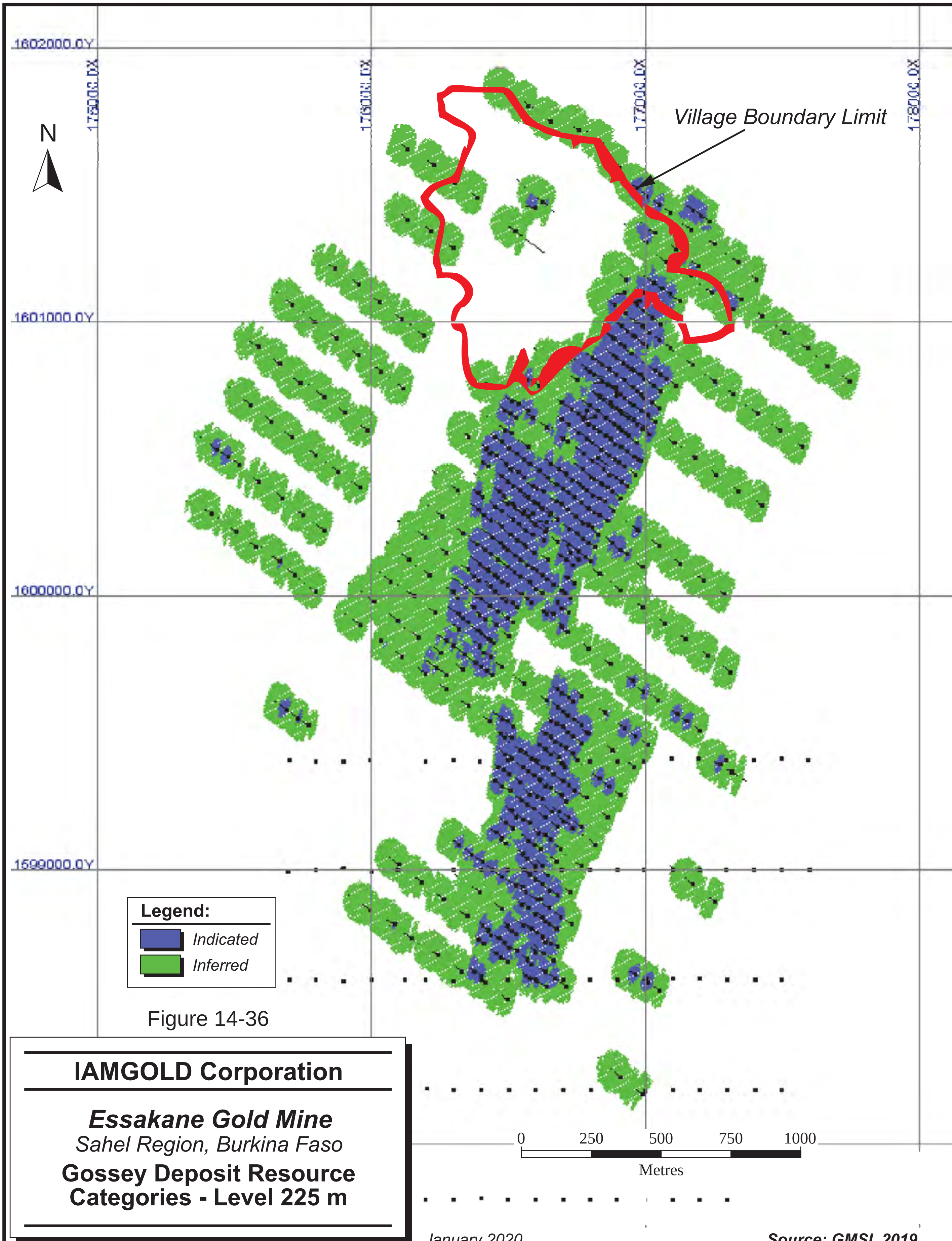


Figure 14-36

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Gossey Deposit Resource Categories - Level 225 m

January 2020

Source: GMSI, 2019.

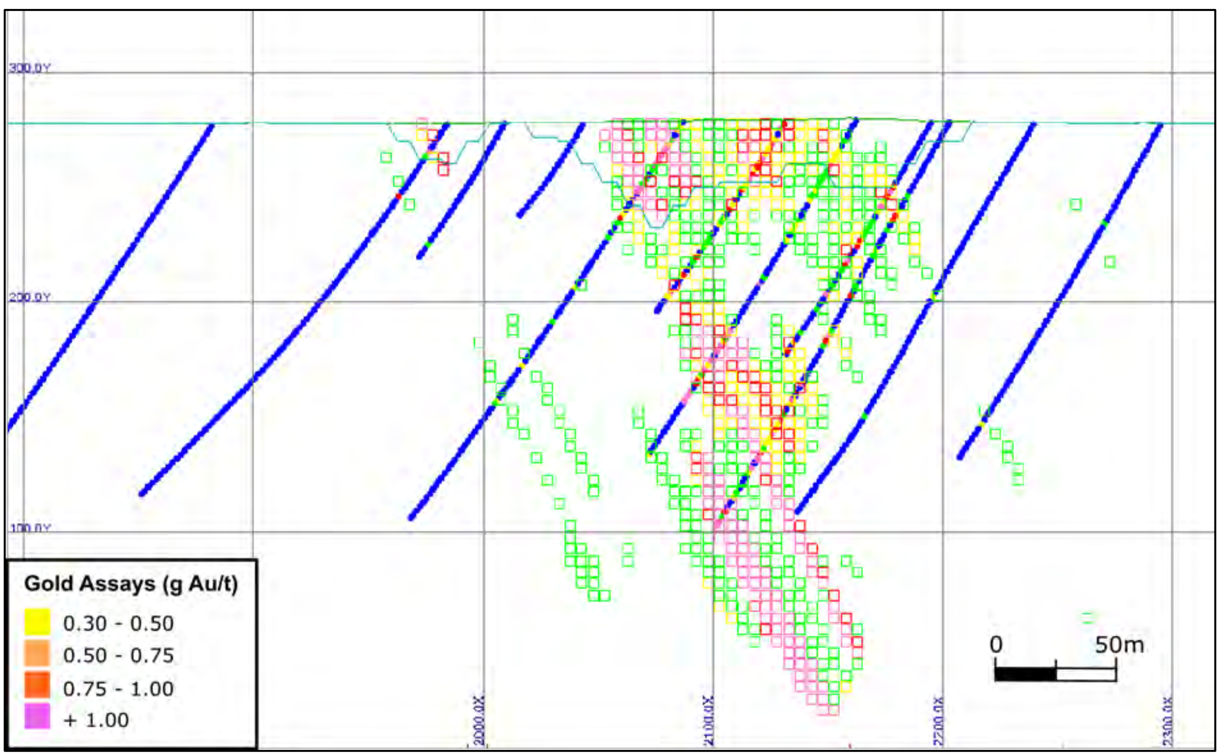
14.2.6 BLOCK MODEL ESTIMATION VALIDATION

Multiple validations were completed on the block model to ensure the block model is a good representation of the composites. The validation process included visual checks, statistical validation of the model, and swath plots.

14.2.6.1 VISUAL VALIDATION

The visual checks consisted of visualization of slices of the block model, mineralized zones, and drill hole database. The slicing was performed vertically on 50 m intervals and horizontally on 5 m intervals. The data source was visually compared with the different model attributes (rock type or domains, percent, weathering type, density, and Au grades) throughout the strike length of the deposit (Figure 14-37). The weathering profile layers, mineralization domains, and associated percentages are well represented in their proper attribute model. The ID^3 method of resource estimation was found to be a good representation of the drill hole composites.

FIGURE 14-37 SECTION 17450 SHOWING COMPOSITE AND BLOCK MODEL GOLD GRADES WITH THE US$1,500/OZ AU INDICATED AND INFERRED PIT OPTIMIZATION OUTLINE



14.2.6.2 GLOBAL STATISTICAL VALIDATION

Comparative statistics were generated for the composites and the blocks for each domain at Gossey considering only blocks within the Indicated and Inferred categories. The results are presented in Table 14-54.

Due to the lognormal distribution of gold grades, outliers in the data have a significant impact on global arithmetic means. In addition, composite clustering and over/under representation of grade domains can impact statistical validation when considering individual domains. For these reasons, arithmetic means should be used only as a guide in validating Mineral Resources. It can be advantageous to compare the grade distributions of composites and blocks across key grade domains (not just descriptive statistics) to better understand the estimation performance.

The CoV values suggest that an acceptable level of smoothing has occurred, largely due to ID^3 interpolator used during grade estimation. Mean gold grades correlate well for domains 1000 and 2000, which contain most of the metal.

TABLE 14-54 COMPARATIVE STATISTICS BETWEEN 2.5 M COMPOSITES AND BLOCKS WITHIN THE GOSSEY DEPOSIT INDICATED AND INFERRED MINERAL RESOURCE CATEGORIES

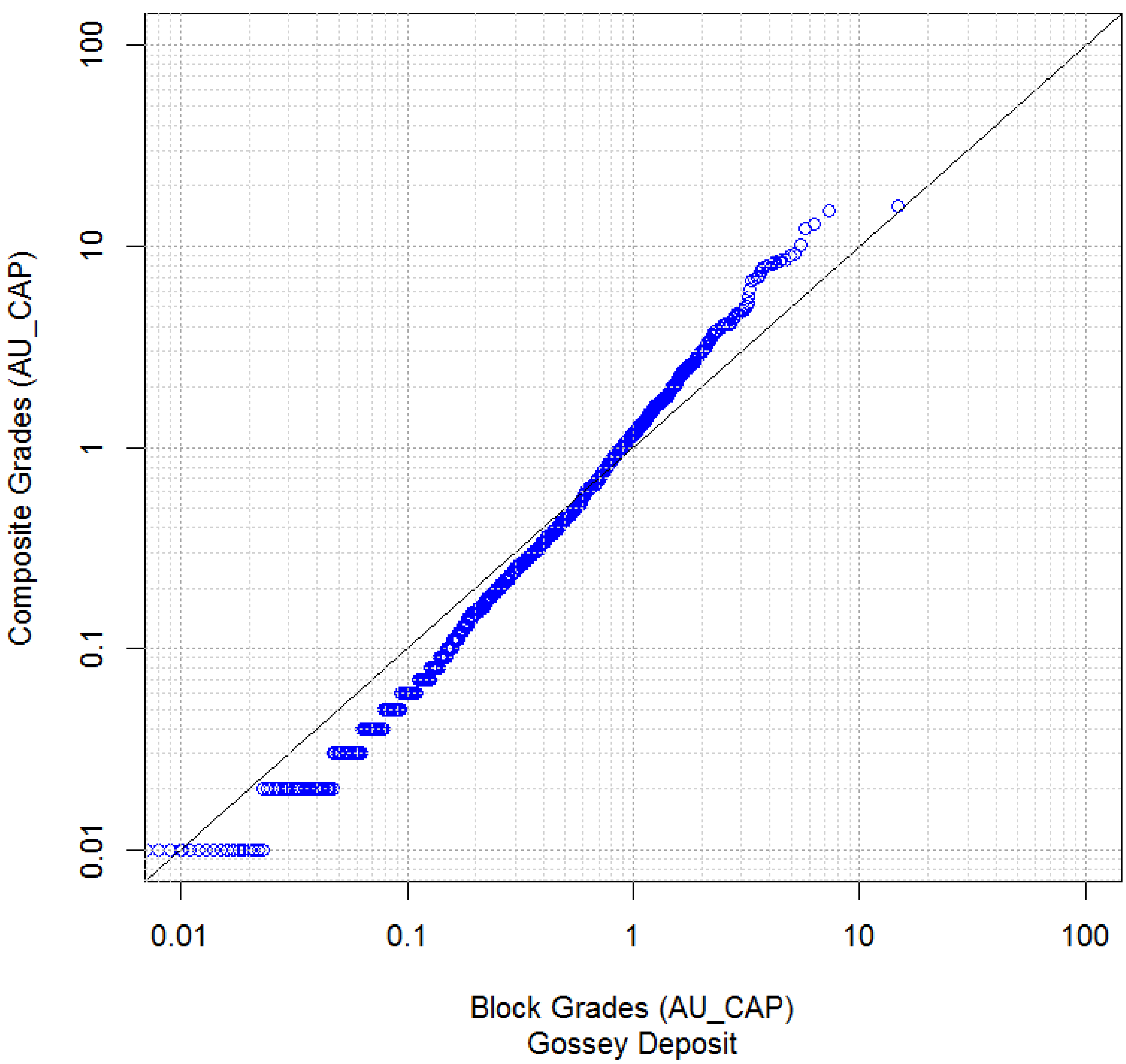
Domain	No. of Composites	Composites (g/t Au)		CoV Composites	Number of Blocks	Blocks (g/t Au)		CoV Blocks	Reduction in CoV	Proportion of Total Composites	Proportion of Total Blocks
		Mean	Median			Mean	Median				
1000	3,698	0.48	0.22	1.95	47,950	0.45	0.29	1.24	36%	67.9%	64.7%
2000	1,196	0.12	0.01	3.93	16,909	0.12	0.01	2.64	33%	22.0%	22.8%
6000	459	0.26	0.12	1.98	8,379	0.19	0.12	1.26	36%	8.4%	11.3%
Other	90	-	-	-	904	-	-	-	-	1.7%	1.2%

Note:

1. Grouped by lithology domain within the US$1,500/oz Au Indicated and Inferred pit optimization

In addition to descriptive statistics, Q:Q plots (a scatterplot created by plotting two sets of quantiles against one another) were generated to assess the distribution of gold grades of composites against blocks on a domain-by-domain basis. These plots are useful in assessing the degree of smoothing (conditional bias) observed during the grade estimation process and can identify any significant over/under estimation of grades. Figure 14-38 shows a Q:Q plot of composite gold grades (Y axis) versus block gold grades (X-axis) for domain 1000.

FIGURE 14-38 Q:Q PLOT OF GOSSEY DEPOSIT GOLD GRADES FROM INDICATED AND INFERRED BLOCKS VERSUS 2.5 M COMPOSITES



Note: For Domain 1000 within the US$1,500/oz Au Indicated and Inferred pit optimization

It can be seen that distribution of block gold grades and composite gold grades for domain 1000 are comparable, and the degree of observed smoothing implies a good local estimation of gold grades within the block model. No significant bias towards the composites or blocks was observed.

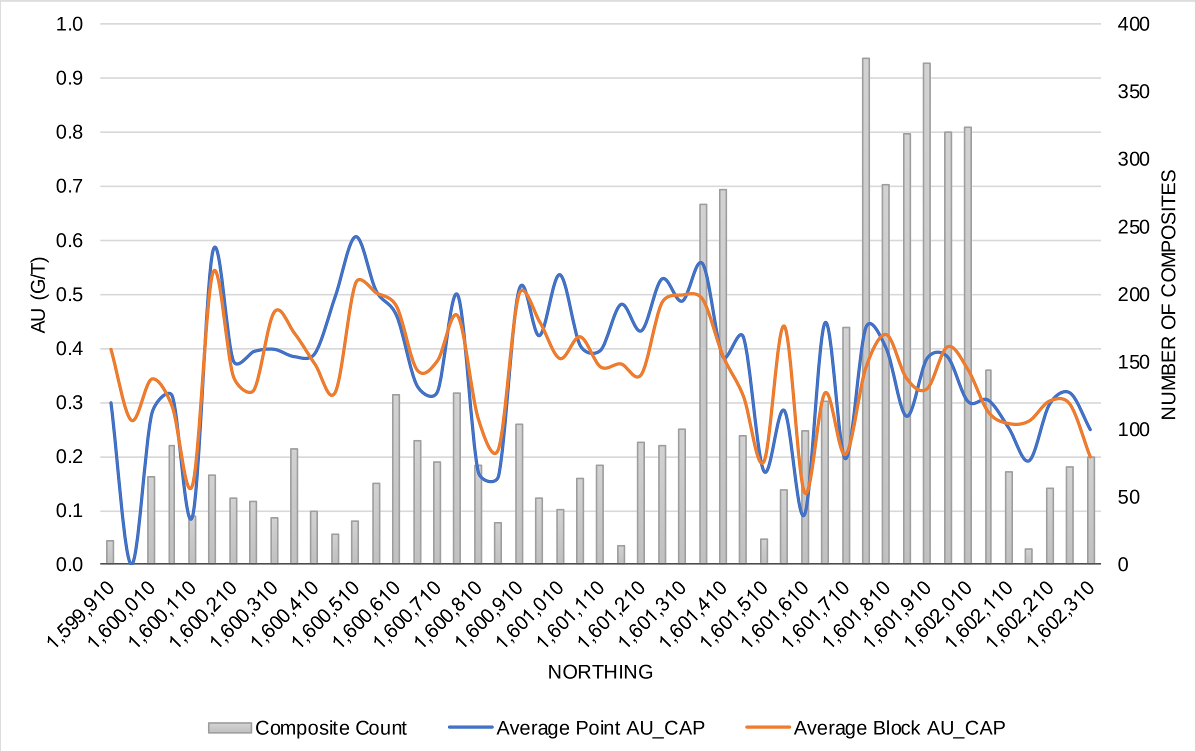
14.2.6.3 LOCAL STATISTICAL VALIDATION - SWATH PLOTS

The swath plot method is considered a local validation, which works as a visual mean to compare estimated block grades against composite grades within a 3D moving window. In addition, it can identify possible bias in the interpolation (i.e. over/under estimation of grades).

Swath plots were generated for domain 1000 at increments of 50 m (Easting) by 50 m (Northing) by 20 m (Elevation) for gold grade. Figure 14-39 illustrates the swath plot for Indicated Mineral Resources within the US$1,500/oz Au MII pit optimization by Northing. Swath plots were created by grouping regolith, saprolite, transition, and fresh rock weathering profiles and isolating only blocks and composites pertaining to domain 1000.

Peaks and lows in estimated gold grades generally follow peaks and lows in composite (or point) grades in well informed areas of the block model, whereas less informed areas can occasionally show some discrepancies between the grades (such as 1601160 mN).

FIGURE 14-39 SWATH PLOT OF GOSSEY DEPOSIT INDICATED MINERAL RESOURCES WITHIN THE US$1,500/OZ AU INDICATED AND INFERRED PIT OPTIMIZATION BY NORTHING – DOMAIN 1000



Reconciliation between composites and block grades at the southern end of the Gossey deposit show a good local estimate.

14.2.6.4 DISCUSSION ON BLOCK MODEL VALIDATION

Overall, the Gossey block model is a good representation of composite gold grades used in the estimation. Global statistical validations show the degree of smoothing is within acceptable limits and no significant over/under-estimation of gold grades has occurred. Local statistical validations show good local correlation of block and composite gold grades, and no excessive extrapolation of grades was observed.

14.2.7 GOSSEY DEPOSIT PIT CONSTRAINED MINERAL RESOURCES

To establish a Mineral Resource estimate for the Gossey deposit, an open pit development scenario is the most suitable due to the geology/geometry, tonnage, and grade. The deposit model was imported into Whittle to determine optimal pit shells based on the Lerchs-Grossman algorithm. The method works on a block model of the orebody, and progressively constructs lists of related blocks that should, or should not, be mined. The method uses the values of the blocks to define a pit outline that has the highest possible economic value, subject to the required pit slopes defined as structure arcs in the software.

At this stage of the Gossey project, all blocks classified as Indicated and Inferred were utilized in the pit optimization process, with a sensitivity analysis run on Indicated, Inferred, and material that could potentially be upgraded to Inferred status with further drilling.

This analysis requires several input parameters such as slope constraints, gold prices, process recoveries, and operating costs. A cut-off grade for each weathering type of mineralized rock (saprolite (including regolith), transition, and fresh rock) was determined in this process.

14.2.7.1 OPTIMIZATION PARAMETERS

The pit optimization parameters were consolidated by the Essakane engineering department with inputs provided by other departments to calculate the break-even cut-off grades for each ore type. The cut-off grades are estimated on the basis of a long-term sustainable CIL plant throughput of 11.0 Mtpa in fresh rock. The cut-off grades, by ore type, are summarized in Table 14-55.

TABLE 14-55 SUMMARY OF GOSSEY CUT-OFF GRADES AT US$1,500/OZ AU

Cut-off Grade	Saprolite	Transition	Fresh Rock
Gossey (g/t Au)	0.33	0.42	0.47

Given the assay technique used for the Gossey drilling campaigns involves a cyanide leach extraction (i.e. LeachWell) of gold from the mineralized samples, thereby confirming the amount of recoverable gold, per sample interval, it was reasoned to conservatively assume recoveries similar to other deposits in the region. As a result, the CIL plant metallurgical recovery assumptions for the Gossey deposit are fixed at 95% for saprolite 92% for transition, and 91.5% for fresh rock.

The mine operating cost inputs for pit optimization are derived from Essakane mining costs and productivities.

Details for the cut-off grades for the Gossey deposit are presented in Table 14-56.

TABLE 14-56 SUMMARY OF GOSSEY DEPOSIT PIT OPTIMIZATION PARAMETERS AND CUT-OFF GRADES

Rock Type	Units	Saprolite	Transition	Fresh Rock
Metallurgical recovery	(%)	95.0%	93.0%	92.5%
Processing rate	(Mtpa)	15.00	12.5	12.1
Avg. power consumption (grinding)	(kWh/t)	5.0	9.0	17.0
Avg. power consumption (fixed)	(kWh/t)	9.1	10.9	11.3
Total fixed processing costs	(MUS$/year)	55.6	58.5	58.5
Processing fixed cost	(US$/t treated)	3.71	4.69	4.85
Other variable costs	(US$/t treated)	0.07	0.07	0.07
Secondary crushing circuit	(US$/t treated)	0.00	0.00	0.00
Liners and grinding media	(US$/t treated)	0.62	1.05	1.24
Reagents	(US$/t treated)	1.76	1.95	2.01
Power	(US$/t treated)	2.09	2.95	4.20
Total Processing Cost	(US$/t treated)	**8.25**	**10.70**	**12.36**
Mining dilution	%	8%	8%	8%
Mineralized material premium mining cost	(US$/t treated)	1.25	1.25	1.25
Mineralized material feed	(US$/t treated)	0.04	0.04	0.04
Total fixed G&A costs	(MUS$/year)	48.1	48.1	48.1
G&A cost	(US$/t treated)	3.21	3.85	3.99
Rehabilitation	(US$/t treated)	0.18	0.18	0.18
Stay in Business (SIB) Capital	(US$/t treated)	0.40	0.48	0.50
Operating Consumables	(US$/t treated)	-	-	-
Total Mineralized Material Based Cost	(US$/t treated)	**13.32**	**16.51**	**18.31**
Break Even Mill Feed Cut-off Grade (at US$1500/oz Au)	**(g/t Au)**	**0.33**	**0.42**	**0.47**
Reference Mining Cost by Deposit			**Gossey**	
Rock Type		**Saprolite**	**Transition**	**Fresh Rock**
Total Reference Mining Cost (Mineralized Material)	(US$/t mined)	2.03	2.47	2.74
Total Reference Mining Cost (Waste)	(US$/t mined)	1.83	2.24	2.32
Incremental bench cost	(US$/t per vert. m)	0.0031	0.0031	0.0031
Geotech domains				
Whittle overall slope angles	(Degrees)	30	35	46

14.2.7.2 PIT CONSTRAINED MINERAL RESOURCE

The compilation of the Mineral Resources was constrained using the LiDAR topography surface provided to GMSI, and from a Whittle pit optimization (gold price of US$1,500/oz) on

Indicated and Inferred Mineral Resource categories, combined. Operating costs and recoveries were derived from the Essakane Mine. Surfaces were produced by Louis-Pierre Gignac (GMSI) on May 15, 2018.

The Gossey pit constrained Indicated Mineral Resource is estimated to be 10.4 Mt at an average grade of 0.87 g/t Au for a total of 291,000 ounces of gold (Table 14-57). The Gossey pit constrained Inferred Mineral Resource is estimated to be 2.9 Mt at an average grade of 0.91 g/t Au for a total of 85,000 ounces of gold. Gold grade distribution and resource categorization for Gossey are illustrated in Figure 14-41 and Figure 14-42, respectively.

TABLE 14-57 GOSSEY DEPOSIT PIT CONSTRAINED MINERAL RESOURCES – MAY 25, 2018

		Indicated			Inferred		
Weathering Profile	**Cut-off Grade (g/t Au)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Contained Metal (000 oz Au)**	**Tonnage (000 t)**	**Grade (g/t Au)**	**Contained Metal (000 oz Au)**
Laterite & Saprolite	0.33	3,916	0.66	83	1,464	0.75	35
Transition	0.42	3,467	0.85	94	986	0.97	31
Fresh Rock	0.47	3,071	1.15	114	489	1.23	19
Total	**Varying**	**10,454**	**0.87**	**291**	**2,939**	**0.91**	**85**

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade which varies between 0.33 and 0.47 g/t Au depending on material type.
3. Mineral Resources are estimated using an average long-term gold price of US$1,500/oz.
4. Bulk density is estimated by Inverse Distance Squared (ID^2) by weathering type.
5. Mineral Resources are inclusive of Mineral Reserves.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

The Gossey deposit is located near the Gossey village for which future development of some portions of the deposit may require either: the development of a re-location program, similar to the one that was completed previously at Essakane; or the consideration of a mining buffer zone to restrict mining activities proximal to the village. No mining buffer zone was applied around the nearby Gossey Village. Should a 350 m buffer zone be maintained around the Gossey Village a portion of the Gossey Mineral Resource will be sterilized. Approximately half of the contained gold ounces would be lost due to the buffer zone restraint, however, the resulting strip ratio is slightly less. Figure 14-40 shows the effect of adding the 350 m buffer zone around the Gossey Village.

IAMGOLD Corporation

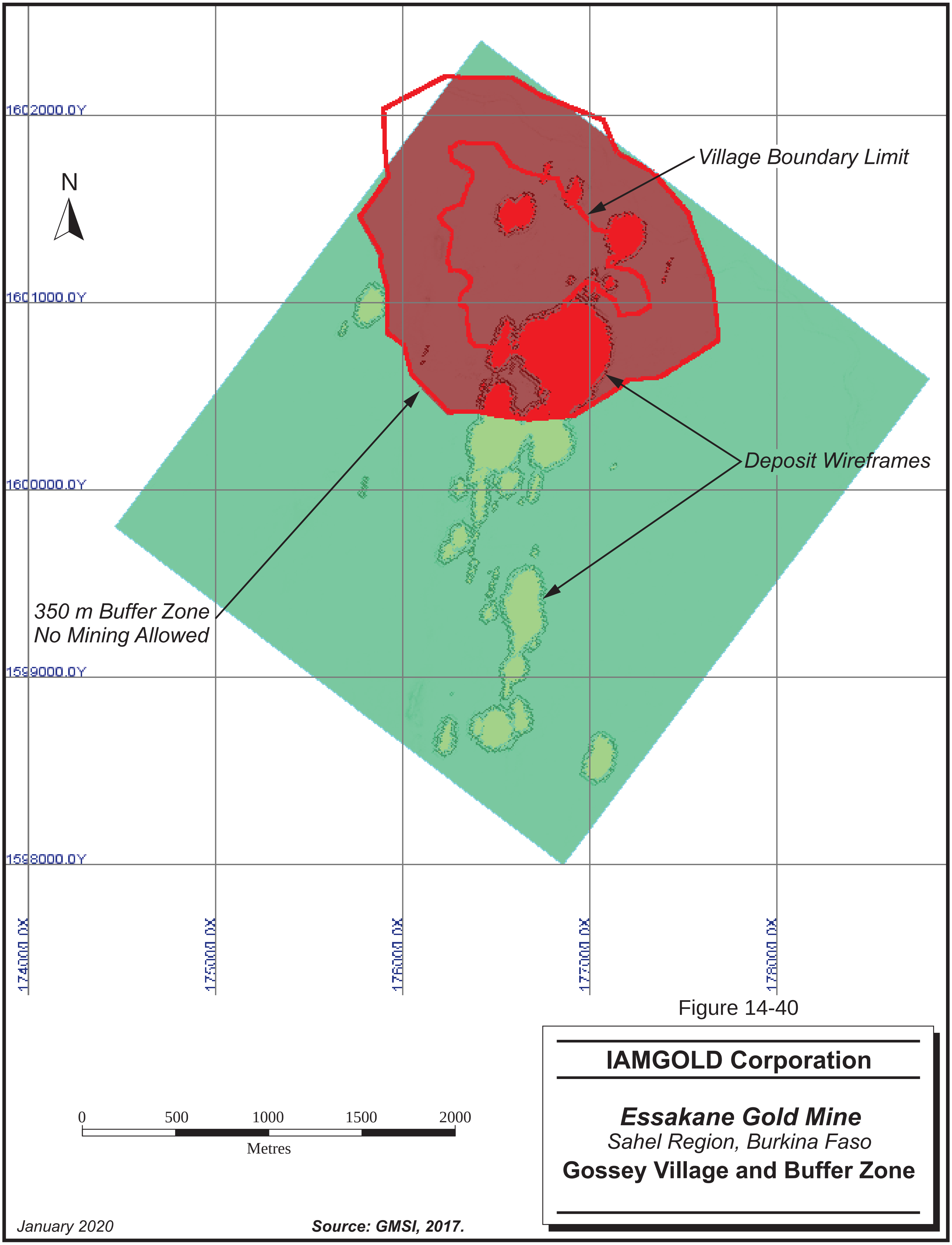


Figure 14-40

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Gossey Village and Buffer Zone

FIGURE 14-41 GOLD GRADE DISTRIBUTION INSIDE THE WHITTLE SHELL OPTIMIZED AT $1,500/OZ AU FOR GOSSEY (INDICATED AND INFERRED BLOCKS)

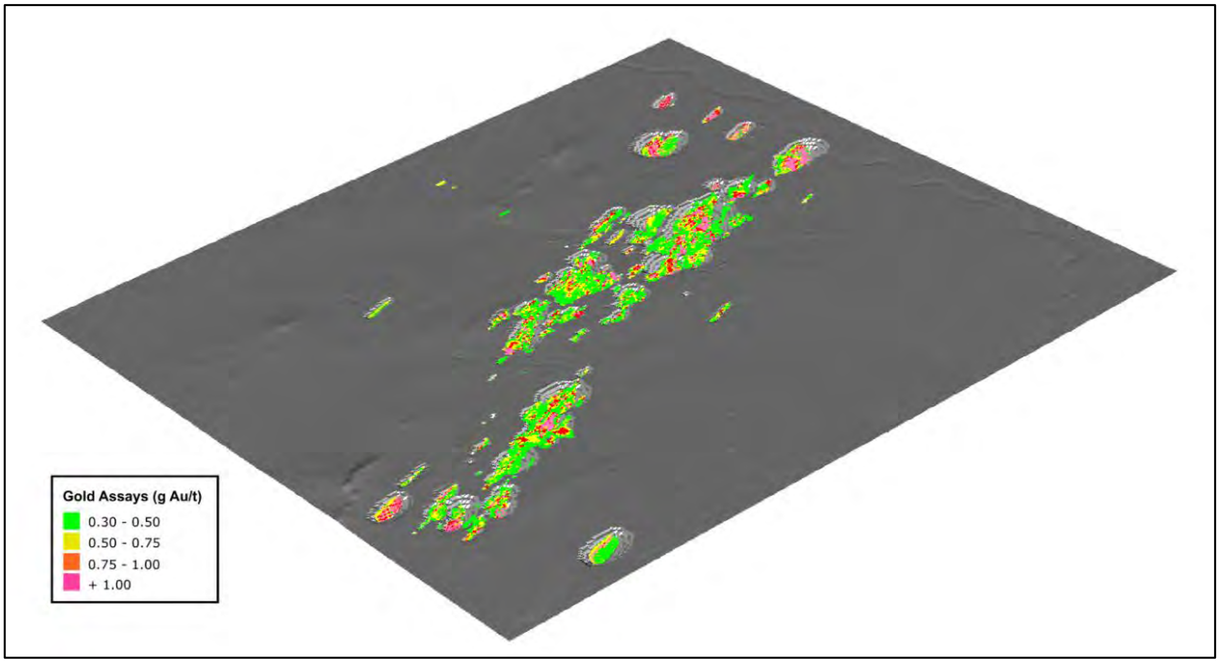
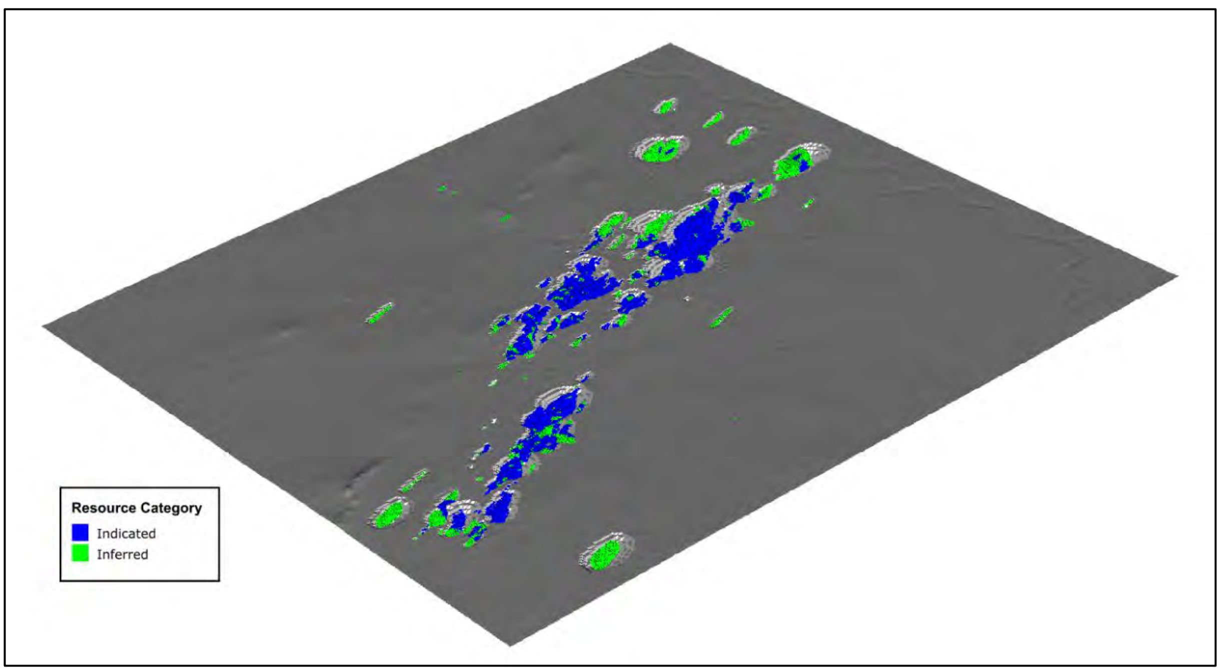


FIGURE 14-42 RESOURCE CATEGORIZATION INSIDE THE WHITTLE SHELL OPTIMIZED AT $1,500/OZ AU FOR GOSSEY (INDICATED AND INFERRED BLOCKS)



14.2.8 MINERAL RESOURCE SENSITIVITY TO CUT-OFF GRADES

Table 14-58 summarizes the sensitivity of the pit constrained Mineral Resources of the Gossey deposit for a series of selected cut-off grades. The sensitivity analysis uses gold cut-off grades ranging between 0.25 g/t Au and 2.0 g/t Au. Figure 14-43 illustrates the grade-tonnage curves for the pit constrained Indicated and Inferred Mineral Resources for Gossey. The sensitivity table and graph use a compilation of the regolith, saprolite, transition, and fresh rock weathering profiles.

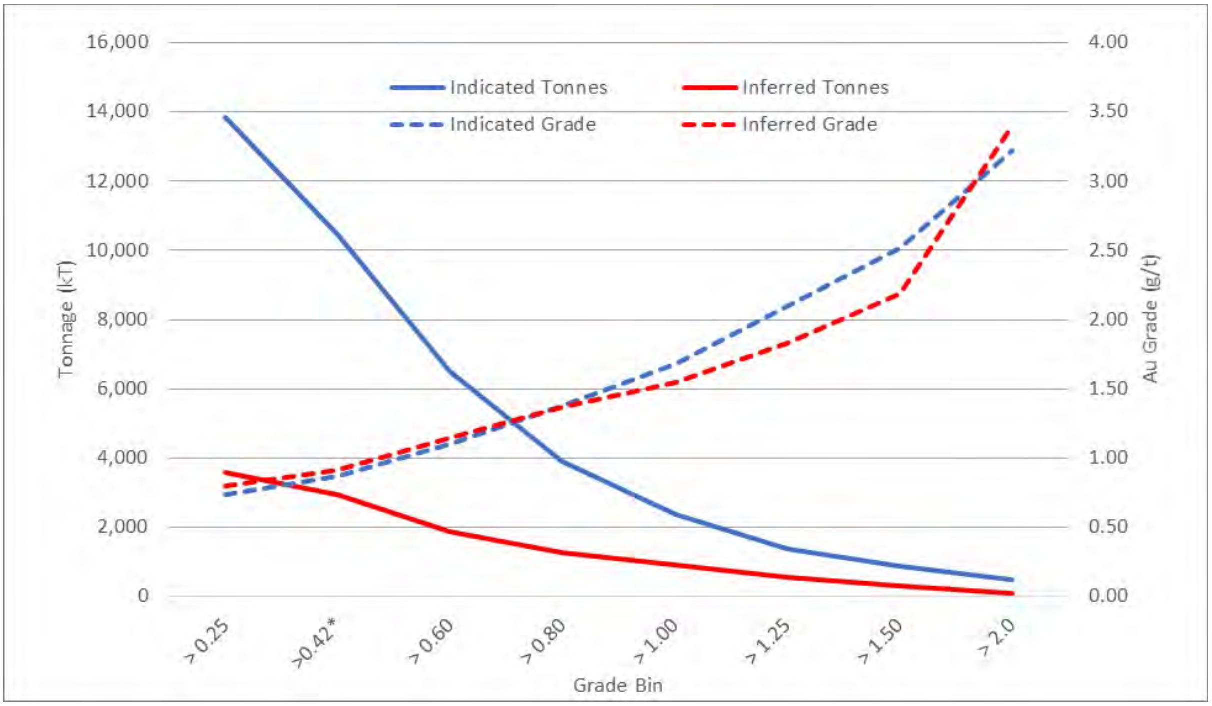
TABLE 14-58 GOSSEY DEPOSIT PIT CONSTRAINED INDICATED AND INFERRED MINERAL RESOURCE SENSITIVITY TO CUT-OFF GRADE

Cut-off Grade (g/t Au)	Indicated			Inferred		
	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
>2.00 g/t	471	3.22	49	91	3.40	10
>1.50 g/t	876	2.52	71	302	2.19	21
>1.25 g/t	1,379	2.10	93	545	1.83	32
>1.00 g/t	2,382	1.68	129	920	1.54	46
>0.80 g/t	3,902	1.37	172	1,257	1.37	55
>0.60 g/t	6,507	1.10	230	1,879	1.15	69
>0.33/0.42/0.47 g/t[1]	10,454	0.87	291	2,939	0.91	85
>0.25 g/t	13,837	0.73	327	3,580	0.80	92

Note:

1. Cut-off grades used for the reporting of the Mineral Resources

FIGURE 14-43 GRADE-TONNAGE CURVES OF PIT CONSTRAINED INDICATED AND INFERRED MINERAL RESOURCE ESTIMATE AT GOSSEY FOR SELECTED GOLD CUT-OFF GRADES



15 MINERAL RESERVE ESTIMATE

15.1 SUMMARY

The Essakane Mineral Reserve estimate for the EMZ, Falagountou West, and Wafaka deposits, at August 31, 2019, is summarized in Table 15-1 and is reported on a 100% basis. The Mineral Reserve estimate was prepared by IAMGOLD's Longueuil Technical Services team. The Mineral Reserve was estimated using a multiple-stage process. Optimal pit shells were run in Whittle to confirm that all designs are appropriately sized.

TABLE 15-1 ESSAKANE MINERAL RESERVE ESTIMATE – AUGUST 31, 2019

Process	Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
CIL	Proven	-	-	-
	Probable	72,690	1.36	3,181
	Stockpile	13,501	0.59	255
	Total CIL	**86,191**	**1.24**	**3,436**
Heap Leach	Proven	-	-	-
	Probable	35,058	0.39	439
	Stockpile	8,049	0.42	110
	Total Heap Leach	**43,107**	**0.40**	**549**
Total		**129,299**	**0.96**	**3,985**
Waste within Designed Pit		261,434		
Ore within Designed Pit		107,748		
Total Tonnage within Designed Pit		369,172		

Notes:
1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves estimated assuming open pit mining methods.
3. Mineral Reserves are estimated using an undiluted cut-off grade which varies between 0.31 and 0.61 g/t Au depending on material type and pit.
4. Mineral Reserves are estimated using an average long-term gold price of US$1,200/oz.
5. Average weighted CIL process recovery of 92.1% and heap leach process recovery of 67.0%.
6. Mineral Reserves are reported on a 100% basis.
7. Mineral Reserves include material from EMZ and Falagountou pits.
8. Numbers may not add due to rounding.

Essakane is in operation and the mine design and Mineral Reserve estimate have been completed to an operational detailed level. The Mineral Reserve estimate stated herein is consistent with the CIM (2014) definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources.

There are no Mineral Reserves for the Gossey deposit.

The responsible QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate.

15.2 RESOURCE MODELS

The August 31, 2019 Mineral Reserve estimate is based on separate updated resource models at year-end 2018 for the EMZ, Falagountou West, and Wafaka deposits. Mining activity is ongoing in the EMZ and Falagountou West pits. All resource models were updated by Essakane S.A. with technical support from various consultants with several changes to the modelling which should improve reconciliation with production in the years to come.

No other models than those described for the EMZ, Falagountou West, and Wafaka deposits in Section 14 of this Technical Report have been used.

15.3 DILUTION AND MINING LOSSES

The EMZ deposit Mineral Reserve estimate includes a mining dilution provision of 10% for saprolite, transition, and fresh rock material. Dilution factors for the EMZ deposit have increased from 8% following an increase in variance in the resource model. The Falagountou West and Wafaka deposits Mineral Reserve estimate includes a mining dilution provision of 8% for saprolite and 10% for transition and fresh rock material. The dilution tonnage is set at zero grade.

15.4 EXTRACTION

The ore extraction rate, or mining recovery, is assumed to be 100%. This assumption is based on several years of operations experience and is supported by reconciliation studies and geological modelling.

15.5 CUT-OFF GRADE

Metal prices used for Mineral Reserves are based on consensus, long-term forecasts from the IAMGOLD corporate team, Essakane Technical Services, and Essakane financial groups. For Mineral Resources, metal prices used are slightly higher than those for Mineral Reserves.

IAMGOLD uses a consistent reserve and resource gold price assumption for all of its operations. The reserve gold price assumption for estimating Mineral Reserves at August 31, 2019 is US$1,200/oz. Other economic assumptions utilized to estimate costs and revenues such as fuel price, exchange rates, and royalty rates are summarized in Table 15-2.

TABLE 15-2 PIT OPTIMIZATION ECONOMIC ASSUMPTIONS

Economic Parameters	Units	Value
Gold price	US$/oz	1,200
By Product Credits	US$/oz	1.20
Long term oil price	US$/bbl	65
CFA exchange rate	CFA/USD	547
Transport and refining cost	US$/oz	2.50
Site diesel price	US$/l	1.15
Site Heavy Fuel Oil (HFO) price	US$/l	0.72
Power cost	US$/kWh	0.18
Royalty (3-5%)	US$/oz	48.00
Community Fund (1%)	US$/oz	12.00
Cost of selling (Cs)	US$/oz	62.50
Discount rate	%	6.00

The pit optimization parameters were consolidated by the Essakane engineering department with inputs provided by other departments to calculate the break-even cut-off grades for each of the pits and ore types. The cut-off grades are estimated on the basis of a long-term sustainable CIL plant throughput of 11.7 Mtpa in fresh rock. The 2019 cut-off grades, by pit, are summarized in Table 15-3.

TABLE 15-3 SUMMARY OF 2019 UNDILUTED CUT-OFF GRADES AT US$1,200/OZ AU

Process	CIL			HL
Cut-off Grade by Pit	Saprolite	Transition	Fresh Rock	Fresh Rock
EMZ (g/t Au)	0.36	0.46	0.57	0.31
Falagountou (g/t Au)	0.40	0.50	0.61	N/A

The lower cut-off grade of 0.31 g/t Au for fresh rock at the EMZ pit is due to it being suitable for heap leaching at the end of the mine life at a rate of 8.5 Mtpa. The split between heap leach ore and CIL ore for fresh rock is at 0.57 g/t Au. The heap leach material only comes from the EMZ area and stockpiles. The EMZ rock is stored in two stockpiles; the heap leach stockpile, and the marginal rock stockpile.

The CIL plant metallurgical recovery assumptions for all deposits are fixed at 95% for saprolite and 93% for transition. Fresh rock has a metallurgical recovery of 92.1% on average, however, it is variable on feed grade. Metallurgical recovery for the heap leach was assumed at 67.0%.

The mine operating cost inputs for pit optimization are derived from current mining costs and productivities. The mine operating costs were estimated on the basis of a diesel fuel price of $1.15/L. The mine operating cost includes mine sustaining capital and capital maintenance items, which total $0.41/t processed.

The Falagountou pit optimization costs are slightly higher than those for the EMZ pit due to additional costs in areas such as RC drilling, production drilling, and rehandling. The ore haulage distance from the Falagountou pit to the CIL plant is 11 km, which results in a total $1.37/t premium relative to the EMZ pit.

Details for the cut-off grades for the EMZ and Falagountou pits are presented in Table 15-4 and Table 15-5, respectively.

TABLE 15-4 SUMMARY OF EMZ PIT OPTIMIZATION PARAMETERS AND CUT-OFF GRADES

Ore Based Cost and Cut-off Grade by Deposit		**EMZ Pit**			
	Units	**CIL**			**Heap Leach**
Rock Type		**Saprolite**	**Transition**	**Fresh Rock**	**Fresh Rock**
Metallurgical recovery	(%)	95.0%	93.0%	Custom	67.0%
Processing rate	(Mtpa)	15.0	12.5	11.7	8.5
Average power consumption	(kWh/t)	13.6	16.9	26.6	-
Total fixed processing costs	(MUS$/year)	45.3	45.3	45.3	36.9
Processing fixed cost	(US$/t treated)	3.03	3.63	3.87	-
Crushing Cost (w/o power)	(US$/t treated)	0.00	0.25	0.25	-
Liners and grinding media	(US$/t treated)	0.71	1.26	1.48	-
Reagents	(US$/t treated)	1.49	1.53	1.55	-
Power	(US$/t treated)	2.37	2.95	4.65	1.55
TSF	(US$/t treated)	0.24	0.24	0.24	-
Mill Stay in Business (SIB) Capital	(US$/t treated)	0.27	0.27	0.27	-
Operating Consumables	(US$/t treated)	-	-	-	1.91
Plant Maintenance and other	(US$/t treated)	-	-	-	0.89
E.Backcharge G&A	(US$/t treated)	0.66	0.66	0.66	-
Total Processing Cost	(US$/t treated)	**8.77**	**10.78**	**12.98**	**4.35**
Mining dilution	%	10%	10%	10%	10%
Ore premium mining cost	(US$/t treated)	-0.03	0.09	0.19	0.19
Total fixed G&A costs	(MUS$/year)	38.76	38.76	38.76	12.5
G&A Project SIB Capital	(MUS$/year)	2.12	2.12	2.12	-
Rehandle	(US$/t treated)	0.00	0.00	0.00	0.75
G&A cost	(US$/t treated)	2.74	3.25	3.46	1.47
Total Ore Based Cost	**(US$/t treated)**	**11.48**	**14.12**	**16.63**	**6.76**
Break Even Mill Feed Cut-off Grade	**(g/t Au)**	**0.36**	**0.46**	**0.57**	**0.31**
Reference Mining Cost by Deposit					
Rock Type		**Saprolite**	**Transition**	**Fresh Rock**	**Fresh Rock**
Total Reference Mining Cost (Waste)	(US$/t mined)	2.14	2.52	2.78	2.78
Incremental bench cost	(US$/t per vert. m)	0.00202	0.00202	0.00202	0.00202
Reference elevation	(m)	260 m			

TABLE 15-5 SUMMARY OF FALAGOUNTOU PIT OPTIMIZATION PARAMETERS AND CUT-OFF GRADES

Ore Based Cost and Cut-off Grade by Deposit		Falagountou Pit		
			CIL	
Rock Type		**Saprolite**	**Transition**	**Fresh Rock**
Metallurgical recovery	(%)	95.0%	93.0%	Custom
Processing rate	(Mtpa)	15.0	12.5	11.7
Average power consumption)	(kWh/t)	13.6	16.9	26.6
Total fixed processing costs	(MUS$/year)	45.3	45.3	45.3
Processing fixed cost	(US$/t treated)	3.03	3.63	3.87
Crushing Cost (w/o power)	(US$/t treated)	0.00	0.25	0.25
Liners and grinding media	(US$/t treated)	0.71	1.26	1.48
Reagents	(US$/t treated)	1.49	1.53	1.55
Power	(US$/t treated)	2.37	2.95	4.65
TSF	(US$/t treated)	0.24	0.24	0.24
Mill SIB Capital	(US$/t treated)	0.27	0.27	0.27
E.Backcharge G&A	(US$/t treated)	0.66	0.66	0.66
Total Processing Cost	**(US$/t treated)**	**8.77**	**10.78**	**12.98**
Mining dilution	%	8%	10%	10%
Ore premium mining cost	(US$/t treated)	1.23	1.39	1.49
Total fixed G&A costs	(MUS$/year)	38.76	38.76	38.76
G&A Project SIB Capital	(MUS$/year)	2.12	2.12	2.12
Rehandle	(US$/t treated)	0.00	0.00	0.00
G&A cost	(US$/t treated)	2.74	3.25	3.46
Total Ore Based Cost	**(US$/t treated)**	**12.74**	**15.42**	**17.93**
Break Even Mill Feed Cut-off Grade	**(g/t Au)**	**0.40**	**0.50**	**0.61**
Reference Mining Cost by Deposit				
Rock type		**Saprolite**	**Transition**	**Fresh Rock**
Total Reference Mining Cost (Waste)	(US$/t mined)	2.19	2.64	2.95
Incremental bench cost	(US$/t per vert. m)	0.00202	0.00202	0.00202
Reference elevation	(m)		260 m	

15.6 MINERAL RESERVE ESTIMATES

The August 31, 2019 consolidated EMZ and Falagountou Mineral Reserves are presented in Table 15-6. No Proven Mineral Reserves have been estimated.

Mineral Reserves were separated based on processing types, to account for the distinct processing CIL plant recoveries. The Falagountou deposits have no Mineral Reserve attributed to the heap leach process as the ore is not confirmed to be suitable for this process.

As of August 31, 2019, there were 82.4 Mt of CIL Probable Mineral Reserves defined in the EMZ pit design and within stockpiles, at an average grade of 1.25 g/t Au for a total of 3,299,000 ounces of gold. The heap leach Probable Mineral Reserves are estimated to be 43.1 Mt in the EMZ pit design and within stockpiles, at an average grade of 0.40 g/t Au for a total of 549,000 ounces of gold.

Additionally, as of August 31, 2019, there were 3.8 Mt of CIL Probable Mineral Reserves defined in the Falagountou pit designs and within stockpiles, at an average grade of 1.12 g/t Au for a total of 137,000 ounces of gold.

The EMZ pit contains 255.3 Mt of waste and the Falagountou pits contain 6.1 Mt of waste resulting in a remaining project stripping ratio of 2.43. As of August 31, 2019, the consolidated Probable Mineral Reserves are estimated to be 129.3 Mt at an average grade of 0.96 g/t Au for a total of 3,985,000 ounces of gold (in-situ).

IAMGOLD's 90% attributable Probable Mineral Reserves total 116.4 Mt of ore and 3,587,000 ounces of gold. Approximately 8% of the Mineral Reserve consists of transition and saprolite ore and the remainder (92%) is fresh rock.

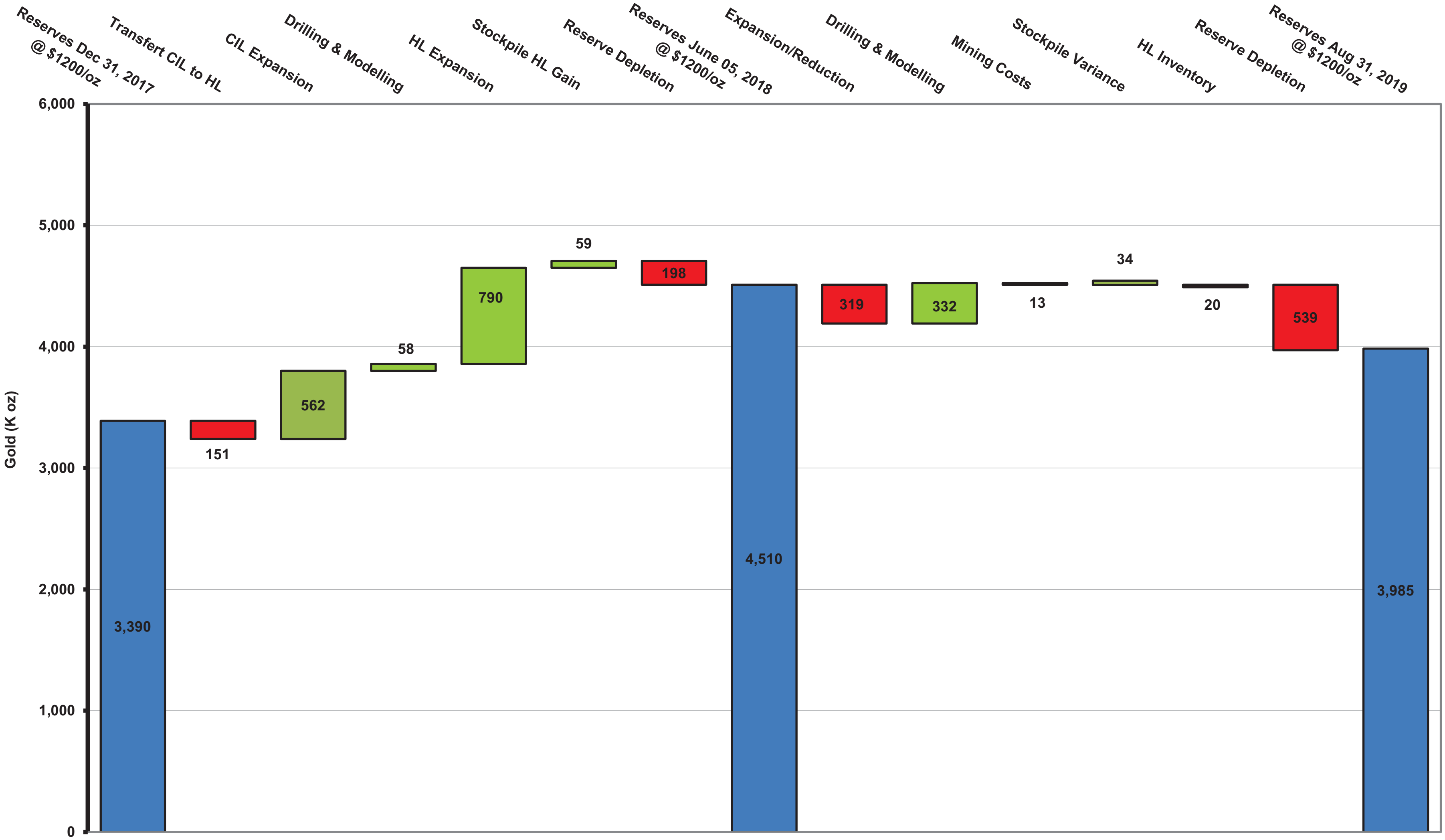
The waterfall graph in Figure 15-1 shows the change in the total ounces of gold contained within the Mineral Reserves from December 31, 2017 through to August 31, 2019. Figure 15-1 shows two phases. The first phase shows the addition of the heap leach process and the second phase shows the increase in pit size at Essakane that resulted from the improved economics.

TABLE 15-6 ESSAKANE CONSOLIDATED MINERAL RESERVES - AUGUST 31, 2019

	Laterite and Saprolite			Transition			Fresh Rock			All Materials		
	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Essakane Main Zone (EMZ)												
Cut-off Grade CIL[2]	0.36 g/t Au			0.46 g/t Au			0.57 g/t Au					
Proven	-	-	-	-	-	-	-	-	-	-	-	-
Probable	1,158	0.57	21	600	0.74	14	68,148	1.39	3,035	69,906	1.37	3,070
EMZ Proven and Probable[1]	1,158	0.57	21	600	0.74	14	68,148	1.39	3,035	69,906	1.37	3,070
Stockpiles	367	0.33	4	8,130	0.54	141	3,979	0.66	84	12,477	0.57	229
Total EMZ CIL Reserve[3]	**1,525**	**0.51**	**25**	**8,731**	**0.55**	**155**	**72,127**	**1.35**	**3,119**	**82,383**	**1.25**	**3,299**
Cut-off Grade Heap Leach (HL)[2]							0.31 g/t Au					
Proven	-	-	-	-	-	-	-	-	-	-	-	-
Probable	-	-	-	-	-	-	35,058	0.39	439	35,058	0.39	439
EMZ Proven and Probable[1]	-	-	-	-	-	-	**35,058**	**0.39**	**439**	**35,058**	**0.39**	**439**
Stockpiles	-	-	-	-	-	-	8,049	0.42	110	8,049	0.42	110
Total EMZ HL Mineral Reserve[3]	-	-	-	-	-	-	**43,107**	**0.40**	**549**	**43,107**	**0.40**	**549**
Falagountou Zone (Falagountou and Wafaka)												
Cut-off Grade CIL[2]	0.40 g/t Au			0.50 g/t Au			0.61 g/t Au					
Proven	-	-	-	-	-	-	-	-	-	-	-	-
Probable	-	-	-	-	-	-	2,784	1.24	111	2,784	1.24	111
Falagountou Proven and Probable[1]	-	-	-	-	-	-	2,784	**1.24**	111	**2,784**	**1.24**	111
Stockpiles	298	0.86	8	84	0.77	2	642	0.76	16	1,024	0.79	26
Total Falagountou CIL Reserve	298	**0.86**	**8**	84	**0.77**	**2**	3,426	**1.15**	127	**3,809**	**1.12**	137
Consolidated Essakane Mineral Reserve												
Total Mineral Reserve	**1,823**	**0.57**	**33**	**8,815**	**0.55**	**157**	**118,661**	**0.99**	**3,795**	**129,299**	**0.96**	**3,985**
IAMGOLD Attributable Reserve	**1,641**	**0.57**	**30**	**7,933**	**0.55**	**141**	**106,795**	**0.99**	**3,415**	**116,369**	**0.96**	**3,587**
	1.4%			**6.8%**			**91.8%**			**100%**		
Waste[4]										**261,424**		
Stripping Ratio (Waste : Ore)[5]										**2.43**		

Notes:

1. Includes Proven and Probable Mineral Reserves in pits designed on US$1,200/oz gold price
2. Undiluted Cut-off grades are based on LeachWELL sampling Au grades, US$1,200/oz operating gold price, applied on ex-pit grade
3. EMZ HL and CIL Reserves are mutually exclusive
4. Includes Inferred Mineral Resources, in addition to the portion of Measured and Indicated Mineral Resources below cut-off grades
5. Excludes Stockpiles



IAMGOLD Corporation

Figure 15-1

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Graph Showing Evolution of the EMZ Mineral Reserves from December 31, 2017 to August 31, 2019

January 2020

Source: IAMGOLD, 2019.

Table 15-7 shows the Mineral Reserve evolution since May 2007 on a 100% basis.

TABLE 15-7 MINERAL RESERVE EVOLUTION

Year	Gold Price (US$/oz)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
August 31, 2019	1,200	129,299	0.96	3,985
December 31, 2018	1,200	148,812	0.92	4,380
June 5, 2018 Heap Leach	1,200	158,197	0.89	4,510
December 31, 2017	1,200	93,126	1.13	3,390
December 31, 2016	1,200	89,676	1.15	3,311
December 31, 2015	1,200	96,463	1.10	3,414
December 31, 2014	1,300	108,821	1.11	3,886
December 31, 2013	1,400	126,806	1.12	4,573
December 31, 2012	1,500	114,377	1.00	3,659
December 31, 2011	1,200	109,245	1.10	3,858
December 31, 2010	1,000 & 850[1]	107,465	1.29	4,461
December 31, 2009	850	92,911	1.44	4,301
March 2009	600	58,122	1.67	3,121
May 2007	650	46,413	1.78	2,649

Note:

1. Falagountou at US$850/oz Au (Jan 2008 pit design)

16 MINING MEASQ ASQTHODS

16.1 GENERAL

Mining is carried out using a conventional drill, blast, load, and haul surface mining method with an owner fleet. The annual mining rate was 50.7 Mt in 2018 with a stripping ratio of 3.05. Approximately 13.0 Mt of ore at an average grade of 1.18 g/t Au for a total of 450,000 ounces of gold were produced in 2018. There are no current mining plans for the Gossey deposit.

Table 16-1 details past production, through December 2018, at Essakane.

TABLE 16-1 ESSAKANE HISTORICAL PRODUCTION

	2010	2011	2012	2013	2014	2015	2016	2017	2018
Ore Mined (000 t)	10,097	10,110	9,562	11,869	12,580	11,518	10,921	11,811	13,866
Waste Mined (000 t)	11,876	15,268	1,689	30,006	32,677	35,690	36,939	36,296	36,825
Marginal Mined (000 t)	957	1,788	25,103	3,257	1,440	1,679	855	926	719
Total Mined (000 t)	22,930	27,166	36,353	45,133	46,698	48,887	48,314	47,993	50,691
Stripping Ratio	1.27	1.69	2.80	2.80	2.71	3.24	3.25	3.10	3.05
Mined Ore Grade (g/t Au)	1.05	1.08	1.04	0.84	0.98	1.14	1.21	1.17	1.12
Ore Milled (000 t)	2,973	7,977	10,762	10,613	11,897	11,716	12,006	13,891	13,031
Mill Grade (g/t Au)	1.49	1.53	1.10	0.89	1.06	1.23	1.22	1.07	1.18
Recovery (%)	95.7%	95.4%	91.9%	91.7%	90.7%	91.7%	89.0%	90.3%	90.9%
Gold Produced (000 oz)	136	375	350	277	369	426	419	432	450
RC Drilling (000 m)	100	209	167	227	195	148	104	78	140
Production Drilling (000 m)	12	257	541	806	724	944	869	886	858
Pre-split Drilling (000 m)	-	-	4	32	21	41	61	124	109
Tonnes Blasted (000 t)	690	12,937	24,818	43,989	37,292	42,218	43,082	49,316	48,970
Explosives (000 kg)	53	2,405	5,813	12,606	11,958	13,740	14,565	16,799	16,446
Powder Factor (kg/t)	0.12	0.19	0.23	0.29	0.32	0.33	0.34	0.34	0.34

Essakane consists of several operating sites. The EMZ main pit is mined in several mining phases and accounts for the majority of the production. The Falagountou, EMZ North, and South (Lao) satellite pits provide additional ore and operational flexibility.

Grade control is accomplished by RC drilling and sampling of the mineralized zone on a 10 m x 10 m pattern, or tighter, as required. For sterile sections of the pit, the grid may be widened out based on the nature of the contacts and/or other geological occurrences.

A fleet of four drill rigs are used for the 229 mm (9.0 inch) production blast holes. All blasting activities on site are executed by an explosives supplier. Holes are loaded with bulk explosive matrix and initiated with electronic detonators.

Grade movement during blasting is a critical issue at Essakane. For this reason, blast movement monitors (BMMs) are systematically used when blasting mineralized areas in order to measure vertical and horizontal displacement which allows for the adjustment of the post blast ore packets.

The mine loading fleet currently consists of three O&K Terex RH-120 shovels, five Caterpillar (CAT) 993K wheel loaders, and two Komatsu PC 2000 390 excavators.

The mine hauling fleet currently consists 26 CAT 785C haul trucks, eight CAT 785D haul trucks, five CAT 777F haul trucks, and three CAT 777F water trucks.

Mine haul roads are planned 30 m wide and are constructed by the mining department to support the mine haul trucks.

A list of Essakane S.A.'s primary mine production equipment fleet is shown in Table 16-2.

TABLE 16-2 CURRENT PRIMARY MINE EQUIPMENT FLEET

Type	Model	Number
Shovel	O&K Terex RH120	3
Excavator	Komatsu PC2000	2
	Caterpillar 390	4
	Caterpillar 345/349	3
Loader	Caterpillar 992/993K	5
Truck	Caterpillar 785C	26
	Caterpillar 785D	8
	Caterpillar 777F	8
Drilling	Atlas Copco PV-235	4
	Sandvik DK45	5
Dozer	Bulldozer D9R	6

Type	Model	Number
	Bulldozer D10T	3
	Wheeldozer 824	2
	Wheeldozer 834	1
Grader	Grader Cat 16M	5
Auxiliary	Caterpillar 966H	1
Tow Haul	Caterpillar 777D	1

Essakane S.A.'s ancillary equipment includes fuel and water trucks, mobile light plants, and service trucks.

Waste material is being dumped in the WRDs located east of the main pit.

Various ore stockpiles sorted per type (saprolite, transition, or fresh rock) and grade (marginal, low, and high grade), are located to the west of the main pit, just north of the primary crusher.

Water runoff from the ore stockpiles and WRDs is collected in ditches and diverted to catchment basins where the runoff is pumped to one of the bulk water storage reservoirs near the TSF.

Other mining infrastructure includes a mine office complex (mine offices, change houses, and canteens), equipment workshop, with overhead cranes integrated with the main warehouse, and external wash bays, blasting and explosives compound including magazines, diesel storage and dispensing facility, and a drill core storage facility.

16.2 DEWATERING

In the pit, sumps excavated below the active mining level are used, as required, to control rainfall runoff during the rainy season, as well as groundwater seepage.

Pump and pipeline systems are in place to lower the water table below active mining faces. Water is collected and used for in-pit and haul road dust suppression and the CIL plant process requirements.

16.3 MINE DESIGN

SRK support and make recommendations for Essakane S.A.'s geotechnical design studies and wall stability analyses. In October 2018, SRK was mandated to carry out a review of the pit slope design and has since provided overall pit slope design and recommendations for the EMZ and Falagountou pits.

A Leica Geomos deformation monitoring instrument is used at Essakane for wall stability monitoring, coupled with a Reutech MSR194 radar.

The EMZ and Falagountou pit design parameters used to ensure slope stability are detailed in Table 16-3 and 16-4, respectively.

TABLE 16-3 EMZ PIT DESIGN PARAMETERS

Parameter	Unit	Saprolite	Transition	Fresh Rock			
Sector (rock only)				East 1	East 2	West	West 48
Bench height	(m)	10	10	20	20	20	20
Berm width	(m)	7.6	9.6	12.7	12.7	11.5	14.5
Bench face angle	(°)	50	65	80	80	80	80
Geotech berm width	(m)	20	20	20	20	20	20
Maximum stack height	(m)	N/A	N/A	120	120	120	100
Inter ramp angle (IRA)	(°)	32	35	51	51	53	48
Ramp Width	(m)	30	30	30	30	30	30
Ramp Gradient	(°)	8	10	10	10	10	10

TABLE 16-4 FALAGOUNTOU PIT DESIGN PARAMETERS

Parameter	Unit	Saprolite	Transition	Fresh Rock		
Sector (rock only)				West	Northeast	South
Bench height	(m)	10	10	20	20	20
Berm width	(m)	7.6	9.6	20.3	13.6	13.6
Bench face angle	(°)	50	65	80	80	80
Geotech berm width	(m)	15	15	15	15	15
IRA	(°)	32	32-39	40	50	50
Ramp Width	(m)	30	30	30	30	30
Ramp Gradient	(°)	8	10	10	10	10

It should be noted that for both pits, pit slope parameters are continuously being optimized as mining progresses.

16.4 FIVE YEAR PLAN AND LIFE OF MINE PLAN

Although the LOM extends until 2026 in terms of mining from which 100.3 Mt of ore at an average grade of 1.06 g/t Au will be extracted, the following five-year plan provides a detailed breakdown in the mining activities through to 2024.

The five-year mine plan provides an overall annual gold production of more than 410,000 ounces, on average. In 2020, the CIL plant will process 9% of saprolite due to the addition of the Essakane Lao pit. Starting from mid-year 2020, the Falagountou satellite pit is expected to be mined out. Production will then take place in the EMZ main pit and the Essakane Lao pit, located south of the main pit. Production will be approximately 57.0 Mtpa with 2020 being a peak production year at 65.5 Mt. Table 16-5 details Essakane S.A.'s five-year mine plan from 2020 to 2024.

TABLE 16-5 ESSAKANE FIVE YEAR MINE PLAN

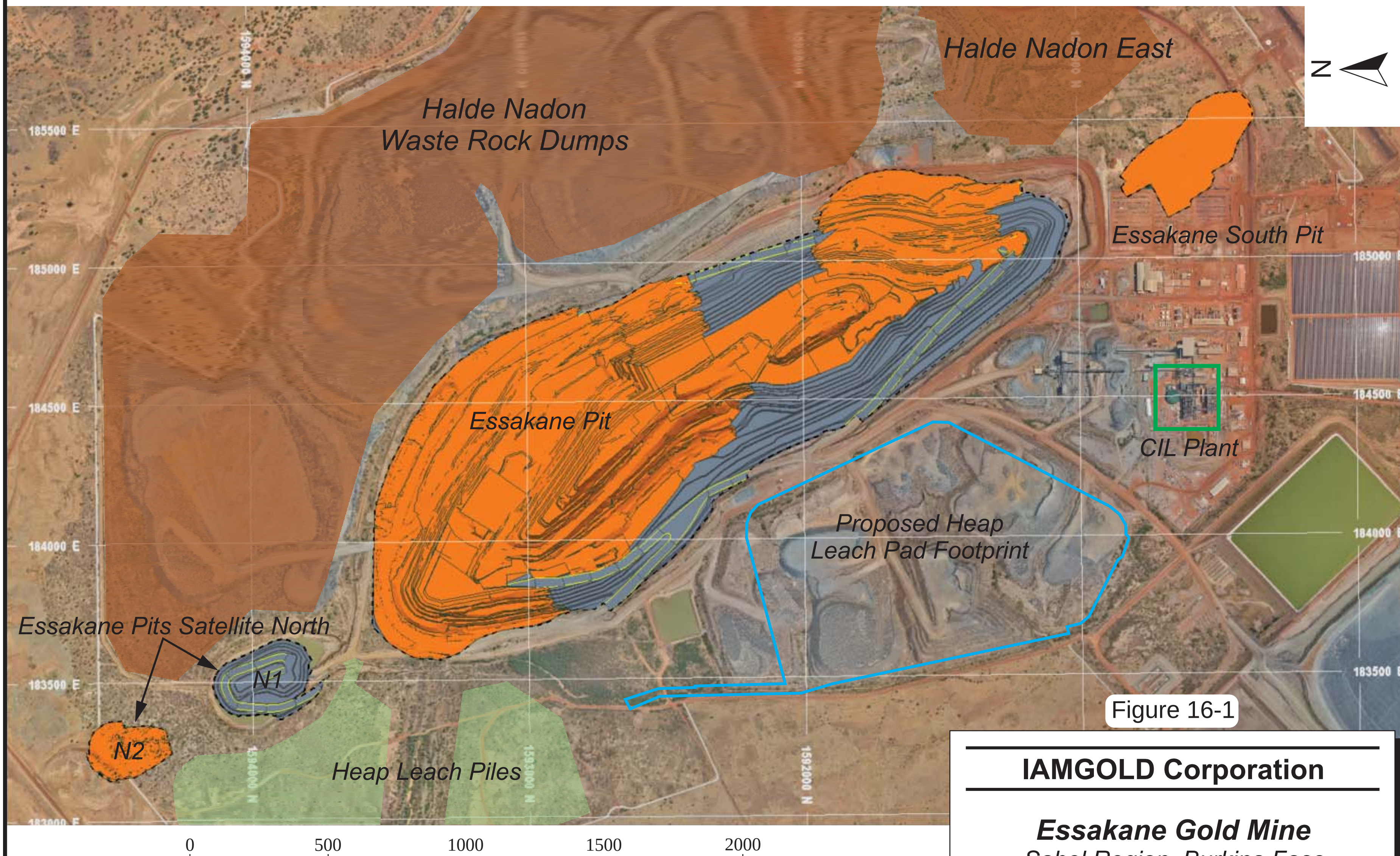
	2020	2021	2022	2023	2024
Ore Mined (000 t)	16,328	11,849	14,755	12,042	14,262
Waste Mined (000 t)	49,191	45,211	42,321	44,184	37,777
Total Mined (000 t)	65,520	57,060	57,076	56,225	52,040
Stripping Ratio	3.01	3.82	2.87	3.67	2.65
Mined Ore Grade (g/t Au)	0.96	1.10	0.99	1.09	0.98
Ore Milled (000 t)	12,599	11,829	12,500	12,143	11,870
Mill Grade (g/t Au)	1.14	1.20	1.11	1.15	1.14
Recovery (%)	92.19%	91.91%	91.86%	91.80%	91.93%
Gold Produced (000 oz)	425	420	410	412	400
RC Drilling (000 m)	184	128	104	91	89
Production Drilling (000 m)	854	694	719	726	659
Pre-split Drilling (000 m)	130	108	115	115	121
Tonnes Blasted (000 t)	62,807	56,287	56,042	56,225	52,040
Explosives (000 kg)	21,685	19,305	19,676	19,575	18,520
Powder Factor (kg/t)	0.35	0.34	0.35	0.35	0.36

Mining in 2020 is concentrated in EMZ pit phases PH3 and PH4, with some stripping activities in the upper benches of phase PH7 (Figure 16-1 to Figure 16-6). For the Falagountou deposit, mining in 2020 will be completed in June (Figure 16-7 and Figure 16-8). Mining of the Essakane South (Lao) satellite pit, located south of the main pit, will commence in 2020 to supply saprolite material to the CIL plant for the following two years. Production in 2021 will follow the same pattern as 2020 with mining in the Essakane South (Lao) pit for saprolite material, and phase PH3, which will be depleted by 2021. Stripping in phase PH6 is initiated in 2022. From 2021 onward, ore will be provided mainly from phase PH4, supported by phase PH7.

Table 16-6 details Essakane S.A.'s LOM plan from 2020 to 2031. No production has been scheduled from the Gossey deposit.

TABLE 16-6 ESSAKANE LIFE OF MINE PLAN

	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	Total
CIL													
Ore Mined (000 t)	10,426	8,268	10,390	8,230	9,062	12,762	8,678	-	-	-	-	-	**67,815**
Mined Ore Grade (g/t Au)	1.29	1.42	1.23	1.41	1.32	1.49	1.49	-	-	-	-	-	**1.38**
Heap Leach													
Ore Mined (000 t)	5,902	3,581	4,366	3,812	5,200	6,715	2,865	-	-	-	-	-	**32,440**
Mined Ore Grade (g/t Au)	0.39	0.38	0.40	0.39	0.39	0.39	0.39	-	-	-	-	-	**0.39**
Total Ore Mined (000 t)	16,328	11,849	14,755	12,042	14,262	19,476	11,543	-	-				**100,256**
Mined Ore Grade (g/t Au)	0.96	1.10	0.99	1.09	0.98	1.11	1.22						**1.06**
Waste Mined (000 t)	49,191	45,211	42,321	44,184	37,777	21,442	4,814	-	-	-	-	-	**244,940**
Total Mined (000 t)	65,520	57,060	57,076	56,225	52,040	40,919	16,356	-	-	-	-		**345,196**
Stripping Ratio	3.01	3.82	2.87	3.67	2.65	1.10	0.42	-	-	-	-	-	**2.44**
CIL Mill Production													
Ore Milled (000 t)	12,599	11,829	12,500	12,143	11,870	10,802	11,396	-	-	-	-	-	**83,140**
Mill Grade (g/t Au)	1.14	1.20	1.11	1.15	1.14	1.65	1.29	-	-	-	-	-	**1.23**
Recovery (%)	92.19%	91.91%	91.86%	91.80%	91.93%	92.59%	92.08%	-	-	-	-	-	**92.07%**
Gold Produced (000 oz)	425	420	410	412	400	531	436	-	-	-	-	-	**3,033**
Heap Leach Production													
Heap Leach Feed (000 t)	-	-						8,500	8,498	8,500	8,500	9,348	**43,346**
Heap Leach Grade (g/t Au)	-	-						0.44	0.46	0.38	0.34	0.34	**0.39**
Recovery (%)	-	-						67.00%	67.00%	67.00%	67.00%	67.00%	**67.00%**
Gold Produced (000 oz)	-	-						81	85	70	63	69	**368**
Total Production													
Tonnes Processed	12,599	11,829	12,500	12,143	11,870	10,802	11,396	8,500	8,498	8,500	8,500	9,348	**126,486**
Grade (g/t Au)	1.14	1.20	1.11	1.15	1.14	1.65	1.29	0.44	0.46	0.38	0.34	0.34	**0.95**
Recovery (%)	92.19%	91.91%	91.86%	91.80%	91.93%	92.59%	92.08%	67.00%	67.00%	67.00%	67.00%	67.00%	**88.49%**
Gold Produced (000 oz)	425	420	410	412	400	531	436	81	85	70	63	69	**3,401**
RC Drilling (000 m)	184	128	104	91	89	138	68	-	-	-	-	-	**802**
Production Drilling (000 m)	854	694	719	726	659	520	217	-	-	-	-	-	**4,388**
Pre-split Drilling (000 m)	130	108	115	115	121	119	78	-	-	-	-	-	**787**
Tonnes Blasted (000 t)	62,807	56,287	56,042	56,225	52,040	40,919	16,317	-	-				**340,637**
Explosives (000 kg)	21,685	19,305	19,676	19,575	18,520	15,559	6,571	-	-	-	-	-	**120,892**
Powder Factor (kg/t)	0.35	0.34	0.35	0.35	0.36	0.38	0.40	-	-	-	-	-	**0.35**



Note: Phase Pit Designs Shown in Orange

Figure 16-1

IAMGOLD Corporation

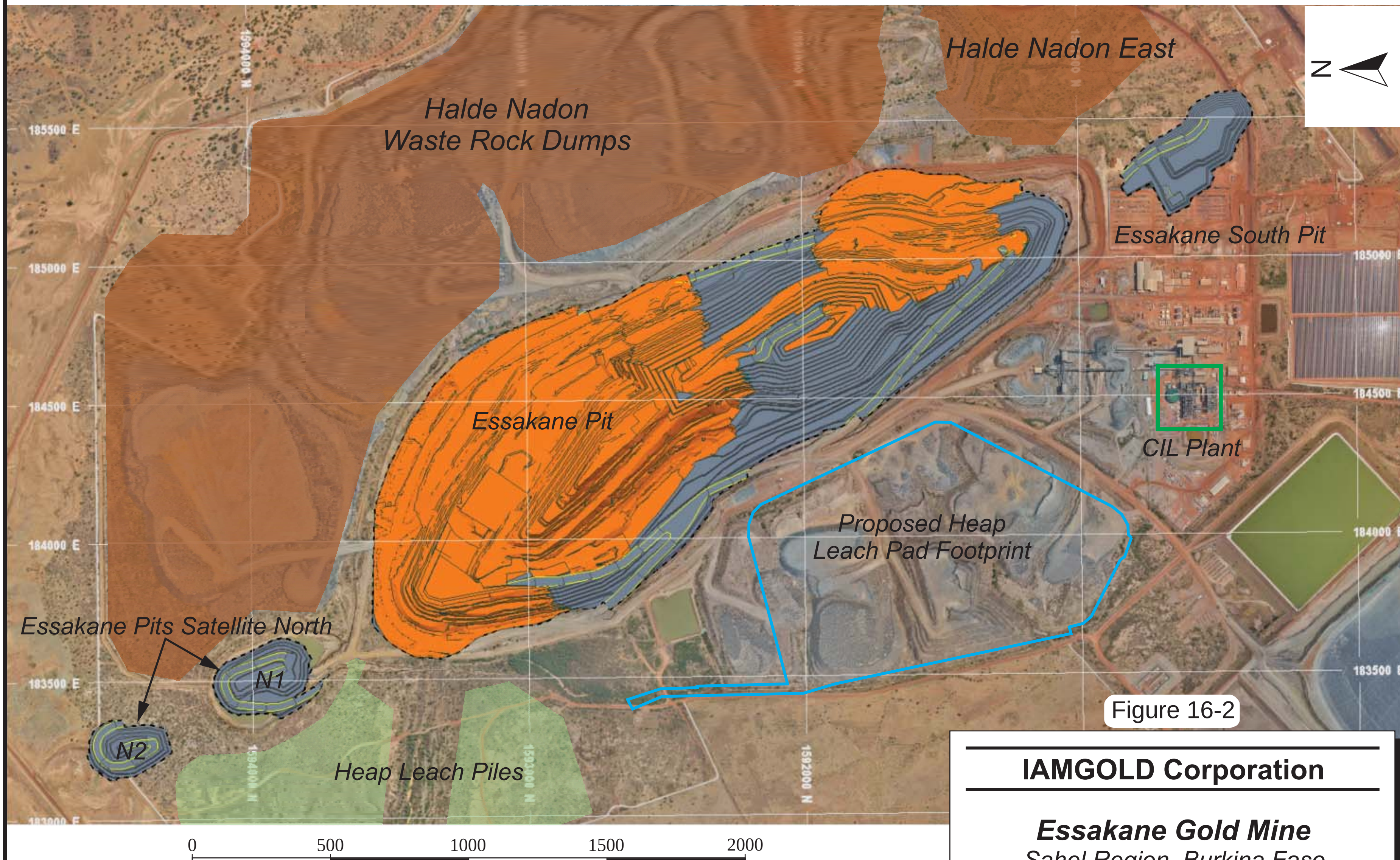
Essakane Gold Mine

Sahel Region, Burkina Faso

Phase 3 Pit Design

January 2020

Source: IAMGOLD, 2019.



Note: Phase Pit Designs Shown in Orange

Figure 16-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

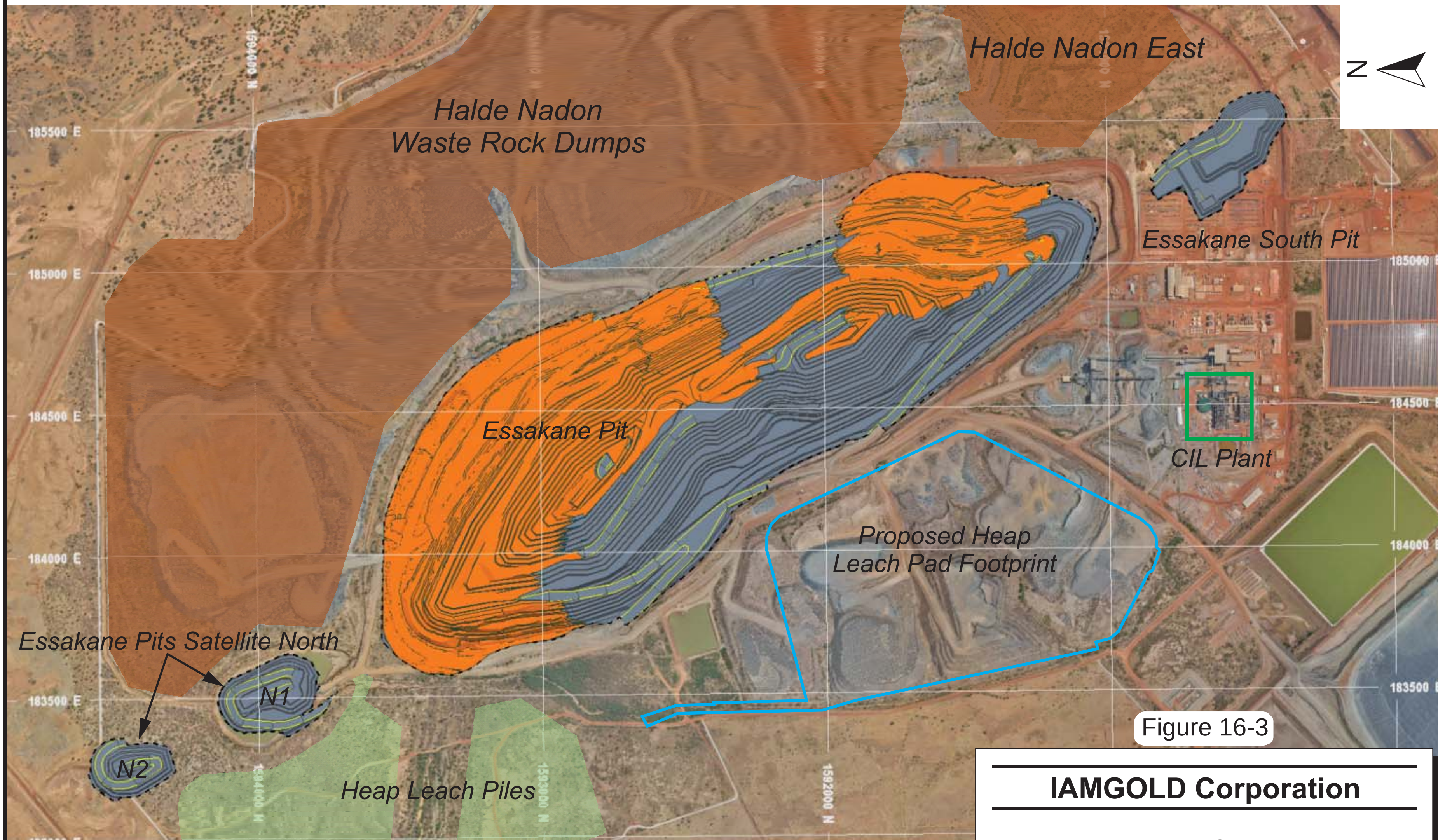
Phase 4 Pit Design

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation

IAMGOLD Corporation



0 500 1000 1500 2000

Metres

Note: Phase Pit Designs Shown in Orange

Figure 16-3

IAMGOLD Corporation

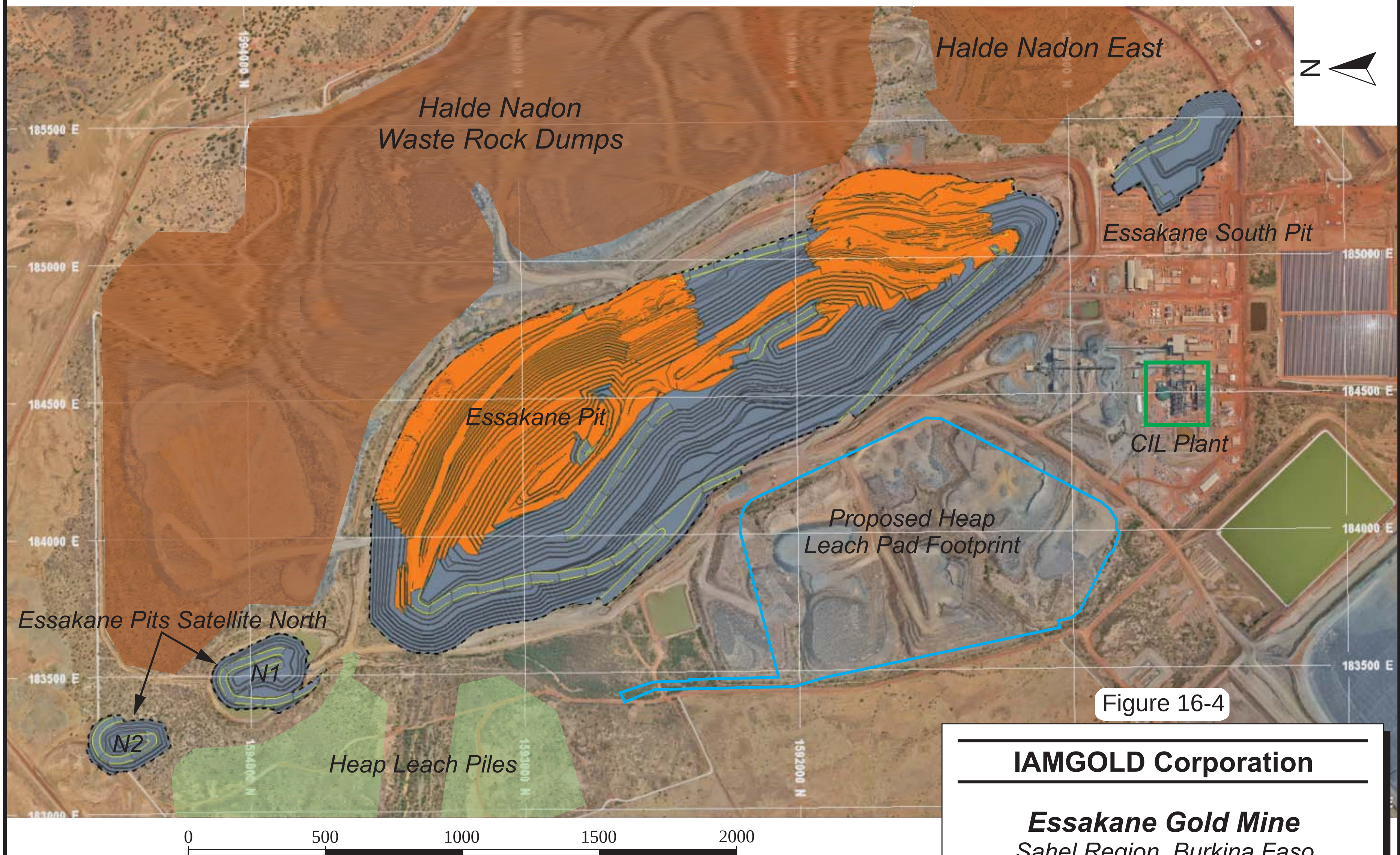
Essakane Gold Mine

Sahel Region, Burkina Faso

Phase 5 Pit Design

January 2020

Source: IAMGOLD, 2019.



Note: Phase Pit Designs Shown in Orange

Figure 16-4

IAMGOLD Corporation

Essakane Gold Mine

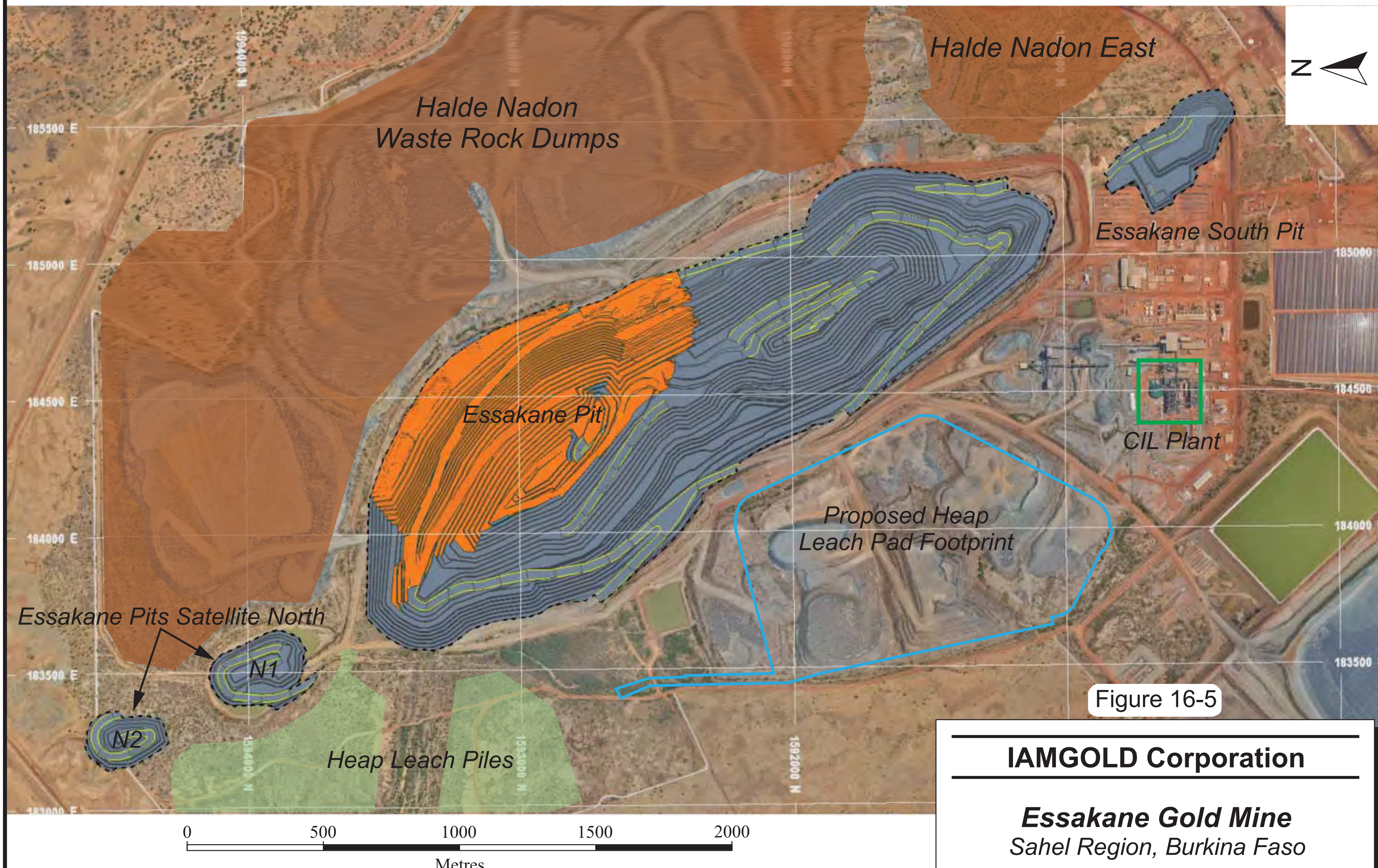
Sahel Region, Burkina Faso

Phase 6 Pit Design

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation



Note: Phase Pit Designs Shown in Orange

Figure 16-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Phase 7 Pit Design

January 2020

Source: IAMGOLD, 2019.

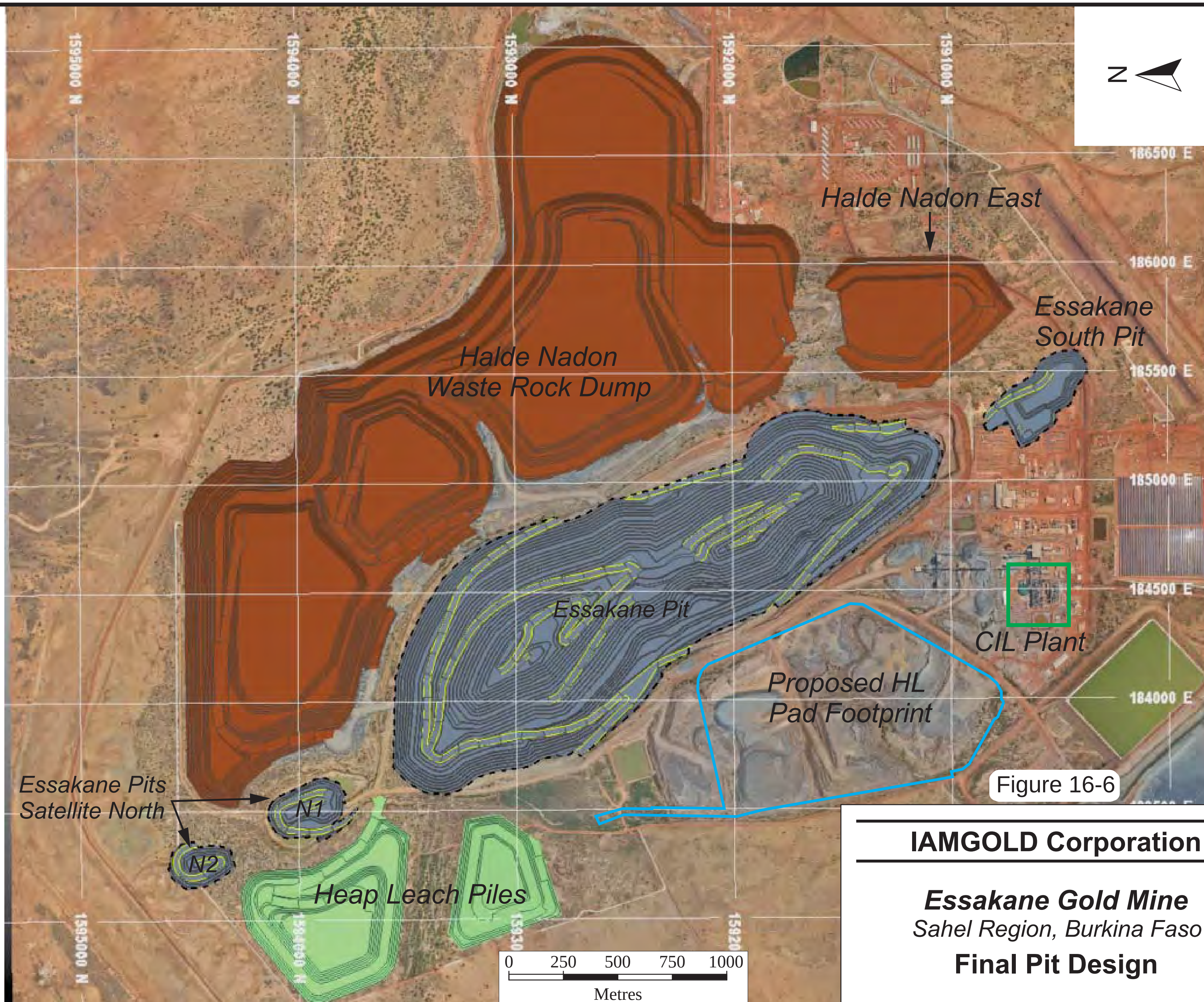


Figure 16-6

IAMGOLD Corporation

Essakane Gold Mine

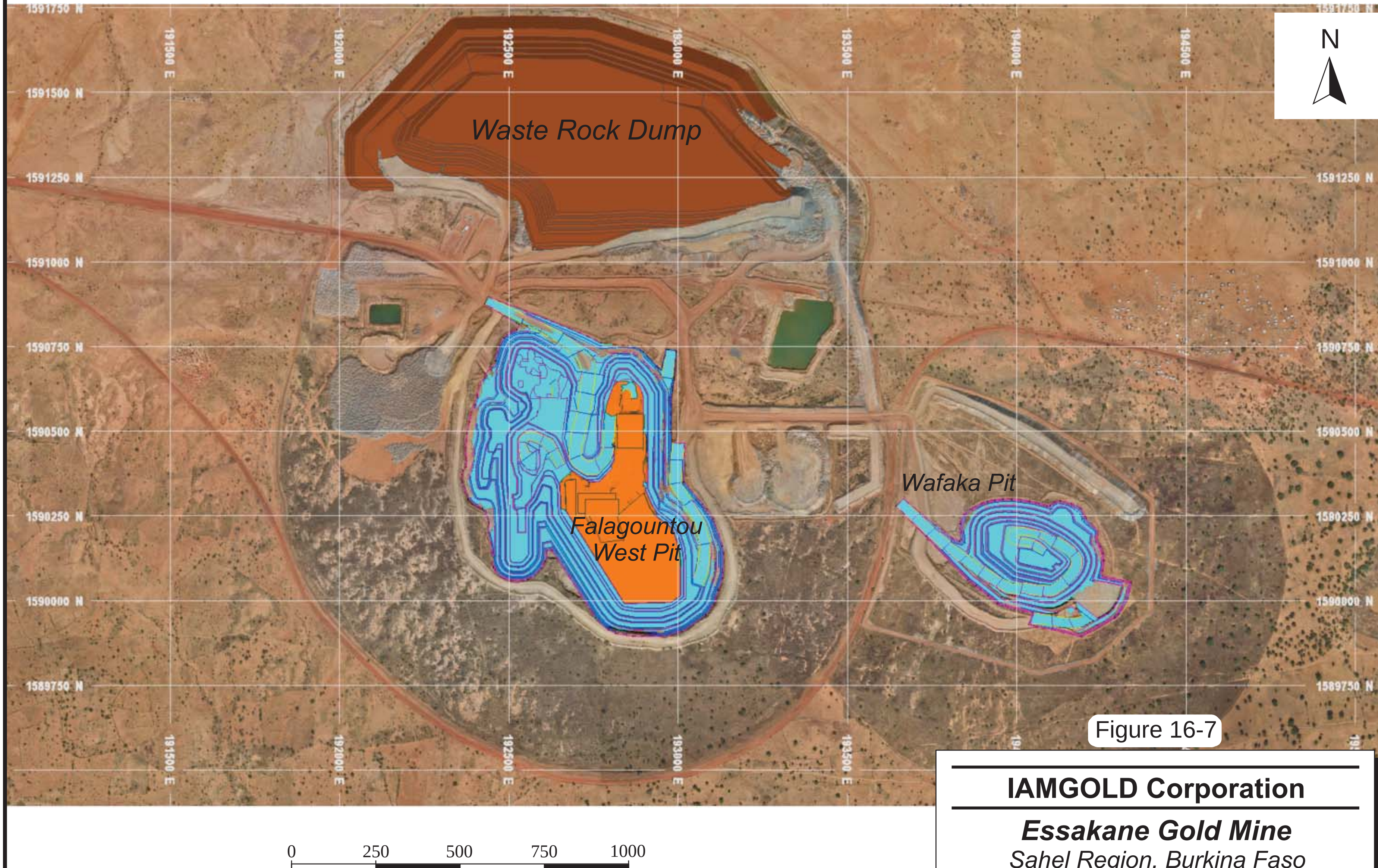
Sahel Region, Burkina Faso

Final Pit Design

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation



Note: Phase Pit Designs Shown in Orange

Figure 16-7

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Phase 3 Pit Design - Falagountou Pit

January 2020

Source: IAMGOLD, 2019.

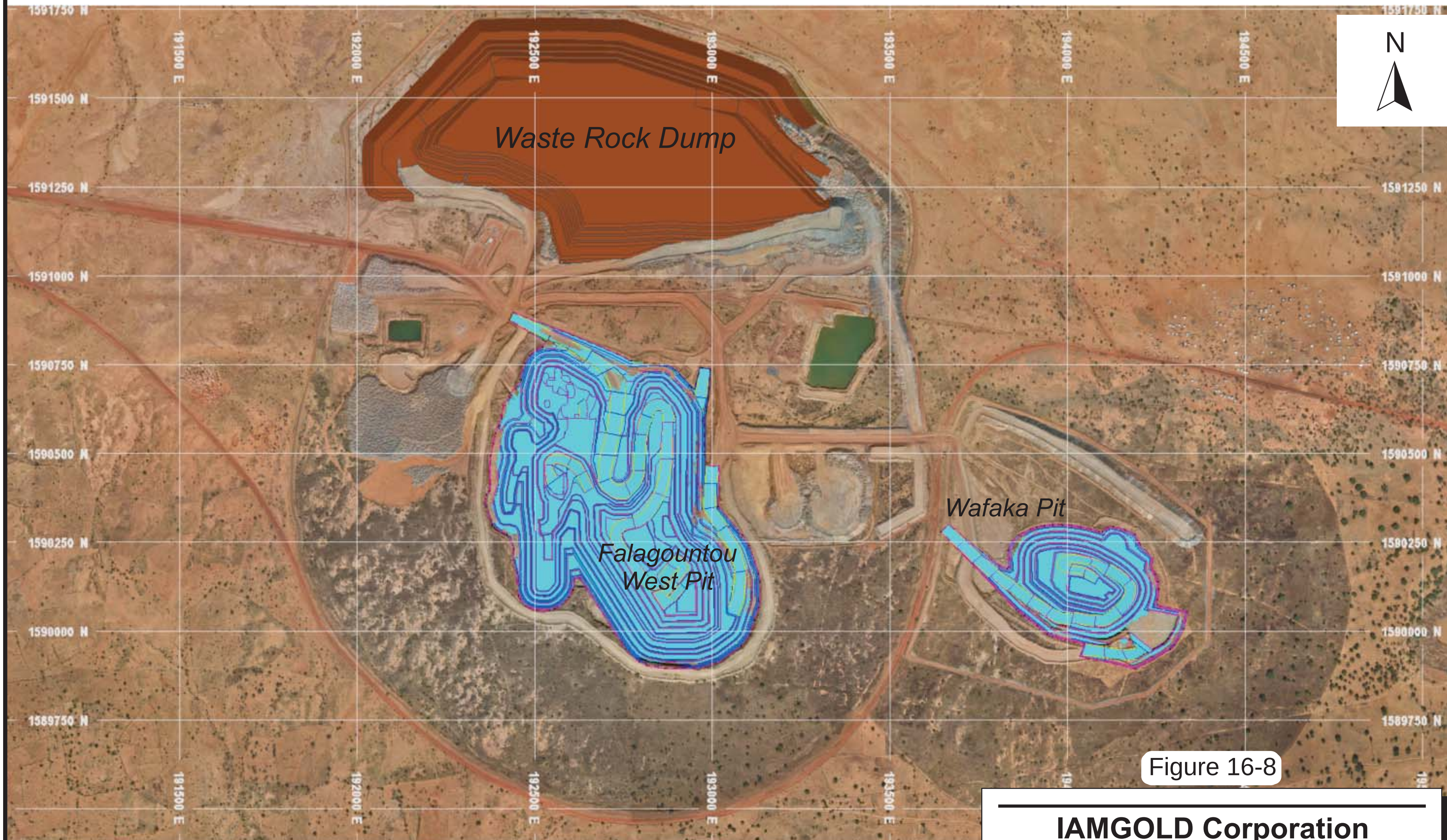


Figure 16-8

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Ultimate Pit Design
- Falagountou Pit

0 250 500 750 1000
Metres

January 2020

Source: IAMGOLD, 2019.

IAMGOLD Corporation

17 RECOVERY METHODS

17.1 CARBON-IN-LEACH RECOVERY METHODS

Ore is currently processed using two stages of crushing, semi-autogenous grinding (SAG), ball mill grinding, pebble crusher grinding (SABC), gravity concentration, and a CIL gold plant. The 2008 UFS proposed a process plant throughput rate of 7.5 Mtpa. During construction, some debottlenecking improvements were made to the design, resulting in a revised nameplate capacity of 9.0 Mtpa based on processing 100% saprolite ore. Due to further operational improvements, plant throughput has increased beyond the constructed design capacity.

Fresh rock CIL plant feed has gradually increased from 2012 onwards. To maintain gold production levels, with increasing proportions of fresh rock in the CIL plant feed, an expansion was completed in 2014. The objective was to double the fresh rock processing capacity from 5.4 Mtpa on a 100% fresh rock basis to 10.8 Mtpa.

The expansion consisted of the addition of a secondary crushing circuit and a second process line (grinding, gravity concentration, and leach-CIL) in the CIL plant and included:

- One secondary crusher of 750 kW
- One SAG mill of 7 MW
- One ball mill of 7 MW
- Two pebble crushers, one on line A and one on line B
- Two gravity concentrators
- Eight CIL tanks

The process plant expansion was commissioned in February 2014, and effectively doubled the fresh rock processing capacity.

The expanded mineral processing flow sheet is shown in Figure 17-1.

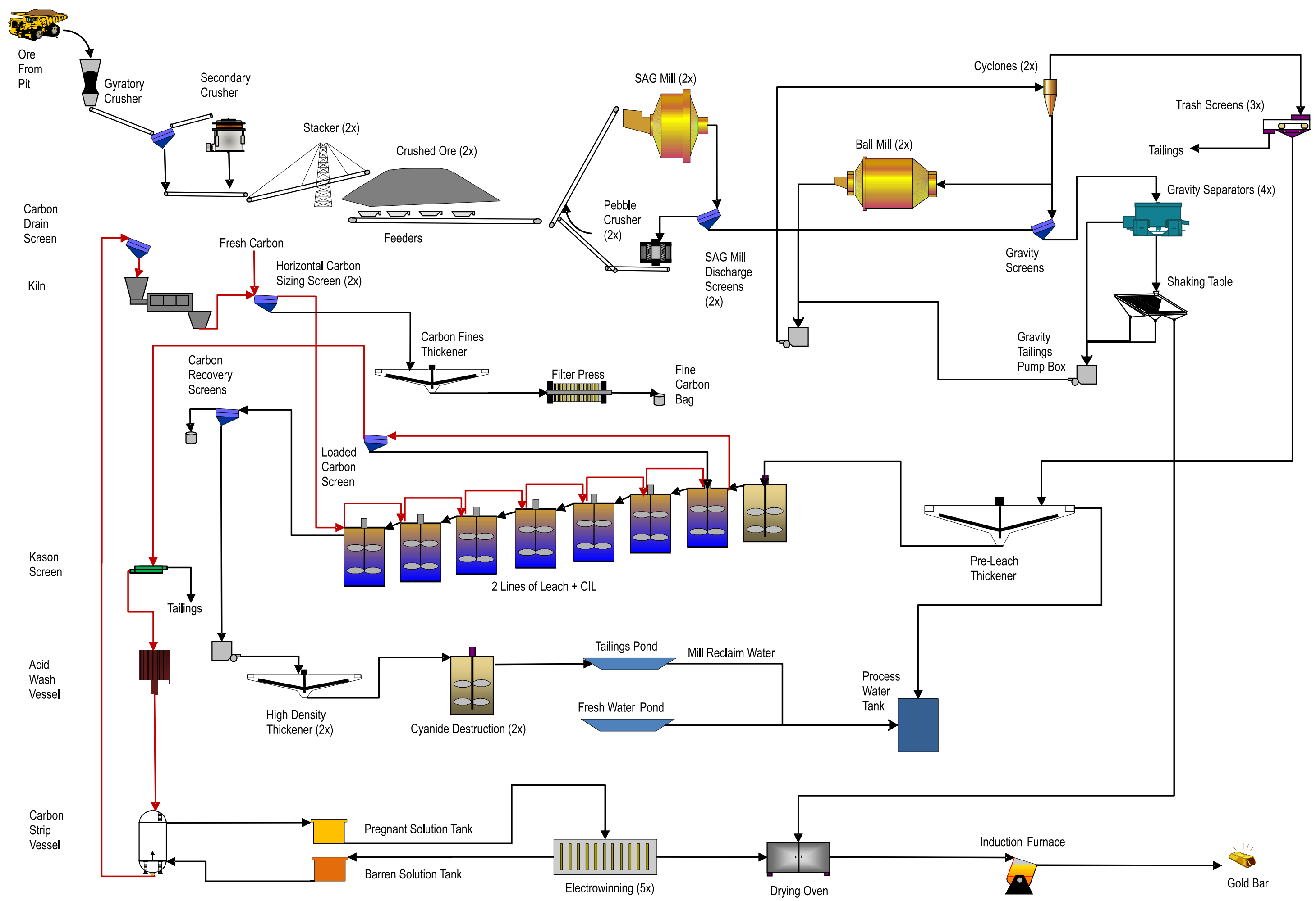


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Figure 17-1

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Mineral Processing Flowsheet for the CIL Plant

January 2020

Source: IAMGOLD, 2019.

The ore is crushed in a gyratory crusher and in a cone crusher. The crushed ore is stockpiled either in a pile for Line A or Line B. The ore is reclaimed with apron feeders and feeds SAG mills on each line. The pebbles from the SAG mills are diverted to their respective pebble crusher in closed circuit. The ore passing through the SAG mill discharge screen feeds a pack of cyclones. Cyclone underflow returns to the ball mill. Cyclone overflow is sent to the pre-leach thickener. A portion of cyclone underflow goes to the gravity concentrators (two on each line).

The thickened ore feeds two parallel lines consisting of one leach tank followed by CIL tanks. Once loaded with gold, the carbon is screened, acid washed, and eluted. The pregnant solution is sent to the gold room for electrowinning, drying, and finally, smelting into doré bars.

Eluted carbon is regenerated in a kiln and reused in the CIL circuit. Carbon fines generated from the circuit are recovered in bags for further gold recovery.

The gravity concentrate feeds an intensive leach reactor. The pregnant solution obtained from the intensive leach reactor is processed by two dedicated electrowinning cells and the sludge recovered is filtered, dried, and smelted together with the sludge recovered from the elution circuit.

Plant tails are thickened and stored in the tailings pond. The water is reclaimed to the plant.

Table 17-1 summarizes CIL plant throughput, head grade, recovery, and gold production since reaching commercial production in July 2010. Figure 17-2 shows the yearly average recovery by gravity, recovery for the CIL plant, and the head grade since 2010.

TABLE 17-1 CIL PLANT PRODUCTION 2010 TO 2018

	Units	2010	2011	2012	2013	2014	2015	2016	2017	2018
Throughput	(000 t)	2,973	7,977	10,762	10,613	11,897	11,716	12,006	13,891	13,031
Head Grade	(g/t Au)	1.49	1.53	1.10	0.89	1.06	1.23	1.22	1.07	1.18
Recovery	(%)	95.7%	95.4%	91.9%	91.7%	90.7%	91.7%	89.0%	90.3%	90.9%
Gold Production	(000 oz Au)	136	375	350	277	369	426	419	432	450

FIGURE 17-2 HISTORICAL RECOVERIES AND HEAD GRADES

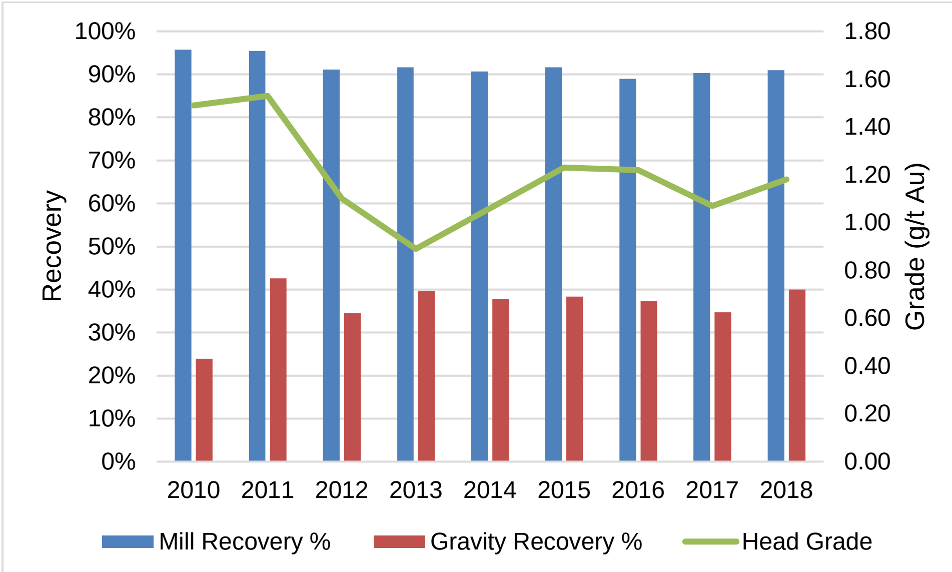


Table 17-2 summarizes the 2018 CIL plant production.

TABLE 17-2 2018 CIL PLANT PRODUCTION

Month	Tonnage (000 t)	Head Grade (g/t Au)	Recovery (%)	Gold Production (000 oz Au)
Jan	1,169	1.17	90.8	40
Feb	1,073	1.35	91.9	43
Mar	1,090	1.18	92.3	38
Apr	1,045	1.31	91.6	40
May	1,140	1.06	90.7	35
Jun	1,056	1.06	89.5	32
Jul	1,215	1.00	91.3	36
Aug	1,101	1.09	90.8	35
Sep	1,054	1.17	92.1	37
Oct	991	1.48	90.5	43
Nov	952	1.27	90.2	35
Dec	1,146	1.10	89.4	36
Total	**13,031**	**1.18**	**90.9**	**450**

Table 17-3 summarizes the actual ore tonnes milled and recovery achieved for 2017 and 2018 compared with the mine plan tonnage and recovery. In 2017, total tonnage treated was higher than the mine plan due to the higher than target percentage of saprolite in the CIL plant feed and optimized SAG mill liners which permitted an increase in CIL plant throughput. The recovery and head grade for 2017 were in line with the mine plan. In 2018, total ore tonnes milled were slightly higher than the mine plan. The recovery for 2018 was lower than the mine plan and the head grade was higher than the mine plan.

TABLE 17-3 2017 AND 2018 ACTUAL CIL PLANT SUMMARY COMPARED TO MINE PLAN

	2017		2018	
	Mine Plan	**Actual**	**Mine Plan**	**Actual**
Ore Milled (000 t)	12,953	13,891	12,945	13,031
Saprolite	14%	11.6%	13.0%	11.8%
Transition	1.0%	2.7%	5.0%	3.5%
Fresh Rock	85%	85.7%	82.0%	84.7%
Mill Grade (g/t Au)	1.10	1.07	1.12	1.18
Recovery	90.8%	90.3%	91.9%	90.9%

No material from the Gossey deposit has been milled in the CIL gold plant.

17.2 MODIFICATIONS TO THE EXISTING CONCENTRATOR

17.2.1 INTRODUCTION

IAMGOLD has conducted a number of investigations to upgrade the existing Essakane milling operation by identifying equipment throughput constraints at various plant capacities. The areas of focus were the crushing and grinding areas, with additional investigations into the CIL and tailings handling areas, the water balance, and the lime addition circuits.

17.2.1.1 SCOPE OF WORK

In November 2018, IAMGOLD requested Lycopodium Minerals Canada Ltd. (Lycopodium) to conduct a feasibility study on selected portions of the CIL plant upgrade project. Throughout the study phase a number of revisions were made to the scope of the study (Lycopodium, 2019a).

A plant capacity of 15 Mtpa was initially targeted. This included 13.5 Mtpa of fresh rock evenly split between Line A and Line B, with 1.5 Mtpa of saprolite added to Line A after the crusher plant. During the bridging phase, the targeted plant capacity was revised to be based on 11.7 Mtpa of fresh rock. In other words, the grinding circuit will be designed to accommodate 11.7 Mtpa of fresh rock equivalent in terms of the total specific energy. This means that more than

11.7 Mtpa total ore can be processed, as long as the required total specific energy for the ore blend is less than or equal to 11.7 Mtpa of fresh rock. The expected ore split ratio according to the latest mine plan is 86.68% fresh rock, 10.34% transition, and 2.98% saprolite. Therefore, the annual tonnages for the different ore types, equivalent to 11.7 Mtpa of fresh rock specific energy, are 10.9 Mtpa of fresh rock, 1.3 Mtpa of transition, and 0.38 Mtpa of saprolite, totalling 12.6 Mtpa at these ore proportions.

The scope of work included:

- Primary screening – evaluate the replacement of existing grizzly screens for improved efficiency, and also the addition of a wet scrubber for improved dust control.
- Gravity circuit – trade-off study and evaluate options for replacement of the gravity scalping screens for improved efficiency.
- Line A and Line B pebble crushing – review pebble discharge belt conveyor capacity and recommend necessary modifications to operate during pebble crusher by-pass scenario.
- Line A and Line B cyclone underflow distribution system – evaluate the transfer boxes and piping configuration and propose modifications to improve flow distribution from cyclone underflow to the mill feed hopper and to the gravity scalping screens.

17.2.2 DESIGN OUTCOMES AND PROCESS DESCRIPTION

17.2.2.1 KEY PROCESS DESIGN CRITERIA

The key process parameters required as inputs into the scope of study are summarized in Table 17-4.

TABLE 17-4 SUMMARY OF KEY PROCESS DESIGN CRITERIA

Parameters		Units	Value	Source
Design Throughput	- 100% Fresh Rock	(Mtpa)	11.7	IMG
Ore Blend per Mine Plan	- Fresh Rock	(%)	86.68	IMG
	- Transition	(%)	10.34	IMG
	- Saprolite	(%)	2.98	IMG
Equivalent Tonnage to 11.7 Mtpa Fresh Rock Specific Energy[1]				
	- Fresh Rock	(Mtpa)	10.9	Calculated
	- Transition	(Mtpa)	1.3	Calculated
	- Saprolite	(Mtpa)	0.38	Calculated
	- Total	(Mtpa)	12.6	Calculated
Ore Properties	- Rock Density	(t/m^3)	2.8	IMG
	- Bulk Density	(t/m^3)	1.6	IMG
Strength Properties	- Crushing Work Index	(kWh/t)	17.1	Test work
	- Abrasion Index	(G)	0.322	Test work
	- Axb Parameter	(-)	22.9	OMC
	- Bond Ball Mill Work Index	(kWh/t)	15.1	OMC
ROM Particle Size	- F_{100}	(mm)	800	IMG
	- F_{80}	(mm)	423	IMG
Crushing Plant Availability		(%)	75.0	IMG
Grinding Circuit Availability		(%)	93.3	IMG
Targeted Crushing Plant Product Size	- P_{80}	(mm)	48	OMC
Targeted Grind Size	- P_{80}	(µm)	125	Existing Data

Note:
1. Calculations are based on specific energy of 17.6 kWh/t, 9.0 kWh/t, and 5.0 kWh/t for fresh rock, transition, and saprolite, respectively.

17.2.2.2 PRIMARY SCREENING

Primary Screen

Primary screening at the existing crushing circuit is currently achieved by using two grizzly screens in series. A number of studies have been conducted to improve the efficiency of this circuit and to allow for greater throughput. Although the plant capacity has been slightly reduced since the bridging phase, the same screen size is still being retained in the new proposed design.

A double deck vibrating screen will be used to replace the two existing grizzly screens. The crushing circuit will feed 1,912 t/h of ore. The primary crusher will produce a crushed product top size as large as 404 mm. The crushed product is screened by the primary screen with the oversize reporting to the secondary cone crusher. The screen undersize will report to the respective grinding line's feed stockpile.

A number of equipment vendors were approached for bid during the study phase. The Schenck double deck horizontal screen (SLK3685) was selected during the bid evaluation mainly due to the fact that the technical specification was met, and also no expansion to the existing screening building will be necessary. The proposed screen will have a top deck aperture of 100 mm x 240 mm, and a bottom deck aperture of 38 mm x 50 mm. A 3D model detailing the demolition and construction work will be provided in order for a preliminary constructability analysis to be conducted, and a provisional construction schedule to be generated.

Primary Screening Dust Collection

A venturi wet scrubber has been recommended for dust collection in the primary screening area. Various hoods will be placed as dust pick-up points to capture dust generated from the vibrating primary screen, conveyors, and transfer points in the area. The dust will then be scrubbed with water and the resulting slurry will be transferred to the pre-leach thickener for processing.

It has been estimated that the dust loading will range between 20 g/m^3 to 30 g/m^3 with an air flow of 78,000 Am^3/h as the scrubbing duty.

17.2.2.3 LINE A AND LINE B PEBBLE CRUSHING

Line A Pebble Discharge Conveyor

The pebble crusher on both Line A and Line B has a maximum capacity of 300 t/h at the current operating closed side setting (CSS) of 12 mm. In the case that the pebble crusher is by-passed, the by-passing flow rate is 300 t/h. Based on observations, the pebble discharge conveyor on each line does not have sufficient capacity to accommodate the crusher by-passing scenario. This is due to a delay in the amount of material remaining in the crusher for approximately 30 seconds. When this delayed amount is combined with the by-passed amount, the total conveyor throughput can be as high as 600 t/h. It was part of the study's scope of work to upgrade the pebble discharge conveyors to allow proper operation during this by-pass scenario.

In order to increase the belt capacity during the by-pass operation, either the belt width or belt speed will need to be increased. It is not practical to increase the belt width on an existing conveyor, hence increasing the belt speed is the only suitable option. The belt will be upgraded to include a variable speed drive to allow the speed to increase for approximately 30 seconds during the crusher by-pass.

As a consequence of the speed increase, the trajectory of the material discharging from the belt will change and may cause wear issues in the existing head chute. To alleviate this, an impact plate or rock box can be included to protect the impact point. This will be investigated further during detailed design.

Line B Pebble Discharge Conveyor

The same solution will be implemented for Line B as per the Line A pebble discharge conveyor.

17.2.2.4 LINE A CYCLONE UNDERFLOW SYSTEM

Currently, the Line A cyclone underflow stream that is flowing to the ball mill is traveling via two 90º turns (one vertical and one horizontal) to go around the gravity screens. This was originally installed using a rubber lined pipe which was later replaced with a box and a launder with a bend. The box is currently relying on an impact plate to dissipate the energy. It has been noted that there are performance issues with this box, particularly the wear on the impact plate. As part of the study, this box has been proposed to be replaced with a kill box which utilizes a pool of slurry to dissipate the energy. The pool is developed by a downstream weir with de-sanding ports.

It is understood that the launder with the 90º bend is also experiencing spillage at high tonnages. This is likely due to the slurry velocity being too excessive to negotiate this bend, hence spilling over the side of the launder. To address this issue, the bend will be replaced with a turning box. Similar to the kill box, this turning box will utilize a pool of slurry to dissipate the energy. The slurry pool is developed by a downstream weir with de-sanding ports. New launders will also be provided to connect these boxes.

17.3 HEAP LEACH RECOVERY METHODS

17.3.1 SUMMARY OF PROCESS DESCRIPTION

Following the end of the existing CIL process plant operations, the heap leach operation will process 8.5 Mtpa of ore from a stockpile with a current estimated LOM for the heap leach operation of approximately five years.

Re-handled ore from a stockpile will be delivered to a local feed bin by a haul truck to feed a three-stage crushing plant consisting of an existing first and second stage crushing area, and a new HPGR tertiary crushing circuit. Cement will be added to the crushed ore via a rotary valve and screw conveyor suspended from a large overhead storage silo. The cement will also act as an alkalinity control reagent while agglomerating the crushed ore. The agglomerated crushed ore will be conveyed via a series of grasshopper conveyors and stacked onto the leach pad with a radial stacker (Lycopodium, 2019b).

The leach pad will be constructed in an up-gradient manner, with ore stacked in multiple lifts. Cyanide bearing solution will be applied on the leach pad for approximately 180 days prior to draining. The solution will be applied counter current to the placement of new ore using a network of piping. Both sprinklers and drip emitters will be used for uniform distribution of the solution. Gold bearing pregnant solution will be drained from the leach pad via gravity to a pregnant solution pond prior to being pumped to the carbon-in-column (CIC) circuit.

The pregnant solution will be processed using one train of six CIC stages with a carbon capacity of 8.5 tonnes for each carbon adsorption column. Loaded carbon will be stripped with hot cyanide-caustic solution in an existing pressure Zadra elution circuit designed for 17 tonnes of carbon. Each CIC train will advance 8.5 tonnes of carbon every other day to the existing elution facility, and the loaded carbon will be processed there once a complete batch of 17 tonnes is reached. Gold concentrated sludge from electrowinning will be dried in an oven and smelted in a furnace to pour into doré bars.

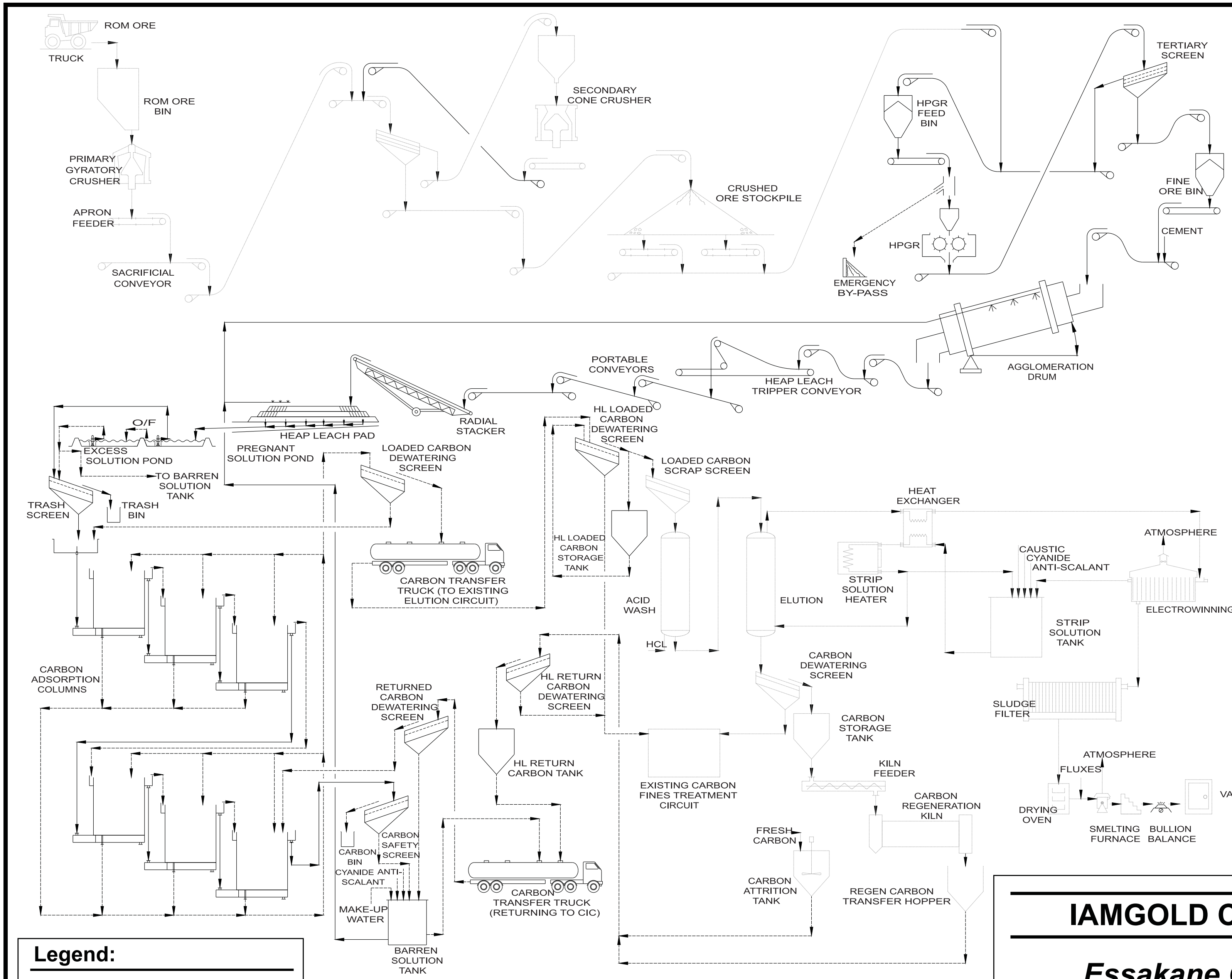
The following areas are currently part of the existing Essakane facility and will require modifications in order to tie-in with the new areas:

- Primary and secondary crushing:
    - modifications will be required to change the existing open secondary crushing circuit to a closed circuit.
- Elution circuit:
    - loaded carbon from the new CIC area will be trucked to and from the existing elution facility.
    - additional dewatering screens and storage tanks will be required for carbon handling.

Water and air services:

- only services near the existing plant can be re-used, additional water and air systems will be required for the new CIC area.

The overall process flow diagram is presented in Figure 17-3. The HLF general layout is presented in Figure 17-4. The heap leach crushing plant layout is presented in Figure 17-5 and the heap leach adsorption plant layout is presented in Figure 17-6.



Legend:

NEW

INTERMITTENT

EXISTING

January 2020

Figure 17-3

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Overall Process Flow Diagram

Source: Lycopodium, 2019.

IAMGOLD Corporation

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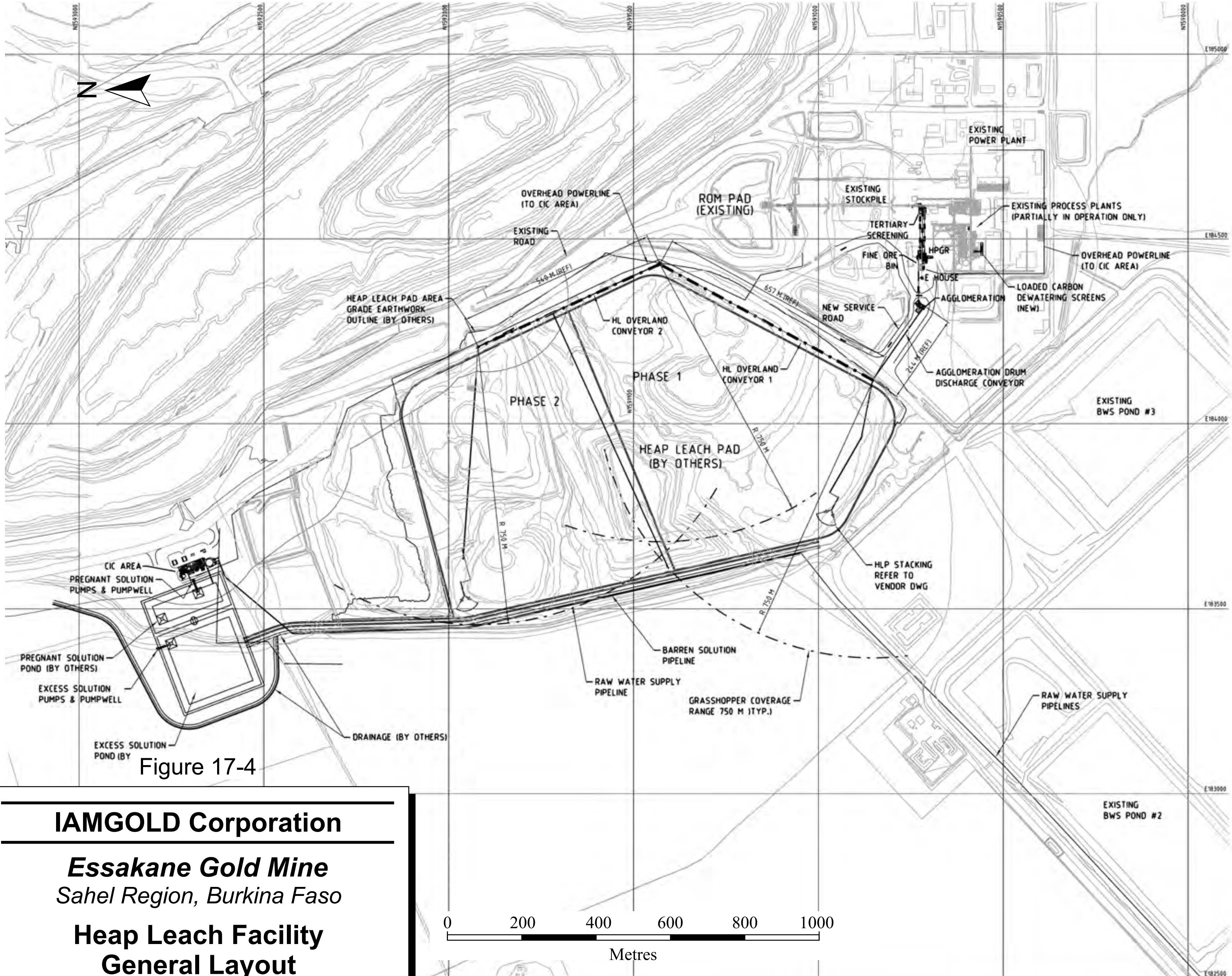

Figure 17-4

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Essakane Gold Mine

Sahel Region, Burkina Faso

Heap Leach Facility General Layout

January 2020

Source: Lycopodium, 2019.

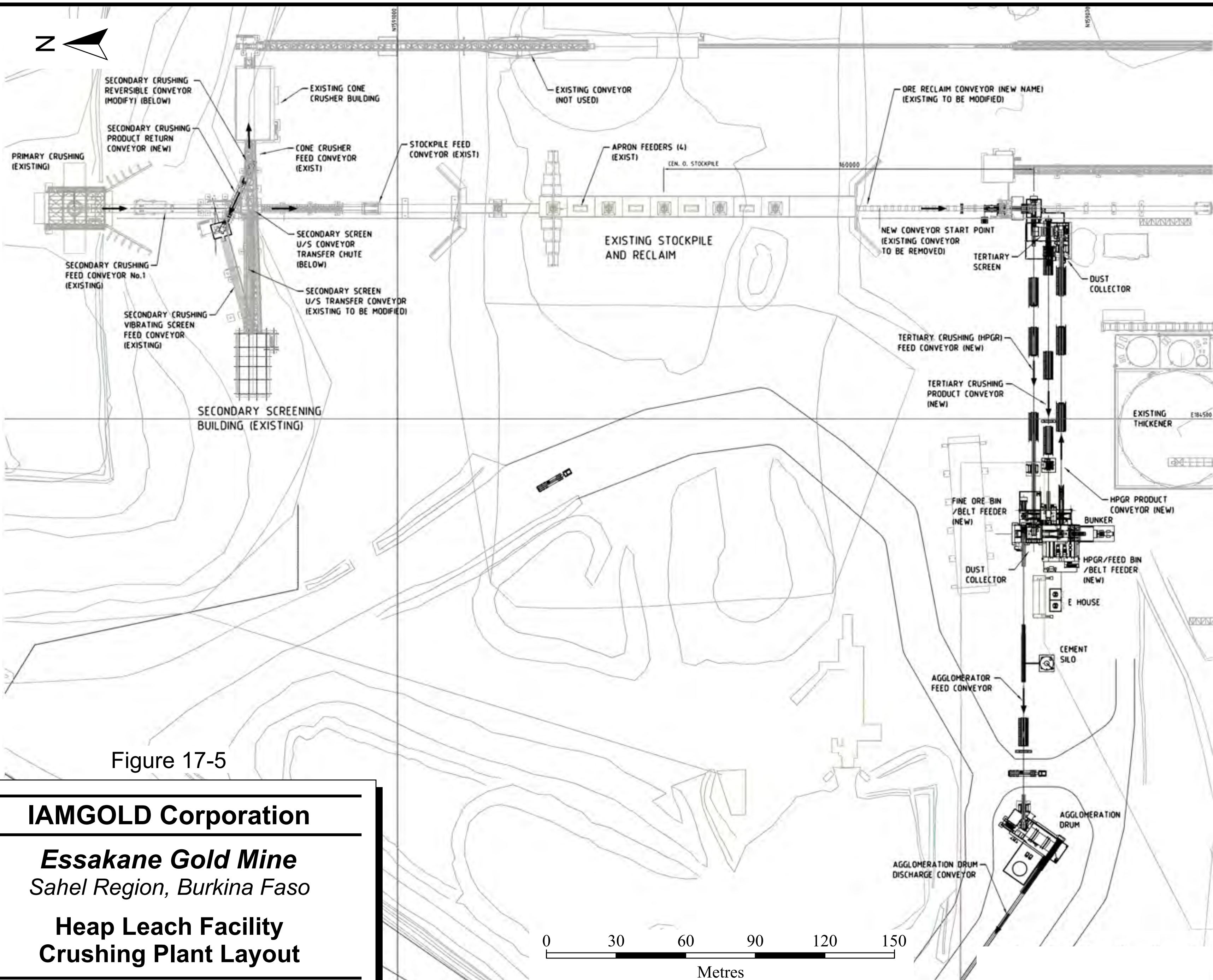

Figure 17-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Heap Leach Facility Crushing Plant Layout

January 2020

Source: Lycopodium, 2019.

IAMGOLD Corporation

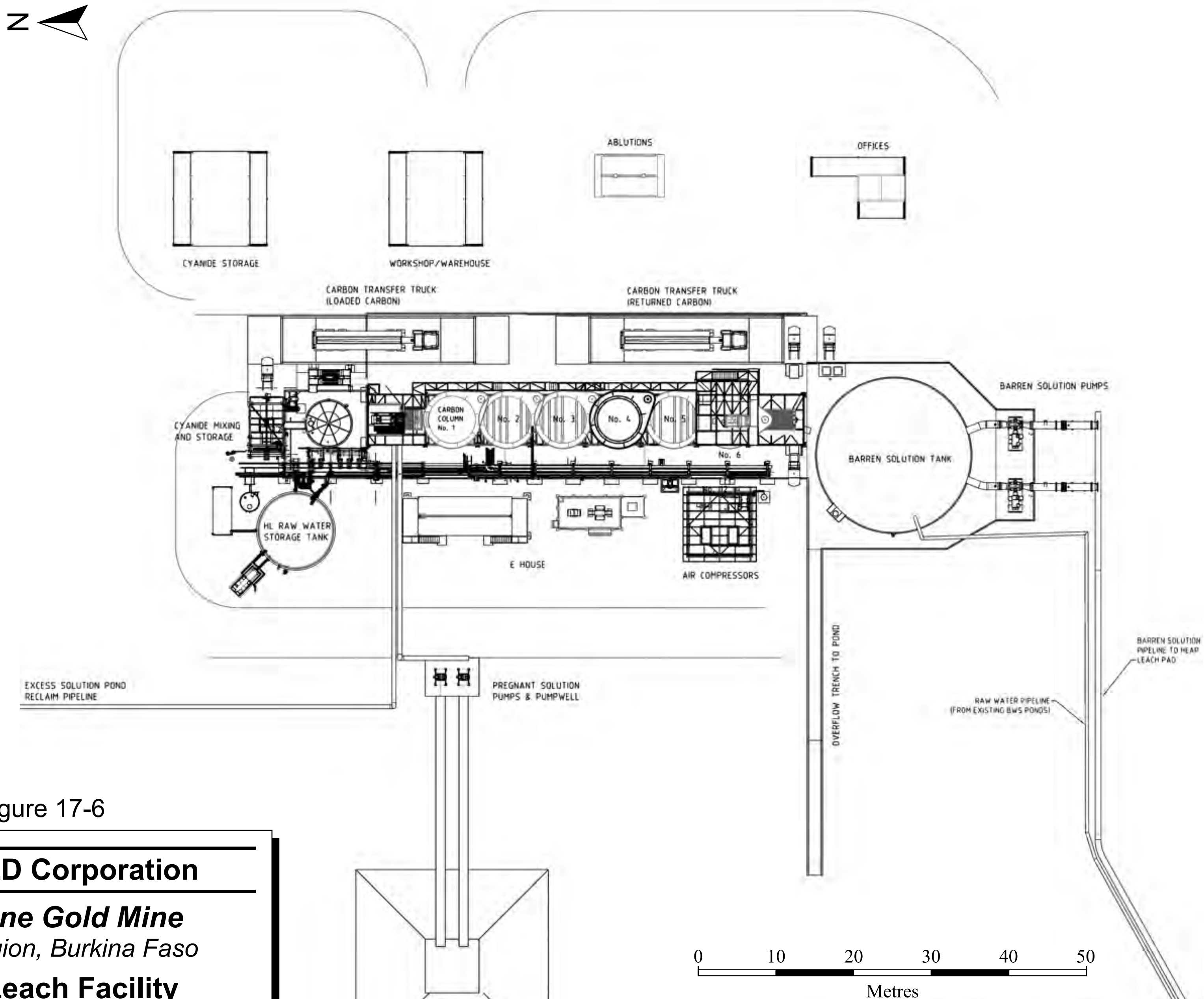

Figure 17-6

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Heap Leach Facility
Adsorption Plant Layout

January 2020

Source: Lycopodium, 2019.

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17.3.2 SUMMARY OF KEY PROCESS DESIGN CRITERIA

The process design criteria listed in Table 17-5 form the basis of the process design and mechanical equipment list.

TABLE 17-5 KEY PROCESS DESIGN CRITERIA

Criteria Description	Units	Nominal	Source
Plant Throughput	(Mt)	8.5	IMG
Estimated LOM Tonnage	(Mt)	44.34	IMG
Life of Mine	(years)	5.2	Calculated
Ore Type	(-) ( )	Argillite, Arenite, and Turbidite	IMG
Head Grade	(g/t Au)	0.37	IMG
Lab Gold Recovery	(%)	69	KCA
Lab to Field Gold Recovery Discount	(%)	2	KCA
Overall Field Gold Recovery	(%)	67	KCA
Crushing Plant Availability	(%)	73	IMG
Crushing Plant Product Size (P_{80})	(mm)	7	Test work
Design CWi – Average (49th Percentile)	(kWh/t)	17.1	Test work / OMC
Design Ai – Average (52nd Percentile)	(g)	0.322	Test work / OMC
HPGR Design Specific Throughput	($t{\cdot}s/(m^3{\cdot}h)$)	250	Lycopodium
HPGR Design Specific Energy	(kWh/t)	2.2	Lycopodium
Plant Availability for Other Plant Areas	(%)	91.3	Lycopodium
Leaching Method	(-)	Heap Leach	Lycopodium
Heap Leaching Cycle Time	(days)	180	KCA
Leach Days Limited by Solution ('Transition' Period)	(days)	60	KCA
Solution Application ('Transition' Period / Remaining Period)	($L/m^2/h$)	10 / 5	KCA
Heap Leach Pregnant Solution Tenor	($g\ Au/m^3$)	0.190	Calculated
Number of Carbon Adsorption Columns (CIC)	(#)	6	Lycopodium
Number of CIC Trains	(#)	1	Lycopodium
Size of CIC Column	(t)	8.5	Lycopodium
Carbon Movement	(t/batch)	8.5	Lycopodium
	(batch/week)	5	Lycopodium
	(t/week)	42.5	Calculated
Carbon Gold Loading	(g/t Au)	1,427	Lycopodium
Elution Circuit Type – Existing	(-)	Pressure Zadra	Lycopodium / IMG
Elution Circuit Size – Existing	(t)	17	Lycopodium / IMG
Frequency of Elution	(strips/week)	2.5	Calculated
Sodium Cyanide Addition	(kg/t ore)	0.26	KCA
Cement Addition	(kg/t ore)	4.0	Test work
Barren Solution Make-up Water	(m^3/h)	337 / 470	Nominal / Design
Water Supply from BWS (bulk water storage) No. 1 and No. 2	(m^3/h)	365 / 1,035	Nom. / Max. Intermittent

Source: Lycopodium, 2019

17.3.3 DETAILED PROCESS AND PLANT DESCRIPTION

17.3.3.1 CRUSHING PLANT

A three-stage crushing circuit has been proposed for the Project to reduce the ROM with a F_{80} 423 mm to a P_{80} 7 mm. Feed to the crushing plant will be accomplished via a haul truck dumping re-handled ore from a stockpile into a feed bin which directly feeds a gyratory crusher. Primary crushed ore will be drawn from the crushed ore bin at a controlled rate of 1,329 t/h via a variable speed apron feeder to send to a closed loop secondary crushing circuit. Primary crushed ore will first be fed to one horizontal vibrating screen where the oversize is further crushed by the secondary cone crusher in closed circuit with the horizontal vibrating screen. The undersize is considered as the final secondary crushed product, which will be transferred to a stockpile for storage. Secondary crushed ore will be reclaimed from the stockpile to feed the tertiary crushing circuit. The tertiary crushing circuit is comprised of an HPGR unit configured in a closed screening circuit. The undersize from the tertiary screens will be transferred by the tertiary crushing product conveyor to the fine ore bin. A metallurgical sampler will be installed at the tertiary crushing product conveyor for metallurgical accounting.

17.3.3.2 AGGLOMERATION AND STACKING

Tertiary crushed product in the fine ore bin will be drawn by a belt feeder to discharge onto the agglomerator feed conveyor. A weightometer will be placed underneath the agglomerator feed conveyor to track and monitor ore tonnage being conveyed. This value will be used to control the speed of the feeders and the rate at which cement is added. Cement will be added on to the agglomerator feed conveyor via a rotary valve and screw conveyor suspended from a storage silo. The ore and cement will be mixed in an agglomeration drum with sprayed barren solution to produce an agglomerated product with approximately 10% moisture content. The cement will also provide alkalinity control for the ore. The agglomerated ore will be transferred through a series of overland conveyors, and then mobile grasshopper conveyors before being stacked onto the leach pad via a radial stacker. Grasshopper conveyor quantity will vary based on the distance from the crushing plant to the exact location of ore being stacked.

17.3.3.3 HEAP LEACHING

The leach pads will be constructed in two phases to minimize initial capital expenditure and prevent unnecessary exposure of pad liner to the environment. Three months prior to actual planned production, 380,000 m^3 of pad overliner material will be required for constructing Phase 1 of the leach pad. Similarly, for Phase 2, 450,000 m^3 of pad overliner material will be required three months prior to planned production. The overliner material can be generated within one month using the primary and secondary crushing circuit with minor setting adjustments.

The design bulk density for the stacked ore is 1.6 t/m^3 and each lift will be approximately 10 m high. The pad will have an overall leach area of approximately 830,000 m^2. A pregnant solution pond will be used to capture solution during a drain-down from the leach pad, and the excess solution pond will be used to capture precipitation and run-off from the surrounding area during a 24-hour, 100-year rainfall event.

Heap leaching at Essakane will be divided into two leaching periods. The first is limited by solution and is referred to as the 'transition' period, and the second period is limited by time. Cyanide solution will be dosed directly into the barren solution tank, and the cyanide bearing solution will then be applied onto the leach pad at a rate of 10 L/m^2/h during the 'transition' period, and 5 L/m^2/h for the remaining time limited period through a piping network consisting of sprinklers and drip emitters. Anti-scalant will also be added to the barren solution tank to prevent scaling in the pipelines. Based on column leach test results, a leach period of approximately 180 days is required to maximize gold recovery. The first 60 days of the 180 days will be considered as the 'transition' period, and the remaining 120 days will be the time limiting period. Upon reaching the leach cycle time, pregnant solution drained from the leach pad to the pregnant solution pond will be pumped to the CIC circuit for further processing. Barren solution from the CIC will be returned to the barren solution tank to be recirculated and sprayed onto the heap leach pad.

The barren solution pump will apply a nominal flow rate of 1,913 m^3/h onto the leach pad, and the CIC circuit will be capable of receiving the full flow drained from the pad. In the case that the pregnant solution pond is full, the overflow will flow to the excess solution pond. The excess solution can be reclaimed with a vertical pump to the CIC circuit for processing or sent to the barren solution tank for recirculating onto the leach pad.

The Project is designed for a 67% field gold extraction from heap leaching. This was based on 69% laboratory gold extraction from column leach test work conducted by KCA. A 2% laboratory to field discount factor typical for 'clean' ores was then applied to account for residual recovery, making the design gold recovery 67%.

17.3.3.4 CARBON ADSORPTION CIRCUIT

The adsorption circuit will consist of one train of six open, up-flow columns, each with a carbon capacity of 8.5 tonnes and will operate as an expanded bed contactor.

Pregnant solution containing soluble gold and silver will be pumped from the pregnant solution pond to the columns to remove gold and silver via carbon adsorption. The adsorption circuit will be operated manually on a daily basis to allow counter-current contact with the carbon to achieve the targeted carbon loading. Solution will enter into the bottom of each column and exit from the top. Dart valves will be used to control flow to a column, and also to bypass the feed to it, if required. The first column will contain solution with the highest gold concentration and carbon with the highest gold loading. As the solution passes through the next five columns, the gold concentration will drop off, leaving the weakest gold concentrated solution to be in contact with the freshest carbon (or most recently regenerated stripped carbon) in the last column. Solution exiting the last column will pass over the carbon safety screen to provide a visual check on whether any carbon is escaping from the columns. The screen underflow will flow to the barren solution tank to be mixed with cyanide solution for re-spraying onto the heap leach pad.

Carbon advancement between the columns will be manually controlled by the operator through opening and shutting different isolation valves around the CIC carbon transfer pump. Loaded carbon will be transported by truck from the first column to the acid wash column. Subsequently, carbon advancement will progress down the carbon adsorption train until fresh or regenerated carbon is added to the last column. The CIC circuit is designed to advance 8.5 tonnes of carbon every 1.4 days (or 5 days per week), making a complete 17 tonne carbon batch every 2.8 days (or 2.5 days per week) for processing in the existing elution facility. The CIC circuit and the existing elution facility are approximately three kilometres apart. Loaded carbon from the CIC train will be recovered using a loaded carbon dewatering screen, and then transported by truck to the existing elution facility for processing. Regenerated barren carbon or fresh carbon will be transported from the existing elution facility to the CIC circuit, also by truck. A portion of the barren solution will be sent to the carbon transfer truck and be used as carbon transfer solution when pumping barren carbon slurry back to the last stage of the CIC. A returned carbon dewatering screen will be available for this step. Pregnant solution and barren solution samplers will also be installed for metallurgical accounting at the CIC.

17.3.3.5 ACID WASH AND ELUTION

The acid wash and elution steps will be operated as per the existing sequence currently in place at the Essakane CIL facility.

Additional pieces of equipment will be added to this existing area in order to provide a means of loading and unloading carbon between the CIC circuit and the existing elution facility. Loaded carbon from the CIC area will be unloaded from the carbon transfer truck by raw water addition into the truck and pumping the diluted carbon slurry to a heap leach (HL) loaded carbon dewatering screen. The screen oversize will be stored in the HL loaded carbon storage tank. Once a batch of 17 tonnes of carbon is reached, raw water will be used to educt the carbon from the storage tank to the same HL loaded carbon dewatering screen, in which the oversize will this time be passed through the existing loaded carbon scrap screen prior to entering the acid wash column.

The volume of the existing carbon bed is approximately 35 m^3 for a batch of 17 tonnes of carbon. The batch of loaded carbon will be transferred every other day to the acid wash column for acid washing with a 3% w/v hydrochloric acid solution. A total of 29 m^3 of acid wash solution will be used to remove scale and other inorganic contaminants to prepare for the elution step. The residual acid solution will be drained to a sump in the elution area and will be used as fine carbon transfer solution in the existing fine carbon treatment circuit. The acid washing sequence will require eight hours to complete. The carbon will be soaked in the acid solution for three hours, rinsed and drained for two hours, neutralized with caustic solution for two hours, and finally transferred to the elution column in the last hour.

Gold and silver on the carbon will be eluted in the elution column with 60 m^3 of solution containing 0.2% w/v cyanide and 1% w/v caustic at a pressure of 300 to 400 kPa, and a temperature of 120°C to 130°C. This solution will be circulated through the heater, heat exchanger, elution column, and five to seven parallel electrowinning cells, until gold content is depleted in the strip solution exiting the electrowinning cells. The pressure Zadra elution step will require approximately 16 hours to complete. The elution system will require two hours to heat up, one hour to add water to the eluate tank, ten hours to elute with electrowinning, one hour to drain, and finally two hours to transfer the eluted carbon to the carbon regeneration kiln.

17.3.3.6 GOLD ROOM

The existing gold room facility will be utilized for smelting the gold sludge produced from the elution electrowinning step to produce the final doré bars.

The electroplated gold sludge will be removed from the stainless-steel mesh cathodes by washing with high-pressure water. The resulting slurry will be pressure filtered with the solid then dried in an oven. The dried sludge will be direct smelted with fluxes in a furnace to produce doré bars. Fume extraction equipment will be available to remove gases from the electrowinning cells, oven, and smelting furnace.

17.3.3.7 CARBON REGENERATION AND HANDLING

After completion of the elution process, the barren carbon will be transferred from the elution column to an existing carbon regeneration circuit. Eluted carbon will be transferred to the eluted carbon dewatering screen which will dewater the carbon prior to feeding the carbon regeneration kiln. The carbon will be heated to 600°C to 700°C and held at this temperature for at least 20 minutes to allow regeneration to occur. The existing kiln has a capacity of 800 kg/h, which will be capable of regenerating 100% of the carbon from the heap leach CIC circuit at only 35% utilization. Regenerated carbon from the kiln will be quenched and pumped to the HL return carbon dewatering screen and recovered for storage in the HL return carbon tank. Once a batch of 8.5 tonnes of carbon is reached in this tank, the carbon will be transferred via eduction into the carbon transfer truck to be returned to the CIC circuit. The carbon transfer solution from the truck will be pumped to the carbon sizing screen in the existing fine carbon treatment area. The screen oversize will be collected in a coarse carbon bag to be returned to the adsorption circuit and the screen undersize will be treated through a carbon fines filter and collected in a bag.

17.3.3.8 REAGENTS

The operations will require sodium cyanide, cement, activated carbon, sodium hydroxide (caustic), antiscalant, and flux. Some of the key reagent systems are described in the sections to follow.

Cement

Cement will be delivered in bulk truckload quantities of approximately 40 t per trip and will be stored in a cement silo with a 250 t capacity to provide approximately two days of inventory.

The cement will be unloaded from the silo via a rotary feeder and screw conveyor onto the tertiary crushing product conveyor for ore agglomeration and leaching pH control.

Caustic Soda

The existing caustic soda mixing system will produce a caustic solution at 15% NaOH to be transferred to a caustic dosing tank and pumped to the acid wash and elution column for pH control.

Cyanide

An existing cyanide mixing system will produce cyanide solution at 20% NaCN by either breaking bags of cyanide into a mixing tank, or by diluting premixed cyanide solution from a cyanide isotank truck with raw water. The cyanide solution will then be transferred to a cyanide dosing tank and pumped to the elution column for use during the carbon elution step.

Another new cyanide mixing system will be available at the CIC area as the existing cyanide system is too far away to be used there. The new system will have a mixing tank with bag breaker to produce a 20% NaCN solution and will be transferred to a storage tank and dosed into the barren solution tank for application onto the leach pad.

Activated Carbon

Approximately 51 t of carbon will be required initially to fill up the train of six carbon adsorption columns. In terms of carbon make-up, the existing plant has a carbon attrition tank which will condition the fresh carbon prior to sending it to the carbon transfer truck for delivery to the CIC circuit.

Anti-Scalant

Anti-scalant reagent, delivered in a 1-t bulk box, will be pumped into the existing eluate tank to minimize scaling during the elution step. Furthermore, to minimize scaling of the solution piping network of drip emitters and sprinklers for the leach pad, anti-scalant reagent will also be pumped into the barren solution tank in the new CIC area.

17.3.3.9 PLANT SERVICES

Air

Plant and instrument air will be supplied from new duty / standby air compressors for the new CIC area. The air will be dried before distribution to the air receiver. For equipment in proximity of the existing CIL plant, plant and instrument air will be supplied from the existing compressed air system.

Water

All raw water and make-up water requirement will be sourced from BWS (bulk water storage) Pond No. 1 and No. 2.

Raw water from BWS Pond No. 1 and No. 2 will be used for the following:

- Dust suppression in the existing crushing and stockpile area.
- Reagent mixing, carbon transfer water, and other usages for the existing elution circuit area and gold room.
- Make-up water to barren solution tank, screen sprays, and reagent mixing for the new CIC and heap leach areas.

In total, the amount of water required on a nominal basis is 365 m^3/h. The instantaneous intermittent flow can potentially be as high as 1,035 m^3/h if all the maximum flowrates occur simultaneously. Based on preliminary assessment, the existing BWS pumps in Pond No. 1 and No. 2 are sufficient to supply the required raw water flow rate.

The main source of make-up water to BWS Pond No. 1 and No. 2 will be from an existing off channel reservoir (OCR).

Potable water will be used for all the safety showers. A potable water line will be extended from the existing potable water system to the agglomeration area safety shower. The new CIC area and new cyanide area will utilize portable safety showers with potable water supplied by a truck as needed.

18 PROJECT INFRASTRUCTURE

18.1 EXISTING INFRASTRUCTURE

18.1.1 GENERAL

General services are an essential component to the success of the Project. Due to the remoteness and complex logistics of the Project coupled with the limited services available in Burkina Faso, the scope and extent of the general services department required to support production is very substantial.

Mine infrastructure consists of a mine office complex (mine and administrative offices, change houses, and canteens), equipment workshop with overhead cranes integrated to the main warehouse and external wash down bays, blasting and explosives compound including magazines, diesel storage and dispensing facility, and a drill core storage facility.

The mine village was built from prefabricated structures and this village was initially used as the construction camp. The site has a satellite communications system. Two office complexes are located in the Project plant area, one to service mine operations, maintenance, and administrative services and the other reserved for the capital project department. The Capital project department offices would be used for the heap leach owner team during construction. The main warehouse is attached to the mine maintenance shops and includes a sizeable storage yard.

The initial mine infrastructure and support facilities constructed between 2009 and July 2010 have been modified and/or adapted for the expansion phase which was carried out from 2012 to 2014. The following summarizes the modifications to the main infrastructure built for the CIL plant expansion and the ones required for the heap leach project are discussed in Section 18.2.

There is no current infrastructure at the Gossey deposit.

18.1.2 MINE TRUCK SHOP AND WAREHOUSE

The existing mine truck shop and warehouse was expanded to accommodate the increased maintenance requirements for the additional mobile mining equipment. An extension of the truck shop was completed in 2019. No additional expansion will be needed for the heap leach.

18.1.3 SITE AND MINE ROADS

A fenced haulage road of 8.8 km was built in 2015 between the Falagountou pit and the crusher ROM pad to accommodate the mine truck traffic. This road crosses the regional road from Falagountou and then stays north of this road, maintaining a distance of 500 m from any dwellings located near the regional road and does not impact any arable land. In 2018, a new three kilometre fenced road was built around the Wafaka pit.

The heap leach area will be within the site fence and the project will reuse existing roads to feed the primary crushers. The existing road system will also be upgraded to allow access to the heap leach pad and CIC plant. Finally, a perimeter road will be established around the heap pad.

18.1.4 COMMUNICATION SYSTEM AND IT

All the network communication and information technology (IT) related hardware are linked to the existing systems at Essakane. A very small aperture terminal (VSAT) system is installed to allow for the transfer of large files and to provide Internet connectivity for employees. Network and cable television access are available in the mine camp. The heap leach area will be linked to the existing plant control room for remote monitoring of the facility.

18.1.5 FUEL OIL STORAGE

Fuel oil storage capacity was expanded to four 500 m^3 light fuel oil (LFO) tanks during the project expansion. The LFO storage area as well as the existing containment area were extended, as well as pumping installations. No further expansion is required for the heap leach project.

18.1.6 EXPLORATION BUILDING

An exploration building, including office space, logging, warehouse, and sampling area is part of the existing infrastructure to accommodate the resource development and exploration groups. No modifications are planned for the Project.

18.1.7 MINE CAMP

The mine camp was initially built to support construction and was then used for operations. Specific upgrades associated with the mine camp and offices were carried out to accommodate the mine expansion:

- The kitchen was enlarged to better accommodate the mine expansion and food preparation.
- A recreation building was added, including a full size multifunction gymnasium and workout rooms.
- Sewage capacities were installed and are sufficient to handle waste from the mine village and the CIL plant area.
- Two existing potable/fire water tanks are supplying the mining camp with a 220 gallon per minute (gpm) pump and five hydrants and fire cabinets.
- The current camp capacity is 711 rooms (1,302 beds).

No additional capacity for the camp is planned for the heap leach project or the CIL plant upgrade.

18.1.8 RIVER DEVIATION

In order to expand the EMZ pit to the north, a five kilometre deviation of the Gorouol River was undertaken and effectively protected the EMZ pit during seasonal rains. No modifications are required for the Project.

18.1.9 POWER GENERATION AND DISTRIBUTION

The current power plant was developed in two phases between 2010 and 2014. During the period between 2010 and 2014, temporary six megawatt (MW) LFO generators from SDMO Industries were installed to supplement power to the original plant. At present, these units are not utilized on the Essakane distribution network.

The first phase has five generators (Wärtsilä 12V32 units each of 5,256 kW based on 35°C) for a total installed power of 26,280 kW. During the second phase, six additional units were installed in order to supply 25,000 kVA of additional power required under the expansion scenario, bringing the total units installed to 11, which includes two spares (one for maintenance, and one for emergency spare).

Additional fuel storage capacity and fuel treatment capacity, as well as day storage capacity, were built accordingly. There is also a spare space in the existing power plant for one more 12V32 generator.

The power distribution has been upgraded based on the existing configuration in order to supply the CIL plant expansion in 2014. Four electrical rooms were installed to supply energy to the new grinding circuit, the pebble crusher, the new screening, and the crushing circuits.

In 2018, a photovoltaic (PV) solar plant was constructed on the Project's site and connected to the existing thermal powerhouse grid. The solar plant has an installed power of 14.92 MWp DC and a delivery capacity of 11.46 MW alternating current. It consists of nearly 130,000 panels separated into three different fields, each field being connected to two inverters and injecting power to the grid through a transformer. The addition of this plant is estimated to reduce carbon dioxide emissions caused by the mine's activities by an estimated 18,000 tons/year.

18.1.10 ASSAY AND METALLURGICAL LABORATORIES AND MILL OFFICE

The metallurgy building consists of two main sections: the metallurgical laboratory and fire analysis area, and the fire assay furnace area. The heap leach expansion will require an extension of the existing laboratories mainly to perform column testing.

18.1.11 ADMINISTRATION BUILDING

All administration offices are modular-type structures placed on concrete floor slabs with adequate sanitary and air conditioning facilities. The administration building features the following divisions: reception, boardroom, kitchen, offices, and map room.

18.1.12 POTABLE WATER AND TREATMENT FACILITIES

Water is currently extracted from underground using borehole pumps feeding into a buried HDPE ring main which, in turn, feeds into a potable water storage tank located within the plant boundary. The water is filtered and chlorinated prior to entry into a tank which is specially lined to ensure that the water is then acceptable for human consumption.

In order to handle waste from the additional construction camps in 2014 and influx of workers, two 100 m^3/day sewage facilities were installed beside the existing facility, one at the mine village and the other at the CIL plant area during the expansion. No further expansion is planned for the heap leach project.

18.1.13 BULK WATER STORAGE AND PUMPING

As there is no continuous access to fresh water for the processing plant, bulk water storage (BWS) basins were constructed. During the rainy season, these BWS structures are recharged to store water in sufficient quantity to secure supply to the plant and for dust control in the mining areas. The recharging point OCR, with a surge capacity installed adjacent to the Gorouol River at 5.4 km from BWS is the source for the water. A dam installed on the Gorouol River raises the water during the rainy season to overflow into the OCR. When sufficient volumes are available in the OCR, pumping, using two pipelines permits the transfer of the water to the BWS basins. There are three BWS basins containing 1.87 Mm^3, 1.54 Mm^3, and 2.60 Mm^3 for a total of 6.01 Mm^3 of water. During a dry year, the operation of the heap leach process will require approximately 1 Mm^3 of makeup water. The BWS system will be upgraded as part of the mine operations and will be able to provide the heap leach makeup water.

18.2 HEAP LEACH FACILITY

Feasibility-level design of the HLF was prepared by SRK Consulting (Canada) Inc (SRK). The HLF consists of the following facility components, each of which is discussed in the following sections:

- Heap leach pad
- Pregnant solution pond
- Excess solution pond
- Surface water management controls

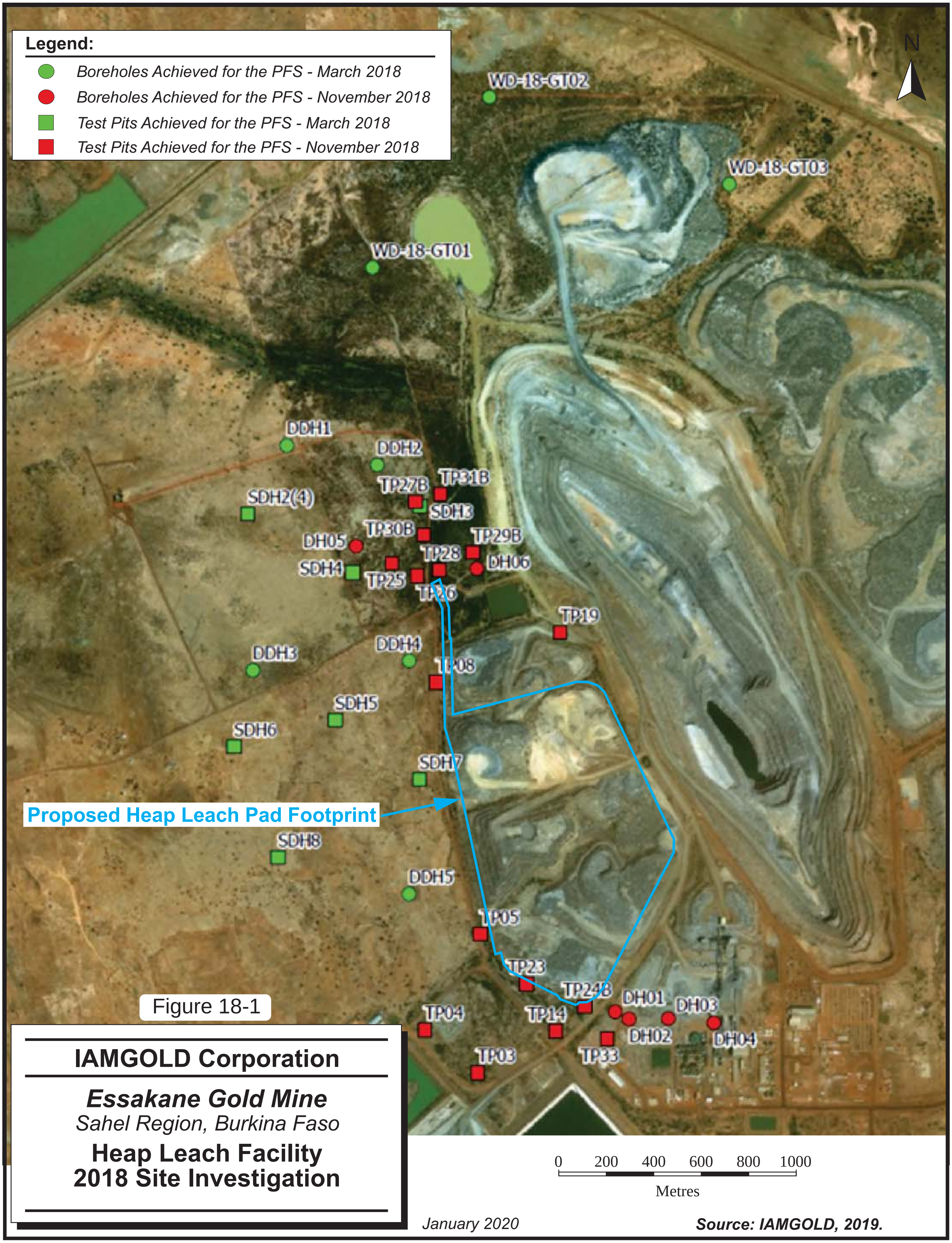
18.2.1 HEAP LEACH FOUNDATION

Foundation conditions for the proposed HLF are summarized in the May 2019 report by SRK titled Essakane Mine 2018/2019 Geotechnical Investigation for the Heap Leach Pad Design (SRK, 2019b). Additional geotechnical data were available from the following reports:

- Geotechnical Investigation of Process Plant Areas (Golder, 2008a)
- Geotechnical Investigation of the Essakane Pit and Zone 3 Borrow Area (Golder, 2010a)
- Geotechnical Investigation of the Tailings Storage Facility and Water Retention Structures (Golder, 2010b)
- Geotechnical Testing of Essakane Tailings and Saprolite for TSF Basin Lining (Golder 2012)

As part of SRK's HLF design, a total of seven test pits were excavated in March 2018 during the PFS and a total of 16 were completed in November 2018 for the FS to determine depth to bedrock and subsurface conditions within the proposed heap leach pad footprint. In addition, a total of eight boreholes were completed in March 2018 with an additional six completed in November 2018. As anticipated, groundwater was not encountered during the site investigation and soils observed within the area of the future HLF were classified into four stratigraphic units: eolian deposits, residual soils (laterite), saprolite, and bedrock.

The locations of the drill holes and test pits are shown in Figure 18-1.


Legend:
Boreholes Achieved for the PFS - March 2018
Boreholes Achieved for the PFS - November 2018
Test Pits Achieved for the PFS - March 2018
Test Pits Achieved for the PFS - November 2018
N
WD-18-GT02
WD-18-GT03
WD-18-GT01
DDH1
DDH2
TP27B
TP31B
SDH2(4)
SDH3
TP30B
TP29B
DH05
TP28
DH06
SDH4
TP25
TP26
TP19
DDH4
DDH3
TP08
SDH5
SDH6
SDH7
Proposed Heap Leach Pad Footprint
SDH8
DDH5
TP05
TP23
TP24B
DH01
DH03
TP04
TP14
DH02
DH04
TP33
TP03
0 200 400 600 800 1000
Metres
January 2020
Source: IAMGOLD, 2019.
Figure 18-1
IAMGOLD Corporation
Essakane Gold Mine
Sahel Region, Burkina Faso
Heap Leach Facility
2018 Site Investigation

In SRK (2019c), SRK makes the following recommendations based on the results of the site geotechnical investigations:

- The eolian soil, as well as the laterite, must be removed from the HLF footprint and from the conveyor footing footprint.
- The saprolite can be used as construction material, if properly compacted.
- The HLF pad must be built over exposed in-situ saprolite to comply with the stability requirement (SRK, 2019c).
- The conveyor footing should be built over exposed in-situ saprolite as described within the foundation geotechnical design memo (SRK, 2019b).

18.2.2 HEAP LEACH STABILITY ASSESSMENT

SRK performed a stability assessment on the preliminary design of the HLF. The natural topography under the proposed HLF slopes northward with an overall grade of approximately 0.25%. In order to achieve the design base of heap leach slope of 0.5% the stability analysis assumed that the heap leach pad will be constructed on an above grade compacted saprolite layer that will vary in thickness from about 3 m along the southern limits to 0.3 m along its northern limit.

Two-dimensional limit equilibrium analyses were carried out using the Rocscience program SLIDE (v7.034) (Rocscience, 2017). Deterministic factor of safety (FOS) values were determined for deep seated failure in saprolite, sliding along the geomembrane, and sliding along in-situ saprolite of the final, resloped heap leach configuration. A FOS of 1.3 was the criterion used for long term overall stability, of both deep seated circular failure and block failure due to sliding, while a FOS value of 1.05 was the criterion used for pseudo-static conditions (NDEP-BMRR 1994, and Hawley and Cunning 2017).

Essakane is located in a seismically inactive region with a low potential for a seismic event. Grünthal et al. (1999) estimate the horizontal peak ground acceleration with an occurrence rate of 10% within 50 years is less than 0.2 m/s^2 (0.02 g). Pseudo-static stability analyses using a horizontal seismic loading of 0.025 g exhibit an acceptable FOS.

The static stability analysis results are summarized in Table 18-1.

TABLE 18-1 STATIC ANALYSIS STABILITY RESULTS

Section	Case	Failure Mode	Factor of Safety [1] (Static)
South	Base Case	Circular through Saprolite	1.71
		Sliding along In-Situ Saprolite	1.72
		Sliding along Liner	1.54
North		Circular through Saprolite	1.76
		Sliding along In-Situ Saprolite	1.76
		Sliding along Liner	1.52
South	Worst Case Material Properties	Circular through Saprolite	1.48
		Sliding along In-Situ Saprolite	1.42
		Sliding along Liner	1.27
North		Circular through Saprolite	1.50
		Sliding along In-Situ Saprolite	1.50
		Sliding along Liner	1.25
South	Interface friction angle 9° [2]	Sliding along Liner	1.16
	Interface friction angle 12° [2]		1.31
	Interface friction angle 14° [2]		1.41
North	Interface friction angle 9° [2]		1.08
	Interface friction angle 12° [2]		1.30
	Interface friction angle 14° [2]		1.39
South	Ore saturated to 12.5 m	Circular through Saprolite	1.68
		Sliding along In-Situ Saprolite	1.70
		Sliding along Liner	1.47
North		Circular through Saprolite	1.71
		Sliding along In-Situ Saprolite	1.74
		Sliding along Liner	1.40

Notes:
1. Acceptable minimum FOS = 1.3 for static analyses
2. All other material properties are from the base case.

The results presented in Table 18-1 indicate that the final proposed HLF is expected to exhibit an acceptable design FOS with respect to the analyzed failure mechanisms in the longer term, using an assumed set of material parameters. Sensitivity testing indicates that liner interface friction angle is the parameter with the greatest influence on the calculated FOS. While the stability analysis indicates that a smooth HDPE or LLDPE liner should meet the acceptable FOS, further analysis using site specific material parameters and interface shear strengths may indicate that textured liner is required.

Testing of the liner interface friction angle and heap leach construction materials should be performed in the next stage of design, and the stability analysis should be updated.

18.2.3 HEAP LEACH PAD DESIGN

The feasibility-level design of the Essakane Project heap leach pad was prepared by SRK using the design criteria and parameters listed in Table 18-2.

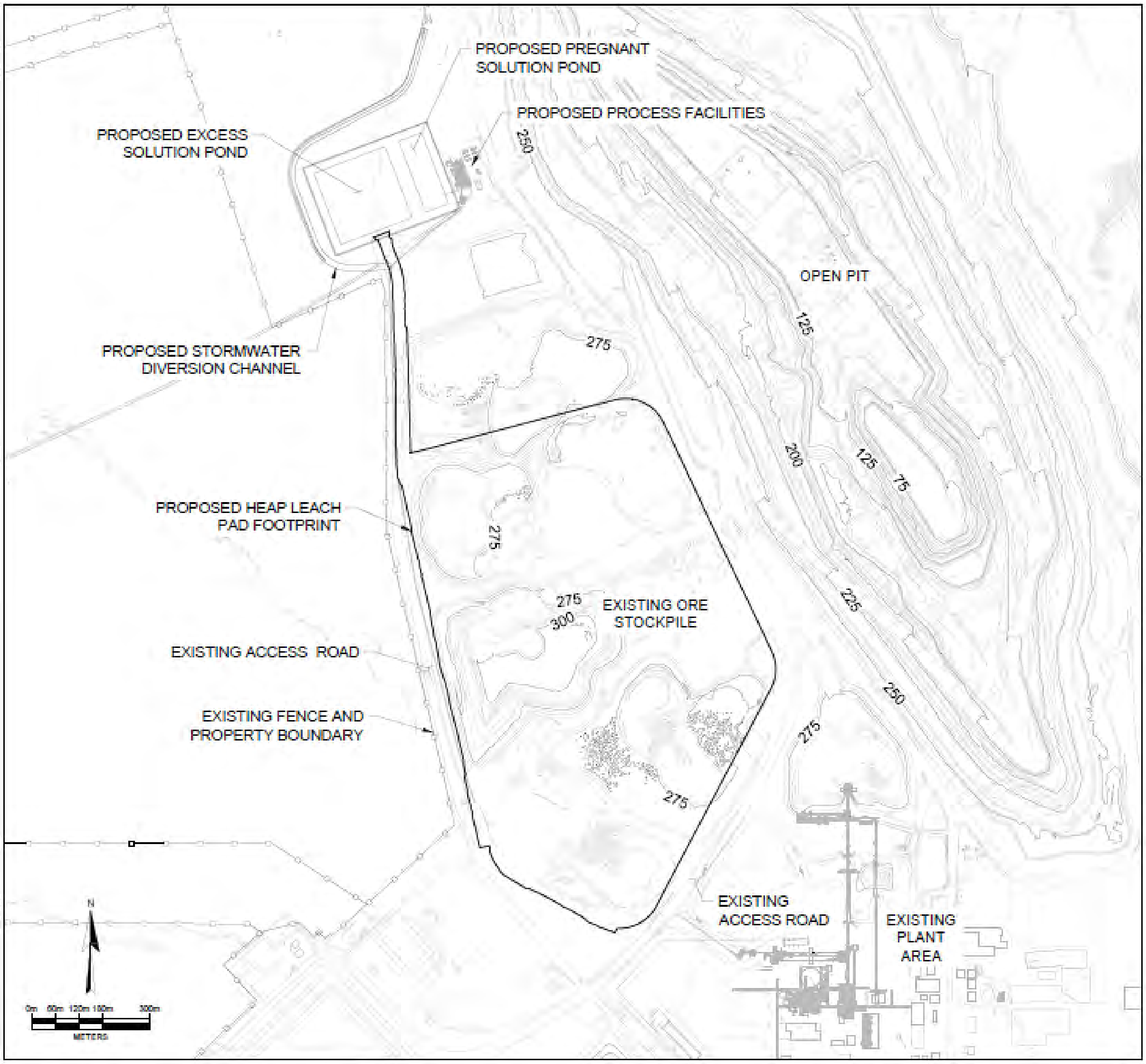
TABLE 18-2 KEY HEAP LEACH FACILITY DESIGN PARAMETERS

Parameters	Unit	Design
Ore Lift Height	(m)	10
Maximum Stacked Ore Height (above liner)	(m)	50
Total Ore Production	(000 t)	49,000
Ore Stacking Rate	(t/day)	23,288
Solution Application Rate	($L/h/m^2$)	10
Leach Cycle – Primary	(days)	60
Leach Cycle – Secondary	(days)	120
Barren Solution Flow Rate	(m^3/hr)	1,913
Water Losses by Retained Moisture in Ore	(m^3/t)	0.084
Net Solution Evaporation	(%)	10
Evaporation Reduction from Bird Balls	(%)	90
Pregnant Solution Pond Capacity (with 1m freeboard)	(m^3)	68,000
Excess Solution Pond Capacity (with 1m freeboard)	(m^3)	175,000
Plant Operating Efficiency	(%)	91

Source: Lycopodium 2019

The leach pad and process facilities are situated west of the existing open pit in an area currently covered by low-grade ore stockpiles, as shown on Figure 18-2.

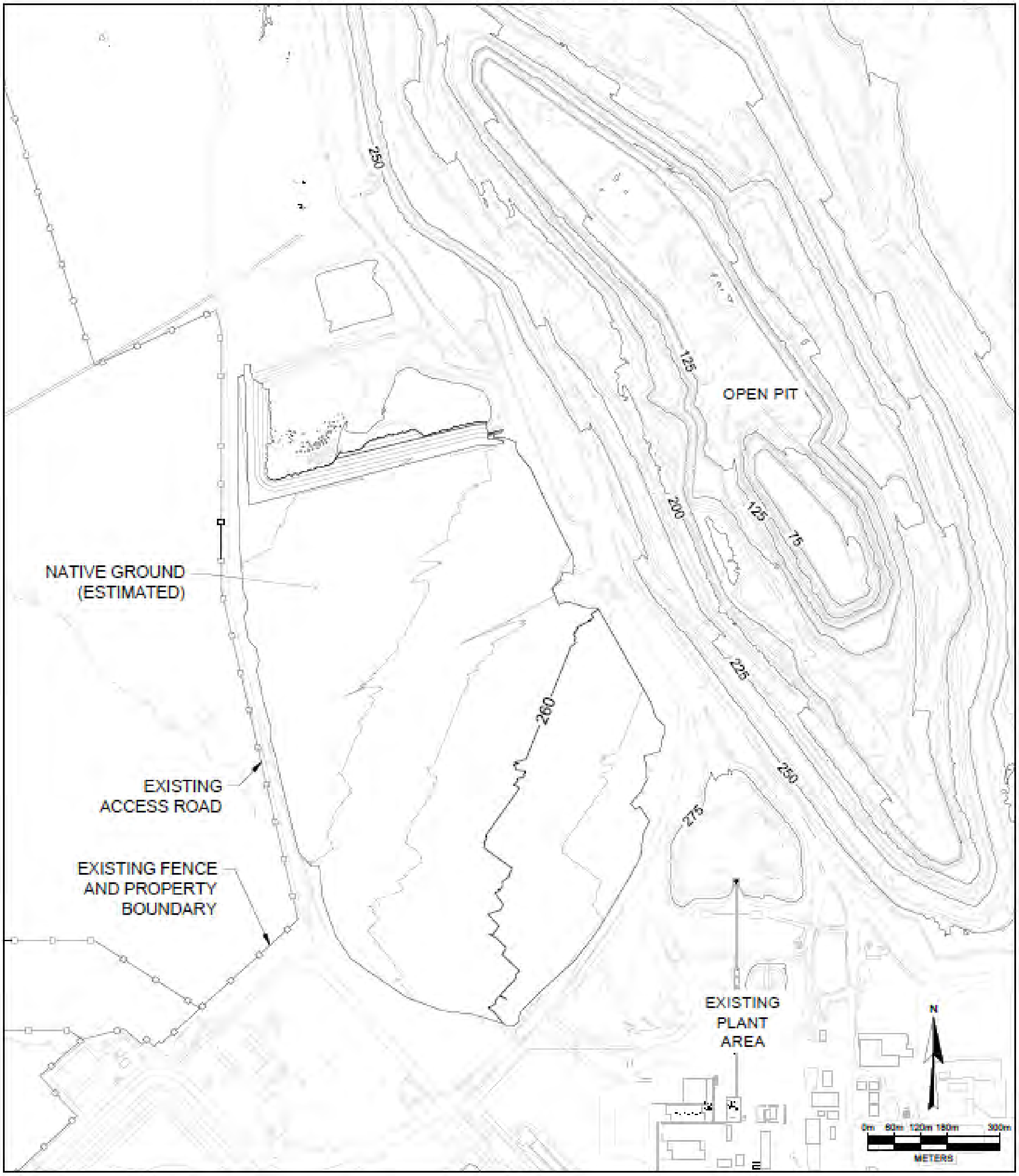
FIGURE 18-2 HEAP LEACH PAD SITE LAYOUT



Source: SRK Consulting (Canada) Inc., 2019

Native topography was estimated by digitizing approximate native ground elevations east and west of the stockpiles. The elevation under the stockpiles ranges from about 265 MASL on the southeast to 250 MASL to the northeast (Figure 18-3) for an overall surface grade of approximately 0.9% from southeast to northwest.

FIGURE 18-3 ESTIMATED NATIVE GROUND SURFACE WITHIN HEAP LEACH FACILITY FOOTPRINT.



Source: SRK Consulting (Canada) Inc., 2019

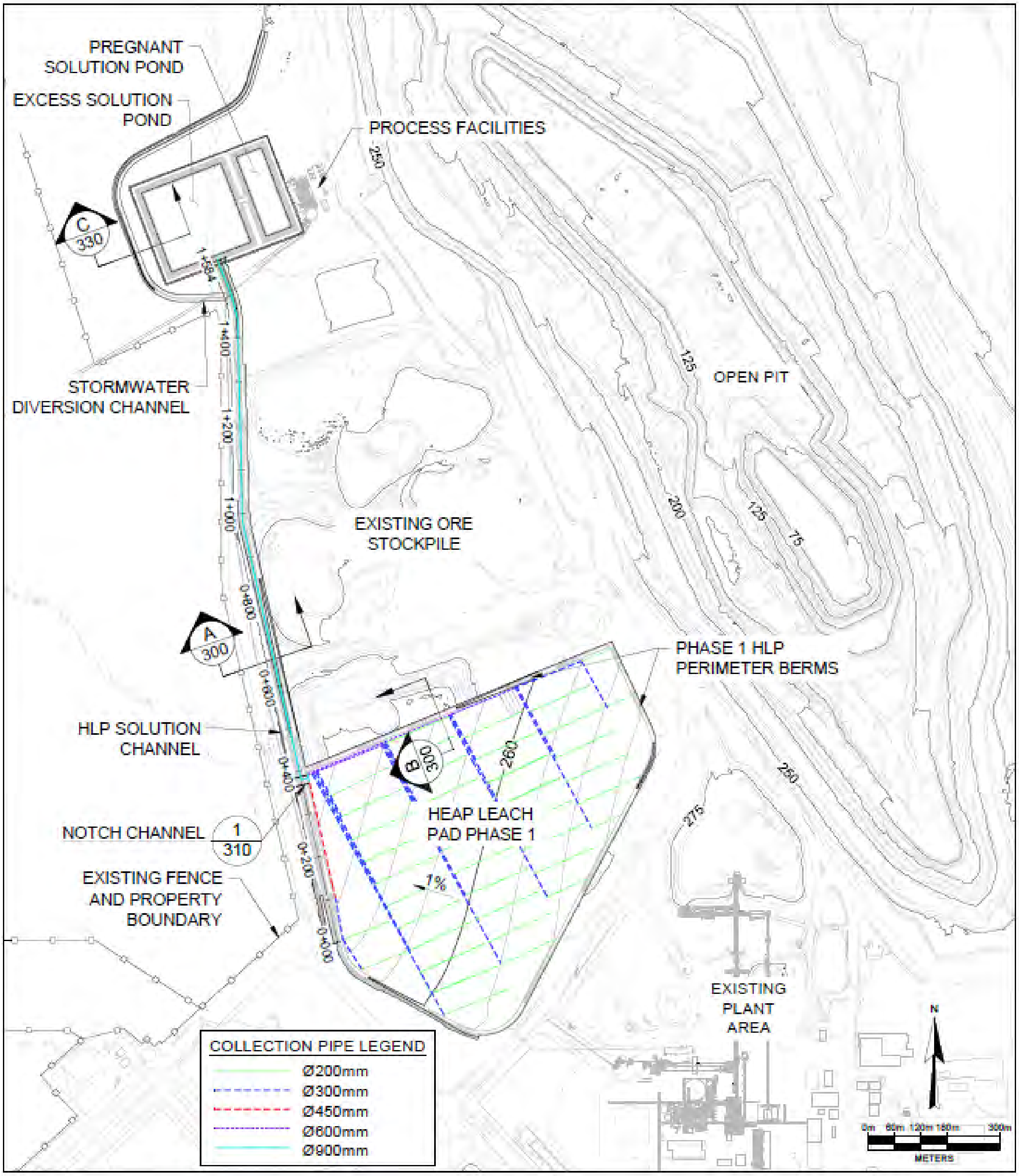
The heap leach pad has been designed for construction in two phases: Phase 1 covers an area of 380,000 m^2 and Phase 2 covers an additional 450,000 m^2, for a total of 830,000 m^2. The leach pad base grading will approximately parallel the estimated native topography and includes cut-to-fill regrading to achieve a consistent grade of 1% from southeast to northwest (Figure 18-4 and Figure 18-5).

Prior to development, the footprint of each HLF phase and the ponds will be cleared of existing stockpiled low-grade ore and, where necessary, cleared and grubbed of existing vegetation, topsoil, and unsuitable soil per the final geotechnical recommendations completed as part of detailed design. According to the feasibility-level geotechnical site investigation report (SRK, 2019b), in situ eolian and laterite soils are not suitable for foundation support and should be removed prior to HLF and associated facility construction. Native saprolite soils are suitable for construction material beneath the HLF if compacted to 95% of maximum dry density at optimum moisture content per modified Proctor testing (ASTM D1557). Cut-to-fill regrading will be utilized where possible to minimize earthworks requirements. Unsuitable soil removed from the base of each phase of the leach pad will be stockpiled for use as growth to provide for final cover to be placed over the regraded HLF at the end of the Project. Excess suitable soil from the Phase 1 footprint (relative to the estimated native ground topography) will be stockpiled for use in Phase 2 leach pad construction.

The leach pad base liner system will be a compacted 0.3048 m (12-inch) thick, low-permeability subgrade layer overlain by a single geomembrane liner. The subgrade layer will consist of 0.3048 m (12 inches) of saprolite scarified, moisture conditioned, and compacted to achieve a hydraulic conductivity of 1x10-6 cm/sec or less. Additional laboratory testing of in-situ soils is recommended as part of detailed design to confirm that the specified permeability can be achieved with local saprolite soils.

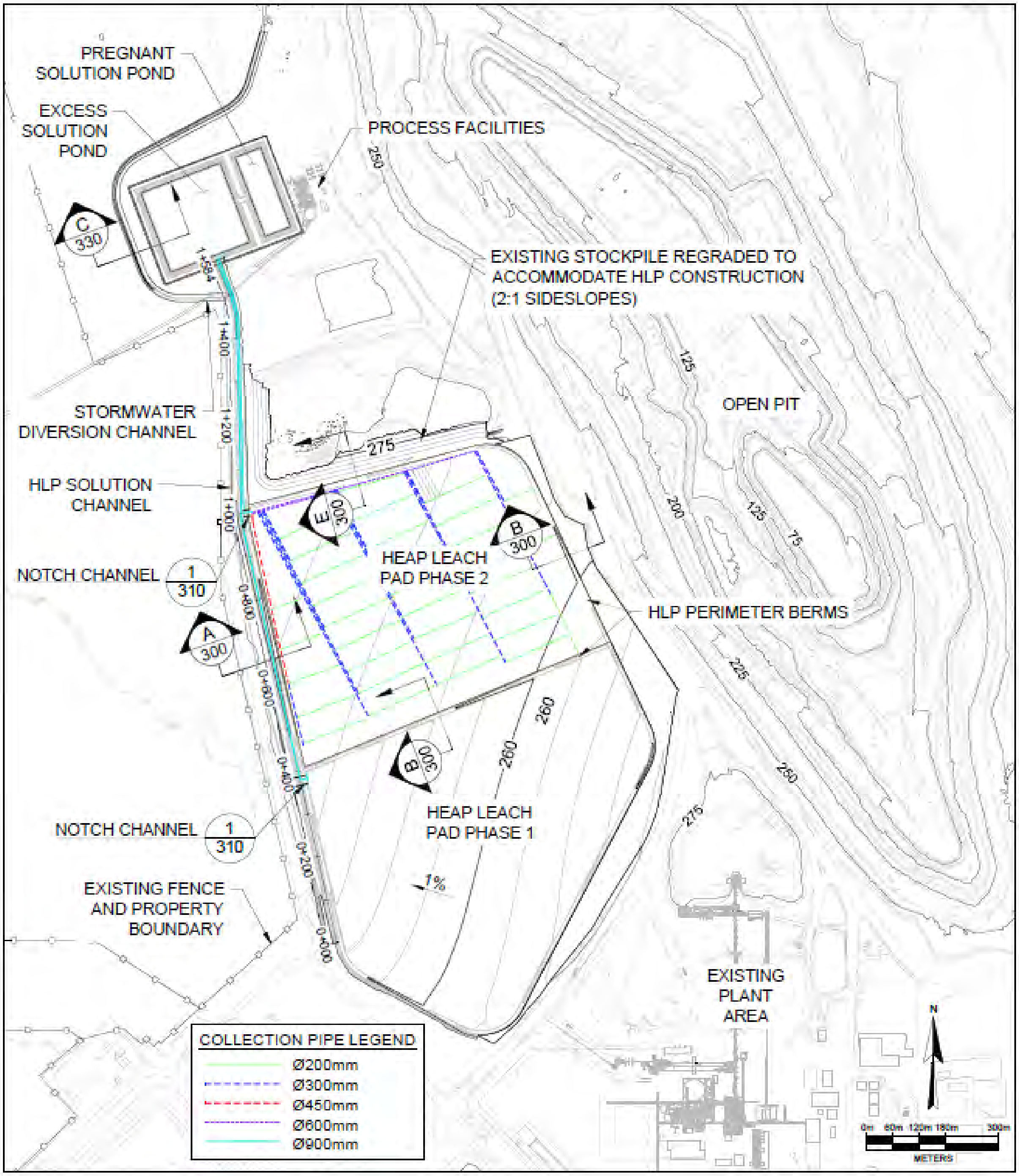
The geomembrane liner will be a single-sided, textured, 2-mm thick high-density polyethylene (HDPE) geomembrane liner. The liner will be placed with the textured-side down to improve interface shear strength with underlying soils. Liner and overliner testing are described in SRK (2019d). Double-sided textured liner should be considered for areas of exposed liner around the heap leach pad and pond perimeter to improve traction for worker safety.

FIGURE 18-4 PROPOSED PHASE 1 HEAP LEACH FACILITY CONFIGURATION



Source: SRK Consulting (Canada) Inc., 2019

FIGURE 18-5 PROPOSED PHASE 2 HEAP LEACH FACILITY CONFIGURATION



Source: SRK Consulting (Canada) Inc., 2019

Pregnant solution will be collected within the overliner collection system, consisting of a 1-m thick layer of crushed rock (overliner layer) and a network of perforated and solid corrugated polyethylene pipes placed at the base of the overliner. Based on the results of material testing described in SRK (2019d), crushed arenite meeting the grain size distribution shown in Table

18-3, with a minimum permeability at least two orders of magnitude higher than the agglomerated ore permeability, is recommended for use in the 1-m thick overliner layer:

TABLE 18-3 OVERLINER LAYER MATERIAL GRADATION

Metric Sieve Size (mm)	U.S. Standard Sieve Size	Percent Passing by Dry Weight
50.8	2-inch	100
38	1.5-inch	85-100
19	¾-inch	50-100
4.75	No. 4	20-50
2.0	No. 10	10-30
0.425	No. 40	0-20
0.075	No. 200	0-10

Source: SRK Consulting (Canada) Inc., 2019

SRK recommends that additional loaded permeability (USBR 5600), interface shear and liner integrity testing of the final overliner material selected be completed as part of detailed design to ensure satisfactory interface shear strength and drainage of the overliner layer relative to the agglomerated ore.

Pregnant solution recovered in the perforated pipes in the overliner layer will be directed into a 900-mm diameter solid-wall HDPE solution pipe in the solution channel for conveyance to the pregnant solution pond. Two 900-mm diameter pipes will be placed in the solution channel, one for each phase of the leach pad. The solution channel will be constructed with the same base liner system as the HLF.

A slope stability analysis was performed for the proposed heap leach pad configuration using the computer program SLIDE (Version 7.037, 64-bit, September 26, 2018, by Rocscience Inc.). The stability analysis methodology and results are described in detail in the slope stability report (SRK, 2019f). Two-dimensional limit equilibrium slope stability analyses were performed to evaluate the stability of the proposed final slope configurations and layout of the facility. SLIDE is equipped with multiple search routines that allow for either circular or non-circular (polygonal) potential critical slip surfaces. The results of the analyses show that for the proposed overall final slope configuration of 3H:1V (horizontal:vertical), the incorporation of a textured geomembrane with the textured side down and in contact with the subgrade saprolite soils is necessary to achieve satisfactory factors of safety against mass failure. As designed, however, the proposed heap leach pad configuration will be stable under both static and seismic loading conditions with acceptable predicted levels of displacement.

18.2.4 WATER MANAGEMENT DESIGN

Water management designs for the Essakane HLF include the pregnant solution pond, the excess solution pond, and surface water controls to manage onsite and upstream water with respect to the leach pad.

The pregnant solution pond will be constructed with a double-layer synthetic liner system consisting of a 2 mm HDPE single-sided textured (textured side up) primary liner above a 1.5 mm HDPE Agru DrainLiner secondary liner. Alternatively, the DrainLiner can be replaced with a 1.5 mm smooth HDPE liner and geonet (6.25 mm [250 mil] Agru Geonet or equivalent). The pond liner system will be equipped with a leak collection and recovery system (LCRS) consisting of a gravel-filled sump to collect potential leakage between the primary and secondary liners. The LCRS sumps will be drained via a dedicated pump placed in a riser access port located on the side slope of the pond.

The pregnant solution pond has been designed to contain 12 hours of operating storage and 24 hours of drain down at the barren solution design flow rate of 1,913 m^3/hr (Table 18-2). Design of the pregnant solution and excess solution ponds and development of a range of potential make-up water pumping rates are described in the water balance technical memorandum in SRK (2019e).

The excess solution pond will be constructed with the same single-layer, base liner system as the leach pad and has been designed to contain the 100-year, 24-hour storm event falling on the total heap leach pad and pond area. During operations, stormwater runoff from the side slopes of the operational heap will flow in an open channel formed by the operational slope of the heap and the perimeter berm. This flow will ultimately report to the solution channel as open channel flow and then directly to the excess solution pond.

Stormwater runoff generated in the upstream watersheds west of the HLF will be diverted north along the west containment berm, i.e., the outside western berm of the solution channel, and then into the clean water diversion channel and around the west side of the excess solution pond to the natural drainage downstream. Design of the stormwater control system is described in SRK (2019c).

19 MARKET STUDIES AND CONTRACTS

Gold is the principal commodity at Essakane and is freely traded at prices that are widely known, so that prospects for sale of any production are virtually assured. All gold produced by IAMGOLD is in the form of doré bars, which is then shipped to a refiner who refines the doré into bullion. The bullion is then sold directly on the open market to gold trading institutions at prevailing market prices.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL AND SOCIAL STUDIES

The Environmental Code (Law No. 006-2013/AN of April 2, 2013) of Burkina Faso stipulates that an Environmental Impact Statement (EIS), or an Environmental and Social Impact Assessment (ESIA), including public enquiry and a mitigation and/or an enhancement plan of negative or positive impacts, be completed prior to the construction of a project that is likely to impact the environment. This requirement is supported by the associated Environmental Decree (Decree No. 2001-342/PRES/PM/MEE) which outlines the scope, content, and administrative procedure of the ESIA.

20.2 ESSAKANE INITIAL PERMITTING

Prior to the beginning of construction work, an ESIA was conducted by Knight Piésold Consulting and submitted to the government on August 8, 2007. This study included an Environmental and Social Management Plan (ESMP) for the Project. The ESIA was completed following a public consultation, from October 3 to November 2, 2007, with key stakeholders, as prescribed under Burkinabé law. In 2008, and following the changes made during construction, an addendum to the ESIA (2008 addendum) was submitted to the Burkina Faso authorities. There was no change at that time to the ESMP as a result of this addendum.

Following this process, on November 30, 2007, the Project was approved by the Burkina Faso authorities (Order No. 2007-083/MECV/CAB) and the mining permit over a 100.2 km² area (Order No. 2008-203/PRES/PM/MCE/MEF/MECV) was granted to Essakane S.A.

20.3 HEAP LEACH PROJECT PERMITTING

The heap leach project triggers an EIS in accordance with Article 4 of Decree No. 2015-1187/PRES-TRANS/PM/MERH/MATD/MME/MS/MARHASA/MRA/MICA/MHU/MIDT/MCT of October 22, 2015 laying down conditions and procedures for carrying out and validating the strategic environmental assessment, the study, and the environmental and social impact notice. Field works to collect baseline data were performed during January and February 2018. Consultations on targeted groups (such as women, elders, youths, gold diggers, farmers, etc.) were performed during the field work. Since the heap leach project incurred significant changes in schedule and location compared to the Prefeasibility study, it is expected that part of the field work, as well as community consultations, will have to be redone.

The ESMP resulting from this study will have to be integrated into the general ESMP of the mine in order to obtain an aggregated ESMP, which will include all the environmental studies carried out within the framework of the exploitation of the Project.

20.4 ESSAKANE EXPANSION PERMITTING

In order to increase the annual gold production, a mine expansion FS (the 2011 FS) was initiated in 2011 by IAMGOLD's project and construction department in collaboration with Essakane personnel and a number of consulting experts.

The expansion project consisted of increasing the total ore and waste mining capacity from 32 Mtpa to 56.5 Mtpa to feed the plant. Additionally, the project focussed on increasing the overall processing capacity from 9.0 Mtpa to 10.8 Mtpa by duplicating the grinding and leaching circuits, in order to adjust to increasingly harder rock and maintain throughput. The LOM would also be extended to 2025. Based on conclusive studies, amendments to the mining plan took place from February 2012 to June 2014.

As part of the mine expansion work (from February 2012 to June 2013), a new addendum to the ESIA and the 2008 addendum was prepared in February 2012 (the February 2012 addendum). The February 2012 addendum covers the expansion phase of the main pit and CIL plant infrastructure, a new satellite pit east of the Project, and the Gorouol river diversion. The ESIA and 2008 addendum already covered an important part of the impacts related to the expansion, including the river diversion.

The February 2012 addendum, which is an appendix to the ESIA approved in 2007, was prepared to analyze the environmental and social impacts of the mine expansion project. It includes, in Chapter 6, an updated ESMP incorporating the necessary adjustments to the initial ESMP to include the expansion changes and to consolidate, in one document, all of

IAMGOLD's social and environmental commitments. An environmental impact assessment was conducted for the river diversion.

These documents were validated on December 5 and 6, 2013 by the Comité Technique d'Evaluation Environnementale (COTEVE- Environmental Assessment Technical Committee), a body created by the government and comprised of experts from various professional communities (non-government organizations (NGO), general population, administration, researchers, universities, and institutes). Following the COTEVE meeting, a second public consultation took place from April 17 to May 5, 2013 in the communes of Gorom-Gorom (Oudalan Province) and Falagountou (Seno Province). Essakane S.A.'s gold mining plan amendment was subsequently approved by Order No. 2014-170/MEDD/CAB.

The heap leach project will be another expansion of the mine. The ESIA report will need to be completed prior the commencement of the detailed engineering phase. This will be the first step towards obtaining the environmental and social feasibility notice.

20.5 GOSSEY PROJECT PERMITTING

Communications with communities were initiated in 2018 during the geological investigation campaign. In light of the growing influx of people who came to settle in the Gossey Project area to benefit from a possible resettlement action plan, the mayor of the commune of Gorom-Gorom, issued a decree fixing the deadline for settlement as May 10, 2018. Beyond this date, no new installation will be taken into account in the inventory of affected property and people. The inventory of properties and people began immediately after the announcement of the deadline. The Gossey Project area was surveyed almost entirely, but the inventory was then suspended, and the communities were informed that the Project was postponed.

There has not yet been a study of the environmental and social impacts of the Gossey Project.

20.6 COMMUNITY RESETTLEMENT PLANS

Essakane S.A. implemented two resettlement plans consistent with Burkinabé laws and best practices recommended by international organizations (World Bank). The first plan started in 2008 (13,000 individuals and 2,981 households affected) and the second plan started in 2012

(3,208 individuals and 555 households affected). In both instances, a consultation process was carried out through the implementation of an Advisory Committee that included representatives from the affected villages and hamlets (High Commissioners, mayors and prefects, and technical service representatives) and representatives from three NGOs (The Organization for Community Capacity Building for Development (ORCADE), Burkinabé Movement on Human and Peoples' Rights (MBDHP), and the League for the Defence of Justice and Liberty (LIDEJEL)).

In both instances, memorandums of understanding were signed, and resettlement follow up committees (CSR) comprising key representatives of affected villages and administrative authorities were created. The CSR committees meet every month to follow up on the progress of the two Resettlement Action Plans (RAPs).

For the negotiation of the second resettlement plan, approximately 500 meetings (formal and informal) took place between June 2012 and December 2013, which led to a consensual framework (12 agreements) through what was qualified by all as a participatory and transparent approach. Additionally, in both instances, public consultations were carried out by the Ministry of Environment.

The heap leach project will not require a resettlement plan as all infrastructure is located within the footprint of existing mining operation on previously compensated land.

20.7 SOCIAL AND COMMUNITY ASSESSMENT

As part of the two population resettlement plans (2008 and 2012), Essakane Consultation Committees were implemented to negotiate with the affected populations in order to reach agreements as part of the memorandums of understanding. Resettlement Monitoring Committees were introduced to ensure full enforcement of the agreements.

As part of the community engagement plan, a Communication Committee, information centres and a community visit program were implemented. Accordingly, community information and consultation programs, community visits from mine representatives and management and participation in concerted action frameworks at a regional and provincial level were planned and implemented. Additionally, grievance management mechanisms (grievance reception and processing) to ensure upward and downward communication were defined and implemented.

A Communication Committee of the Essakane Gold Mine (CCME) made up of representatives from the population, the administration, and the mine (over a hundred participants), meet each quarter to review concerns of the communities and the completion status on community investments and engagement.

As part of the community investment plan, socio-educational infrastructures are being built (wells, medical centres, schools, etc.). Programs to fight malaria and HIV/AIDS and increase road safety awareness, were developed for the benefit of neighbouring populations.

Rural development activities (agriculture, animal husbandry, etc.) are primarily undertaken as part of the livelihood restoration program. Since 2014, a community investment program has been financing community projects through communal development plans.

A program of village forests, tree nurseries, and school tree copses has also been developed to promote environmental protection.

A Community Management Program (PMC) encompasses all engagement actions and community development projects of the community relation development department. Key performance indicators of the PMC are reviewed on a quarterly basis.

20.8 WASTE ROCK AND TAILINGS DISPOSAL

20.8.1 WASTE ROCK DISPOSAL

Storage areas for waste rock have been planned and designed to reduce haulage distances between pit ramp exits and storage areas. These areas were selected following consultation with neighbouring populations in order to minimize the impact on these populations (proximity to houses, cemeteries and other archaeological sites, etc.). Finally, the areas were selected with view of minimizing the impact on water resources.

The 2011 FS included the following storage approach:

- For the EMZ pit: two WRDs - the main dump is east of the pit (footprint of approximately 320 ha, height of 104 m, capacity of 380 Mt), the second dump is north of the main pit (footprint of approximately 24 ha, height of 17 m, capacity of 5.2 Mt).
- For the Falagountou pit: one WRD near (north side) the Falagountou pit (footprint of approximately 55 ha, height of 37 m, capacity of 29.6 Mt).

The plan originating from the 2011 FS will vary since future storage will differ in terms of quantity or even potentially in terms of footprint.

Geochemical studies have demonstrated that the waste is non-acid generating, however, is leaching some arsenic. Based on the precautionary principle, a runoff water quality monitoring program is in place. Ditches were installed around the main WRD to collect runoff water and direct it to the ponds.

Progressive rehabilitation of the WRDs commenced in 2011.

20.8.2 TAILINGS DISPOSAL

The mine tailings site was originally designed by Golder Associates Ltd. (Golder). Inner dams and impervious cells are designed by SNC-Lavalin (Golder, 2008b). Tailings are thickened to recover process water and are deposited in the lined cells. The site footprint is 442 ha, delimited by 10 m high and 10 m crest wide perimeter dams, and with internal raise dams and lined cells. The TSF currently stores approximately 107 Mt of tailings and is expected to store up to 85 Mt for the remaining LOM period. To ensure the infrastructure's stability, daily, monthly, and yearly inspections are carried out. Geochemical studies have shown that tailings are non-acid generating, however, the tailings leach arsenic and contain process water with cyanide. Tailings water confinement is ensured by deposition in lined cells and by a perimeter hydraulic barrier with more than 40 pumping wells.

A program for environmental monitoring (ground water quality, fauna, and dam stability inspection) and progressive rehabilitation of the tailings site is in place, at and around, the tailings site.

A tailings site steering committee meets on a regular basis to review the operational monitoring of the tailings site and to provide guidance to improve environmental performance. A governmental technical committee also review the tailing management facility environmental performance on a regular basis.

20.9 ESSAKANE PIT WASTE ROCK STABILITY ASSESSMENT

SRK carried out a geotechnical and stability assessment for the proposed ultimate WRD designs (SRK, 2018). The proposed WRDs include the Halde Nadon and Halde Nadon East. The dumps are designed with 30 m high lifts, separated by berms ranging from 15 m to 20 m. The overall slope angles range from 15° to 26°. Two-dimensional limit equilibrium analyses were carried out using the Rocscience program SLIDE (v7.0) (Rocscience, 2017). Deterministic FOS values were determined for a shallow, surface and a deep-seated, foundation failure mechanisms through the proposed dumps. The results indicate that the proposed WRD slopes are expected to exhibit acceptable design FOS with respect to the analyzed shallow and deep-seated stability, both during construction (i.e., short term) and in the longer term. Shallow-seated instabilities are expected during the construction of the dump, and the materials along the face may slump to an angle of repose for waste rock materials. These shallower instabilities can be managed through regular geotechnical inspections and monitoring.

20.10 SITE MONITORING

A comprehensive monitoring program is in place (at all stages of the LOM) at the site as well as in the neighbouring villages. This program encompasses water quality monitoring (potable water, groundwater, domestic waste water, surface water, and community well water), air quality (dust and greenhouse gas emission), soil, biodiversity (fauna and flora), noise, vibration, weather, and follow up and assessment of the community investment program (health, education, potable water access, agriculture, animal husbandry, etc.).

20.11 WATER MANAGEMENT

The water management plan includes pit dewatering, waste rock runoff capture, diversion systems, and storage ponds. Water on site is classified into three categories:

- Non-contact water: runoff from undisturbed areas, including flow in the Gorouol River
- Contact water: runoff from WRDs and open pits, which may contain high suspended solids concentrations and arsenic
- Process water: water mixed in the process plant and recovered from the TSF thickeners and dewatering pumps

Key objectives for water management are as follows:

- Provide a reliable water supply to the process plant
- Facilitate mining of the deposits by limiting inflows to the open pit and by timely removal of precipitation inflows
- Reduce slope stability risks by routing and storing water away from sensitive pit walls
- Divert clean water away from the mine site, where possible, and capture contact water

A combination of channels and berms strategically capture and divert contact water to control ponds via gravity and avoid any effluent.

20.11.1 WATER SUPPLY

To supply the mining camp with potable water, three wells were drilled outside the site northwest of the proposed HLF. Due to the relatively high hardness of the well water, a reverse osmosis treatment plant was installed. All other domestic water is only treated by chlorination.

For industrial water needs, part of the water is recycled from the TSF. Tailings are thickened to a density of 60% solids before they are discharged into the TSF. Water recovered from thickeners, and excess water in the tailings cells, is reused in the process plant. Runoff water from the TSF is also pumped to the process plant during the wet season.

Water from the Gorouol River is used to supplement recycled water. A dike was raised from the south bank of the Gorouol riverbed by 1.5 m creating an OCR. The water flows by gravity from the river into the OCR from which water is pumped to the BWS ponds (total of 3) adjacent to the TSF.

Additional inflows to the BWS ponds include contact water collected in the Warren Basin, the North Satellite Pit, and the Gorouol Pond (once it is constructed). Inflows to the Essakane Pit (north and south mining zones) are pumped via transfer stations to either Warren Basin or the Gorouol Pond. Pumping systems direct water collected in both facilities to the BWS ponds. Waste rock runoff from the Nadon Dump facilities is collected in the North Satellite Pit and a series of small ponds; Nadon North, South, and East ponds. The Nadon Ponds are left to evaporate or used for dust suppression as needed. The North Satellite Pit is dewatered to the BWS Ponds as a priority to reduce slope stability concerns along the North Essakane pit wall.

20.11.2 WATER MONITORING

A water quality monitoring program (surface water, groundwater, industrial water, potable water, and domestic wastewater) is in place. Additionally, the quantity of water resources is monitored (river flow, water table level, and water meters, etc.). The dykes of the dam and the ponds are inspected regularly (on a daily, monthly, and yearly basis).

20.12 MINE CLOSURE REQUIREMENTS AND COSTS

A conceptual rehabilitation and closure plan (PRF) were developed in 2009 and updated in 2013 and 2018. Asset retirement costs are updated annually and the final closure cost whenever the mining development plan is amended. In addition, Essakane S.A. opened an account with the Bank of Africa (BOA) in which funds are put in escrow as part of the Mining Environment Preservation and Rehabilitation Fund (Order No. 2007-845/PRES/PM/MCE/MEF of December 26, 2007). Notwithstanding this fund, a progressive mining rehabilitation process commenced in 2011, shortly after the start of production.

A closure plan PFS (Closure PFS) will be conducted prior to mine closure. This step will involve a complete review of the plan to validate the base information and to verify the status of progressive rehabilitation. At that time, consultations with stakeholders will be organized in order to identify their concerns and interests. The Closure PFS will include a risk analysis as well as a social impact analysis and will define the closure and monitoring activities. The Closure PFS should take approximately one year to complete.

A closure plan FS (Closure FS) must be conducted after the Closure PFS and must be approved by the relevant authorities. Following stakeholder consultation, the Closure FS will define the terms and conditions of all rehabilitation activities, including planning, costs, objectives, objective criteria, environmental monitoring, reporting, community legacy, and land use and site restoration conditions.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

The capital cost requirement over the LOM includes the following:

- Resource development costs
- Capitalized waste stripping
- Sustaining capital expenditures (for CIL plant and site in general)
- Mine equipment additions and replacements
- Equipment overhaul costs
- Equipment capital spares (CSPARES)
- Tailings dam capital expenditures
- Heap leach project capital expenditures
- CIL plant upgrade project

No capital costs have been estimated for the Gossey deposit.

21.1.1 LIFE OF MINE CAPITAL EXPENDITURES

Total capital spending over the remaining LOM (2020-2031) amounts to $652 million, representing $5.16/t processed (including heap leach) or $191/oz of Au sold.

The total sustaining capital spending planned, excluding capital waste stripping (cash portion) in 2020, is $41.4 million out of a total of $199.4 million over the LOM (2020-2031). In 2020, the Project's capital cost, including capital waste stripping, is $124.4 million $ or $293/oz Au sold.

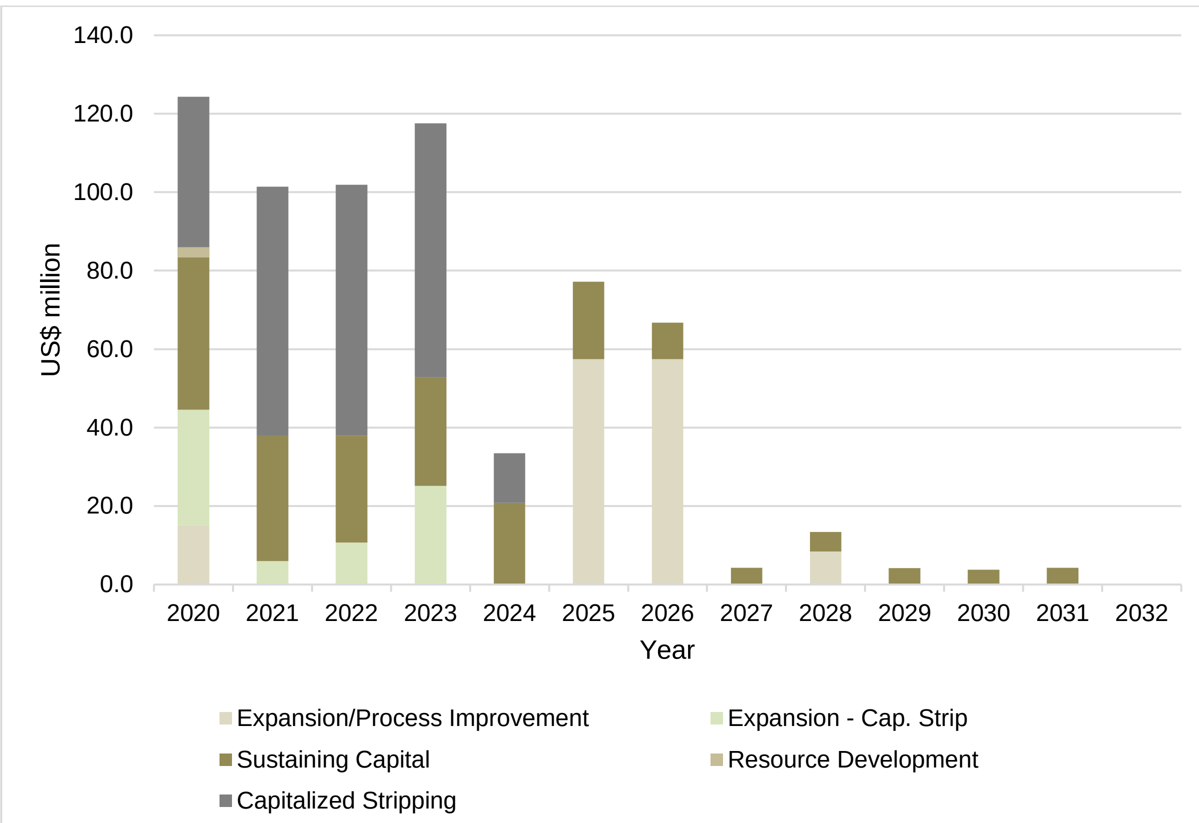
The LOM mine sustaining capital costs are mainly related to the acquisition of mobile equipment, equipment capital spares, and equipment purchases ($23.8 million in 2020, $26.8 million in 2021, and $23.1 million in 2022), with the aim of renewing the aging fleet and supporting production until the LOM ends in 2031.

Total expansion capital is estimated at $138.4 million. A total of $15.1 million is included in 2020 for the CIL plant upgrade and tailings dam capital expenditures. The heap leach project capital expenditures total $57.4 million in 2025, $57.4 million in 2026, and include a sustaining capital cost of $8.5 million in 2028 for the heap leach pad extension.

Total capital waste stripping, inclusive of expansion capital stripping, is the largest capital element estimated at $314.5 million or $92/oz of Au sold over the LOM and represents 48% of the LOM capital cost. Capital waste stripping continues until 2024 after which all mining will be in ore until the end of the heap leach.

Figure 21-1 shows the total LOM capital expenditure distribution.

FIGURE 21-1 LIFE OF MINE CAPITAL EXPENDITURES



No capital costs have been estimated for the Gossey deposit.

A summary of life of mine capital expenditures is presented in Table 21-1.

TABLE 21-1 LIFE OF MINE CAPITAL EXPENDITURES

Items		2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2020-2032
Expansion/Process Improvement	(US$ M)	15.1	-	-	-	-	57.4	57.4	-	8.5	-	-	-	-	**138.4**
Expansion - Cap. Strip	(US$ M)	29.4	6.0	10.7	25.2	-	-	-	-	-	-	-	-	-	**71.3**
Sustaining Capital	(US$ M)	38.8	31.9	27.2	27.6	20.8	19.7	9.3	4.3	5.0	4.2	3.8	4.2	-	**196.9**
Resource Development	(US$ M)	2.6	-	-	-	-	-	-	-	-	-	-	-	-	**2.6**
Capitalized Stripping	(US$ M)	38.4	63.5	63.9	64.8	12.7	-	-	-	-	-	-	-	-	**243.2**
Total Capex	**(US$ M)**	**124.4**	**101.4**	**101.9**	**117.6**	**33.5**	**77.2**	**66.7**	**4.3**	**13.4**	**4.2**	**3.8**	**4.2**	**-**	**652.4**

21.1.2 HEAP LEACH CAPITAL COST ESTIMATE

21.1.2.1 INTRODUCTION

The heap leach capital cost estimate totals $114.9 million of initial capital and $8.5 million of sustaining capital for a total of $123.4 million. The initial capital cost includes all direct and indirect costs for the implementation phase of the project which covers the period starting from the project approval date, finishing with the pre-commissioning activities. Additional costs for transfer to operations, commissioning, start-up, and ramping-up to full production are considered to be part of the operating cost.

This capital cost estimate reflects the scope of work for the heap leach expansion project which includes an additional crushing circuit, material handling equipment, an additional CIC plant, a heap leach pad, and ponds. The processing capacity is calculated based on 8.5 Mtpa of crushed material.

The heap leach project will be developed in one major construction phase for the majority of the scope and one additional sustaining capital expenditure phase:

- Phase I – Capital Cost (Total project scope including first phase of the heap leach pad)
- Phase II – Sustaining Capital (Heap leach pad extension only)

21.1.2.2 ASSUMPTIONS

The following items present the assumptions that have been taken into account during the preparation of the study estimate:

- Based on seven days per week, 11 hours per day for the construction contractor.
- Based on a two week in / one week out rotation schedule for the construction contractor.
- Assumes that labour skills will range from medium to high, i.e., no unskilled nor low skill labour.
- Assumes that the origin of skilled workers is mainly from West Africa.
- Estimate unit pricing for civil works is based on assumptions that borrow pits for crushed material and unselected material is not over 2.0 km radius and excavated material dumps are not exceeding 800 m distance.
- Gas and fuel included in construction equipment is priced at US$0.90/L and is included in direct cost unit rates.
- Basic currency is United States dollar.

Table 21-2 shows currency exchange rates and the portion of each currency considered in the capital cost estimate.

TABLE 21-2 CURRENCY EXCHANGE RATE

Currency Code	Currency Name	Exchange rate	Content (%)
USD	US Dollar	1.00000	88.16
EUR	Euro	1.20000	7.53
CAD	Canadian Dollar	0.80000	3.83
ZAR	South African Rand	0.07692	0.09

21.1.2.3 EXCLUSIONS FROM HEAP LEACH CAPITAL COST

The following is a non-exhaustive list of exclusions:

- Escalation
- Risk
- Risk mitigation plan
- Currency fluctuation
- Hazardous waste
- Financing charges
- Delays caused by community relation, permitting issues, project financing, etc.
- Carry-over work
- Working Capital is excluded from the capital cost
- First Fills
- All costs beyond commissioning and start-up, i.e., ramp up and operations
- Sunk cost

21.1.2.4 TYPE OF ESTIMATE

The heap leach project capital cost estimate is considered to meet the requirement of a Class 3 estimate as defined in American Association of Cost Engineers (AACE) International Recommended Practice No. 47R-11. As such, AACE provides a broad range for accuracy within each estimate class. Typical accuracy ranges for the AACE Class 3 estimates are -10% to -20% on the low side and +10% to +30% on the high side.

The heap leach project capital cost estimate reflects a “Self-Perform” execution mode by IAMGOLD.

Some elements, packages, or areas of the estimate may not achieve the target level of accuracy individually, however, the overall accuracy achieved has been evaluated in careful consideration of the level of definition achieved in major engineering deliverables, execution strategy, and pricing. The sum of all estimate elements falls within the parameters of target accuracy.

21.1.2.5 SUMMARY OF CAPITAL COST ESTIMATE

Table 21-3 provides a summary of the heap leach initial capital cost by major work breakdown structure (WBS) area.

TABLE 21-3 HEAP LEACH CAPITAL COST BY MAJOR AREA

Major WBS Area	Capital Expenditures	TOTAL ($000)
100	Infrastructure	749
300	Mill General (Material Processing including Heap Leach)	79,340
400	Plants & Equipment	282
700	Mining (Covered under Operating Cost)	Operating Cost
900	Project Indirect Costs (Excluding Contingency)	24,057
998	Contingency	10,443
997	Escalation	Excluded
999	Risk	Excluded
Grand Total		**114,871**

Initial and sustaining heap leach capital costs are shown by WBS in Table 21-4.

TABLE 21-4 HEAP LEACH INITIAL AND SUSTAINING CAPITAL COST

WBS	WBS DESCRIPTION	Capital Cost Phase I ($000)	Sustaining Capital Phase II ($000)
	Direct Costs	**80,370**	**8,449**
123	Bulk Water Storage Reservoir (BWS)	749	-
301	Ore Receipt – Crushing / Stockpiling	5,734	-
307	Acid Wash and Elution	708	-
308	Carbon Regeneration & Recovery	3	-
327	Process Control (Integration to current process)	162	-
330	Plant Electrical	194	-
360	Heap Leach Process Area – General	714	-
361	Heap Leach – Crushing	21,707	-
362	Heap Leach – Conveying & Stacking	25,538	-

WBS	WBS DESCRIPTION	Capital Cost Phase I ($000)	Sustaining Capital Phase II ($000)
364	Heap Leach – Pad & Ponds	14,188	8,449
365	Heap Leach – CIC Plant	8,596	-
366	Heap Leach – Reagents	546	-
367	Heap Leach – Air Services	254	-
368	Heap Leach – Water Services	994	-
406	Support Equipment	282	-
	Indirect Costs	**34,499**	**-**
910	Construction Equipment & Tools	1,058	Not Required
920	Construction Engineering	3,755	Excluded
925	Construction Management	5,230	Excluded
930	Construction Freight	4,891	Excluded
935	Construction Room & Board	315	Excluded
940	Construction Transportation	1,207	Excluded
945	Vendor Representatives	1,218	Not Required
950	Initial Fills	100	Not Required
955	Capital Spares	1,593	Not Required
956	1 – 2 Years Operational Spares	In Operating Cost	Not Required
957	Commissioning Spares	209	Not Required
964	Construction Services	1,355	Excluded
985	Corporate Administration	3,124	Excluded
998	Contingency	10,443	Excluded
997	Escalation	Excluded	Excluded
999	Risk	Excluded	Excluded
	Grand Total	**114,871**	**8,449**

21.1.2.6 CORPORATE ADMINISTRATION

Owner's Costs have been evaluated by management and transferred to estimating. The following list of items is considered to be covered under owner’s cost in the estimate:

- Owner’s team
- Security equipment
- Training
- Safety gear and construction supplies
- Compensation fees
- Project insurance including comprehensive general liability and insurance for construction equipment and tools, builder’s all-risk insurance (to be validated)
- QA/QC equipment (Concrete testing, soil compaction, etc.)
- Commissioning cost

21.1.2.7 CONTINGENCY

The contingency evaluation was structured by packages, followed by discipline and/or area, and limited to direct costs and indirect costs excluding contingency, owner's costs, escalation, and risk. The contingency was established at 10%, which is in line with IAMGOLD's guideline for a FS study.

21.1.2.8 ESCALATION

All costs were developed in present pricing valid to the estimate base date. All forward escalation from the estimate base date through to the end of the project is excluded from present FS estimate.

21.1.2.9 RISK

The cost estimate excludes risks. The estimate summary does not identify a separate value for capital risk and the estimate contains no value for cost mitigation.

21.1.2.10 HEAP LEACH SUSTAINING CAPITAL COSTS

Sustaining capital expenditures are estimated to be $8.45 million. The sustaining costs are required to enlarge the heap leach pad and ponds. Any additional piping distribution is considered part of the operating cost. The estimation of direct sustaining capital costs follows the initial capital unit rates which are valid at the estimate base date. No forward escalation has been applied to those unit rates. Indirect costs have been excluded for sustaining capital activities.

21.1.2.11 HEAP LEACH RECLAMATION AND CLOSURE COSTS

The reclamation and closure costs of the heap leach project have been integrated in the Essakane closure cost and included in the LOM.

21.2 OPERATING COSTS

The Project's operating costs are estimated on the basis of the physical quantities of the mine plan, realistic equipment productivity assumptions, overall equipment efficiencies, and updated consumable prices. No operating costs have been estimated for the Gossey deposit.

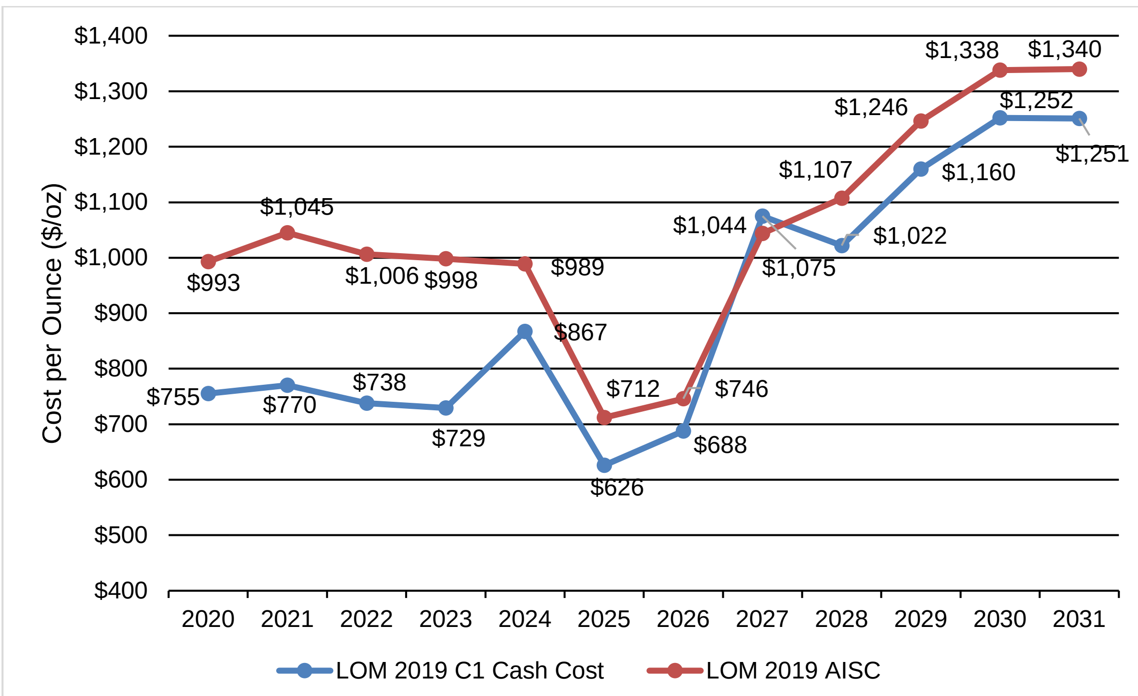
Average operating costs over the LOM and over the Five Year Plan (2020 to 2024) are shown in Table 21-5.

TABLE 21-5 LIFE OF MINE AND FIVE YEAR PLAN OPERATING COSTS

Area	Units	LOM Average	Five Year Plan (2020-2024)
Mining	US$/t mined	3.02	2.88
CIL Processing	US$/t milled	11.71	11.59
Heap Leach Processing	US$/t milled	4.35	-
G&A	US$/t milled	4.27	-

The LOM operating cost metrics per ounce of gold produced are presented in Figure 21-2. The LOM total cash cost per ounce of gold sold is US$778 while the all-in sustaining cost (AISC) per ounce of gold sold is US$949.

FIGURE 21-2 OPERATING COST METRICS

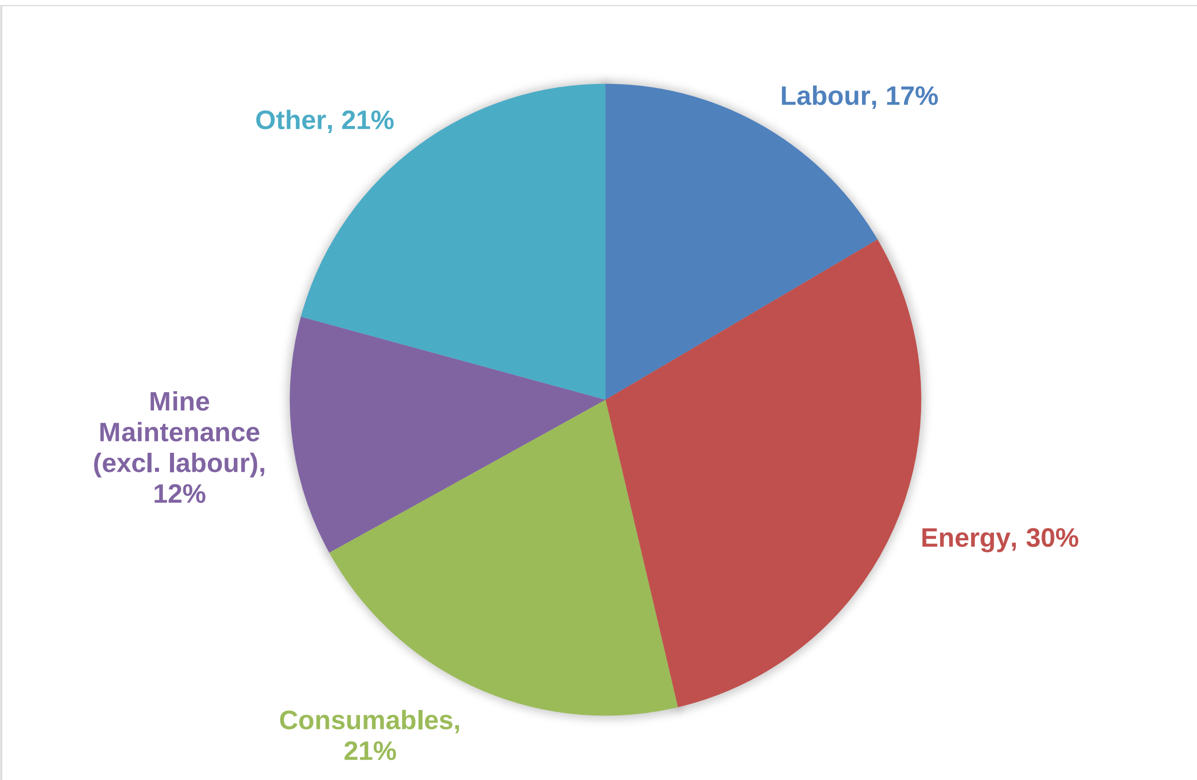


Note that AISC measures do not have any standardized meaning prescribed by International Financial Reporting Standards (IFRS) and differ from measures determined in accordance with IFRS. AISC is intended to provide additional information and should not be considered in isolation or as a substitute for measures of performance prepared in accordance with IFRS. This measure is not necessarily indicative of net earnings or cash flow from operating activities as determined under IFRS.

21.2.1 MINE OPERATING COSTS

Average mine operating costs over the LOM are estimated at $3.02/t mined and average $2.88/t mined over the next five years. The LOM schedule manages to keep the mining cost around the average throughout the years by carefully selecting waste storage locations, thus minimizing haulage distances. An increase in mining cost is observed for the last three years of CIL operation (2024-2026) as all mining activities occur at a greater depth. Fuel represents $0.90/t mined and 0.78 L/t mined over the LOM, which represents 30% of the mine operating cost. The top four mining cost categories represent approximately 80% of the mine operating costs (Figure 21-3).

FIGURE 21-3 MINING COST CATEGORIES

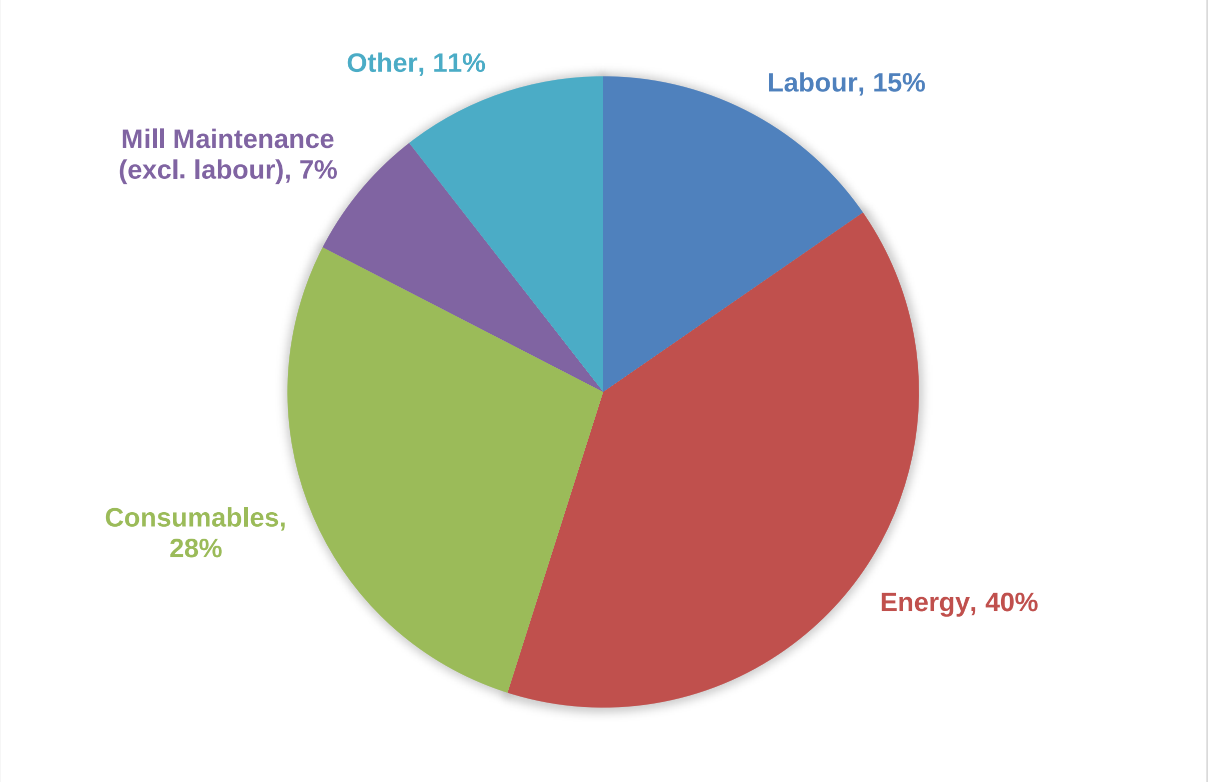


21.2.2 CIL OPERATING COSTS

The average LOM CIL milling cost for 2020-2026 is estimated at $11.71/t milled and an average of $11.59/t milled over the next five years (2020-2024). Heavy fuel oil related to power generation is the primary cost and represents 33% of the CIL operating cost. The energy cost (self-generated power and solar) is $4.63/t milled for the CIL plant. The total energy cost is 40% of the CIL overall costs. It is followed by cyanide (8.4%, $0.99/t milled) and grinding media (8.3%, $0.97/t milled). The higher price of HFO ($0.72/L) has raised projected costs for power generation since the previous LOM, considering that the annual consumption is approximately 64.5 million litres; taking into consideration the energy from the solar plant.

The top four milling cost categories represent approximately 90% of the CIL plant operating cost (Figure 21-4).

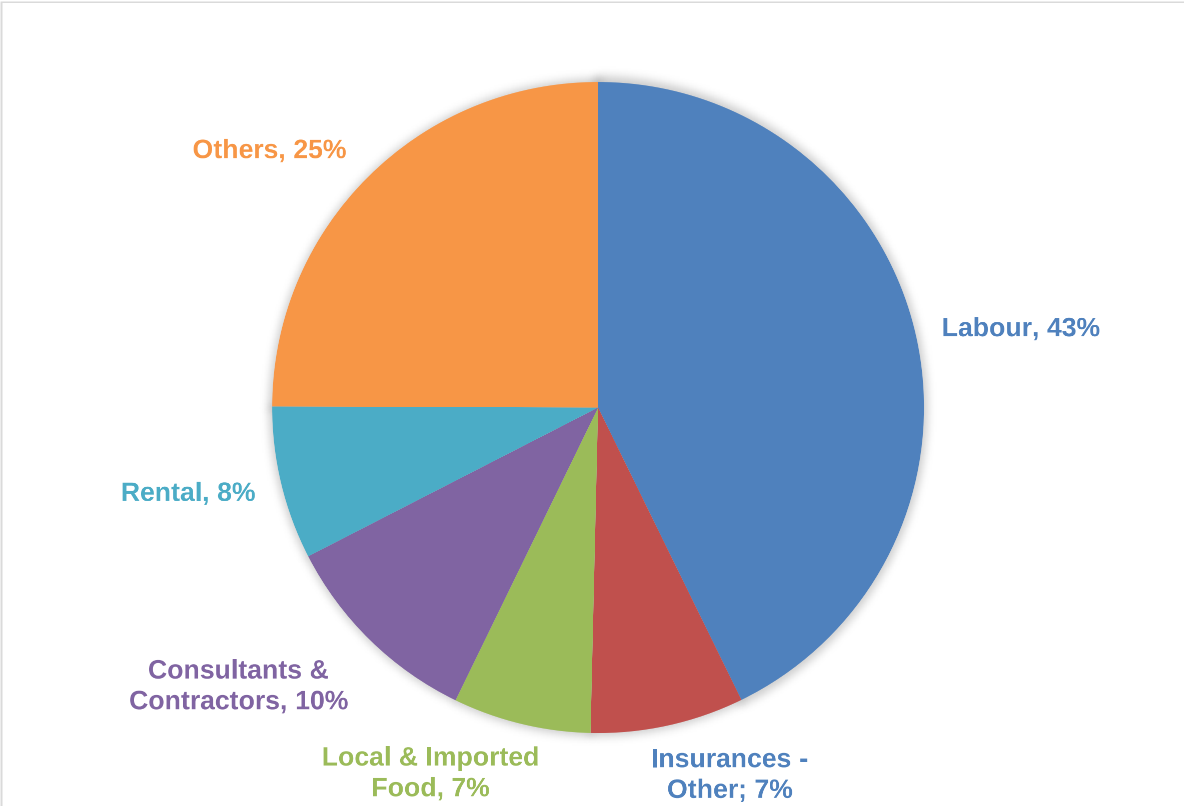
FIGURE 21-4 CIL MILLING COST CATEGORIES



21.2.3 G&A COSTS

The average LOM general and administration (G&A) cost is $4.27/t milled and assumes processing at an average of 11.9 Mtpa at the CIL plant from 2020 to 2026. Note that a proportion of the G&A costs are fixed costs (i.e.: taxes) and are difficult to compress should the milling rate decrease. The top five G&A costs over the LOM represent 75% of the total G&A costs (Figure 21-5) and are detailed as following: labour (43%), insurance (7%), local and imported food (7%), consultants and contractors (10%), and rental (8%).

FIGURE 21-5 G&A COST CATEGORIES



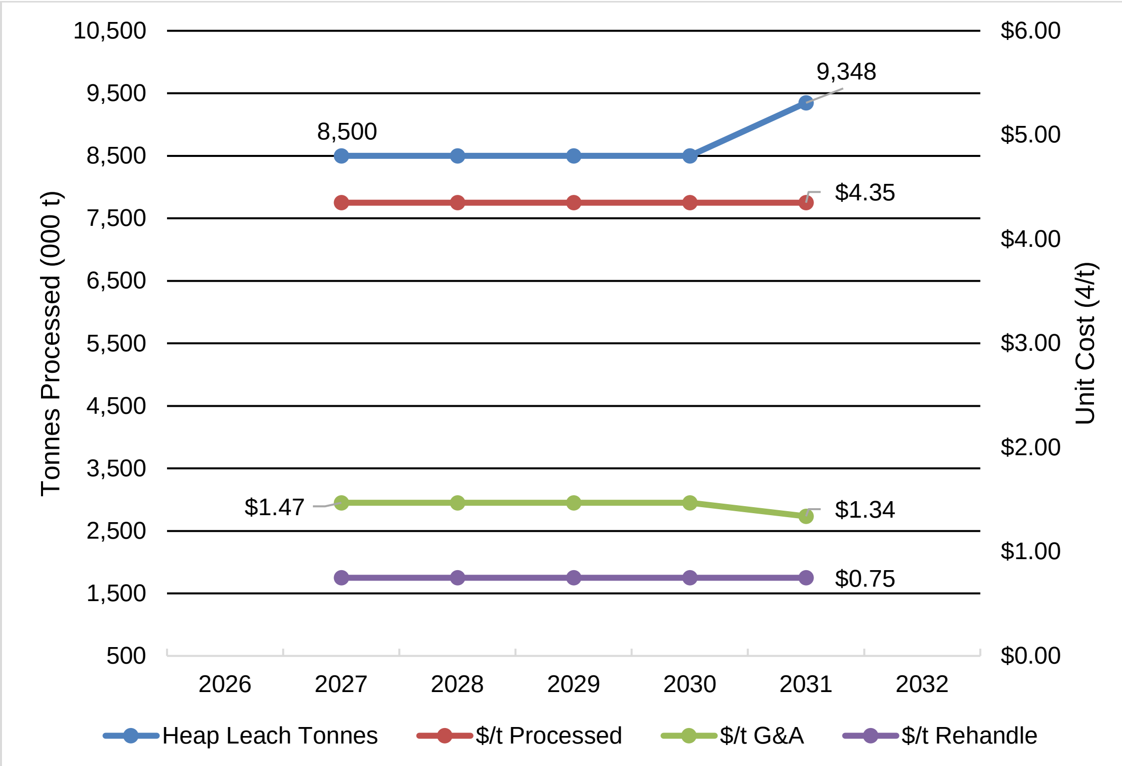
The G&A costs for the heap leach phase are discussed in the heap leach section below.

21.2.4 HEAP LEACH OPERATING COST

The average heap leach processing cost (2028-2031) is \$4.35/t processed (excluding rehandling) and is assumed steady over the LOM. Rehandling unit cost is estimated at \$0.75/t during the heap leach project. The related average G&A unit cost is \$1.44/t.

The average processing rate is 8.5 Mtpa excluding the last year of the heap leach production (2031). Figure 21-6 represents the heap leach processing cost.

FIGURE 21-6 HEAP LEACH COST METRICS



Operating costs for the heap leach expansion Project have been estimated by Lycopodium with input from IAMGOLD.

Sources of general data and assumptions used as the basis for estimating the heap leach operating costs are listed below:

- The process design criteria from this Study.
- The average production rate of 8.5 Mtpa.
- Manpower requirements were developed by Lycopodium with input from IAMGOLD.
- The unit cost of electrical energy is $0.172/kWh.
- The unit cost of diesel fuel is $1.15/L.
- Refining costs, royalties, and net proceeds tax are excluded from the operating costs but are applied to the financial model.

The heap leach operating costs are premised on a design throughput of 8.5 Mtpa of dry ore. The crushing plant and heap leach pad loading will operate a nominal 6,395 hours per year with a 73% utilization and the adsorption process plant will operate 24 hours per day, 365 days per year with 91.3% utilization, or a nominal 8,000 hours per year.

The heap leach operating cost estimate has been compiled from a variety of sources and is based on a LOM average ore grade of 0.37 g/t Au.

The operating cost estimates for the heap leach operation include the following major categories:

- Operating consumables
- Plant maintenance costs
- Power
- Labour (Operations and maintenance)
- Laboratory costs

Heap leach operating costs are summarized in Table 21-6.

TABLE 21-6 HEAP LEACH OPERATING COST SUMMARY

Cost Centre	Base Case ($/t processed)
Operating Consumables	1.91
Plant Maintenance	0.66
Power	1.55
Labour (O & M)	0.21
Laboratory	0.01
Total Cost	**4.35**

The heap leach operating costs have been developed in accordance with industry practice for heap leach operations. Inputs into the costs also include the current operating cost of existing operation areas that will remain as part of the future operation when the heap leach project commences.

Loaded carbon from the CIC circuit will be transported to the existing elution, carbon regeneration, and gold room facility where gold will be stripped from the carbon, electrowon, and smelted into doré bars. Eluted carbon will be regenerated at this facility before it is returned to the CIC circuit. An average of 2.5 elutions (strips) and one smelt will be performed weekly. The cost for these services were calculated from basic principles.

The operating costs for the heap leach operation were derived from a variety of sources including:

- Metallurgical test work
- Consumable pricing from suppliers and from existing IAMGOLD Essakane operation
- Advice from IAMGOLD
- Lycopodium data and estimating methodologies
- First principle calculations

22 ECONOMIC ANALYSIS

22.1 FORWARD-LOOKING INFORMATION

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this Technical Report include, but are not limited to, statements with respect to future gold prices, the estimation of Mineral Resources, the estimated mine production and gold recovered, the estimated capital and operating costs, and the estimated cash flows generated from the planned mine production. Actual results may be affected by:

- potential delays in the issuance of permits and any conditions imposed with the permits that are granted.
- differences in estimated initial capital costs and development time from what has been assumed in the FS.
- unexpected variations in quantity of mineralized material, grade, or recovery rates, or presence of deleterious elements that would affect the process plant or waste disposal.
- unexpected geotechnical and hydrogeological conditions from what was assumed in the mine designs, including water management during construction, mine operations, and post mine closure.
- differences in the timing and amount of estimated future gold production, costs of future gold production, sustaining capital requirements, future operating costs, assumed currency exchange rate, requirements for additional capital, unexpected failure of plant, equipment or, processes not operating as anticipated.
- changes in government regulation of mining operations, environment, and taxes, unexpected social risks, higher closure costs and unanticipated closure requirements, mineral title disputes, or delays to obtaining surface access to the property.

22.2 VALUATION METHODOLOGY

The Project has been evaluated using discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including sustaining and expansion capital costs, operating costs, taxes, and royalties. These are subtracted from revenues to arrive at the annual cash flow projections. Cash flows are taken to occur at the end of each period.

The appropriate discount rate can depend on many factors, including the type of commodity, the cost of capital for the firm, and the level of project risks (e.g. market risk, technical risk, and political risk) in comparison to the expected return from the equity and money markets. The base case discount rate for this Technical Report is 6%, which has been used to evaluate the Essakane mine projects. The discounted present values of the cash flows are summed to arrive at the Project's net present value (NPV).

22.3 BASIS OF ANALYSIS

The financial analysis was based on:

- royalty rates as per Burkina Faso mining legislation.
- the subset of the Mineral Resources disclosed in Section 14, defined as Mineral Reserves as included in the mine plan presented in Section 15.
- the mine plan described in Section 16.
- mill feed treated in the process plant described in Section 17.
- support from the projected infrastructure requirements outlined in Section 18.
- doré marketing assumptions described in Section 22.
- permitting, social and environmental regime discussions in Section 20.
- capital and operating cost estimates detailed in Section 21.

22.3.1 METAL PRICING

For the purposes of the financial analysis, the assumed gold price for the LOM is $1,350/oz. The gold price was the consensus forecast of the following sources: bank analysts' long-term forecasts; historical metal price averages; and prices used in recent publicly-disclosed comparable studies.

22.3.2 EXCHANGE RATE

For the purpose of the capital cost, operating cost, and financial analysis, the assumed USD/EURO exchange rate for the LOM is 1.20. The exchange rate was the consensus forecast of the following sources: bank analysts' long-term forecasts; historical exchange rate averages; and prices used in recent publicly-disclosed comparable studies.

22.3.3 TRANSPORT, INSURANCE, AND REFINING

The doré will be picked up from site and shipped to a refiner in Switzerland who refines the doré into bullion. An indicative quote for transportation, insurance, and refining costs stands at approximately $2.50/oz Au, which has been used in the financial model for the Project.

22.3.4 WORKING CAPITAL

Working capital modelling of cash outflow and inflows are included in the cash flow model. The calculations are based on the assumptions that accounts payable will be paid on average within 45 days. Consumables inventory are assumed to decrease by 60% by the end of the CIL phase in 2026. The remaining inventory will steadily decrease by $5 million per year until the end of the mine life. Value added tax (VAT) receivable is expected to remain stable until 2026 where it starts to decrease. Throughout the LOM, finished goods inventory is expected to remain stable as yearly production is assumed to be entirely sold within the same year, except for 2027 where the remaining inventory balance from the CIL phase is planned to be sold.

22.3.5 ROYALTIES

The royalties' base and rates are as follows:

TABLE 22-1 ROYALTIES RANGES

Gold Price Level	Royalties
Less or equal to US$1,000/oz Au	3%
Greater than US$1,000/oz Au but less than US$1,300/oz Au	4%
Greater than or equal to US$1,300/oz Au	5%

Given that the gold price assumption taken is US$1,350/oz for this LOM, royalties have been calculated at a rate of 5%. They amount to approximately $231 million over the remaining life of the Project.

22.3.6 TAX

Taxation considerations included in the financial model comprise the following items:

- Corporate income taxes in Burkina Faso at a rate of 17.5%
- “Patente” tax, specific to Burkina Faso and based on fixed assets value and revenue
- Land tax

The tax calculations are underpinned by the assumption that the after-tax analysis does not attempt to reflect any future changes in taxes legislation in Burkina Faso.

22.3.7 FINANCING

The financial model includes the financing assumption that all expenditures, including capital, are financed by operating cash flows, except for a $30 million amount for capital lease of mobile equipment to enhance the existing fleet. An interest rate of 8.85% per annum is assumed for this finance lease.

22.3.8 INFLATION

There is no adjustment for inflation in the financial model; all cash flows are free of inflation.

22.4 ECONOMIC ANALYSIS RESULTS

Table 22-2 summarizes the financial results with the NPV 6% highlighted. The after-tax NPV at 6% for the remaining LOM (2019-2031) is $874 million. Only Essakane Probable Mineral Reserves are mined in the LOM. No mining of the Gossey deposit is included in the LOM.

The LOM total cash cost per ounce of gold sold is US$778. The AISC per ounce of gold sold is US$949. Note that AISC, as reported, is based solely on costs associated with this mine site and does not take into account head office or any other corporate costs not directly associated with the Project.

TABLE 22-2 SUMMARY OF FINANCIAL MODEL

Key Performance Indicators		T39 2019	2020	2021	2022	2023	2024	2025 (LOM)	2026	2027	2028	2029	2030	2031	2032	R4 2020-2032
Mining Statistics																
Ore Mined	(000 t)	18,423	16,328	11,849	14,755	12,042	14,262	19,476	11,543	-	-	-	-	-	-	**100,256**
Waste Mined	(000 t)	25,962	25,167	20,774	15,741	13,490	33,597	21,442	4,814	-	-	-	-	-	-	**135,025**
Capital Waste Mined	(000 t)	11,726	24,024	24,437	26,580	30,693	4,180	-	-	-	-	-	-	-	-	**109,915**
Total Tonnes Mined	**(000 t)**	**56,110**	65,520	57,060	57,076	56,225	52,040	40,919	16,356	-	-	-	-	-	-	**345,196**
Milling Statistics - CIL																
Throughput	(000 t)	12,841	12,599	11,829	12,500	12,143	11,870	10,802	11,396	-	-	-	-	-	-	**83,140**
Gold Production - CIL	**oz**	**420,215**	**424,751**	**419,666**	**410,441**	**411,778**	**400,053**	**530,769**	**435,588**	**-**	**-**	**-**	**-**	**-**	**-**	**3,033,045**
Unit Costs																
Mining Cost / t Mined	$/t	3.10	2.82	2.84	2.81	2.93	3.03	3.40	4.47	-	-	-	-	-	-	**3.02**
Mining Cost / t Milled (net of ore stockpiled & deferred stripping)	$/t	9.09	7.51	8.50	6.43	6.55	11.02	10.63	7.67	-	-	-	-	-	-	**9.61**
Milling Cost / t Milled	$/t	7.17	7.04	7.26	6.89	6.99	7.04	7.26	7.12	-	-	-	-	-	-	**7.08**
Power Cost / t Milled (CIL only)	$/t	4.65	4.37	4.67	4.52	4.62	4.57	4.96	4.74	-	-	-	-	-	-	**4.63**
G&A Cost / t Milled	$/t	4.14	4.19	4.44	4.12	4.21	4.28	4.51	4.13	-	-	-	-	-	-	**4.27**
Total Cost / t Milled	**$/t**	**25.05**	**23.12**	**24.87**	**21.96**	**22.37**	**26.91**	**27.37**	**23.66**	**-**	**-**	**-**	**-**	**-**	**-**	**25.58**
Heap Leach Operation																
Tonnage Processed	(000 t)	-	-	-	-	-	-	-	-	**8,500**	**8,498**	**8,500**	**8,500**	**9,348**	-	**43,346**
Gold Production (HL)	oz	-	-	-	-	-	-	-	-	**80,734**	**84,944**	**70,243**	**62,899**	**69,426**	-	**368,246**
Heap Leach Processing Cost	$/t									4.35	4.35	4.35	4.35	4.35		**4.35**
Total Combined Production & Sales (CIL+HL)																
Total Gold Production	oz	**420,215**	**424,751**	**419,666**	**410,441**	**411,778**	**400,053**	**530,769**	**435,588**	**80,734**	**84,944**	**70,243**	**62,899**	**69,426**	**-**	**3,401,291**
Total Gold Sales	oz	**430,003**	**424,751**	**419,666**	**410,441**	**411,778**	**400,053**	**530,769**	**435,588**	**102,083**	**84,944**	**70,243**	**62,899**	**69,426**	**-**	**3,422,641**
Gold Price	$/oz	1,282	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	**1,350**
Silver Price	$/oz	16.64	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	**18.00**
Total Cash Costs ($ / oz)	**$/oz**	**820**	**755**	**770**	**738**	**729**	**867**	**626**	**688**	**1,075**	**1,022**	**1,160**	**1,252**	**1,251**	**-**	**778**
TOTAL CAPEX	**US$ 000**	**110,611**	**124,357**	**101,364**	**101,863**	**117,547**	**33,485**	**77,179**	**66,739**	**4,259**	**13,406**	**4,215**	**3,757**	**4,230**	**-**	**652,400**
										-	-	-	-			
All-In Sustaining Costs ($ / oz sold)	**$/oz**	**982**	**993**	**1,045**	**1,006**	**998**	**989**	**712**	**746**	**1,044**	**1,107**	**1,246**	**1,338**	**1,340**	**-**	**949**
Free Cash Flow																
Operating Cash Flows	US$ 000	172,606	230,960	232,747	226,506	233,128	160,883	344,124	200,890	55,376	73,665	27,806	4,588	41,190	10,199	**1,842,054**
Sustaining Capital Expenditures	US$ 000	(49,258)	(79,823)	(95,394)	(91,120)	(92,401)	(33,485)	(19,744)	(9,304)	(4,259)	(4,957)	(4,215)	(3,757)	(4,230)	-	**(442,689)**
Non-sustaining Capital Expenditures	US$ 000	(61,353)	(44,534)	(5,970)	(10,743)	(25,146)	-	(57,435)	(57,435)	-	(8,449)	-	-	-		**(209,711)**
Lease Principal Payments	US$ 000	(952)	(6,272)	(6,566)	(6,372)	(6,833)	(3,540)	(493)	0	0	-	-	-	-		**(30,075)**
Lease Interest Payments	US$ 000	(1,132)	(1,749)	(1,464)	(1,020)	(560)	(130)	(12)	(0)	(0)	-	-				**(4,935)**
Restricted Cash	US$ 000	313	(3,454)	(3,454)	(3,454)	(3,454)	(3,454)	(3,454)	-	4,000	4,000	4,000	4,000	4,000	24,354	**23,630**
Dividends Paid to Government	US$ 000	-	-	-	-	-	-	-	-	-	-	-				**-**
Free Cash Flow	US$ 000	**60,224**	**95,128**	**119,898**	**113,797**	**104,734**	**120,275**	**262,985**	**134,151**	**55,117**	**64,259**	**27,592**	**4,831**	**40,960**	**34,553**	**1,178,274**

NPV		2019-2032
NPV (US$ 000)	6%	**$874,331**

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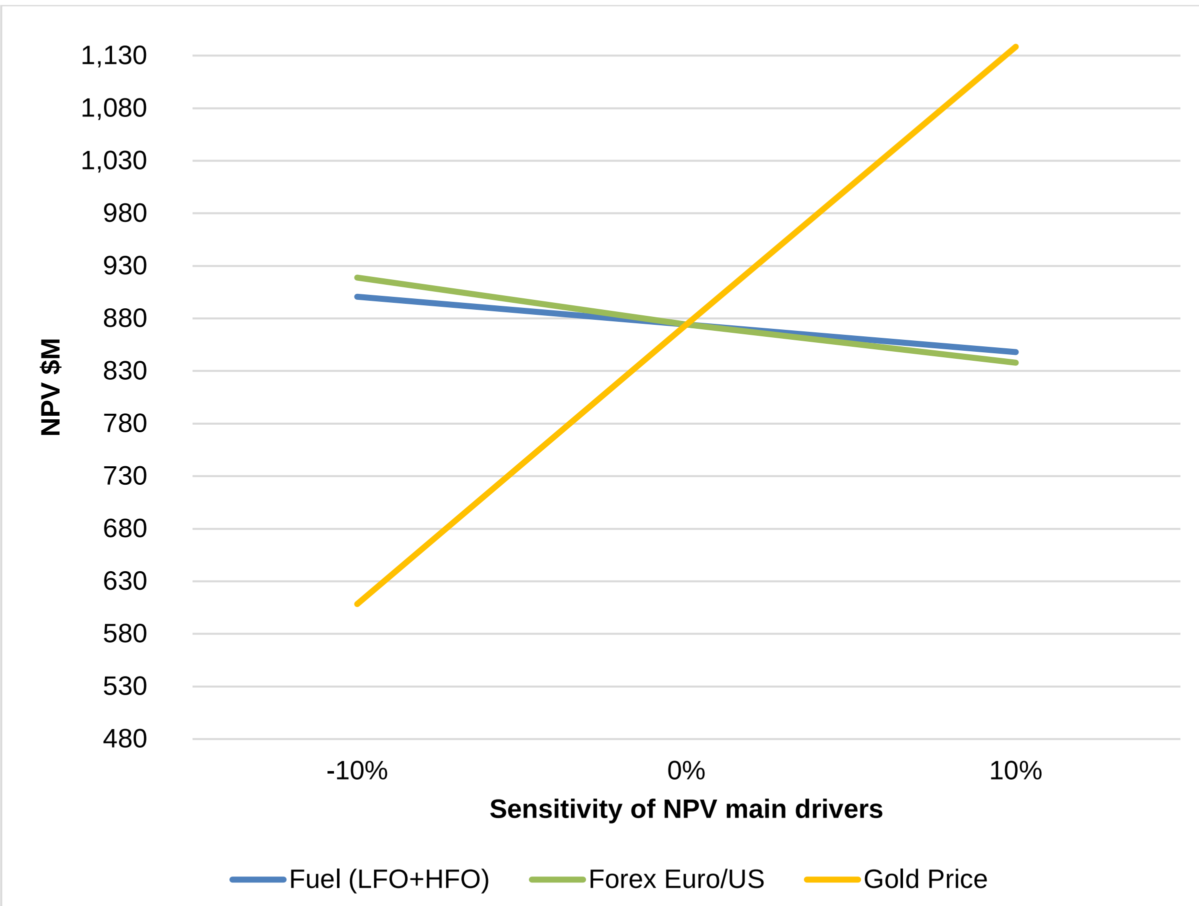
22.5 SENSITIVITY ANALYSIS

Three important parameters (fuel price, foreign exchange (Forex), and gold price) greatly impact the NPV. A simulation was performed to understand the impact on the site NPV by fluctuating these parameters by 10%. As expected, the NPV reacts the most with the variation of the price of gold (Figure 22-1). The impact of the gold price increasing by 10% is $264 million. It is followed by Forex by more than $45 million when the Euro becomes stronger compared to the US Dollar. Finally, the fuel price (HFO and LFO) impacts the NPV by $26 million when the unit price varies by 10%.

FIGURE 22-1 NPV SENSITIVITY ANALYSIS



22.6 CLOSURE AND RECLAMATION COSTS

Payments of $76.5 million are estimated for the closure and reclamation cost of Essakane. This amount was updated in March 2019 to reflect the updated projected disturbance until the end of the mine.

23 ADJACENT PROPERTIES

Adjacent properties to the Essakane deposit are held by exploration companies including SHANIEL Sarl, EXMA, and Diallo Aboubakar who own the Saouga-OA, Yevelde, and Bamguel licences located west and south of the Essakane Exploration Permits (Figure 23-1). There is no relevant information from these adjacent properties available for disclosure in this Technical Report.

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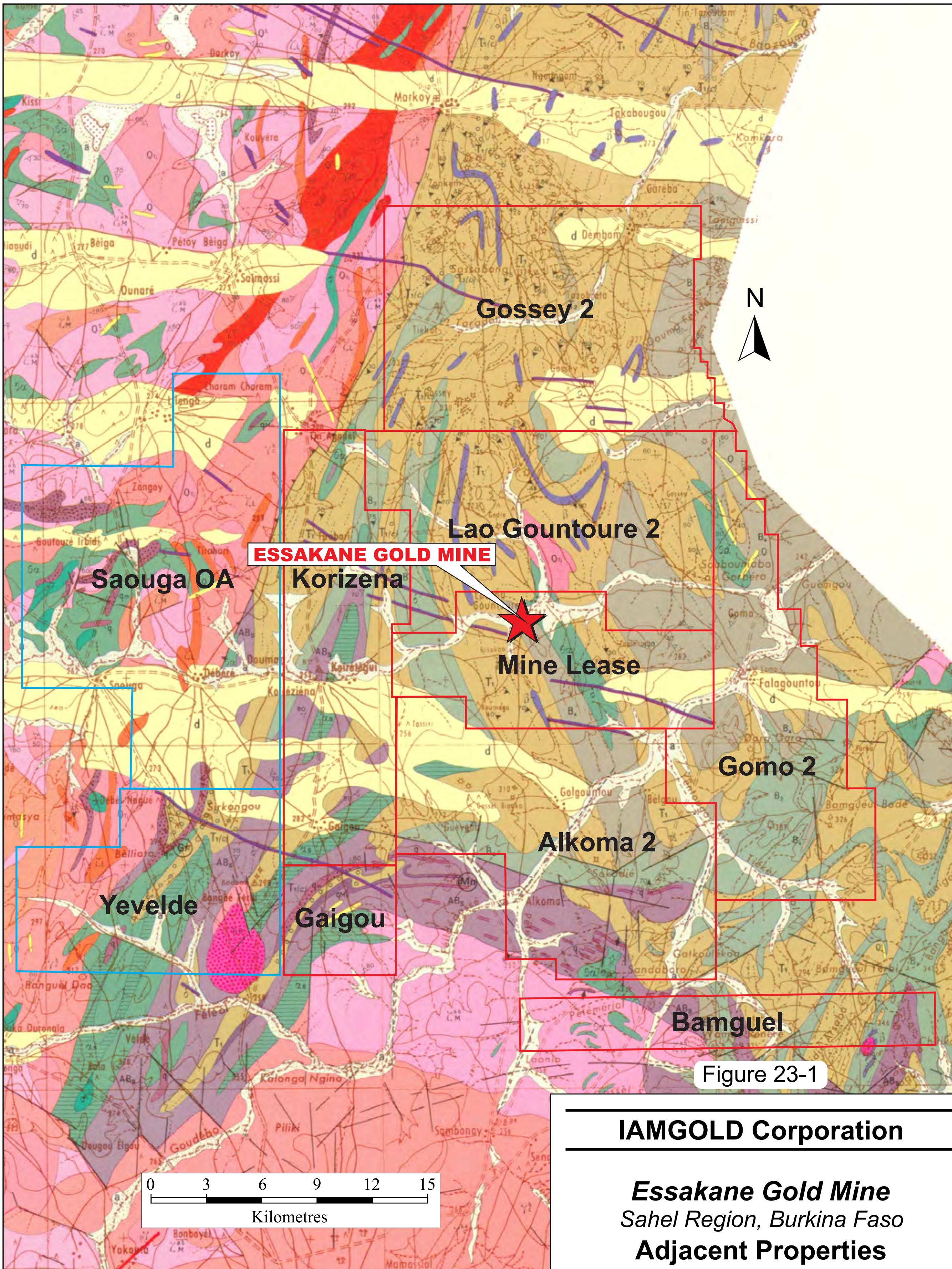

Figure 23-1

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Essakane Gold Mine

Sahel Region, Burkina Faso

Adjacent Properties

January 2020 ***Source: Ministere du Plan et des Travaux Publics, 1970.***

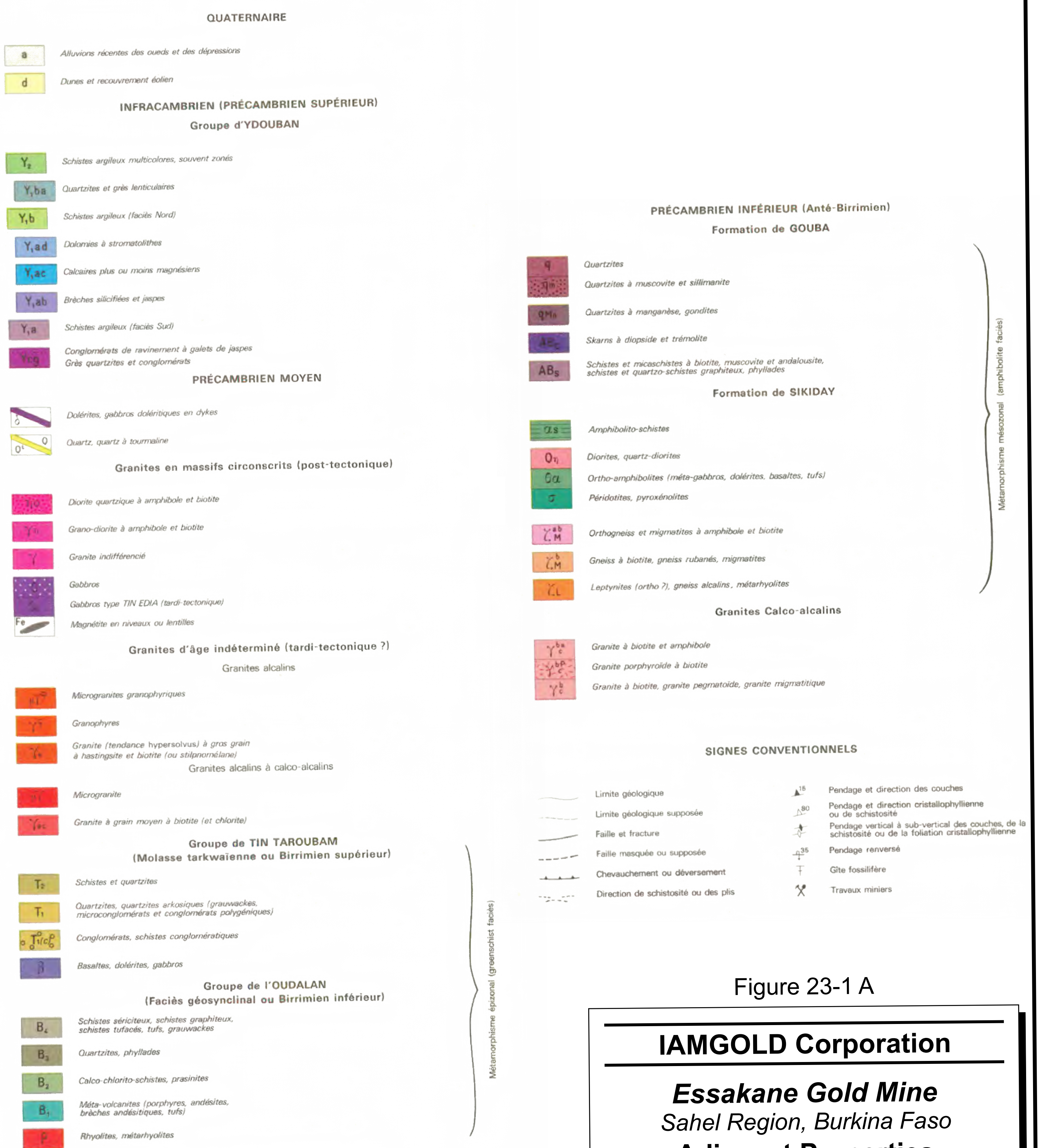

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Figure 23-1 A

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Essakane Gold Mine

Sahel Region, Burkina Faso

Adjacent Properties Geology Legend

January 2020 ***Source: Ministere du Plan et des Travaux Publics, 1970.***

24 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

24.1 PROJECT EXECUTION PLAN

The CIL plant upgrade project is directly managed by the Essakane project team. The project was approved in July 2019 and is expected to be completed in Q3 2020.

For the heap leach project, the project execution will be directly managed by the IAMGOLD project management team. The engineering will be contracted out to qualified firms. The construction work will be mainly contracted out to local and regional contractors under the supervision of the project management team. Project control functions such as scheduling, cost control, procurement, project logistics, and site supervision will be executed directly by the IAMGOLD project management team.

An Owners' Steering Committee will be formed to oversee the Project. The Project will be initiated in 2025 and completed in 2026. Production is expected to start in 2027.

24.2 RISK MANAGEMENT

A risk analysis was performed for the CIL plant upgrade project during the FS. The main risks are presented in Table 24-1.

TABLE 24-1 MAIN PROJECT RISKS

Risk	Risk Response/Mitigation
Customs clearance and delays	1) Make sure the paperwork requirements and procedures are known early at the start of implementation 2) Ensure that all vendors and contractors have clear shipping instructions, packing requirements, labeling etc. 3) Confirm prior to release from factory that instructions have been followed
Primary Screen mechanical & structural modifications take longer than the specified shutdown period.	1) Consider an alternative flowsheet option to be put temporarily in place to keep production going (bypass) 2) Feed from stockpile to extend the 6-7 day planned shutdown duration) 3) Develop an execution plan and confirm activities duration with contractors
Health and safety incidents during intensive construction periods	1) Ensure a good ratio of stakeholders and supervisors for all contractors 2) Establish procedures, training workshop, and adequate induction session 3) Define and communicate prevention plans and emergency procedures

The opportunities which may improve the Project are shown in Table 24-2.

TABLE 24-2 MAIN PROJECT OPPORTUNITIES

Opportunity	Opportunity Response
Low capital cost option: maximize the ounces production while reducing the investment.	1) Gravimetric circuit improvement 2) Maximize plant throughput
Improve HL recovery	1) Evaluate the use of less cement in the agglomeration circuit 2) Test transition material for heap leach 3) Use higher Cn concentration at the end of heap life to defeat preg-robbing.

25 INTERPRETATION AND CONCLUSIONS

25.1 ESSAKANE GEOLOGY AND MINERAL RESOURCES

IAMGOLD has the following conclusions and observations on the EMZ, Falagountou West, and Wafaka deposits:

- Mineral Resources have been prepared in accordance with the CIM (2014) definitions.
- Work completed to date by the geological staff is appropriate.
- The geological model employed by Essakane S.A. geologists is reasonably well understood and is well supported by field observations in both outcrop and drill intersections.
- The resource model has been prepared using appropriate methodology and assumptions. These parameters include:
    - Treatment of high assays
    - Compositing length
    - Search parameters
    - Bulk density
    - Cut-off grade
    - Classification
- The block model has been validated using a reasonable level of rigor consistent with common industry practice.
- The current drill spacing in the EMZ deposit is judged adequate to develop a reasonable model of the mineralization distribution and to quantify its volume and quality with a good level of confidence in all three areas of the Project.
- Based on visual verification, the models (Rock Type, Density, and Au Grade) were found to be globally representative of the known geological and structural controls of mineralization at the EMZ deposit.
- Statistical analysis demonstrates that the block model provides a reasonable estimate of the Mineral Resources for the EMZ deposit.
- Validation of the block model using different interpolation methods indicated that tonnages, grades, and gold contents are similar.
- Swath plots for Indicated and Inferred Mineral Resources by vertical sections for the EMZ and North Satellite areas indicate that peaks and lows in gold content generally match peaks and lows in composite grades; no bias was found in the resource estimate in this regard.
- The information stored in the Falagountou West and Wafaka database was reviewed and found it to be in good standing.

- Drill hole spacing on the Falagountou West and Wafaka deposits are judged adequate to develop a reasonable model of the mineralization distribution and to quantify its volume and quality with an acceptable level of confidence.
- The OK method was judged to be the most suitable to replicate composite grades throughout the Falagountou West and Wafaka deposits
- Sampling and assaying have been carried out following standard industry QA/QC practices. These practices include, but are not limited to, sampling, assaying, chain of custody of the samples, sample storage, use of third-party laboratories, standards, blanks, and duplicates.

25.2 GOSSEY DEPOSIT GEOLOGY AND MINERAL RESOURCES

GMSI has the following conclusions and observations on the Gossey deposit:

- Mineral Resources were classified into Indicated and Inferred categories according to the CIM (2014) definitions.
- The geological interpretation for the Gossey deposit is based primarily on DD drilling data and geological interpretations by representatives of Essakane S.A. The geology of the deposit is relatively well understood.
- The mineralization is found mainly in the arenitic lithologies, and occasionally found within the diorite intrusive. The strongest mineralization is found along the contacts between these two lithologies. The mineralization controls of the deposit are well understood.
- The protocols followed to collect sample data are considered sufficient for NI 43 101 purposes. Stringent protocols are in place to ensure that sampling and assaying of drill samples are undertaken to a high standard, and that QA/QC data is checked frequently to identify any errors that may arise. Sampling has been undertaken based on geological logging and is adequate for the mineralization style and size of the deposit.
- QA/QC samples submitted as part of the 2017 drilling campaign returned values within expectations. QA/QC data from previous drilling campaigns were also reviewed and were found to be in good standing. All analyses were undertaken by the on-site laboratory at the Essakane mine site, with frequent umpire checks submitted to an external laboratory (SGS Ouagadougou). GMSI considers all matters relating to QA/QC in line with NI 43-101 requirements.
- Réjean Sirois, P. Eng. and James Purchase, P. Geo, from GMSI, observed the RC drilling campaign during a site visit on March 27-31, 2018 and a laboratory tour was also undertaken. GMSI found the drilling methods and sample recoveries acceptable.
- The geological model was undertaken in Leapfrog GEO where 3D wireframe solids of lithologies and weathering profiles were produced and are judged representative of the style of deposit observed at Gossey.
- Mineral Resources were estimated within the lithology domains using GEOVIA GEMS from 2.5 m long composites using four interpolation passes of ID^3. Each search ellipse was incrementally larger than the previous, and dimensions were based on drill hole spacing.

- The block model was validated against the drill hole composites through global and local validation methods, including visual comparisons, descriptive statistics, swath plots, and Q:Q plots. No production data was available to validate the accuracy of the model to true known grade. Block grades were found to reproduce composite grades sufficiently in the block model.
- The Mineral Resources are reported within a Lerchs-Grossman open pit shell (based on Indicated and Inferred Mineral Resources) and are effective as of May 25, 2018. A 0.33 g/t Au cut-off for saprolite and laterite material, 0.42 g/t Au for transitional material, and 0.47 g/t Au for fresh rock were used to report Mineral Resources. A gold price of US$1,500/oz was used for the pit optimization. The pit constrained Mineral Resource for the Gossey deposit is as follows:
    - Indicated Mineral Resource is estimated at 10.4 Mt at an average grade of 0.87 g/t Au, totalling 291,000 ounces of gold.
    - Inferred Mineral Resource is estimated at 2.9 Mt at an average grade of 0.91 g/t Au, totalling 85,000 ounces of gold.
- No mining buffer zone was applied around the nearby Gossey Village. Should a 350 m buffer zone be maintained around the Gossey Village a portion of the Gossey Mineral Resource will be sterilized. Approximately half of the contained gold ounces would be lost due to the buffer zone restraint, however, the resulting strip ratio is slightly less.

25.3 MINING AND MINERAL RESERVES

IAMGOLD has the following conclusions and observations:

- The mine design and Mineral Reserve estimate have been completed to a level appropriate for an FS.
- The economic assumptions and methodology used for estimation of the Mineral Reserves are appropriate.
- The Mineral Reserve estimate is consistent with the CIM (2014) definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources, and do not include any Inferred Mineral Resources.

25.4 METALLURGICAL TESTING AND MINERAL PROCESSING

KCA has the following conclusions and observations:

- The results of the metallurgical test programs indicate that the ore types tested are amenable to standard heap leaching methods. The available test results are sufficient to support this FS. Gold recovery is estimated to be 67% and reagent requirements are low.
- Heap leach metallurgical testing has been carried out by KCA. KCA has identified the following risks that may affect the economics of the heap leach project:
    - No metallurgical test work has been completed on the material in the stockpiles and the effects of weathering is unknown.

  - Due to the low grade of the heap leach ore and the presence of coarse gold, individual tests give ranges of assays and there could be some error in recovery estimates.
  - Some of the ore at Essakane is preg-robbing which can have long term effects if placed in the lower lifts of the heap.
- KCA has identified the following opportunities that may affect the economics of the heap leach project:
  - It may be possible to vary the cement addition rate by lift, dependant on the ultimate load of additional lifts stacked on top.

25.5 ENVIRONMENT

IAMGOLD has the following conclusions and observations:

- No outstanding technical issues were identified for environment and permitting.
- Communications with communities were initiated in 2018 during the geological investigation campaign. In light of the growing influx of people who came to settle in the Gossey Project area to benefit from a possible resettlement action plan, the mayor of the commune of Gorom-Gorom, issued a decree fixing the deadline for settlement as May 10, 2018. Beyond this date, no new installation will be taken into account in the inventory of affected property and people.
- There has not yet been a study of the environmental and social impacts of the Gossey Project.

26 RECOMMENDATIONS

The FS recommends initiating the detailed engineering for the CIL plant upgrade to 11.7 Mtpa. The project schedule is estimated to be 12 months and is expected to be commissioned in Q3 2020. For the HL, study assumptions will be validated on a yearly basis during the LOM process. Some additional test work will also be initiated to evaluate low grade transition material within the CIL Mineral Reserve that may be amenable to HL with the addition of agglomeration.

As construction of the HL facility is not required until 2025, the business retains the option of re-evaluating the economics of the construction project at that point in time. Given that the CIL process generates higher recovery and doesn't require additional capital investment, it is recognized that there may be a case where the existing stockpiled ore planned for the HL process generates superior economics by processing through the existing CIL circuit, especially in scenarios with higher gold prices than have been used for the current study.

26.1 ESSAKANE GEOLOGY AND MINERAL RESOURCES

IAMGOLD has the following additional recommendations for the EMZ, Falagountou West, and Wafaka deposits:

- A more complex structural model should be integrated in the next update in order to have a better understanding of mineralization features at a smaller scale.
- The area covered by the pit shell in this study reached areas with lower confidence in the geological model (west flank and lower layer). Diamond drilling should be carried out in the upcoming year in order to improve the geological model.

26.2 GOSSEY DEPOSIT GEOLOGY AND MINERAL RESOURCES

GMSI has the following additional recommendations for the Gossey deposit:

- Additional exploration work should be carried out as the Gossey deposit remains open to the northeast, where some of the best intersections of the 2018 drilling campaign were returned (23 m at 1.18 g/t Au, 26 m at 2.26 g/t Au, and 14 m at 1.17 g/t Au). The most northern drill line is located on the northeast margin of the Gossey Village, and artisanal workings continue further northeast for 1 km. There is scope for significant additional tonnage to be discovered along this highly prospective trend.

- Increase the proportion of DD drilling (currently 14%) within the pit constrained Mineral Resource to increase confidence in the gold grades and provide more bulk density data.
- Periodically review the RC drill sample splitting procedure. During the site visit it was noted that the riffle splitters were often bias towards one side (poorly aligned), therefore affecting the ability to obtain a representative sample. In addition, sample splitting was undertaken outside in windy conditions resulting in a loss of fine material during splitting.
- A budget of $1.0 million is proposed to drill the northeast extensions of the deposit.

26.3 METALLURGICAL TESTING AND MINERAL PROCESSING

- A recent metallurgical study indicated a risk for a lower gold recovery related to the amount of graphitic ore present in future mining zones, according to the LOM. Essakane S.A. has undertaken a mitigation plan that needs to be completed. Additionally, a geometallurgy survey, which is currently ongoing, will help determine where the graphitic ore originates and serves as a basis for better mill feed sequencing in order to optimize CIL plant operating parameters as a function of graphitic carbon concentration in the feed.
- KCA has the following recommendations:
    - Metallurgical testing should be conducted on stockpile material to check if weathering has any effect on recoveries.
    - Test transition material.
    - Testing with reduced cement addition.
- SRK has the following recommendation:
    - For the HLF additional loaded permeability (USBR 5600) interface shear and liner integrity testing of the final overliner material selected should be completed as part of detailed design to ensure satisfactory interface shear strength and drainage of the overliner layer relative to the agglomerated ore.

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28 DATE AND SIGNATURE PAGE

This report titled "Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso" with an effective date of November 6, 2019 and dated January 31, 2020 was prepared and signed by the following authors:

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29 CERTIFICATE OF QUALIFIED PERSON

29.1 VINCENT BLANCHET

I, Vincent Blanchet, ing, as an author of this report entitled “Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso”, with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am a Geological Engineer with IAMGOLD Corporation of 1111, rue St-Charles Ouest - Tour Est, Suite 750, Longueuil, Québec, J4K 5G4.

2. I am a graduate of Université Laval, Québec in 2008 with a Bachelor’s Degree in geological engineering (B.Eng.).

3. I am registered as a Professional Engineer in the Province of Québec (OIQ # 146574). I have worked as a geological engineer for a total of 10 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - I have practiced my profession continuously since 2008 and have been involved mainly in gold mine and gold project, in North America and Australia.
   - I have been working for IAMGOLD Corporation since 2016 as a Geological Engineer.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have been involved at Essakane Gold Mine as a Geological Engineer since July 2016 and regularly visit the site.

6. I am responsible for Sections 1.3.6, 12.1, 14.1, and 23 of the Technical Report. I share responsibility with my co-authors for Sections 3, 25, 26, and 27.

7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101 since I am a full time employee at IAMGOLD Corporation.

8. I have had prior involvement with the property that is the subject of the Technical Report. I am full-time employee of IAMGOLD and I have been involved with Essakane since 2016.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Sections 1.3.6, 12.1, 14.1, and 23, and parts of Sections 3, 25, 26, and 27 of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) "*Vincent Blanchet*"

Vincent Blanchet, ing.

29.2 PHILIPPE CHABOT

I, Philippe Chabot, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso”, with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am a Mine optimization expert with IAMGOLD Corporation of 1111, rue St-Charles Ouest, Tour Est, Suite 750, Longueuil, Québec, J4K 5G4.

2. I am a graduate of Université Laval, Québec in 2004 with a Bachelor’s Degree in Mining Engineering.

3. I am registered as Order of Engineers of Québec in the Province of Québec (OIQ #139359). I have worked as a mining engineer/geologist for a total of 14 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - I have been working for mining companies as an engineer since 2004.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have visited the Essakane mine site on a regular basis since 2015.

6. I am responsible for Sections 1.3.7, 1.3.8, 15, 16, and 21.2.1 of the Technical Report. I share responsibility with my co-authors for Sections 3, 25, 26, and 27.

7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101 since I am a full time employee of IAMGOLD Essakane S.A., Burkina Faso and I own shares of IAMGOLD Corporation.

8. I have had prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Sections 1.3.7, 1.3.8, 15, 16, and 21.2.1 and parts of Sections 3, 25, 26, and 27 of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) “*Philippe Chabot*”

Philippe Chabot, ing.

29.3 STÉPHANE RIVARD

I, Stéphane Rivard, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso”, with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am Director Metallurgy with IAMGOLD Corporation of 111, rue Saint-Charles Ouest, Tour Est, Suite 750, Longueuil, Québec, J4K 5G4.

2. I am a graduate of Université Laval, Québec with a B.Sc.Eng. Degree in Metallurgical and Material Science Engineering in 1994.

3. I am registered as a Professional Engineer in the Province of Quebec (O.I.Q. licence number 118538). I have practiced my profession continuously since my graduation. My relevant experience for the purpose of the Technical Report is:
    - IAMGOLD Corporation, as Director Metallurgy overseeing projects such as Côté Gold project, Boto Gold project, Saramacca Gold project, and Essakane Heap Leach project and also providing site metallurgical governance for Essakane, Rosebel, and Westwood mines
    - Cambior Inc. at Bouchard-Hébert mine as metallurgist
    - Noranda Inc. at Gallen mine as chief metallurgist
    - Ausenco Engineering company Director M&M and Project Managers: Goldcorp Century Gold project, Lundin Gold Fruta Del Norte project, Algold Resources Tijirit project, Kinross Tasiast expansion project
    - Metchem Canada Inc. as Manager metallurgy: Goldcorp Eleonore Project metallurgist

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Essakane mine site multiple times. The last visit being on September 24 to 27, 2019.

6. I am responsible for Sections 1.3.9, 1.3.10, 13.1 to 13.4, 17, 21.2.2, and 21.2.4 of the Technical Report. I share responsibility with my co-authors for Sections 3, 25, 26, and 27.

7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101 since I am a full time IAMGOLD employee.

8. I have had prior involvement with the property that is the subject of the Technical Report since I have been an employee of IAMGOLD Essakane S.A. since August 2014.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Sections 1.3.9, 1.3.10, 13.1 to 13.4, 17, 21.2.2, and 21.2.4 and parts of Sections 3, 25, 26, and 27 of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) "*Stéphane Rivard*"

Stéphane Rivard, ing.

29.4 DENIS ISABEL

I, Denis Isabel, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso”, with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am Director Health Safety and Sustainability with IAMGOLD Essakane S.A. of 146, rue 13.49, quartier Zogona, 09 BP 11 Ouagadougou 09, Burkina Faso.

2. I am a graduate of Université Laval, Québec in 1980 with a B.Sc. in Geology and in 1981 with a B.Cs.A. in Engineering Geology. I am a graduate of Institut National de la Recherche Scientifique, Québec in 1983 with a M.Sc. in Water Sciences and in 1988 with a Ph.D. in Water Sciences.

3. I am registered as a Professional Engineer in the Province of Quebec (Reg.# 36006). I have worked as an environmental engineer for a total of 33 years since my graduation. My relevant experience for the purpose of the Technical Report is that, in my successive positions of Professor at the Engineering Geology department of Université Laval, President of Enviroconseil, Vice-President Mining Environment at SNC-Lavalin and Director Environment and Sustainability at Ausenco, I was involved in many relevant projects:
   - Participation in environmental and social impact assessment studies for mining projects and manager of an environmental and social impact assessment study for an expansion project at a mine site.
   - Participation in Scoping, Prefeasibility and Feasibility studies for mining projects
   - Management of environmental studies related to the environmental permitting of mining infrastructures like tailing management facilities and mine water management facilities
   - Management of mine closure studies.
   - Management of mine closure construction 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 in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have been working at the Essakane mine site since November 2017.

6. I am responsible for Sections 1.3.13, and 20 of the Technical Report. I share responsibility with my co-authors for Sections 3, 25, 26, and 27.

7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had prior involvement with the property that is the subject of the Technical Report. I have been involved with this property since 2016.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Sections 1.3.13, and 20, and parts of Sections 3, 25, 26, and 27 of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) "*Denis Isabel*"

Denis Isabel, ing.

29.5 LUC-BERNARD DENONCOURT

I, Luc-Bernard Denoncourt, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso”, with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am Projects Manager with IAMGOLD Corporation of 1111, rue St-Charles Ouest - Tour Est, Suite 750, Longueuil, Québec, J4K 5G4.

2. I am a graduate of Université Laval, Québec in 2002 in Mining Engineering.

3. I am registered as a Mining Engineer in the Province of Quebec (OIQ #129874). I have worked as a mining engineer and project manager for a total of 18 years since my graduation

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Essakane mine site several times in 2017, 2018, and 2019.

6. I am responsible for Sections 1.1, 1.2, 1.3.11, 1.3.12, 1.3.14, 1.3.15, 2, 3, 18.1, 18.3, 19; 21.1, 21.2.3, 22, and 24 of the Technical Report. I share responsibility with my co-authors for Sections 3, 25, 26, and 27.

7. I have been working for IAMGOLD since 2015 as a Project Director. I am a full time employee of IAMGOLD Corporation, Canada and I own shares of IAMGOLD Corporation.

8. I am not independent of IAMGOLD Corporation as set out in Section 1.5 of National Instrument 43-101 as per NI 43-101 s.8.1(2)(f) and I did receive from my employer participation incentive securities (“options”) and company shares in 2016, 2017, 2018 and 2019.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Sections 1.1, 1.2, 1.3.11, 1.3.12, 1.3.14, 1.3.15, 2, 3, 18.1, 18.3, 19; 21.1, 21.2.3, 22, and 24 and parts of Sections 3, 25, 26, and 27 of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) “*Luc-Bernard Denoncourt*”

Luc-Bernard Denoncourt, ing.

29.6 FRANÇOIS J. SAWADOGO

I, François J. Sawadogo, M.Sc., MAIG, as an author of this report entitled "Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso", with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am Chief Geologist with IAMGOLD Essakane S.A. of 146, rue 13.49, quartier Zogona, 09 BP 11 Ouagadougou 09, Burkina Faso.

2. I am a graduate of University of Ouagadougou, Burkina Faso in 1995 with a Master's degree in Geological Sciences, Fundamental and Applied Geology (M.Sc.).

3. I am registered as Member of the Australian Institute of Geoscientists (MAIG # 6108) and Member of the Australasian Institute of Mining and Metallurgy (MAusIMM # 309642). I have worked as a geologist for a total of 24 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - I have practiced my profession continuously since 1995 and have been involved in many gold mines and gold projects in West, Central, and North Africa.
   - I have been working for IAMGOLD Essakane Gold Mine since September 2014 as Chief Geologist overseeing both the production and near mine geology.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have been involved at Essakane Gold Mine as a Chief Geologist since September 2014.

6. I am responsible for Sections 1.3.1 to 1.3.5, 4 to 11 of the Technical Report. I share responsibility with my co-authors for Sections 23, 25, 26 and 27.

7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101 since I am a full time employee at IAMGOLD Essakane Gold Mine.

8. I have had prior involvement with the property that is the subject of the Technical Report. I am full-time employee of IAMGOLD and I have been involved with Essakane since 2014.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Sections 1.3.1 to 1.3.5, 4 to 11 and parts of Sections 23, 25, 26, and 27 of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) "*François J. Sawadogo*"

François J. Sawadogo, M.Sc., MAIG

29.7 TRAVIS J. MANNING

I, Travis J. Manning, P.E., as an author of this report entitled "Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso", with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am Senior Engineer for Kappes, Cassiday & Associates located at 7950 Security Circle, Reno, Nevada USA 89506.

2. I graduated with a Bachelor of Science degree in Metallurgical Engineering from the University of Nevada in 2002.

3. I am a Registered Member of the Society for Mining, Metallurgy and Exploration (4138289 RM). I am a Professional Engineer in the State of Utah (No. 6880159-2202). I have worked as a Metallurgical Engineer for 15 years.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Essakane mine site on 20 June 2017.

6. I am responsible for Sections 13.5, 13.6, and 13.7 of the Technical Report. I share responsibility with my co-authors for Sections 1, 25, 26, and 27.

7. I am independent of IAMGOLD Essakane S.A. and related companies applying all of the tests in Section 1.5 of National Instrument 43-101.

8. I have had no prior involvement with the Essakane Project.

9. I have read National Instrument 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.

10. As of the effective date of this report, to the best of my knowledge, information and belief, Sections 13.5 to 13.7, and parts of Sections 1, 25, 26, and 27 of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading;

Dated 31st day of January 2020

(Signed & Sealed) "*Travis J. Manning*"

Travis J. Manning, P.E.

29.8 R. BREESE BURNLEY

I, R. Breese Burnley, P.E., as an author of this report entitled "Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso", with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am Principal Engineer of SRK Consulting (U.S.) Inc., 5250 Neil Road, Suite 300, Reno, NV USA 89502.

2. I graduated with a B.Sc. degree in in Geology in 1991 from the University of Nevada, Reno. In addition, I obtained an M.Sc. in Geological Engineering in 1993, also from the University of Nevada, Reno.

3. I am a registered Professional Engineer in the State of Nevada (No. 16225). I have worked as an engineer for a total of 26 years since my graduation from university. My relevant experience includes site investigations, conceptual and detailed design, construction supervision, management and operational assessments, mine reclamation permitting and closure design and permitting at numerous industrial and mining properties throughout the western United States and South and Central America.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Essakane Mine, but rather have been apprised of site conditions by my colleague at SRK, Michel Noel, P.Eng., who has visited the site on a number of occasions and possesses extensive knowledge of site conditions.

6. I am responsible for the preparation of Section 18.2 of the Technical Report. I share responsibility with my co-authors for Sections 1, 3, 25, 26, and 27.

7. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

8. I have not had prior involvement with the property that is the subject of the Technical Report.

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, Section 18.2 and parts of Sections 1, 3, 25, 26, and 27 of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) "*R. Breese Burnley*"

R. Breese Burnley, P.E.

29.9 RÉJEAN SIROIS

I, Réjean Sirois, ing., as an author of this report entitled "Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso", with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am a Geological Engineer acting as Vice President Geology and Resources for G Mining Services Inc. with an office at 7900 Taschereau Blvd, Building D, Suite 200, Brossard, Quebec, Canada, J4X 1C2;

2. I am a graduate of l'Université du Québec à Chicoutimi with a B.Sc. (Geological Engineering) in 1983.

3. I am a Professional Engineer registered with the "Ordre des ingénieurs du Québec" (OIQ-Licence: 38754). I have worked as a geological engineer for a total of 34 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - I have been involved with the Essakane mine since the acquisition of Orezone by IAMGOLD in 2008.
   - I have practiced my profession continuously since 1985 and have extensive experience in estimating Mineral Resources for all kinds of deposits located in South and North America as well as in Southern and West Africa.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Gossey Project from March 27 to March 31, 2018.

6. I am responsible for Sections 12.2 and 14.2 in regard to the estimation of the Mineral Resources of the Gossey gold deposit of the Technical Report. I share responsibility with my co-authors for Sections 1, 25, 26, and 27.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101;

8. I have prior involvement with the property that is the subject of the Technical Report since I was an Employee of IAMGOLD between 1987 and 2012 and the resources and reserves of the Essakane mine and surrounding gold prospects were under my responsibilities as former Manager of Mining Geology between 2008 and 2012.;

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains Sections 12.2 and 14.2 in regard to the estimation of the Mineral Resources of the Gossey gold deposit and parts of Sections 1, 25, 26, and 27 of the Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of January 2020

(Signed & Sealed) "*Réjean Sirois*"

Réjean Sirois, Ing.

29.10 JAMES PURCHASE

I, James Purchase, P.Geo., as an author of this report entitled "Technical Report on the Essakane Gold Mine Carbon-in-Leach and Heap Leach Feasibility Study, Sahel Region, Burkina Faso", with an effective date of November 6, 2019 and dated January 31, 2020, do hereby certify that:

1. I am a Geologist acting as Director Geology and Resources for G Mining Services Inc. with an office at 7900 Taschereau Blvd, Building D, Suite 200, Brossard, Quebec, Canada, J4X 1C2;

2. I am a graduate of University of Liverpool, UK with a B.Sc. (Geology) in 2006;

3. I am a Professional Geologist registered with the "Ordre des Géologues du Québec" (OGQ-Licence: 2082). I have worked as a geologist for a total of 11 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - I have been involved with the Essakane mine and associated deposits since 2017.
   - I have practiced my profession continuously since 2008 and have extensive experience in mineral exploration and Mineral Resource estimation for various commodities in Australia, Canada, and West Africa.
   - I have worked in my current role with G Mining Services Inc. since February 2017.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

5. I visited the Gossey Project from March 27 to March 31, 2018;

6. I am responsible for Sections 12.2 and 14.2 in regard to the estimation of the Mineral Resources of the Gossey gold deposit of the Technical Report. I share responsibility with my co-authors for Sections 1, 25, 26, and 27.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101;

8. I have prior involvement with the property that is the subject of the Technical Report since September 2017, where G Mining Services Inc. completed a preliminary internal Mineral Resource for the purpose of the planning of infill drilling;

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains Sections 12.2 and 14.2 in regard to the estimation of the Mineral Resources of the Gossey gold deposit and parts of Sections 1, 25, 26, and 27 of the Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st day of January 2020

(Signed & Sealed) "*James Purchase*"

James Purchase, P.Geo.